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SME MINING ENGINEERING HANDBOOK

THIRD EDITION

EDITED BY PETER DARLING

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Preface

It was Robert Peele who gathered a group of 46 specialists almost a century ago to write the now classic *Mining Engineers' Handbook*, published in 1918 by John Wiley & Sons, with second and third editions published in 1927 and 1941. When Wiley declined to publish further editions, SME began publication of the *SME Mining Engineering Handbook.* SME published the first edition in 1973 and a second edition in 1992.

From that initial publication in 1918, the handbook has been acknowledged as the repository of all essential information and useful, practical detail relating to virtually every aspect of mining.

From the beginning of my mining career, I have relied heavily on this book as a source of inspiration and information. With the honor of being asked to take on the somewhat challenging role of managing editor for this third edition, I owe a debt of professional gratitude to several generations of managing editors, authors, and technical reviewers who set the seemingly impossible high standards in the previous editions of the handbook.

From the onset, several objectives were established. These objectives included (1) to produce a book that would stand shoulder to shoulder as an equal alongside previous editions, (2) to maintain its tradition of being the "handbook of choice" for every practicing mining engineer, (3) to be practical rather than theoretical in its content and approach, and (4) to be international in its appeal and examples.

Mining is an international business, and the importance of sharing experiences, knowledge, and examples from around the world cannot be overemphasized. A water "problem" in one part of the world may be considered "normal" in another. A "deep" mine on one continent may connote "very deep" on another. Tailings dams in some countries need to be earthquake proof, whereas in other countries they must be free draining.

This edition attempts to take the best of the best from around the world and package it in a standard and logical format for the benefit of the global industry. This internationalism is shown by the subjects covered in each chapter as well as by the nationalities of the world-class authors and technical reviewers represented. It is noteworthy that most significant mining operations have graciously shared their knowledge, techniques, experience, and alternative viewpoints.

The handbook has moved with the times to cover the issues that are exercising the industry, the innovations that are exciting, and how the industry is dealing with changing attitudes toward a number of its constituents such as energy (both electrical generation and carbon/petroleum based), water management, resource maintenance, and the whole subject of sustainable development. One of the significant areas in which this handbook differs from previous editions is that it includes several chapters on both the social and environmental issues often associated with mining, and, more importantly, how these issues and their impacts can be mitigated and managed.

It is important to note that this is a handbook and not an encyclopedia of everything mining. Several subjects previously included as dedicated chapters are now covered in less detail within other chapters. This move was not designed to marginalize or dilute the importance of certain subjects but was done to acknowledge the significant shifts in the way mining has changed as a result of new technologies. It also reflects how the work and the responsibilities of today's mining engineer have changed and developed.

Attempts to encapsulate the essence of so diverse a discipline as mining engineering could not have been possible without the unselfish contributions of the hundreds of authors, coauthors, technical reviewers, and mentors who are very much the unsung heroes of this publication. Much gratitude and thanks are due to the many talented and world-class professionals who have given so freely, patiently, and enthusiastically of their time, hard-earned experiences, and masterful knowledge on a plethora of mining and related subjects to ensure that this handbook was produced on time and to the meteoric standards that the industry has come to expect.

One of the many delights of managing this project has been the acquaintance (usually electronically) of so many helpful, patient, enthusiastic, and friendly people from within the various mining associations, academic establishments, mining companies, and consultancies, as well as retired engineers and specialist editors—everyone an expert in their field. Without their cooperation, steady guidance, constructive comment, and encouragement, the managing of this edition would have been an impossible task. I cannot name them all, and to mention some but not others would be ungracious, but tremendous appreciation is due.

These experts are headed by the authors themselves, almost every one of whom is a world leader in their specialist field. Often with the briefest of guidance and without any form of remuneration, they have passionately undertaken their writing tasks. Their enthusiasm, commitment, and professionalism formed the bedrock on which the handbook has been based, and, as a result, they have produced superb and exceptional texts. I salute and thank you all.

Next, I acknowledge the many technical reviewers who, often at short notice and, as this project progressed, with an ever shortening lead time, have been called on to read through many drafts before articulating and listing their comments, suggestions, and observations. It typically was not an easy task and often may have been a marathon requiring skills, diplomacy, and knowledge that went well beyond their original brief. I apologize and thank you.

Thanks are extended to SME's book publishing team who have been thoroughly professional and efficient in their handling of this edition. This team has checked every comma, word, phrase, sentence, and illustration. They have ensured that copyright rules have not been flaunted in the quest for expedience, checked and rechecked references, standardized units of measure, and performed a hundred and one other tasks to ensure that this finished product is a source of information in a format that at best pleases and at worst does not annoy. Their attention to detail in this exacting task is very much appreciated.

I also thank SME for affording me this opportunity to repay some of the debt that I believe is owed to an industry that has kept me fed, watered, enthralled, and enthused for more than three decades.

Finally, I trust that any errors in opinion, facts, or perceptions in the handbook are few in number and will not overtly detract from the usefulness of and, I dare say, enjoyment of this third edition of the *SME Mining Engineering Handbook*.

Peter Darling
February 2011

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PART 1

Mining: Setting the Scene

CHAPTER 1.1

Mining: Ancient, Modern, and Beyond

Peter Darling

The extraction and processing of minerals is an essential part of the way the world and its various civilizations function and interact. To forget the contribution that mining has made (and continues to make) is to take for granted the significant progress that civilization has made since the last Ice Age. It is also to ignore the fact that the very structure upon which we depend is built on—and with—the products of mining.

ANCIENT MINING

The extraction, smelting, and refining of metals date from the onset of the Bronze Age, if not before. Unfortunately, no manuscripts concerning mining from this early period survive, and we are left with only fragments discussing mining in the works of the classical Greek and Roman authors. For example, Pliny the Elder (1984), in his *Naturalis Historiae* (Natural History), discusses the mining of gold, silver, mercury, and lead and mentions the use of pillars as well as fire-setting and quenching with vinegar in order to break rock. Strabo (1988) mentions the mining of tin in the British Isles as well as the copper mines of Rio Tinto in Spain and the silver mines of Laurium in Greece.

Indeed, it is thought that tin has been mined continuously in Cornwall, Great Britain, from ancient times until the closing in 1998 of the South Crofty mine, which incorporated the long-lived and famous Dolcoath, Carn Brea, and Cook's Kitchen *setts*. Other mines dating from the Middle Ages or earlier include the Falun copper mine in Sweden, which was mined from the 11th century until 1992; the Wieliczka salt mine in Poland, which was mined from the 13th century until 2007; and the Rio Tinto mine in Huelva, Spain, which has been mined periodically since Roman times.

POST-RENAISSANCE MINING

Modern mining is considered by many to date from 1556 when Georgius Agricola (Georg Bauer), the city doctor in Chemnitz, Saxony (Germany), published his famous monograph, *De Re Metallica* (Agricola 1950), which documented contemporary mining practices in the Erzgebirge ("ore mountains") mines in Saxony. Other mining-related works by Agricola include *Bermannus*, a dialogue by a learned miner; *De Ortu et Causís Subterraneorum*, a work on physical geology; and *De Natura Fossilium*, the first systematic mineralogy. Among the equipment documented by woodcuts in *De Re Metallica* are hand- and horse-operated rope hoists, piston dewatering pumps powered by large water wheels, and ventilation systems consisting of pipes fed by large air bellows. Rock breakage was by fire-setting, with gads and mauls hammered into cracks to expand them.

Mining as described by Agricola changed little until the introduction of gunpowder about 1670 and Thomas Savery's fire engine in 1698. Savery's engine, however, could only lift water about 15 m (50 ft) (Barton 1969), and it remained for Thomas Newcomen in 1712 to develop a pumping engine, consisting of a cylinder and piston connected via an overhead beam to a pump rod. James Watt's patent in 1765 consisted of two improvements to Newcomen's design (the use of a separate condenser and utilizing steam at slightly above atmospheric pressure) that resulted in the Cornish pump. The ability to pump out water extended significantly the depths to which mines could be exploited.

The introduction of gunpowder made it easier to break rock, but the process remained slow and dangerous. Powder was poured by hand into the blastholes and could be ignited by a spark. The inventions of the safety fuse by William Bickford in 1831 and dynamite by Alfred Nobel in 1867 did much to improve the safety of rock breakage. The introduction of steam- and compressed-air-powered drills in the 1860s increased productivity considerably. Refinements in drill steels and bits and the development of mechanized loading and haulage equipment bring us to the present.

ECONOMIC DEVELOPMENT AS A RESULT OF MINING

The quest for minerals has without doubt brought wealth and civilization and accelerated development to many parts of the world. Mature industries need resources for sustainability, and, once they have been depleted in one location, more must be found (usually elsewhere). Although one may question the motives and methods of those involved in early mining operations, their tenacity and determination should never

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be underestimated. For once the initial enthusiasm of being a pioneer miner had given way to the dull but enduring glow of reality; progress followed quickly in the footsteps of these first buccaneers.

The U.S. gold rushes in California and later Colorado did as much for the opening of the American West as the railroads or agriculture did. Moreover, towns such as Deadwood, South Dakota, and Tombstone, Arizona, which were made famous by dime-store novels and later the movies, were actually mining towns. West and southern Africa were opened up by the slave trade, later by Christian missionaries, and then farmer-landowners. However, it is the mining industry and agriculture that have sustained these regions ever since. Indeed, such is the nature of the ultra-deep mines in South Africa that even today this country's mining engineers are considered the real pioneers when it comes to solving the problems associated with deep-level mining.

Latin America, often considered a relative latecomer to the pursuit of mineral wealth, has nevertheless contributed significantly to the world's riches. Indeed, the gold and silver of the Incas was the basis of Spain's preeminence in the 16th century, and the gold mines in Morro Velho, Brazil, and copper–silver mines of Cerro de Pasco, Peru, have been mined since the 18th century. Indeed, immediately after World War I, Argentina was the third richest country in the world (after the United States and Great Britain) based in part upon, as its name suggests, silver mining.

Australia, a vast continent in its own right, is not without reason called the "Lucky Country," due in part to its vast mineral wealth. Mining was as responsible for the opening of much of Australia's vast interior as were the sheep and cattle stations. Copper mining began in the 1840s in the South Australian towns of Moonta, Kadina, and Wallaroo. The 1850s saw gold rushes in Ballarat and Bendigo, both in Victoria. Flotation was adopted early at Broken Hill, New South Wales; and Mount Isa in Queensland has been responsible for many innovations in underground mining. Even today, barely two centuries since the initial settlement at Botany Bay, New South Wales, in 1770, mining still contributes much to the nation's fortunes, as well as to the economic and industrial expansion of so many of its Asian neighbors.

MODERN TRENDS IN MINING

Although current trends in mining have been covered in considerable detail in a subsequent chapter, several aspects of this topic deserve further examination here.

Ore-Body Characteristics

The industry has already accepted the fact that the majority of the world's future minerals will come from low-grade, super-large, high-tonnage, and ultra-mechanized operation. This trend already started with the move toward high-tonnage longwall operations for coal, huge open pits for copper (and sometimes gold), and the mining of ore that 50 years ago would be considered interesting but uneconomical to extract and therefore virtually unexploitable. Smaller operations will still have a place, but the trend toward massive block caving operations, high tonnage, and automated mining will only increase in tempo.

Indeed, with the quickly disappearing likelihood of discovering a new and easily mined high-grade ore body, the time may soon come to view in-situ leaching of certain low-grade and high-tonnage mineral deposits as attractive. Although host materials must be relatively permeable and the mineral readily extractable by acid or alkaline leaching for this method of extraction to be practical, nevertheless it may be attractive to mine some mineral deposits of low grade and relatively large tonnage by this method, where traditional methods would prove uneconomical. Although it is evident that not every mineral can be extracted in this way, in many instances such deposits will respond to economic in-situ leaching.

Social and Environmental Factors

Unfortunately, it may appear to some that gone forever is the notion that mining is an honorable adventure, often played out in inhospitable and distant regions, that aims to produce the very building blocks upon which world civilization advances itself. Today, mining is often portrayed as a beast that destroys everything in its path and creates wealth for the few and ongoing misery for the many. This portrayal is unfair, although mining companies often do little to refute it and then reinforce it through their opposition to environmental regulation.

Critics accuse the mining industry of having ignored environmental and social issues. Such criticism ignores the fact that technologies for addressing environmental problems such as acid drainage did not exist a generation ago. Moreover, factories, chemical and food processing plants, and thermal power plants, which are generally closer to population centers than are mines, sometimes produce more pollution than do mines. Nevertheless, we no longer live in an age where mining can take place in splendid isolation. Mining engineers should take heart from the fact that *engagement* rather than *confrontation* is the watchword when it comes to the future. Although it may be too much to expect the world at large to respect mining engineering as a profession, its citizens may in time acknowledge the essential role that mining performs and the constraints under which it operates.

While the majority of the industry already believes that mining must develop and operate in harmony with the environment, the real challenge is to convince the social structure, government agencies, and nongovernmental organizations that mining can (and will) be done in a safe and environmentally compatible manner. Mining's acceptance of its responsibility relating to environmental and social issues is now engrained into virtually every aspect of the industry, in much the same way as safety has become. This reeducation has paid dividends in terms of reversing the image of the mining industry, which had acquired the significant negative press irrespective of what it did. A great deal of credit needs to go to those executives of senior mining companies who grouped together several years ago to foster the Global Mining Initiative. This was certainly an idea whose time had come, but to have grasped the nettle and financed its launch and (more importantly) its implementation took a great deal of foresight and courage at a time when it seemed that nothing the industry tried could stem the tidal wave of negative opinion.

The majority of mining companies have made massive advances to address both environmental and social issues. Nevertheless, the extraction of natural resources now attracts the sort of responsibility and scrutiny that few other global industries are subjected to. The mining industry has evolved (and continues to evolve), although it is perhaps too much to expect those outside the industry to wholeheartedly acknowledge what the industry does in difficult circumstances.

Still, outside the realm of the legal/governmental license to operate, mining has nevertheless accepted the general

obligation of a social license to operate with the community at large in the areas in which it operates. Such informal contracts are being increasingly accepted as a must-have commodity whose goodwill value cannot be underestimated.

Economic Uncertainty

The first quarter of 2008 saw a series of global economic disasters that combined to create a financial tsunami that affected virtually every country and every industry, irrespective of their importance and how well they were being run. As this crisis began to unfold, there was a short-term belief that mining, as an essential industry, would escape the majority of the fallout. However, a contraction in manufacturing industries inevitably results in reduced demand for raw materials such as metals. Thus, it was inevitable that the mining industry would also follow a downward financial trajectory of its own.

Despite the fact that this brutal and surprisingly swift lurch toward the economic abyss was totally unexpected, the mining industry has fortunately survived and appears to have returned to its precrisis level. Nevertheless, there have inevitably been some considerable impacts. Confidence in this primary industry has been shaken with major layoffs implemented at some operations, others put on care-and-maintenance, and planned expansions at others postponed or cancelled altogether. Although confidence has suffered a temporary battering, the underlying economics of mining remain sound.

The anticipated amalgamation of several of the larger mining companies came perilously close to becoming a reality. In several high-profile instances, shareholders who saw no need for any new conglomerates were skeptical and uncharacteristically vocal. Customers saw such mergers as a dangerous trend that would weaken their bargaining position, as it gave too much leverage to an even fewer number of raw material providers. Financial institutions and analysts saw several mergers as a logical rationalization, and when none materialized appeared to be content that these would happen once ready money was again available to finance such acquisitions.

Personnel Assets

Virtually every industry claims that its personnel are its major asset, and the mining industry is no exception. As recently as a generation ago, mining offered a career that combined a kaleidoscope of trades (explorer, adventurer, pioneer, and engineer) while also providing a potential for overseas travel and long-term career prospects in a profession that was both respected and essential. However, a number of factors have combined to relegate mining to a career of lesser option for many who, although voicing a preference for something "less ordinary" than a nine-to-five, bureaucratic-bound existence, appear to have no desire for the type of lifestyle that mining has to offer.

The industry is, and always has been, keen to ensure that its engineers are properly taught and prepared, and has quite rightly believed that traditional university education and on-the-job training provides the best preparation for a safe, efficient, and productive working life in mining. However, the number of educational establishments providing mining engineering as a formal undergraduate-level course has diminished considerably since about 1990. Indeed, even the world-renowned Royal School of Mines in London and the highly regarded Michigan Technological University in Houghton, Michigan (United States) no longer offer mining engineering as an undergraduate subject. In many other mining institutes, the number of graduates has dropped alarmingly.

The permanent loss of knowledge as experienced artisans leave the mining industry before their skills can be passed on to others is becoming a concern in many parts of the world. The lack of skilled mining personnel has begun to emerge, at various mining conferences and similar high-level meetings, as a major concern for the industry. Although the current poor economic situation has seen workers laid off and operations, projects, and expansions mothballed, the overall trend is for fewer professionals in positions of responsibility, as well as a more casual and migratory workforce. The impact upon safety and productivity is not to be dismissed so lightly.

The significance of this issue is not in itself the need to keep up numbers but it is (1) the loss of knowledge before it can be passed on, (2) the amount of time and effort required to bring new employees up to speed so they can perform their various tasks efficiently, and (3) more alarmingly, the high number of injuries and accidents associated with new hirings. The lack of replacement miners, technicians, artisans, and qualified engineers entering the industry in sufficient numbers must be addressed before it causes the closure of marginal ongoing operations, the shelving of expansion plans, and the abandonment of seemingly viable prospects. However, it is the permanent loss of knowledge that should cause trepidation for the industry's leaders.

Fortunately, the news is not all bad: The industry is trying (and succeeding) to drastically reduce the number of personnel actually involved in ore extraction (both underground and in pit). In addition, the industry is moving (or rather being forced to move) toward exploiting mines that are larger in terms of daily tonnages with automation playing a significant part in what may rather casually be described as simply a "rock moving operation." Unfortunately, we are many years away from the totally automated, unmanned mine.

Automation

Advances in automation and robotics, nevertheless, have the potential to significantly change the way mining is conducted in many current and future operations. The major advantages of, and drivers for, automation are in hazardous, repetitive, and/or labor-intensive tasks. Although automation has not yet significantly impacted global mining and processing, it has, however, started to show its value in terms of improved safety to the point that the industry has a real sense that it will be a significant factor in the future.

Automation is particularly suitable for a number of key operations: production drilling, the real-time recognition of material being drilled, automated loading, accurate and automatic movement of equipment (be it drill rigs, loading shovels, or haul trucks), monitoring of moving parts for maintenance purposes, and diagnostic intervention. The emerging view is that automation can (and will) lead to immediate improvements in productivity. Dragline operations are in the vanguard for research aimed at remote and fully automated working. By limiting operational variations, a consistent input will ensure maximization of shovel loads, for example. In addition, improved precision means that mining tasks are consistently performed at their required location, exact rate, planned orientation, and precise time. Finally, because automation enables production consistency (especially in consistent and regular ore bodies), it means that the mine design does not need to be updated on an ongoing basis because of production variations.

With the advances being made in certain aspects of technology (e.g., wall profile recognition), the time has come when repetitive tasks, such as multiple vehicle haulings from large underground operations, are a reality and are being run from a control cabin at the surface. This facility not only removes an operator from underground but means that an 8-hour shift means 8 hours at the controls. In addition, operator fatigue is negligible. However, it is not all positive news. Side-impact incidents, which are a significant issue for load-haul-dump traveling at speed in narrow tunnels, have still not been eliminated, while productivity per hour is still lower than for manned machines. Obviously, more work still needs to be done.

Tar Sands

The world seems to have a love–hate relationship with oil: It loves the freedom that petroleum provides (i.e., instant power in a barrel) but dislikes the environmental issues associated with its extraction, refining, and burning. Tar sands, which are mixtures of sand and bitumen, are found in about 70 countries. They are distinct from oil shales, which are sedimentary rocks containing kerogen and kerabitumen, a mixture of organic compounds (from which the extraction and conversion of oil is much more expensive than tar sands).

Some tar sand deposits are accessed by drilling, after which superheated steam is used to melt and migrate the tar to a collecting chamber from where it is pumped to the surface. However, most of the *economically* viable tar sand deposits are accessed from the surface and mined conventionally. By far the largest surface tar sand deposits are found in Alberta and Saskatchewan in western Canada and contain an estimated 1.7 trillion barrels of oil. Of the Canadian oil contained in tar sand, approximately 173 billion barrels are considered economically extractable using current surface or subsurface technology, with 20% of this volume being recoverable by surface mining.

Although the cost of extracting oil from surface tar sands is high, the process does not contain any of the high risks associated with traditional oil drilling (i.e., dry holes, blowouts, hydrogen sulfide [or sour gas], explosions, and high equipment rental costs). The technology and equipment originally employed to mine these surface tar sands were borrowed from the brown coal surface mines of Germany. However, bucket-wheel excavators have by and large given way to traditional truck-and-shovel operations.

One of the major issues associated with tar sand extraction is that the oil content tends to be low (of the order of 10% to 12% by volume) and that one of the by-products of tar sand production is usually vast quantities of coarse sand. Indeed, the world's largest dam is a tailings dam associated with tar sand production in Alberta, Canada. In addition, the rock formations surrounding the tar sands tend to be heavily water bearing and require depressurization. Water-saturated organic deposits of peat material, known as *muskeg*, are also present in bogs and poorly draining creek systems. Thus water management, which also includes treatment and storage, pose significant mining engineering challenges, although the biggest issue is the operational carbon footprint, which is much bigger than traditional petroleum exploration and production.

Of course, the subsurface exploitation of tar sands is also of interest to the mining industry, although the technology employed is more akin to the methods of directional drilling and hydrofracturing, which are the mainstay of today's petroleum exploration and production industry. Solution mining is an established extraction technique and is as applicable to the subsurface extraction of tar as it is for sulfur or uranium.

This is a relatively new industry, and it will be interesting to see how and in what direction it matures. However, while the quantities of oil involved are huge, the cost of extraction will always be an issue. The fact that oil can fluctuate by as much as US$10 a barrel in a single day does not provide the degree of stability that this important (and emerging) mining sector deserves.

Alternative Uses of Coal

The 1992 edition of this handbook covered the subject of in-situ burning of coal (Chaiken and Martin 1992), and although this method holds many prospects for the industry, perhaps its greatest potential is to extract power from narrow, deep seams. However, with the near-universal requirement for clean coal, the opportunity to burn any type of coal irrespective of its rank, heating value, and ash or sulfur content has passed. Nevertheless, another potential means of extracting energy from coal has emerged: coal-bed methane production. In the near future, the pre-mining extraction of methane from coal beds will become common practice, especially from seams which are small, narrow, deep, or near populated areas. The energy potential for coal-bed methane production is enormous.

The method of drilling into and the extraction of methane gas from coal beds use well-established oilfield directional drilling, reservoir hydrofracturing, and production technology. As this method of pre-mining gas capture becomes more common, its greatest usage has and will continue to be on coal seams deemed too small, too narrow, too deep, or too close to centers of population to warrant traditional coal extraction.

BEYOND THE HORIZON

Chapter 1.3 looks at the anticipated direction for the industry in the next decade or so. However, what will this industry look like at the start of the next century? Although no one knows for certain, many subtle indicators highlight what mining in the 22nd century will encapsulate and the direction it will take.

Climatic Variation

Over time, the hysteria associated with *global warming* will have matured to the universally accepted term *climatic change*; however, the debate will continue as to the level of anthropogenic contribution. Nevertheless, in the meantime, a common-sense approach to resource husbandry and the advantages that can go hand in hand with such a widely encompassing strategy will have altered the mining/minerals extraction and processing industry forever.

Nowadays, manufacturing businesses, industrial concerns, and even corporate headquarters are being encouraged to reduce their carbon footprint. Mining, processing, and refining organizations are no exception in being pressured to reduce their environmental impact. In the future, the need to further reduce their impact will have become a way of life, utilizing considerable imagination and thinking outside the box. Mining companies hitherto concerned with providing coal for power generation will now be known as *power providers*, and their businesses will consist of supplying methane from in-situ coal bed production and generating power from nontraditional sources such as geothermal, solar, wind, and wave/tidal hydropower sources.

Electricity Generation

In the 1970s and 1980s, it was not uncommon in Britain or Continental Europe to see a Citroën 2CV car being driven in the slow lane of a motorway, with flowers and rainbows painted on its side panels. There was often an "Atom Kraft—nein danke" (liberally translated as "atomic power, no thank you") sticker displayed prominently. Many people from this era developed a no-compromise view that nuclear power was an abomination. Admittedly, they were only one generation removed from associating the power of the atom with the devastation of Hiroshima and Nagasaki, while the disaster at Chernobyl was too real to be dismissed as a once-in-a-million-year occurrence. Nevertheless, to put matters into context, the reactors at Chernobyl were of an antiquated design without containment and were poorly constructed.

In the last few years, a new generation earnestly searching for alternatives to power produced by the burning of coal or natural gas is looking at nuclear power generation anew. Unfortunately, there is still some lack of public confidence in an industry that has been shown to have safety issues. A second factor relates to waste management, which poses a different set of safety issues, but which has an engineering solution, although a different one than building safe atomic power stations.

The generation of power at an atomic plant (under normal circumstances) leads to little pollution but does result in the creation of three levels of waste:

1. **Low-level waste:** contaminated clothing, tools, paper, filters, and rags, which are known as contact waste and contain limited amounts of low-level radioactivity.
2. **Intermediate-level waste:** fuel casings and reactor components, graphite from reactor cores, and sludges from radioactive liquid effluents.
3. **High-level waste:** spent reactor fuel and transuranic residues from reprocessing spent fuel into weapons.

Although the high-level waste is automatically destined for recycling, thereby providing yet more energy, it is the intermediate- and low-level wastes that are proving to be a burden. Both are currently being stored in vast secure temporary facilities in anticipation that engineering will finally resolve the issue of how to recycle (at least partly) this growing mountain of radioactive waste. The alternative is to produce an internationally acceptable means of reducing this waste volume into a manageable size, bind it up with a medium that is inert and stable, and then devise a means of storing this in an environment that is both secure and yet allows the material to be easily retrieved. In comparison, building safe reactors is a relatively simple question of belt-and-braces design and construction, incorporating fail-safe systems, strict controls, and a strict adherence to operating procedure. On the other hand, the storing and recycling of intermediate- and low-level waste will require the combination of numerous engineering disciplines, skills, and thus far unheard of innovations. We are not so far away from wrestling this waste problem into a manageable issue, and from that point on this major hurdle to the long term and (more importantly) the sustainable implementation of nuclear power will finally have received the well-deserved fillip it needs.

Except in areas of abundant water or high relief, where hydropower is a possibility, nuclear power is probably the best the world can expect in terms of clean, effective, predictable, controllable, nonseasonal, high-volume, and efficient power generation. It would be unfortunate to squander this gift for any longer than necessary by continuing to look to the burning of coal and gas as the first, and often the only choice, in terms of power generation.

Conspicuous Power Consumption

Regardless of the way that power is generated, it will remain a strategic issue and in the future will govern the way the industry operates in certain regions. In 1980, it was estimated that in certain regions almost 40% of the energy consumption was directed toward mining, crushing, grinding, and milling. Although things have changed considerably since then, it is a fact that many of the larger mills still operating were installed decades ago. Though these mills remain (within engineering limits) as efficient as they ever were, the question has been posed of whether there is a better way to extract the "goodness" out of the ore.

As the majority of ores now being extracted will be low grade, deep, and chemically challenging, the in-situ leaching of massive blocks will become commonplace. Such operations will now fall into four categories: already fractured, self-fracturing, easily fractured, or those that require help in order to create the fracture-rich environment that will enable leaching to occur.

While today only eight minerals have been commercially mined by in-situ or solution mining (sulfur, salt, trona, nahcolite, potash, boron, uranium, and copper), by the progressive technology of bioleaching, other minerals may become commercially available in the future. Nevertheless, the practice of bringing ore to the surface to enable processing to take place will not disappear altogether. Such practices will now be confined to long-established operations where the infrastructure and processing plant have long since paid for themselves, operations that are high grade and low volume in nature, or operations where the geochemistry of the minerals and host rocks is such that leaching has failed to establish an effective means of producing pregnant liquors in sufficient quantity or at a rate that justifies embracing a new modus operandi. However, the potential environmental impacts on shared groundwater resources are likely to affect the places where in-situ leaching can be used.

Value of Water

In much of the world, fresh water is a scarce commodity. Because of the quantities of water required for drilling, crushing, grinding, and aqueous processing methods, the availability of sufficient water is critical to mining. Even in many nondesert parts of the world, water has become a matter of life and death. Indeed, Israel, Cyprus, and the Yemen are among the countries that, for the first time, have acknowledged fresh water as being a strategic commodity and have elevated its capture, "creation," and storage to a level akin to other national security items. In many other parts of the world, water is a serious issue but has yet to be elevated to a strategic commodity.

In the future, however, the use of water in mining and processing will become a real issue. The salvaging of water from processing will become, after an initial period of rapid growth and innovative thinking, an established part of everyday mining operations. Water, often a never-ending problem in underground operations, could well be the starting point for a symbiotic relationship with nonmining surface activities such as farming. Elsewhere, finer initial rock breakage and dry milling/grinding similar to that employed by the aggregate and

cement industries will become normal practice, and wet processing will be confined to fewer operations.

Nevertheless, water may also become one of the ways by which mining can be seen as a net contributor to the community in which it operates. It is a dichotomy that many mining operations (whether they operate from the surface or from underground) still pump millions of gallons of water out and away from their operations every month. Therefore, the dewatering of workings to allow development and mineral extraction will be seen as a positive by-product of mining and indeed a positive legacy in those arid and semiarid parts of the world where agriculture is an ongoing struggle against drought.

Materials' Stewardship

The Doe Run lead operations in Missouri (United States) have been in production in one form or another since the U.S. Civil War. One of the interesting aspects of the current operation is that the company has been actively practicing the notion of *commodity stewardship* for more than 30 years. The company took the long-term view that their responsibility to the United States was to see that the nation's needs for metallic lead were catered for in every sense of the word. We all understand the notion of mining, refining, and marketing raw metallic lead, but the company also assumed the mantle of collecting and recycling lead once the components in which it had been employed had reached their end of life.

Doe Run set up a nationwide collection network at which used car batteries (the major and most readily available source of secondhand lead in the United States) could be deposited free of charge. These batteries were then drained of their acid, which was sold back to the industry; the plastic battery housing was likewise recycled; and the lead metal was returned to the Doe Run processing plant where it was recycled to produce new lead. The logic behind this strategy is that the company knows about lead processing, it already owns the necessary plant and equipment to recycle the metal, and as such it is an integral part of the whole lead market.

This concept of stewardship has been used to varying degrees within the nuclear industry, which recycles its own products. It will only be a question of time before other metallic industries take on (or are forced to practice) the notion of commodity stewardship, which is the logical next step forward in recycling.

Ethics and Good Neighbors

In the future, mining organizations, and especially the larger ones, will see themselves as being resource and service providers. Whether this involves power, water, forestry goods, or the mass recycling of metals and glass, they will view their business in a holistic way. What shareholders will make of this arrangement has yet to be tested, but in recent years we have seen a shift away from shareholder positions based purely on financial returns. For example, hedge funds that invest in enterprises based on a particular ethical position (be it green, nonpolluting, good neighbor, anti-apartheid, or nonmilitary) have been available for many years.

The next logical step for mining companies is to ensure that in addition to their core business of finding, mining, and refining minerals, they also engage in other associated activities. A mining company could, therefore, find itself farming large tracts of land that it had reclaimed after strip mining had passed through the area, running offshore wind farms, or providing hydroelectric power and agricultural irrigation services.

The fact remains that today's society cannot exist without mines and the products of mining. It is no good simply saying that we will allow others to run roughshod over environmental and social issues as long as these issues do not impact us. Alternatively, by simply outlawing the consumption of metals and minerals, a resource-scarce global environment would be created that would relegate large areas of the world and its peoples to zero advancement, while ensuring that developed countries remained on the plus side of the inequality equation.

Beyond the Pale

Many would consider that future mining trends must encompass an appreciation and inclusion of two further untapped regions that are ripe for exploration and exploitation: marine mining and mining in space. However, neither of these frontiers holds much in the way of either economical or realistic possibility within the next hundred years.

The offshore production of sulfur from the Gulf of Mexico using the Frasch process is one of the few marine mining success stories. Marine mining (with the exception of creating large, near-shore coffer dams for the exploitation of diamonds in southwestern Africa, and the offshore extraction of sulfur) does not hold anything like the attraction that surface mining does. Although manganese nodules and other sea-floor mineral deposits contain huge tonnages of metals, they are unlikely to be mined for the foreseeable future for political reasons. Though the technology for mining the nodules was demonstrated in the late 1960s by the *Glomar Challenger* and others, as the deposits lie in the depths of the ocean beyond the territorial limit of any coastal nation, international agreement is required before development can occur. The failure of the 1972 Law of the Sea conference and many subsequent discussions have aptly demonstrated the unlikelihood that seafloor mining will occur anytime soon. Indeed, the only potential for a further advance in marine mining is in the winning of sand and gravel for construction purposes.

Any potential for mining in space is bound to be hampered by four fundamental issues:

1. Excessively high costs of transportation
2. Delivery and maintenance of mining equipment
3. Allowable processing methods and their requirements
4. Ownership of a planet or star and the willingness (or otherwise) of the owner to allow mining

The only reason to mine in space would be to acquire metals and minerals not available on earth, but would the excessive cost for such a high-risk project (in what would be, after all, a scientific adventure) ever be justified? There is no doubt that the major advantage that space can offer mining is as a spinoff in technological improvements, in much the same way that space and military technology and materials have already provided for a plethora of other high-level industries. In the longer term, space mining *may* provide raw materials for use in space itself (water, hydrogen, metals, etc.) to overcome the "gravity well" of Earth.

CONCLUSION

Although some may believe that mining is a sunset industry that is beginning a slow decline into oblivion, this is simply not the case. There is no doubt that the industry is a mature one,

and that, like many industries, it has had to adapt to changes in technology and perceptions. Unfortunately, it has not always been as quick to respond to public opinion as perhaps it should have. Nevertheless, mining, particularly that conducted by the larger companies, has moved with the times, and while no one is pretending that things have not always been done well in the past, no one is claiming any reason to be complacent now.

Of course, any discussion that attempts to focus on mining and mining companies under the working title of "past, present, and future" must acknowledge that mining companies are not equal in terms of their means and manner of operation. The markets they cater for are numerous, the regions where they operate are various and varied, and their attitudes toward their own personnel and those with whom they work can differ significantly.

It would be pointless to attempt to paint every mining company, minerals magnate, or corporation executive as perfect. Mining is a tough profession and frequently attracts those of similar mettle. Neither does this engineering discipline wish to lower the bar of acceptability concerning what mining companies can and should be able to do to a point that encourages a free-for-all vis-à-vis standards. There are many mining companies, most of which are large, international, and multi-commodity in character, that strive daily to raise the bar of acceptability, and it is to them that the industry often looks for continued inspiration and leadership.

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CHAPTER 1.2

Current Trends in Mining

Marcus Randolph

SUPPLY AND DEMAND

Two words dominate current discussions about the supply and demand of minerals: *globalization* and *supercycle*. Globalization's application is much wider than mining, although a shrinking world has profoundly impacted the mining industry. Supercycle has recently come into use because rapidly industrializing, less-developed countries have stimulated minerals demand. Fifty years ago the mining industry was dominated by local companies supplying single products to local customers. High transportation costs, poor communications technology, and a lack of companies that had the capability to invest outside their national boundaries led to an industry dominated by small producers operating on either a local or national scale.

Globalization changed all this. Easy telephone calls, long-haul planes, and huge ships meant that mining companies could develop mines and sell their output globally. At the same time, the technology used in production became more efficient—generally by getting bigger. Plants and earth-moving equipment became larger and underground longwalls got longer. Increasingly, the low-cost source of supply was huge, high-quality deposits, the product of which was shipped all over the world. The result was the cost of supply went down and mineral prices declined.

This trend was enhanced by the post–World War II reconstruction of Europe and subsequent growth boom plus the rise of major Asian economies with little or no indigenous metals resources. It also fueled the segregation of basic raw materials supply and processing, as raw materials became sourced where the best resources were located and processing occurred in countries where materials were in high demand.

Figure 1.2-1 tracks the price of copper for the period 1800 to 2008 in U.S. dollars per pound, in real 2008 dollars. In this graph, the effect of inflation has been removed.

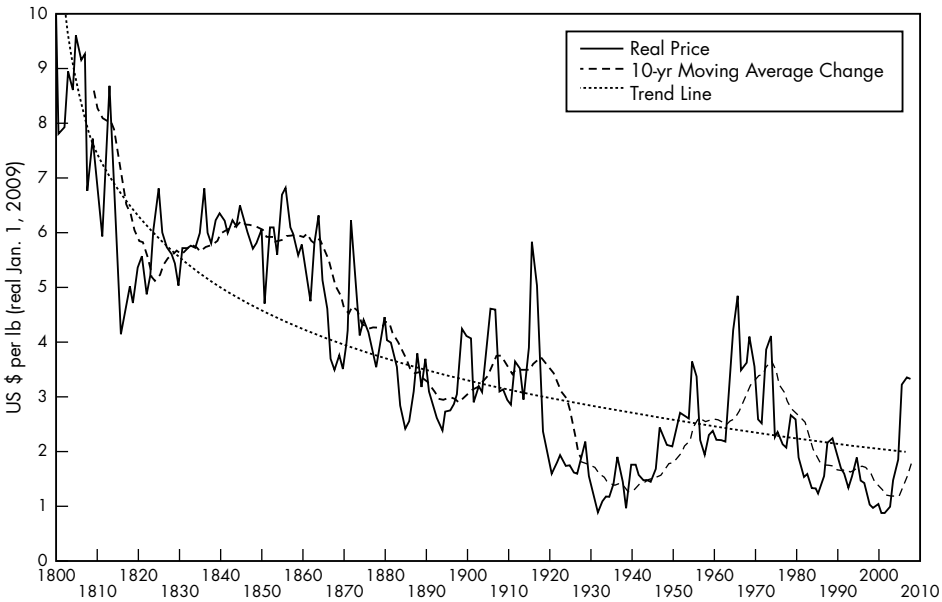
The first observation on Figure 1.2-1 is that the general price trend is downward, due to the impact of globalization and, to a lesser extent, technology. It is much less expensive to produce lots of copper from a few large mines in Chile than it is to supply it locally from many small mines—and, as extraction costs fall, so do prices.

The next critical observation is that, while the overall trend is clearly downward, during periods of up to 50 years prices have steadily risen. The current buzzword for these periods is supercycles, and they generally follow a major event. The largest of these was the post–World War II period when much of the world was rebuilding and metals demand grew strongly. A substantial body of experts currently believes we are in another supercycle. This period of strong demand growth is driven by the industrialization of the BRIC (Brazil, Russia, India, and China) countries. The people in these countries, which number approximately 2.7 billion (40% of the world's total population), are seeking a better way of life, which they have learned about through television, the Internet, and global tourism, and believe that this better way of life is increasingly achievable to them.

Industrialization does not affect the demand for all products equally. One of the early beneficiaries was the raw materials that go into making steel, which is an essential input in the construction of roads, buildings, and general infrastructure. Figure 1.2-2 shows the steel intensity per capita at various levels of gross domestic product (GDP). As this graph shows, steel usage rises rapidly until the economy has grown to about US$15,000 per person. During this early industrialization, iron ore and metallurgical coal, the two primary raw materials in making steel, have seen demand growth (and price) exceeding long-term averages. Should the BRIC countries continue to industrialize, the large population bases of these countries would support another supercycle. Over the next 10 to 20 years, this future growth (or not) of the BRIC countries will be the issue that will have the biggest impact on the mining industry.

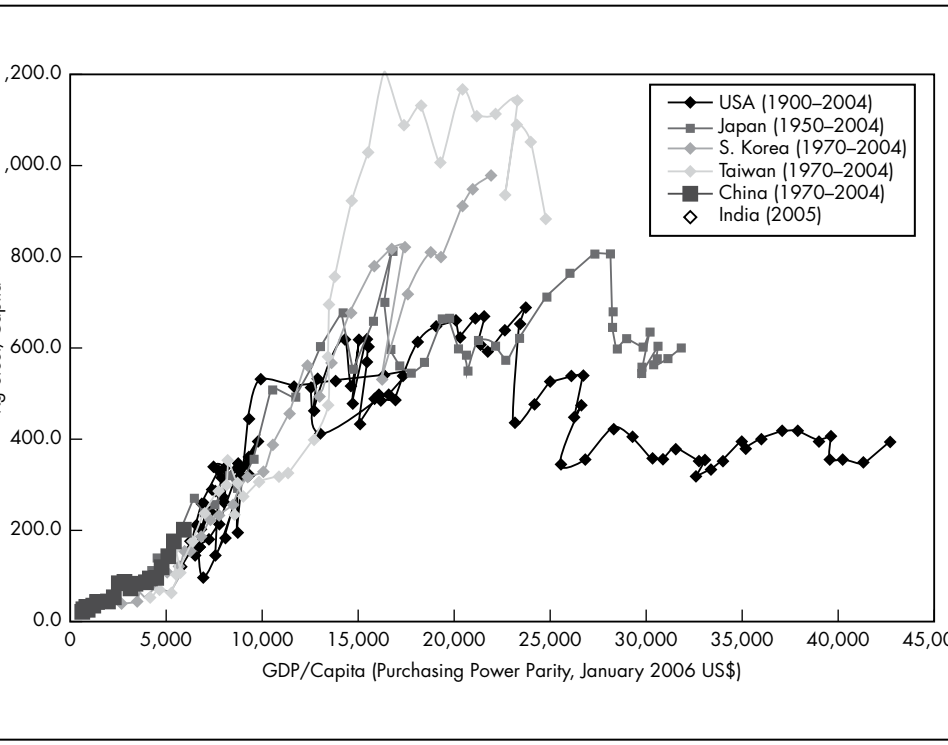
The large growth in natural resource requirements is causing debate now, as it did following the substantial expansion of demand after World War II, about the adequacy of the world's resource base to meet the world's needs. The earth has a large storehouse of minerals, and nearly all mining products are much more common in the earth's crust than petroleum. As the requirement for mining products grows, the issue will not be whether we will run out, but how much will these minerals cost? The answer to this question will be determined by

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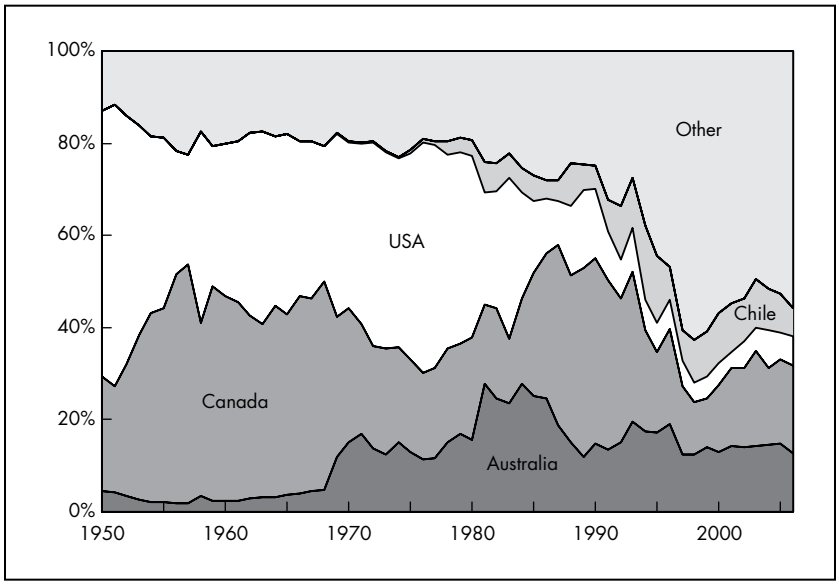
Courtesy of BHP Billiton.

Figure 1.2-1 Real copper prices, 1800–2008



Courtesy of World Bank.

Figure 1.2-2 Steel intensity per capita



Courtesy of Metal Economics Group and BHP Billiton estimates.

Figure 1.2-3 Nonferrous exploration expenditures by country

the ability of mining and mineral processing technology to stay ahead of demand growth.

Marketing

No assessment of current trends would be complete without commenting on the major changes that have occurred in how mining products are sold. In the 1970s, producer prices (where producers unilaterally declared the price) were commonly replaced by auction prices. Effectively, producers of copper and aluminum began to sell to warehouses operated by the London Metal Exchange. Buy and sell prices were agreed upon through an open platform/open outcry system, similar to early stock exchanges, through auction prices. After these prices were announced and published, they became the basis for setting the price of raw materials.

The use of an auction process has two major benefits. The first is that prices change rapidly with demand. Effectively, the price is continuously set at a level where there is a willing buyer and a willing seller, which effectively represents true supply and demand fundamentals of that product in real time. Second, the auctions allowed the creation of financial products that are valued by both the buyer and seller. These financial products allow prices to be agreed upon for future sales, effectively allowing companies to reduce their exposure to changing raw materials prices. They also made it possible for companies to manage pricing risk between the purchase of raw materials and the consequent sale of the product (matching input cost and revenue).

The current trend is toward greater use of screen-traded, open auction pricing for an ever increasing range of raw materials. The early adopters were commodities that could easily be delivered to a single, consistent specification. Increasingly, screen trading is expanding to include a wider range of specifications and products, such as coal and iron ore, which have substantial variations in quality.

MINERALS EXPLORATION

Location

Mines have a single characteristic that is different from most other industries: Development, and investment, must occur where the resource is located. As miners, we don't have a choice about where to locate our mines; they need to be where the deposits are. We do, however, have a choice in how much we are willing to trade mining higher-grade and higher-quality resources in higher-risk countries for lower-grade and lower-quality resources in stable countries with developed infrastructures. This trade-off between discovery risk and political risk is the history of minerals exploration and has driven a recent trend toward exploring in less-developed countries.

Figure 1.2-3 illustrates this shift of exploration expenditure into less-developed countries. In the 1950s, nearly 80% of the world's exploration expenditure went into the United States and Canada. By the year 2000, the United States and Canada were attracting barely 25% of exploration dollars. Instead, the major beneficiaries were Latin America and Africa, with Australia and Chile developing major mining industries after early exploration success.

What is clear is that, while there is a trend away from the highly developed countries, there has not yet been a major increase in exploration in the highest-risk countries. Although many of these extreme-risk countries offer easier-to-find, high-quality deposits, they have not yet received substantial attention from the global explorers. The reason for this is the long-term nature of finding and developing a mine combined with the limited risk appetite of investors and significant operating and fiscal challenges. From the start of exploration until a mine is in production generally requires around 10 years. Recovery of the initial investment frequently takes another 10 years, effectively meaning that 20 years of operational and fiscal stability is required for a mining project to pay back its initial investment. Higher-risk regions may be geologically

attractive, but their political stability is usually believed by mine developers to be insufficient to support 20 years of exploration, development, and mine operation.

Exploration Technology

During the last 50 years, the exploration sector has moved from reliance on observations of geologic features exposed at the surface to an ability to detect hidden mineralization at increasing depths. This has largely been achieved through developments in geophysical exploration tools. Advances in microelectronics in the late 1950s supported the development of smaller equipment that could quickly measure and process large amounts of data. This smaller equipment was mounted in planes and helicopters and produced a quantum leap in the industry's ability to cost-effectively screen large areas of land.

Additionally, refinements in both passive (magnetics, gravity) and active (electromagnetics) methods provided increasing depth penetration potential. Direct detection methods are now capable of outlining larger targets buried at depths of up to 1 km. While this depth is still short of the economic limit of mining, it is a vast improvement in capability versus 50 years ago, and ongoing improvements will ensure the industry can increasingly detect mineralization that is deeper and covered by post-mineralization sediments.

In the last 25 years, targeting capability has also been enhanced by satellite spectral scanners. Dedicated satellites are now providing geologists with a substantially improved ability to map topography, structure, and alteration. The development of faster processing capability linked to computer chip speed means that an individual geophysicist carrying a laptop computer has the ability to process complex inputs from a variety of sources.

Exploration Competitors

The starting point for a successful minerals exploration program is to obtain access to highly prospective ground. This access is typically granted in the form of an exploration license by the government where the ground is located. Effectively this means that a good explorer must have a combination of relationships, capability, and reputation, which makes it attractive to the government granting the license.

This structure has resulted in four competitors in the minerals exploration arena:

1. Local public or parastatal companies that have good relationships with the host government
2. Junior exploration companies, typified by small Canadian and Australian exploration companies, that are willing to take high risks to generate a discovery and then usually on-sell it to a large producer with operating and financial capability
3. Global mining companies that are exploring for deposits they hope to develop
4. State-owned enterprises (SOEs), supported by national governments, that are seeking to acquire natural resources globally to support their countries' development

The development of group 4, the SOEs, is a recent trend and is typified by China. Under this model, the government sponsoring the SOE manages a coordinated program that frequently ties together government-to-government soft loans, infrastructure development, and access to resources for the SOE. The goal of this strategy is to ensure a supply of critical natural resources to support country growth and development.

ORE RESERVES AND VALUATION

The fundamental source of a mine's value is its potential ore reserve. The word *potential* is used because rarely is the full extent or quality of an ore body known until it is fully mined out. One of the key decisions that the developer faces before building a mine is therefore the level of certainty required about the ore body's size, quality, and shape before the commitment of funds is made. In estimating the potential, most good analyses will consider the probability that various sizes and grades of ore bodies will ultimately be found and mined.

Reporting requirements mandated by stock exchanges are requiring companies to disclose their ore reserves to standards set in codes, such as Australia's Joint Ore Reserves Committee Code. The reporting of ore reserves is generally an external exercise to provide investor assurance and governance and is different from the assessment of ore potential that is required to support an investment decision. Investment decisions need to factor in the likely outcome for the total resource size and quality, as well as the order in which it will be recovered. The critical difference is that investments and valuations are based on what is expected, whereas reported ore reserves are based on what is known.

Investment decisions establishing ore reserves and/or expected reserves also need to consider the ore-body structure, processing recoveries, and impurities. The single biggest physical factor that causes mining projects to produce lower-than-expected investment returns is misunderstanding the ore body's characteristics. The biggest errors occur in geological continuity, ground conditions, and metallurgical recoveries. Either the ore body isn't continuous, it has unexpected faults or other geologic features, the ground conditions (in the ore or the adjacent waste) are harder or softer than expected, or assumptions of recoveries and payable product are overly optimistic. All result in higher costs, and several high-profile operations had to close after start-up because of unexpected and structurally unsustainable costs resulting from serious errors in predicting one of these issues.

Unfortunately, companies do not appear to be getting better at understanding the ore reserves supporting their investment decisions. The possible explanations for this are the following:

- Drilling and sample testing are typically the slowest and most expensive parts of feasibility studies. When corners get cut, drilling gets underdone.
- Understanding the adequacy of a drilling program is complex, and the people approving the project rarely have the time or capability to judge the appropriateness of the estimated resource.

Despite the importance of understanding ore-body structure and metallurgy, the biggest unknown in mine valuation is mineral prices. This is also the factor that has the biggest impact on the project's returns. Companies are improving their capabilities in this area, with virtually all midsize and larger producers investing substantial efforts into understanding the likely direction of their key markets. However, as the movement of mineral prices are heavily driven by demand, which flows from global growth, prices are volatile and price estimates remain substantially subjective. This is especially

true over the long term, which is the time frame that is relevant to virtually all mining development projects.

MINE OPERATIONS

Most mine operations have two separate activities: mining and mineral processing. The high capital cost and long life of a mining operation necessarily lead to relatively lower rates of technological change, broadly shared with other industries with similar characteristics. The activities currently used in the mining and processing areas of most mines look similar to those used 25, even 50 years ago. Drilling and blasting, the use of trucks and shovels, crushing, grinding, flotation, smelting, and refining are all still core elements of the industry. As is the case with most technologies, however, the extent of perceived change depends on how high a helicopter view is taken. One of the most pervasive overall changes is the size of the equipment and the scale of the operations in which they are used.

In 1970, the average truck purchased for use in an open-pit mine was 90 t (metric tons). By 2008, the average size of a newly purchased mining truck had doubled to just over 180 t. Plant size has followed a similar trend. The Escondida mine in Chile has been either the first or the second largest copper mine in the world since it started operating in the late 1980s. Since construction, it has undergone multiple expansions, as shown in Table 1.2-1.

Table 1.2-1 Escondida's expansion history

Metric	1990	2008
Daily ore production, t/d	35,000	240,000 concentrator feed, 60,000 oxide leach feed, 300,000 sulfide leach feed
Total material moved, t/d	280,000	1.4 million
Annual copper production, t/yr	320,000 in concentrate	1.1 million in concentrate, 300,000 million in cathodes
Average copper grade	2.9%	1.5%
Processing technology	Concentrate flotation	Flotation, oxide leach, sulfide leach, electrowinning
Mining technology	Drill, blast, truck, shovel	Drill, blast, truck, shovel, ore conveyors

The path that Escondida has followed would be a typical one for a large mine. The initial project was designed to mine the easy-to-access, high-grade ore. The limitation on initial plant capacity was the ability to sell the concentrate, as Escondida does not have an associated smelter and was an early mover in supplying raw materials to nonintegrated processing facilities in Asia. Since its initial start-up, substantial value has been added through a series of expansions that used the same infrastructure footprint as the original construction. As these occurred, each generation of larger equipment was added, and incremental processing technologies were introduced to optimize recovery.

Escondida also reflects the general trend in mining toward large open pits. At the beginning of the 1990s, more than 90% of mine production was by underground means. By the year 2000, 85% of mine production was from open-pit mines. Current indications are that the percentage of production from open-pit mines has peaked. Large open-pit mines create big, permanent holes, leave large piles of waste rock, use large quantities of water, produce large amounts of noise and dust, and consume vast amounts of petroleum to power the mobile equipment. The tolerance of society for activities such as these is declining.

At the same time, the ore bodies that remain to be exploited are becoming deeper as existing mines deepen and improved exploration technology finds deposits under deeper cover. Figure 1.2-4 shows the depth to the top of mineralization by year. Between 1930 and 2000, the depth of the average discovery in Australia, Canada, and the United States increased from surface outcropping to 295 m. This depth to the top of the ore body requires more prestripping than any but the largest mines can support.

Additionally, the current mega-mines are reaching the economic limits with their stripping ratios. The world's great open-pit copper mines, including Chuquicamata (Chile), Grasberg (Indonesia), and Bingham Canyon (United States), are all nearly 1,000 m deep. The amount of waste removal required to continue to access the ore body is becoming extreme. All of these operations either have begun to develop and/or operate underground mines underneath their current pits or are in the later stages of engineerinsg studies to do so. To meet the tonnage requirements of these large mines, all of them are considering bulk mining methods, with block caving being the mining method of choice. In South Africa, the Palabora copper mine has already made this transition. The combination of environmental pressures and depth is expected to further this trend toward increased development of large underground mines.

Remote and Autonomous Operation

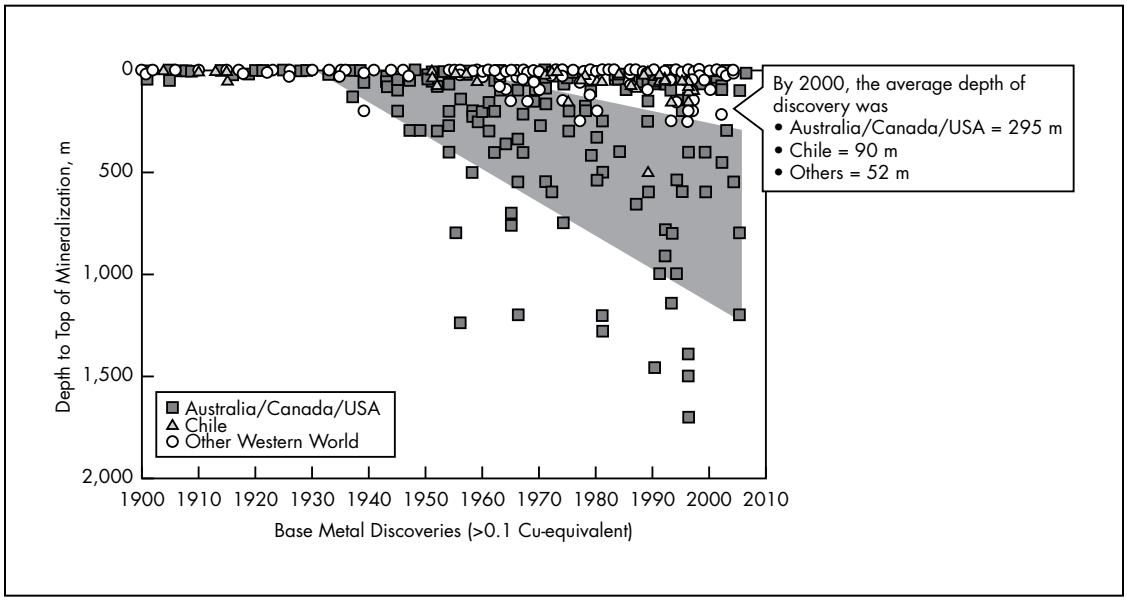
The technology shift in mining that merits close monitoring is the increased operation of equipment either remotely or autonomously. Remote operations, where the operator is not proximate to the equipment, can dramatically improve safety performance. This method of operation started to become more common in the early 1990s. By the mid-1990s, equipment suppliers were presenting the potential of autonomous equipment, where no operator was required. Since then, technology changes, including a high-resolution Global Positioning System, wireless technology, remote sensing, and robust pit management software, are making autonomous operations increasingly viable.

Productivity improvements and reduced equipment wear and tear are driving the development of autonomous technology for mobile mining equipment. Safety and the possibility of relocating staff away from remote and unattractive locations are driving the development of remote operations technology. Both remote and autonomous operations are technology trends that should be watched closely as they can materially reduce mining costs.

Mineral Processing Technology

Mineral processing changes have also continued to evolve. As in mine operations, mineral processing equipment has become bigger. Larger grinding mills, flotation cells, and furnaces have made for simpler plant layouts and have brought down costs.

One of the high-level drivers for other technology changes has been the overall trend toward lower ore grade and/or the need to match processing routes to other ore-specific challenges. As grade declines, the focus is on increasing recovery and tonnage throughput, with less metal value ending up in the tailings impoundment. This has led to measures that provide a greater understanding and control of processes. The development of computer-automated quantitative mineralogy, on-line



Courtesy of BHP Billiton.

Figure 1.2-4 Average depth of newly discovered ore deposits

materials monitoring, in-line process monitoring, sophisticated process modeling, and centralized control have all contributed to improved mineral recoveries and lower costs. Indeed, it is now a rare plant that does not have a central control room with substantially computerized controls.

Large step-change or revolutionary innovations are rare in most industries, and this is also the case for mining and minerals. Nevertheless, they have happened. The most notable change has been the shift from pyrometallurgy toward hydrometallurgy in nonferrous processing. For example, the advent of solvent extraction electrowinning (SX-EW) technology in the 1970s has steadily led to the growth of leaching in the copper industry such that SX-EW now accounts for more than 20% of world copper production. Escondida is a good example of this trend, illustrating the key drivers of ore-specific processing, resulting in chemical heap leaching of oxide ores; and the need to treat low-grade sulfide ores, resulting in biologically assisted heap leaching. Carbon-in-pulp technology had a similarly wide impact on the gold industry.

Of course, many innovative processes or subprocesses are scattered across the industry, and, although these are sometimes significant in novelty and impact, they tend to be applicable to a smaller subset of ore bodies or ore types rather than being generic. A path of process innovation has been followed for some decades by the end user of energy coal, the power industry, in pursuit of advanced combustion and gasification technologies, as well as carbon capture and storage approaches. The desire to reduce greenhouse gas emissions is a strong driver and remains a work in progress.

Major investments have also been made in new technologies for hydrometallurgical nickel laterite processing and the pyrometallurgical production of direct reduced iron. The development of laterite processing technology has been resource driven as a result of the dearth of new nickel sulfide discoveries and the easy availability of large nickel laterite deposits. In both cases, the technologies are still evolving and have not yet become sufficiently simple or predictable to substantially change the economics of their industries. Major technology shifts in nickel and direct reduced iron have struggled to be successful, in part because

- They are competing against technologies that have been optimized over many decades;
- Development time is long and implementation comes at a high capital cost; and
- The products (nickel and iron) have highly volatile pricing, making it difficult to stay committed to an uncertain technology that, in its early stages, is missing cost and performance targets, and is therefore cash negative during much of the price cycle.

Although it is often difficult to predict where major change is likely, a trend to watch is the continued displacement of pyrometallurgy with hydrometallurgy in the nonferrous area and particularly the continued evolution, and ultimate success, of leaching technologies across a wider range of minerals.

VERTICAL INTEGRATION

By definition, mines need to be located where the ore bodies are located. Whereas the mine location is fixed, the level of processing attached to the mining operation is not. Historically, mines have been associated with the minimum level of processing to produce a concentrated product. The goal of concentration has been to minimize transportation costs.

The minimum level of processing required at a site has varied substantially according to the product mined. Largely this is because some products do not have sufficiently developed markets to allow intermediate products to be sold at full value and because locations away from the mine site can facilitate lower total production costs. As an example, there is only a small international market for bauxite and a moderate-sized market for alumina, and prices for these products tend to be fixed as a percentage of the aluminum price rather than the fundamental supply–demand balance for bauxite and alumina. This lack

of freely traded markets for intermediate products has generally resulted in bauxite miners producing alumina at the mine site, and then transferring it to distant, but owned, aluminum smelters, which are generally located in areas that can provide inexpensive power. This structure minimizes production costs but also makes it difficult to be a substantial participant in the bauxite mining industry unless your operations are vertically integrated through alumina refining and aluminum smelting.

Two current trends are pulling parts of the mining industry into or away from vertical integration. Pushing the industry toward vertical integration is the desire of governments to capture maximum rent from ore bodies and maximize job creation. As a trade-off for allowing access to mineral rights, governments are increasingly demanding construction of processing facilities in-country. Effectively, governments are looking to force mine developers to substantially increase investment and job creation as a trade-off for awarding mineral rights, even if this means processing facilities are not optimally located.

The stronger force is toward less integration caused by decreasing transportation costs, the generally lower construction and operating costs available from locating plants in Asia (especially China), and the opportunity to build larger processing facilities in central locations that can aggregate feed from multiple mines. This trend toward disintegration is expected to continue and will make it increasingly easy for mining companies to focus on mine-only operations, with processing handled by a separate company at a distant location.

SUSTAINABILITY

Environment, community, and safety are major issues for the mining industry and the trend is for increased scrutiny in these areas. The reasons for this are the following:

- **Historic:** Improperly operated and/or closed mines are still a legacy of our past. Old mines with major contamination and/or environmental degradation problems are all too common.
- **Social:** Communities, lifestyles, and even social fabric are all substantially impacted by nearby mine development.
- **Emotional:** The natural resources of a country are initially the property of the people of that country. A company, and particularly a foreign one, profiting from the extraction of natural resources is a sensitive issue.
- **Real:** The purpose of mining is to concentrate elements (minerals) for use by society. While the production of concentrated minerals (copper, gold, lead, etc.) is desirable, a side effect of mining tends to be the concentration (and release) of some undesirable elements (lead, arsenic, sulfur dioxide, radioactivity, etc.).
- **Progress:** Society expects ongoing improvements in industrial performance across all industries, and the mining industry must keep pace with progress in other industries.

The mining industry has been well aware of these issues, and its performance in the areas of environment, community, and safety is improving. In part this change is driven by higher levels of social awareness within the companies themselves. However, much of the change is also due to recognition by senior management that a good track record on environmental, community, and safety issues is a requirement for getting (and holding) access to resources.

Simply put, it is increasingly difficult for a mine operator with a bad track record on environmental or social issues to obtain or hold the license to operate that it needs to be successful in most developed and many developing countries. Governments, communities, financial institutions, and investors do not want to support or facilitate actions that are perceived to be unreasonably harmful to people. This has resulted in stronger regulation, higher levels of community involvement in the approval process, strengthened laws regarding indigenous rights, and the adoption of voluntary principles of sustainability and human rights by most large mining companies.

It has also resulted in a bifurcated industry structure whereby large, public companies are heavily scrutinized and typically seek to meet or exceed minimum legal requirements. They also tend to work closely with nongovernmental organizations (NGOs) and governments. The scrutiny and standards that apply to smaller companies are typically less than for a large one. However, governments and partners in less-developed countries are not always happy with the big company approach. This desire for quick investment supports the faster approach that smaller companies typically adopt.

The International Council on Mining and Metals (ICMM) was established in 2002, and most leading mining companies are members. Through the ICMM, the industry has put forward its proposal on sustainable development, including climate change, and has undertaken to report its progress and have its reports verified. Additionally, progressive companies have been engaging directly with their key stakeholders and making progressive decisions to enhance their licenses to operate. In many instances this involves commitment to environmental management strategies that go beyond strict legal compliance and have resulted in increased costs.

Climate Change

While there are some contrarian views, most of society believes that human-made climate change is causing harm and that carbon emissions are the primary cause. Society has therefore demanded that the subject of climate change receive maximum government and industry attention. Through the consumption of its products (primarily the burning of coal), the mining industry indirectly accounts for about 32% of global greenhouse gas emissions (IEA 2008). Given the industry's large contribution to carbon emissions, shareholders, regulators, and society at large expect the industry to reduce its footprint.

Although many mining companies have made commitments to reduce carbon emissions, this issue is expected to become increasingly contentious. The likely focus will be coal and uranium mining. Through its consumption, coal is the single largest contributor to increased levels of carbon in the atmosphere. Nuclear power, which uses uranium, has long been out of favor, largely due to issues associated with the disposal of nuclear waste and society's long memory of early nuclear incidents and accidents. However, nuclear power does not generate carbon, and, as a result, it is increasingly coming back into favor as a source of carbon-free power. Technology improvements, including carbon sequestration at coal-powered generating stations and the development of nuclear reactors with reduced levels of waste generation, will be key trends to follow.

Although it is not clear how the balance between climate change and low-cost energy will evolve, it is obvious that the

mining industry must be part of the solution and that energy produced from the mining of coal, uranium, and tar sands will continue to provide a major part of the world's energy supply.

Safety

The mining industry has historically had a poor safety record, and the industry is viewed as a dangerous one. While injury rates and fatalities continue to be unacceptably high, the industry overall has dramatically improved its safety performance. For context, the construction of the Panama Canal in the early 1890s involved the movement of 300 Mt of material and cost 26,000 lives. BHP Billiton's Escondida mine moves 360 Mt every year with the expectation that this will be achieved without the loss of a single life. Safety performance is still an issue, but dramatic improvements have been made and must continue to be made.

HUMAN RESOURCES

The design and material flow sheets for modern mines are complex and unique, the equipment is large and expensive, and operations are frequently remote. Capable people, at all levels of the organization, are therefore essential to operating a successful mine or mining company.

Despite this requirement for large numbers of capable people, the mining industry has faced and is facing a global skills shortage. The best evidence of this is the decline in university graduates in mining engineering, earth sciences, and metallurgy. In Australia, the supply of mining engineering graduates declined to just 100 in 2005, down from 200 as recently as year 2000. In the United States, there were only 86 mining engineering graduates in 2005. Similar acute skills shortages have developed for mechanical and electrical tradespeople and to a lesser degree for skilled equipment operators.

The reasons for this shortage of skilled people in mining include the following:

- **Remote locations:** Mine locations are frequently rural and remote. This lifestyle is generally less attractive than living in large cities, particularly to unmarried new graduates and families with high-school-aged children.
- **Industry unattractiveness:** Particularly in developed countries, the image of mining industry work is (incorrectly) of low-skilled, dirty jobs in an outdated, boom/bust industry. This image does not attract high-quality entrants.
- **Lack of awareness:** As the relevance of the mining industry has declined in the developed world, potential entrants are increasingly unaware of the opportunities the industry offers.

Increasingly, the trend is to fill the void left by a lack of new entrants in the developed world by recruiting skilled workers from developing countries. In 2007, the mining industry employed 8% of Australia's skilled migrants but only 1.3% of its native workers. A number of companies have started recruiting new graduates in less-developed countries with strong mining or manufacturing cultures. As a result, South Africa, Russia, and China have started to become targeted countries for recruitment of new graduates with technical degrees. These steps will help, but they are not enough to attract the talent the industry requires. To lure skilled workers, the industry is increasingly promoting the benefits of a career in mining, including career development, high pay, the ability to choose between an urban and rural lifestyle, and substantial opportunities to travel.

Finally, there is a strong trend toward fly-in/fly-out (FIFO) operations for new development projects. These FIFO operations are an alternative to creating remote residential towns to support nearby mining activities and are seen as a more attractive lifestyle alternative for many potential employees. This move toward FIFO operations also supports a sustainability agenda, because when the mine is depleted, the towns that were built to support it are frequently left without an economic base.

GOVERNANCE

There is unquestionably a trend to increased governance of public companies in general and mining companies in particular. These come from many sources:

- **Unions:** Although union membership has been falling worldwide, the statement that "you get a union when you earned one" is still valid and provides an important check on the management of industrial relations.
- **Nongovernmental organizations:** NGOs are passionate about their individual issues, and they are growing. Areas such as human rights, protection of indigenous people, biodiversity, and the environment all attract strong funding and provide a voice on mining issues that is widely heard and considered. The trend is for these groups to become more pragmatic in their approach, and many NGOs have moved from simply being against mining to engaging companies in substantial discussions about how mining can be done more sustainably.
- **Regulators:** Stricter reporting requirements and unbiased verification are clear regulatory trends through legislation such as the Sarbanes–Oxley Act of 2002 in the United States. Industry groups such as the ICMM are also substantial forces for self-regulation through their members' commitments to codes of conduct and reporting. The trend is for industry to increasingly rely on self-regulation and voluntary compliance to standards that exceed legal requirements.
- **Financial institutions and shareholders:** Equity analysts are increasingly independent from their lending institutions, and banks and investors have much better access to information about the companies they choose to financially support. For these reasons, it is harder for bad companies and projects to attract funding.
- **Society:** The volume of information and speed with which it is transmitted through sources such as the Internet are making it harder for unacceptable actions to go unnoticed.
- **Boards of directors:** Members of the boards of directors of public companies strongly dislike being associated with any type of scandal or perceived violation of a company's "social license to operate." Most boards devote large amounts of time to ensuring that high standards of governance are upheld.

Lastly, there is a clear trend toward more disciplined management decision making. As mining companies have become larger and more global, the costs of the decisions they are making, particularly to build ever larger individual mining projects, are increasing. This has brought with it a higher level of discipline to operating and capital decisions that have benefited all aspects of operation, including safety and environmental performance.

This discipline includes a more sophisticated approach to analyzing future price scenarios, assessing investment and operating decisions, and understanding risk. It also includes aggregating purchases of consumables, measuring and reporting performance in key operating areas, and requiring adherence to minimum standards of operation across entire companies. Evidence of the success of these actions is the general reduction in capital cost blowouts by the leading companies, improved investment decision making, and the ability to establish and report against a wide range of operating targets. Ten years ago BHP Billiton could not track its company-wide energy use, its carbon emissions, its usage and the prices paid for key spare parts, its instantaneous financial exposure to its key customers, or its equipment uptime. Nor could it easily dig into the quality of the supporting data for fundamental documents such as its ore reserve statements or its financial results. Today it routinely measures (and reports) all of these and much more.

However, the resources required to complete this work are large and therefore tend to be within the domain of the larger companies. This disciplined approach will need to be increasingly adopted by smaller companies or the industry will face increased regulation, which will further increase the value gap between the industry leaders and the rest of the industry.

INDUSTRY STRUCTURE

In the mining industry's evolution, the winner is clear. Lower transportation costs, bigger equipment and projects, higher standards for operating and ethical performance, and strong demand growth have all favored big, global companies with deep financial and operating capabilities. The gap in size between the leading mining companies and the rest of the pack has become quite large. In 2009, the world's largest mining company (BHP Billiton) is 40% larger than its nearest competitor (Vale), which is itself 40% larger than its next largest competitor. As a group, the large companies are active in multiple products and countries and could credibly claim to have the capability to efficiently build and operate mines in any of the major minerals, anywhere in the world.

The industry leaders also dominate the ownership of the best mineral deposits. A few midsized companies have large, high-quality deposits, but they tend to be companies that are protected from large acquirers by either their ownership structure or their potential liabilities. Companies in this group include state-owned miners such as Chile's CODELCO and companies with two-tiered ownership structures such as Teck Resources (previously known as Teck Cominco).

SUMMARY AND CONCLUSIONS

The mining industry is going through a period of rapid change. Globalization and a potential supercycle, caused by strong demand from newly industrializing countries, are permanently changing the industry. The industry-leading companies that have emerged are bigger, financially stronger, and much more capable than their predecessors. They operate across borders, produce a wide variety of mining products, and generally seek to operate in a manner that exceeds minimum compliance with international standards.

Although the companies within the industry have substantially changed, most mining operations and the technologies they apply are similar to the practices in the 1960s to 1980s. Although similar technologies are employed, individual mines are much larger, and the percentage that are open cut is higher. Operations are also increasingly moving to hydrometallurgical recovery processes and de-integrating themselves from downstream activities.

The industry is also receiving a high level of scrutiny. Resources are increasingly scarce and securing access to sufficient resources to facilitate economic development has become part of government policy in some countries. Catastrophic "unsafe events" continue to be unacceptably common and are heavily reported. Lastly, mining activities consume and alter large quantities of water and terrain and, as mining operations become larger, the scale of such alterations is becoming increasingly visible.

Society demands more from the mining industry, and the industry is responding. Increased raw material demand has brought with it growing recognition that the mining industry can be an attractive employer and that it does meet an essential need of society. This new status is increasingly attracting a new and more capable set of industry leaders, and ones that are increasingly focused on not only providing essential resources to society but doing so in a manner that is socially and environmentally sustainable.

REFERENCE

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CHAPTER 1.3

Future Trends in Mining

Tom Albanese and John McGagh

INTRODUCTION

Imagine for a moment the mine of the future, where knowledge of the ore body, its mineralogy, size, and value are known precisely, based on a range of three-dimensional (3-D) geological images captured nonintrusively long before mining commenced. The mine plan covers not only the initial target ore body but all future extensions until the reserve is exhausted. Nothing is left to chance. Imagine a mine with a zero environmental footprint and zero net energy consumption, where all processes are continuous, with process control systems that monitor and optimize performance, and where all moving equipment is autonomous and controlled from afar. Few people are visible on or under the ground, and the work environment is safe and healthy. Highly skilled workers operate the mine from air-conditioned control rooms in major capital cities. These jobs are well paid and highly prized.

Can we imagine this future, and is it that far away? The pace of change in the industry has increased dramatically, with strong market pull and strong technology push. The mine of the future may be closer than we think, and many of the enabling technologies exist today. The trends likely to shape our future will be explored through this chapter.

Demand

Although the pace of change continues unabated, the nature, rate, magnitude, and impact of change are not constant and know no boundaries. No one predicted the coming of the information age and the enormous global impact of the Internet. The mining industry is changing in step with global demands, but the challenges of supplying minerals and metals to a world experiencing exponential change are great. The future will be very different.

The mining industry is experiencing a dramatic change, one that profoundly affects our industry, an unprecedented change that creates an enormous challenge and an immeasurable opportunity. The world is rapidly becoming urbanized, with an additional 1.4 billion people predicted to move into cities within 20 years. Although the population shift will be universal, it is being led by China and India. People who move to cities require houses, roads, schools, power stations, and stadiums, and they gain the wealth to purchase consumer goods, such as refrigerators, cars, and air conditioners. With urbanization comes a greater demand for metal. It is estimated that the average per-capita requirement for metal products is 155 kg for China's rural communities and 817 kg for China's urban dwellers.

Demand for all base metals, particularly iron, copper, and aluminum, will likely double from 2010 to 2025, due largely to this population shift. Putting this in perspective, the additional demand for iron ore in that time period is equal to the capacity of five Rio Tinto Pilbara operations, which produce close to 200 Mt per year. It is also estimated that the world will consume as much copper from 2010 to 2035 as it has during the last century.

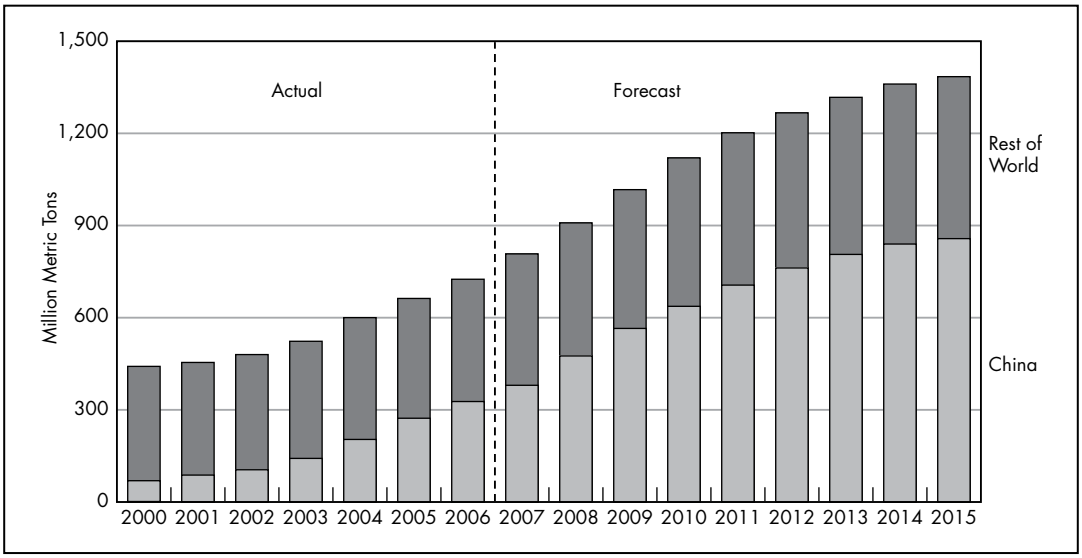
China's iron ore imports are expected to double from 2010 to 2016 (Figure 1.3-1), following many years of growth that has made China the world's largest consumer of traded iron ore, copper, and aluminum (Table 1.3-1), together with nickel, steel, and coal. From 1990 to 2006, China's steel production more than tripled, with iron ore imports increasing 20-fold during this period. China is clearly the new force in commodity demand. The industrialization of China and India is changing the economic world order.

Supply

Satisfying this huge growth in demand is the mining industry's greatest challenge, and one that must be confronted head on. The industry must think and work differently to keep pace with this burgeoning demand. The old ways will not be good or fast enough. Change is essential.

Mine output rates must increase. Existing assets must be extended to yield more. Lower-grade reserves must be tapped. Exploration and discovery must become more efficient. The search for new high-value reserves must accelerate. These outcomes must be delivered during a global industry skills shortage and against a background of diminishing surface deposits and rising costs. Moreover, in today's society, everyone wants more for less. Higher outputs must be achieved at lower unit costs. Working against this need for lower costs are increasing energy costs, the threat of climate change, and the higher cost

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Source: Albanese 2008.

Figure 1.3-1 Seaborne iron ore imports

of mining deeper ore bodies and lower ore grades, possibly in more challenging geopolitical environments.

Efficiencies must be found in all operational areas, from exploration to extraction. The solution to efficiency improvement lies in the development and implementation of new and innovative technologies. Companies that innovate are more likely to be rewarded with lower costs, improved competitive positions, superior returns to shareholders, and sustainable businesses.

And the mining industry must deliver these outcomes in an environmentally sustainable way. The planet is warming because of human activity. Atmospheric levels of greenhouse gases are increasing. The mining industry is not insulated from the effects of global warming, and we must play our part in dealing with it. As miners, we must take sustained action to reduce the environmental impact of our operations. We have no choice. If we do not reduce the size of our footprint, those who are in a position to give us a license to operate will no longer do so. Our aim must be to achieve both zero emissions and zero net energy consumption. A suite of technologies that could support such a vision is under development.

All of this must be achieved in a world where stakeholder consultation is assumed and affected communities benefit from mining activities through and beyond the life of a mine. Consultation with local communities and other stakeholders must continue to evolve through all stages of a project, including the ultimate mine closure. This necessity increases as the search for new tier 1 reserves takes exploration to less accessible and more sensitive remote areas, often in Third World countries.

The mining industry works under intense scrutiny, and rightfully so. We live in the information age. People are more informed, and information is available to many people at the touch of a button. They are aware of the environmental challenges confronting this and future generations. They are more likely to act on what they see and take action against those who do not accept that the risks to our future are real and against those who act irresponsibly. We must deal with the intense scrutiny that comes with this new age. Not only must our house be in order, but we must ensure, through better communication, that the wider community knows it is.

Table 1.3-1 Growth in China's share of global consumption of metals (%)

	2001	2002	2003	2004	2005	2006	2007E*	2011E
Aluminum								
China	15	16	19	20	22	25	31	41
USA	22	22	21	20	20	18	15	12
Copper								
China	16	18	20	20	22	23	24	26
USA	18	16	15	14	14	12	11	11
Iron Ore								
China	30	32	34	39	46	51	53	54
USA	5	5	5	4	4	3	3	3

Source: Albanese 2008.
*E = estimate.

Finally, while innovation may hold the key, today's new technology could well be next year's standard practice, so innovation must be a continuous process through the economic highs and lows. A cultural change is needed. The goal is an environment in which workers constantly seek new and better ways of doing things and in which innovation is rewarded. New ideas must be continually developed and nurtured. The same systems and cultural changes that brought the world higher quality, better customer service, and improved safety can drive innovation in the mining industry.

The challenges are universal and demanding—the increasing demand for commodities; grades and their decline with time; mineralogy and the need to handle more complex ores; the need to find new reserves; disposal and minimizing of wastes; and the availability of water, power, and skilled labor. These challenges are combined with increasing expectations from the community and concerns about sustainability and safety and climate change, forcing a more targeted

approach on energy. The opportunities and the rewards are great. Those mining companies that meet the challenges will be in a stronger competitive position. A vision for the future is provided in the following sections.

EXPLORATION AND GEOLOGY

Logic would suggest that it is easy to find things that have already been found. In the mining world, it is hard to argue the point when just about every square kilometer of the developed world has been surveyed to some extent. It follows that if we are to keep pace with demand, exploration and discovery must become more efficient and the technology used to detect and characterize mineral deposits on and below the earth's surface must become more capable. Vast amounts of money are being spent on exploration. In 2002, global exploration expenditure was in the region of US$2.5 billion, and by 2007, it had risen to more than US$10 billion. The identification of the geologically rare tier 1 deposits is the highest prize. Such deposits grow with exploration, commonly have other tier 1 deposits nearby, and support production expansions. Their discovery is a necessary part of the total solution to satisfying growing global demand for minerals and metals.

Exploration

The aim of exploration geology is to find mineralized target areas for development into profitable mines. To define an economic deposit involves a number of steps—from initial small-scale sampling to larger-scale characterization. History has repeatedly shown that the probability of converting exploration targets into economic deposits is low. In the future, therefore, the key challenge for exploration geology is to increase this probability of success by identification of

- A wider range of deposit types, including lower-grade ores, deposits with different mineralization styles, and ores with greater variability, possibly in areas already explored;
- Deposits that do not occur at the surface or are covered and possibly in areas already explored, near existing ore bodies, or even below existing mine sites;
- Targets that are potentially more remote;
- Deposits in more politically sensitive or unstable regions; and
- Deposits in more environmentally sensitive regions.

The discovery of ore bodies or mineral resources in any of these categories will present financial, political, and scientific challenges.

Interestingly, Davy analyzed all kimberlites/lamproites discovered from 1966 to 2003 (excluding those in Siberia and Russia) and observed the following (Davy 2003):

- The frequency of discovery doubled through the 1990s compared with the 1980s as more money was spent on exploration and more junior diamond explorers were active.
- More world-class projects were discovered in the 1990s, lending support for the view that, with improved methods and new technology, world-class deposits are still there to be found.

Before discovery, however, the rights to explore a prospective area of land must first be acquired, and this is not without difficulty, especially in a competitive market. Company reputation will influence the outcome, and financial considerations are also critical. Different skill sets will be required. After government approval is given, exploration in remote, unstable, or environmentally sensitive regions must be efficient, and less-intrusive methods for detecting minerals must be employed.

Geology

In addition to target identification, geology has to better predict how ores are expected to behave during the stages of mining and metals extraction. To optimize cash flow, such ore knowledge is applied throughout the value chain. In the medium to long term, ore characterization methodologies will be improved through better measurement techniques, predictive capability, and early decision making.

Improved Measurement Techniques

New technology allows for higher-quality results that can be provided in a shorter time frame, hence increasing resolution. Such improved characterization will allow for better definition of the reserve, which impacts the economic value of the deposit.

Increased ore-body knowledge and associated technical developments allow more complex ore bodies to be potentially exploited. And better characterization of the resource is used strategically; that is, which ore bodies need to be progressed through the prefeasibility, feasibility, or order-of-magnitude stages.

In normal circumstances, confidence in data collected during project development from exploration to feasibility study increases as the project progresses. Some of the measurement and testing technologies that are likely to be further developed in the future include automated core logging, core imaging, and on-line and near-online analyzers.

Improved Predictive Capability

Improvements in 3-D modeling capability will increase the ability to predict both mining and processing behavior from measured primary data. Key elements that need to be known for base and precious metal mining operations include blasting, crushing, grinding, liberation, and recovery characteristics. Other related issues include tracking deleterious elements and minerals, providing inputs into the environmental management of waste rock, and increasing the energy efficiency of processing equipment:

- **Improved prediction of ore-body behavior in mining.** More knowledge at the early stages of projects improves decision making on mining methods; for example, for the prediction of fragmentation, crushing, and grinding and for the optimization of blending strategies. In underground mining, improved cave models can be used to optimize draw strategy.
- **Improved prediction of processing behavior.** Metallurgical data in the block model improve decisions about processing methods and allow for the prediction of performance for specific ore types and ore blends. The data are also used for concentrator optimization and metallurgical accounting.
- **Improved prediction of behavior into the environment.** It may be possible to minimize the environmental impact of mining and predict environmental impact and cost by incorporating environmental data in the block model.

Improved Early Decision Making

Higher data density (but lower cost) and increased predictive capability will enable the industry to more confidently reject exploration targets that are deemed uneconomic. Exploration provides a significant return on investment. Despite that, the cost is high and future tier 1 assets will be harder to find, so technological advancements and process improvements that shorten the discovery cycle and increase the probability of success need to be developed and implemented.

SURFACE MINING

The advent of surface mining stands, arguably, as the most significant change to the fundamentals of the mining process. The move to open-pit mining, which started in the 1890s with advancing mechanization, has dramatically simplified the process of extracting minerals. A rich history of innovation has brought surface mining to where it is today, with mine output rates that were unimaginable even a few decades ago. But what does the future hold? Surface mining is subject to a wide range of internal and external pressures, so change is essential to meet the challenges ahead. It is no longer just about moving as much rock as safely and cheaply as possible. Recently, automation and remote control of mine processes have taken center stage, and this is likely to continue into the foreseeable future as advances in communication systems, measurement systems, and computational power provide unlimited scope for development. As well as these technologies, there are still many areas where both step-change and incremental improvement can add tremendous value to the surface mining sector, and the industry appears to be poised to pursue these opportunities.

Fragmentation

Fragmentation in hard-rock surface mines is almost entirely dependent on explosive rock breakage, and this is unlikely to change in the foreseeable future. In terms of effectiveness and cost, blasting provides the ability to liberate large quantities of material to a size that can be moved using standard excavation and transport equipment.

Given that blasting lies at the core of the mining process chain, it is not surprising that considerable research has gone into explosive formulation, initiation techniques, and simulation. The Hybrid Stress Blast Modeling research project is an example of current research that is exploiting the increase in computing power to apply sophisticated numerical modeling codes to the process of blasting (Batterham and Bearman 2005). The knowledge of fragmentation and muckpile formation that can be yielded by this approach will enable blasting to be better matched to downstream requirements. This is part of the move toward an optimized mining process, free of disruptions from poor blast performance.

Alternatives to explosive fragmentation in surface mining are limited by the amenability of the ore body, in terms of both material properties and geological structure. The barrier to widespread use of mechanical excavation is the difficulty of cutting hard rock and the high cost of machine wear and tear. In mines where material is amenable to mechanical cutting, significant proportions of production are being delivered without blasting. In these instances the driver tends to be selectivity, linked to the fact that the ore thickness is significantly thinner than the normal blast-sized smallest mining unit. Therefore, if mined using traditional open-pit bench heights, the degree of dilution would be excessive. If selectivity is not a prime driver, then factors such as reduction of diesel consumption or environmental sensitivities that preclude blasting or adverse pit floor conditions could support mechanical cutting and continuous material movement.

Blasthole Drilling

Blasthole drilling offers the opportunity to gather more information on the strata and rock encountered during drilling. Today, the data associated with drilling—torque and pull-down force—are either not logged or are used in a fairly basic manner. In some instances, rock, or strata, recognition is performed by correlation of drilling parameters with rock hardness, but the technology's acceptance is not widespread despite numerous positive applications and case studies. The most often cited reason for the lack of acceptance is the need to retrain the algorithms at the heart of the system as the drill moves into different domains.

In the future, real-time feedback from the drilling rig will be regarded as routine. In addition to the drill parameters and rock recognition, sensors in the drill will perform a variety of duties ranging from elemental ore analysis to the measurement of geotechnical rock mass characteristics. Discrimination using a variety of measured and derived properties will move the industry toward greatly improved digging to ore–waste boundaries. Advanced blast design packages will become more accepted and more sophisticated, with the packages linked directly to the charge loading trucks. The linkage will be wireless and will replace the manual exchange of data, thus leading to the planned loading of a range of explosive types and densities. The correct delivery of optimized blast designs will ensure greater predictability in fragmentation and muckpile shape, which in turn will lead to improved digging conditions and reduced operating costs.

The advanced drill-blast-load loop is heavily dependent on the deployment of a variety of sensors. Every time we touch a material, we must learn something about it. The use of sensors and their integration into standard operating procedures will enable miners to increase operational effectiveness even when there is a skills shortage.

Materials Movement

A major challenge lies in how best to get material out of a mine. In early open-pit mines, locomotives moved much of the material in the larger pits such as Bingham Canyon, Utah (United States). The move to trucks was a major step forward in flexibility and has driven the increase in open-pit mining. In open-pit mining, equipment size matters and equates to productivity: more material moved in a given time. For this reason, the trend will be for ever larger equipment. Although the trend has been focused on the size of haul trucks, to load these larger vehicles the size of loading equipment has also increased commensurately. Currently, haul trucks with payload capacities of up to 365 t (metric tons) carry loads from the mine face to the tip point, a 12-fold size increase in payload capacity since 1950. To satisfy the enormous appetites of these trucks, excavators with buckets of up to 45 m^3 and payloads of more than 100 t are in use, enabling even the largest haul trucks to be loaded with four passes, thereby ensuring a quick turnaround.

But what is the ideal size of a haul truck: larger, smaller, or the current size? The answer is uncertain, but the trend toward larger vehicles shows no sign of slowing. Fewer, larger trucks reduce flexibility, increase risk, reduce mining selectivity, and drive up the size of ancillary equipment. Larger trucks and

excavators must be exceptionally reliable to improve availability and ensure that productivity targets are met. Smaller trucks drive up cost because of number, maintenance, and larger work force. The trend in size will be strongly influenced by what best suits an automated mine operation, where reduced cycle times and increased availability will deliver productivity gains. Limitations to further size increases may also come from engineering and material constraints.

Trucks, predominantly diesel-electric, provide flexibility and can move anywhere. Despite increases in efficiency, the diesel use is significant, and its reduction is a major challenge to the industry. Alternative energy sources for trucks must be developed. Driven by the need to reduce greenhouse gas (GHG) emissions and reduce dependence on petroleum feedstocks, the global automotive industry is moving rapidly to develop alternatives to the embedded internal combustion engine. Hybrids may be part of the solution. Hydrogen fuel cells offer some promise, and biodiesel based on waste biomass may be a viable alternative fuel for internal combustion engines. Certainly, the automotive industry's experience will flow onto the mining industry, and early adoption of a viable alternative can be expected.

Electrically augmented trucks fed from an electric pantograph (overhead power lines) are deployed at some sites, with their original installation driven by the fuel crisis of the 1970s. Their reduced flexibility and difficulty in changing the size of trucks due to the fixed overhead infrastructure limits their widespread application. Given the current fuel situation, development in the field could be expected.

Alternatives to haul trucks must be considered, particularly in view of ever-increasing energy costs. The obvious alternative is a conveyor system for flat areas or high-angle conveyor systems to reduce diesel-intensive uphill hauls, but there are drawbacks. A conveyor is more fixed and can transport well-fragmented material but cannot take run-of-mine blasted material, unlike haul trucks. For conveyors to be effective, the top size of material must be controlled and this can only be guaranteed currently by size reduction through crushing or mechanical cutting. In-pit crushing is a solution to this dilemma, which has been deployed at various sites over the years, but the challenge has always been such units' mobility. Recent developments in mobile crushers and the use of conveyors have created greater opportunities with future developments in this field expected to widen the application of the technology. Further, mechanical excavation could provide the consistent material flow suitable for a conveying system in amenable materials.

Long-haul, or out-of-mine transport, presents a further set of challenges in the future. Long-range overland or aerial conveyors offer some alternatives. Many significant overland conveyors have been deployed to great effect, and recent developments in aerial conveying systems could provide further alternatives where terrain is unfriendly to the overland version or where the system must traverse environmentally sensitive areas. From an energy perspective, conveyors of both types offer the option to use regenerative technology to feed power back into the energy system.

Pumping has not traditionally been considered as a material movement system, but with improved knowledge of rheological flow properties, there are moves to examine the pumping of slurries containing much larger particles.

The traditional transport option for long distance, including mine to port, is rail. Locomotives are currently the focus of considerable research into alternative fuels, including the development of a hybrid diesel-electric locomotive that not only reduces emissions but reduces fuel consumption by capturing and storing energy dissipated during braking. It promises both cost and environmental benefits. The efficiency of the overall rail network is also a major consideration, and, in addition to the application of advanced optimization models, there is a move to autonomous train operation.

Planning and Scheduling

As the mining industry moves toward more complete integration of production systems, planning and scheduling will change dramatically. Whereas plans and schedules for mining, maintenance, and logistics were once developed in relative isolation, the trend is toward whole-of-business planning and scheduling. Distinctions between long- (strategic), medium- and short-term planning may remain, if only for convenience, but business processes and software systems will evolve such that plans and schedules developed with different time horizons will influence and be influenced by others:

- Plans and schedules will become adaptive, responding to increased granularity in space and time information.
- Real-time sensing of material geometallurgical properties will influence the mining sequence and downstream processing in close to real time.
- Short-term production schedules may even respond to short-term fluctuations in market needs.

Although the next step-changes in mining methodologies may not be immediately apparent, every change introduces new challenges for planners and schedulers. Software systems for mine planning and scheduling will evolve to cater to these and other mining options.

Formal optimization algorithms have long been used to design optimal pit shells, aiming to maximize project net present value. But optimization is likely to be applied much more systematically throughout the production process, not only from mine to mill but from pit to port. Decisions that relied on experience in the past may one day be supported by almost continuous re-optimization of the production process. Genetic and evolutionary algorithms will complement parallel efforts to solve large mixed-integer linear programming techniques. Optimization algorithms will account for uncertainty in all parts of the production process, from variability in geometallurgical properties to reliability and availability of fixed and mobile plant to fluctuations and trends in costs and commodity prices.

As the mining industry moves toward automation and autonomy, the movements of individual vehicles will be planned and scheduled at ever-decreasing time scales. Some vehicles will effectively control themselves. Short-term mine plans may define the broad parameters, but conventional dispatch systems may become a thing of the past.

For the foreseeable future, explosive rock breakage and the use of haul trucks and excavators will remain an integral part of hard-rock surface mining. Dramatic increases in the use of automation and remote control of mining equipment will shape the future of surface mining. Underlying all future developments will be the ability to significantly increase the sensing, measurement, and monitoring of critical geological, geometric, and equipment-related parameters. Every time an opportunity arises to gain knowledge by taking a measurement,

this opportunity must be followed up. The effective integration and use of these data will provide the backbone of future advances in surface mining and will enhance the ability to deploy the automated systems that are such a critical part of the future.

AUTOMATION AND REMOTE OPERATION

The automation of mining processes is a technological step-change that will provide part of the solution to the industry's most pressing challenge: achieving higher outputs to satisfy the projected continuing growth in commodity metal requirements.

Automation also addresses the shorter-term imperative of maintaining a suitably qualified work force at remote mine sites, which is an industry-wide problem. Younger generations are reluctant to leave the comforts of urban life, where they see their futures. Although work forces can be maintained in mining regions, the cost of doing so is extremely high, not only in direct wages, training costs, and penalties that have to be paid to professionals and skilled workers alike, but also in housing and other infrastructure needed to support the work force.

Benefits of Automation

Automation increases the level of control in what is inherently a chaotic process by applying more stringent rules to decision-making processes and removing the randomness inherent in isolated decision making. Applying a controlled process to variable mine geology and ever-changing topography results in higher productivity and lower cost. Automation involves the collection and use of data; for example, gathering data from the blasthole drilling process, which enables hole placement and blast design to be better controlled and blast outcomes to be predictable and optimum.

Another benefit of automation comes from increasing the utilization and performance of haul trucks and other high-cost capital items. With improved control comes a reduction in the expected levels of wear and tear and breakdowns, enabling preventive maintenance to be better planned and performed. Moreover, the amount of wear and tear will be reduced because the autonomous machine is operated constantly within its design envelope. Costly breakdowns and unplanned maintenance should be avoided, as the cost of the repairs are higher than planned ones, but more importantly, the disruption to the production process cascades through the system with costly knock-on effects. Attempts to control wear and tear through driver regulation have had limited success because such regulations are not easily enforceable. Higher availability and utilization means higher productivity and lower unit costs. Another significant benefit is the large fuel savings that can be achieved by optimizing the vehicle operating parameters, a vital consideration in times of high oil prices and concern about GHG emissions.

Clearly, the time is right for automation, but it will not happen overnight. The technology for a fully autonomous mine must be developed, but it is unlikely that any one single company could take on the challenge alone. The disparate, independently developed pieces of the automation puzzle will need to be connected and synchronized. This will require the industry to adopt automation standards that allow this to happen. Even so, the cost of automating all of the functions in mining will be a lengthy and costly endeavor. If there is no sustainable competitive advantage from in-house developments, most mining companies will not want to incur the high costs and will prefer to buy technology from specialist suppliers. Mining companies will need to develop a method for overcoming this nexus because the provision of a complete turnkey automation package by a single supplier is unlikely to happen in a timely manner.

Underground mining, where the imperatives for change are much greater, was the first bastion to fall to equipment automation. Space is tight, the dangers are greater than surface mining, and health issues are of greater concern. Unmanned vehicles are now more common. Vision and guidance systems enable a remotely controlled vehicle to know precisely its location in a mine by comparing the camera view with stored images. Vision systems improve the ability of a remotely controlled vehicle to approach a rock pile and optimize the load collected. The combination of these semi-smart machines with effective communications infrastructure enables tele-remote operation of underground machinery by operators sitting in safe and benign office-like environments and allows machinery to be operated in areas where the dangers preclude human operation.

Vision for the Future

The mine of the future might include

- A mine site where automated blasthole drill rigs perfectly position every hole, conduct analysis during the drilling, and tell the explosives delivery vehicle what explosives load and blend to be charged to each hole;
- An excavator that can "see" the difference between ore and waste in the muckpile, can separate the two, and automatically load the driverless haul truck before dispatching it;
- Driverless trains fitted with an array of sensors that enable them to see beyond the horizon and that can travel in a convoy as though linked by some invisible thread;
- A haul truck that automatically reports to the workshop as scheduled maintenance becomes due; and
- A haul truck with none of the design constraints that come with having a driver—no cabin, windows, air conditioning, nor headlights; that is more symmetrical, possibly able to travel in two directions equally; and that comes with the current energy system and drive train—all-wheel drive and steering, electric motors driving each axle, power generators, and storage systems under body.

If these and other systems were put together, it is easy to imagine the mine of the future operating similar to a rock factory where all functions work in unison, more like a production mine than the variable mines seen today.

Automated Mine Site

In surface mining, "islands of automation" in haul trucks, blasthole drill rigs, shovels, surveying, and blasting are being developed. These independent developments must be integrated, which will multiply the benefits that would otherwise be achieved. Integration avoids unnecessary duplication of enabling systems such as navigation and provides operational standards and links all data sets. To avoid possible choking of the available bandwidth, developments in wireless communication are needed. Although individual pieces of equipment will need to become smarter to reduce the communications requirement, a central "brain" to conduct the disparate mining activities must be developed and implemented.

Automation will require the transfer and manipulation of huge amounts of data. Autonomous operations, such as drilling, surveying, blasting, and loading, will each link to the brain or autonomous backbone, which provides the coordination and sharing of resources that will be essential to the autonomous mine. The know-how to develop this backbone will likely be developed in-house by the mining companies in order to tailor it to the mining process. Perhaps in the future, as technology advances, it will be supplied as a turnkey system from original equipment manufacturers (OEMs).

A key capability of the backbone or brain will be the ability to effectively fuse the data from the disparate sources around a mine. Data fusion differs from data warehousing. Whereas data warehousing requires the storage and use of data to extract value, data fusion integrates data that offer a conflicting view of the world prior to the data being used. Data fusion is essential for a process that integrates and automates several functions.

An example of the need for data fusion is to precisely know the position of an autonomous moving vehicle in a mine. A Global Positioning System (GPS) provides a good indicator of a vehicle's position, but it is not fail-safe, so a backup is needed. Inertial navigation systems can provide information on position as can wheel encoders that measure the distance a vehicle has moved. A fast-moving vehicle such as a truck will likely have all three. To integrate these three sets of data and apply uncertainty theory to determine the most likely position of the vehicle, data fusion is required using algorithms. All of this data handling must be performed rapidly to ensure feedback to the vehicle and the autonomous brain controlling the array of resources in the mine. This is but one example of data fusion requirements in an autonomous system, and it heralds the future types of employees that mining companies will need to design and run information processes.

The experience from the development of an autonomous mine will impact future mine planning. For example, the precise control of haul truck movement may create an opportunity to build narrower and longer haul roads.

Technology Development

As discussed, the vision of a fully automated remotely controlled mine is deliverable but will take many years, substantial investment in research and development, and a broad collaborative network involving OEMs and leaders in automation. The creation of a fully automated mine could not be achieved by even the world's largest miner working in isolation. It will take the skills of large and patient companies to develop an autonomous haulage system. To deal with the robotics required in a fully automated mine, it requires the combined brainpower of large teams of dedicated research workers such as those employed at the Rio Tinto Centre for Mine Automation, based at the University of Sydney (Australia). Others will contribute to the development of advanced sensors. The proving ground for new technology is the mine itself. When all components are proven and the system is fully integrated, this template of the autonomous mine will be deployed. Components of the system, such as driverless trains, may be deployed earlier.

Driverless Train

Rio Tinto has announced that it will automate its iron ore railway in the Pilbara region of Western Australia. Within 5 years, driverless trains will be operating on most of the 1,300 km of track that serves its Pilbara operations. The cost is high. This will be the first time automation has been used in a heavy-haul railway of this scale, though the technology successfully operates on many metropolitan passenger railways around the world, where it is safe and reliable. Automated operations will integrate with the existing train management system and will bring efficiency gains through greater scheduling flexibility and the removal of delays. Additional safety systems are being developed to meet safety levels required for automated trains. Rio Tinto is working closely with the Western Australian Office of Rail Safety to ensure that all safety requirements are met.

Operations Center

Rio Tinto has established an operations center in Perth, Australia, to manage operations in the Pilbara mines, about 1,300 km away. This is a key step on the path toward a fully automated mine-to-port iron ore operation. At full operation, it will house hundreds of employees who will work with Pilbara-based colleagues to oversee, operate, and optimize the use of key assets and processes, including all mines, processing plants, the rail network, ports, and power plants. Operational planning and scheduling functions will also be based in the operations center, where staff will also manage power distribution and maintenance planning. Although the goal is a more efficient operation, an additional benefit of establishing an operations center within a capital city is that it will directly confront the high cost of basing employees at remote sites. This center is but one part, albeit a very important one, in a fully automated operation that includes driverless trains, autonomous trucks, and autonomous drills.

In mining, the traditional coal face is where many of the worst accidents happen and occupational illnesses are sown (Cribb 2008). An inestimable benefit of automation and remote operations is the improvement in human health, safety, and well-being as a result of moving people out of the danger zone. So although the absolute number of jobs might not change with automation, the overall safety performance of the company will improve as a direct result of worker displacement.

Computing Power

The mining industry has experienced significant growth in the utilization of computers since the mid-1980s due to widespread adoption of personal computers. For iron, aluminum, and copper mining, it is expected that the computing power required over the next 20 years will increase by an order of magnitude. The upgrading of personal computers across most sectors of the mining industry represents a major share of this growth. The remainder is driven by the needs of various applications that target improvements in productivity, cost, quality, safety, and reliability, including

- Mining and plant scheduling and optimization,
- GPS-based applications,
- Automation,
- Finite element analysis/simulation in plant design and troubleshooting, and
- Adaptive plant control based on predictive models.

Mine Workers

Automation may or may not mean fewer workers in the industry. It may be that, through automation, fewer workers are employed

at the mine site or mine output is doubled with the existing work force. Regardless of the impact at the mine site, specialist jobs in data processing, systems maintenance, electronics, and so forth will be created at locations possibly thousands of kilometers from the mine. These new workers will be housed in high-tech, air-conditioned offices or control rooms, a long way from the conditions experienced at a mine site. Mine operations in more politically sensitive regions may well be controlled by workers sitting in an operations center in a neighboring or distant country.

Automation and remote operations directly impact mine workers, and success in introducing change cannot be assumed. Much effort needs to go into planning, and communication is crucial. The work force must be prepared for such change through a well-planned cultural transformation process; if not, barriers to change will be erected. By being given relevant information, workers must come to understand that change is necessary for survival. At the same time, they must accept that the ways of the past, while good for their time, will not guarantee future prosperity. Finally, they must also understand and accept alternative ways and must embrace the process of change. Although the future of the industry or their employer may be important, to most workers, income stability is all that matters, so this must be addressed in any change process. Perhaps automation's most exciting potential, though, is its power to win a new generation of gifted youth to mining through the marvels of mechatronics and artificial intelligence (Cribb 2008).

While automation in the mining industry has been littered with many false starts, the challenges facing the industry today demand autonomous solutions. The rewards for being at the forefront of automation are great, but the penalties for inaction are far greater. Mine automation will take leadership, resources, good planning, cooperation between suppliers and users, and a lot of patience.

UNDERGROUND MINING

A number of large mining companies pursue a strategy of owning and operating large-scale world-class mines, typically in the form of large open pits. However, the depth at which open-pit mines can be developed is limited. Although larger and more efficient trucks will enable open pits to operate to greater depths, it is likely that the economics of strip ratio and large-scale waste management will be the prime control on depth. For example, it is anticipated that an increasing share of Rio Tinto's production, particularly in the copper and diamond groups, will come from underground operations (Clayton 2008) and that the majority of investment in the future will be in the form of large tonnage block cave mines.

The challenges of block caving include high capital costs; long lead times before revenue generation; and complexity in design, construction, and operation. These projects should be conceived of as rock "factories"—mines built to a specified quality and schedule—and then operated in a predictable manner in terms of production rate, grade, and costs.

Knowledge

The industry's block caving experience has driven a number of new development concepts, which are different from those for a more typical mine. However, the rate of development needs to increase rapidly. This change in concept requires a change in project definition, planning, and implementation. In particular, an early and deep understanding of ore-body (and waste rock) characteristics, design, and constructability are critical.

Improving resource and reserve knowledge can provide substantial competitive advantage. It is important to identify at an early stage those resources that fit the required extraction profile and are amenable to bulk mining. Ore-body knowledge is critical to the overall design and construction plan. Block caves require greater upfront ore-body knowledge, because the final extraction level needs to be planned in detail before construction can commence.

Design

Past block cave design has mainly been based on application to weaker rock masses than those proposed today and will be required in the future, and, as such, much design work is currently based on inappropriate rules and outdated experience. Current design methods in block caves are largely based on empirical techniques developed in the 1970s and 1980s, and more advanced techniques are still in their infancy. There is a clear need for a superior understanding of how a rock mass will cave and the characteristics of caved material, particularly the fragmentation. As the key driver of block cave mines, fragmentation determines bulking and rock flow characteristics that must be understood for optimal mine layout, infrastructure, and operational design. Fragmentation determines optimal drawpoint spacing, which, in turn, strongly influences recovery, dilution entry, and conditions.

Customized Development Design

Improved characterization of the rock mass through which the drift will be developed, via a more rigorous approach to site investigation and face mapping, will yield benefits. For example, ground support techniques have not evolved substantially since their inception in early 1970. Better design and products could reduce costs by 10%, saving many millions of dollars. Such savings could also be achieved in the other caving-type operations. In order to support the substantial levels of investment associated with block caves, functional and reliable design tools are required, which will result in more reliable cave designs.

Reliability in Constructability and Construction

Block caves require large initial capital investment before revenue is generated. As such, they are similar to civil construction projects such as road tunnels where revenues are not realized until the project is complete. The construction of three block caves with a capacity of 110 kt/d will require

- Approximately 16 shafts (8 to 10 m in diameter) 1,500 to 2,000 m deep with four to five in various stages of construction per year over 12 to 15 years, and
- Approximately 900 km of horizontal development over 12 to 15 years.

The quality of mine construction is critically important, as repairing and retrofitting the footprint after production starts is expensive and interferes with operations. Therefore two significant drivers are

1. Time to construct, related to time-cost of money; and
2. Quality of construction, related to operating availability and effectiveness.

Because of the long lead times to cash flow and the construction costs, time to construct the development is vital to a block cave. When projects miss their plan rates of development, this seriously impacts the overall project economics.

The importance of construction quality cannot be overlooked. Lack of attention to quality is a major contributor to slow production start-ups and ongoing operational issues. Quality is much more critical to block cave operations than to other underground operations because of the costs associated with retrofitting. It is 10 times more expensive to repair after the fact than to specify fit-for-purpose during design. More importantly, as repairs are undertaken, production delays are incurred.

If ore bodies are adequately defined and designed, and constructed to perform to plan, the reliability of production will almost certainly be greatly enhanced. Reliable production requires reliable systems and, importantly, automation. The construction to plan must include the ability to develop the mine to plan.

Construction of Underground Infrastructure

Traditionally, underground development has been regarded as an ongoing operating expense. The key driver was the unit cost, and advance rates tended to be a secondary consideration. This led to a general acceptance of rates that were below par and were substantially less than those achieved in the civil industry. Real mine data show that, although equipment technology has improved, performance has deteriorated. The value of a proposed block cave mine is heavily influenced by the speed, cost, and quality of the development work to put the mine in place. Currently, in these circumstances, the key driver is the advance rate of the primary access and critical infrastructure, while unit cost, although important, is secondary.

A major portion of future copper and diamond production will be from underground mines. These block cave mines require a significant portion of all development to be completed before production can commence. As a result, future production will require many kilometers of development each year over a 15-year period.

Today within the mining industry, a single end tunnel is typically advanced at an average rate of about 5 m/d, which has decreased threefold since the 1960s. Over the same time period, equipment performance has increased fivefold and cost per meter of tunnel has increased tenfold. Conversely, the civil tunneling industry has seen a steady increase in advance rates in recent years, and this begs the question as to why mining projects achieve 5 m or less while civil projects achieve 10 m/d.

Five major reasons contribute to this variance:

1. **Knowledge:** A substantial site investigation is undertaken prior to developing any civil tunnel.
2. **Planning:** Civil tunnels are planned in detail.
3. **Face size:** Larger faces in civil tunnels usually allow multi-tasking.
4. **Resources:** Civil projects are focused on developing tunnels, and more money is spent per meter of development in order to achieve schedule.
5. **Technology:** A system approach is applied that includes different equipment than the conventional mining drill-and-blast.

Future significant step-change improvement in the rate of construction of underground infrastructure will require the following initiatives:

- Speed and quality of underground infrastructure construction, including successful implementation of new mechanized excavation technologies and shaft logistics
- Development of innovative support system for different excavation systems and ground conditions
- Reliable prediction of rock behavior to properly select and implement construction technologies
- Use of smart approaches of working with the rock mass to minimize risks and uncertainties

Output Rates

The goal in mining is to achieve planned output rates in a safe and environmentally responsible way. With moves from open-pit to underground mining as one option for extending the life of a mine, or with a preference for underground mining because of its lower environmental impact, output targets will undoubtedly be influential. While this may, at first, seem unreasonable in view of the greater technical difficulties accompanying underground mining, output maintenance may be crucial to the viability of any mine extension project. The cost of developing a high-output underground mine as an extension of an existing open-pit mine may well be lower than the cost of finding and developing a new tier 1 reserve.

As mentioned, achieving economic output rates via block caving methods provide numerous challenges. The difficulty lies in operating sufficient drawpoints to create the required muck mass and having a materials handling system capable of moving that amount of rock. Here, the development work is all related to the mine plan and the layout of the production block. For example, preliminary plans for the Grasberg block cave in Indonesia (Brannon et al. 2008) suggest that 1,100 drawpoints are required to deliver an output of 160,000 t/d.

Planning

When planning an underground mine it is important to have detailed knowledge of the ore body, the ore grade, its mineralogy, its shape and dimensions, intrusions, and contamination. Knowing how a mine will behave during mining operations is fundamental. The conversion of an open-pit mine to a block cave mine adds even greater complexity because of the potential for pit failure and the dilution effects that come with ongoing deterioration of the pit wall. In addition, the extent of the underground mine network inevitably causes higher stresses that must be considered in the mine planning to ensure a successful transition from open pit to underground. The timing of the transition is not negotiable, because caving can cause instability in a pit, so all surface mining activities must cease before ore can be taken from a block cave mine. Such timing issues are considered in plans for two major transitions to block cave mines currently being investigated, namely the Grasberg (Indonesia) and Bingham Canyon copper mines.

The technology used in block cave mines is not new. What is new is the scale of the mines now being planned, which takes the industry into uncharted territory. For this reason, the planning process for the conversion of an open-pit mine to an underground mine is measured in decades rather than years. Improved modeling of the mine would deliver immeasurable savings in development costs, but to create such models, the learning from existing large-scale projects must first be captured.

Bingham Canyon

As an example, studies of Bingham Canyon (Brobst et al. 2008) and what option to choose (open pit, underground, or closure) when the current pit mining operations finish around

2019 provide an interesting insight into the time and effort needed to ensure that all possibilities are considered and the best option is chosen. The study timeline follows.

- 1997: study commenced.
- 2006: order-of-magnitude study complete.
- 2006: prefeasibility study commenced.
- 2009: prefeasibility study for expanded open pit due for completion.
- 2013: prefeasibility for block caving methods due for completion.
- 2019: current operations due to cease.

As well, the following tests have been conducted during the studies:

- 160 km of drilling
- 500 unconfined compressive strength tests
- 500 tensile strength tests
- 300 triaxial tests
- 250 direct shear tests

In parallel with this, more than 15,000 individual structures along 44 km of exposed bench in the pit have been measured and logged. This work provides knowledge of the ore body and surrounds and enables plans to be continually refined. One can only imagine the worth of having, at the outset, more detailed underground knowledge that might be delivered via an advanced, nonintrusive sensing process.

Operations

Studies have been conducted into drifting speed (Nord 2008) and the impact of tunnel cross-sectional size and shape, shot length, and the optimum timing of support activities versus activities at the mine face. This knowledge is of great value when linked to productivity and equipment utilization objectives.

The key to the future lies, firstly, in developing sensing technologies that will provide a better picture of the subsurface structures, and, secondly, in using advanced computer modeling (a) to predict the broader impact of mining an ore body and (b) to optimize all processes to achieve planned outputs at lowest cost. Because there is only one opportunity in developing and implementing a plan, the uncertainty must be removed during the planning process as much as possible. The only certainty is that the growing global demand for minerals will stimulate changes in underground mining methods, some of which will be predictable and some will not be foreseen.

ADVANCED PROCESSING

The science and practice of mineral processing have been and continue to be driven by the same internal and external pressures that have shaped other facets of the mining industry. At the forefront is strong global demand for virtually all minerals and metals, and this situation is set to continue.

Of greater relevance, given diminishing surface reserves, the industry is required to mine ever-deeper deposits and to process ores of lower quality and more complex mineralogy. This, together with increasing requirements for zero environmental emissions, reduced energy consumption, and sustainability, will require even more sophisticated processing methods.

However, underground mining is traditionally more energy intensive than surface mining. Deeper, lower-grade ore bodies will require more energy to mine and process. Larger quantities of gangue material need to be brought to the surface and then disposed. Against this background, and with higher energy costs and the need to reduce GHG emissions to combat global warming, efficiency improvements and less-energy-intensive processing technologies are essential. Automation, remote control, improved sensors, and real-time analysis will play a key role in mineral processing developments as they will in other mining operations.

Comminution and Energy Usage

Large amounts of energy are needed to crush and grind rock finely enough for subsequent separation of the minerals of interest. Comminution is the most energy-intensive activity in the current mineral concentration flow sheet, consuming around 30% to 50% of the total energy requirement. In plants required to grind a very hard ore (nominally Bond work index in the range of 15–25 kW·h/t) to finer liberation sizes, this requirement can be as high as 70% (Cohen 1983). In the broader perspective, it has been reported that comminution activities in the United States account for as much as 1.5% of U.S. total energy consumption (Charles and Gallagher 1982). In the context of typically quoted energy efficiencies of less than 5%, comminution is an obvious focus for improvement for tumbling mills that represent a majority of downstream size reduction.

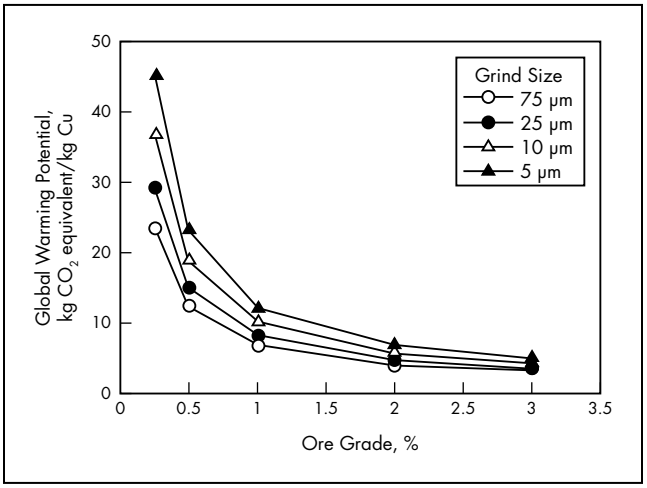
Compounding this situation are industry trends toward lower ore grades, which translate into even more intense comminution processing, hence even higher energy usage to recover the same quantity of mineral. As ore grade decreases, process energy requirements rise rapidly, even for the same liberation size (Figure 1.3-2).

However, the grind size is not a static target. In an effort to increase recoveries, today's grind size target is much finer than it was 50 years ago. At one time, a grind size for lead–zinc processing of <70 μm was regarded as fine, whereas today the grind size is more likely to be <7 μm. This is due to the requirement for subsequent processing, including froth flotation, where finer sizes result in increased recovery. Therefore, despite the development of more efficient grinding mills, there has still been a significant increase in the overall energy consumption.

It may well be that high-pressure grinding rolls (HPGRs) will become a key technology for hard-rock comminution, providing high capacity at lower energy intensity. Recent results (Anguelov et al. 2008; Michael 2007) suggest that replacing semiautogenous grinding mills with HPGRs in a circuit can reduce comminution energy requirements by about 25%.

Flotation and Larger Particles

Like comminution, flotation remains a key technology in mineral processing and one that has seen steady improvements over many years. Flotation performance is highly dependent on particle size. For best performance, a particle size in the range of 20 to 100 μm is required. Poor recovery of fine particles is typically associated with entrainment, whereas poor recovery of coarse particles is associated with inertial forces that prevent the large particles from being recovered. With increasing pressure to reduce the energy and costs associated with comminution, the desire to increase the particle size in flotation increases. Research will be needed to develop improved froth flotation processes that enable these coarser



Source: Norgate and Jahanshahi 2007. © CSIRO Australia 2006.
Figure 1.3-2 Relationship between ore grade and embodied energy

particles to be separated and to ensure that metal recovery is not compromised.

The trend is toward larger flotation cells to reduce capital and operating costs associated with flotation (Outotec 2007). Cell sizes have increased from around 50 m^3 in the early 1990s to 300 m^3 in 2007, and all signs are that this trend will continue. These large cells present challenges to adequate mixing and suspension of solids. Larger cells can require higher shear rates to maintain the solids in suspension, which can exasperate recovery of coarse particles. The use of computational fluid dynamics has become an essential tool for understanding the detailed performance and for designing devices to optimize flow profiles in the cell.

Because water is becoming a scarce resource in many regions, pressure is mounting to manage this resource more carefully. This will undoubtedly serve to stimulate process development wherever water is consumed in the mining industry. One likely emerging trend will be the so-called dry processing, where water is replaced by air as the separation media. For example, the rotary air classifier has an action similar to that of a conventional wet jig and has been successfully applied to gold ore processing (Piggott 2000). Another example of dry mineral processing is the rotary classifier developed by Australia's Commonwealth Scientific and Industrial Research Organisation (CSIRO 2009).

Mining and Reducing Materials Movement

Two other trends will affect mineral processing. Mechanical miners using rotating cutters have shown promise in rocks up to 200 MPa, but they produce a quite different size distribution than blasting and excavation. The use of mechanical cutters opens the possibility for sorting before final comminution, which would reduce energy usage significantly. An added benefit is that mechanical mining and excavation is more amenable to automation than conventional blasting and excavation.

New ore-sorting technologies will automatically sense and optimize conditions according to the composition of the host rock. This process will reject gangue minerals and hence significantly reduce the mass of rock required for processing to yield the same amount of product. New sorting technologies will

- Dramatically increase the ore grade before processing,
- Make low-grade ore deposits more economical to mine, and
- Reduce the comminution of gangue minerals. This will significantly reduce the energy consumption per metric ton of product and reduce quantity of tailings generated per metric ton of product, thus reducing associated environmental and community impacts.

New ore-sorting and grinding techniques in the future will enable ores to be processed underground, further reducing waste movement and potentially compounding the benefits already mentioned. Underground processing will require equipment that is smaller, lighter, and more mobile, possibly made from advanced composite materials.

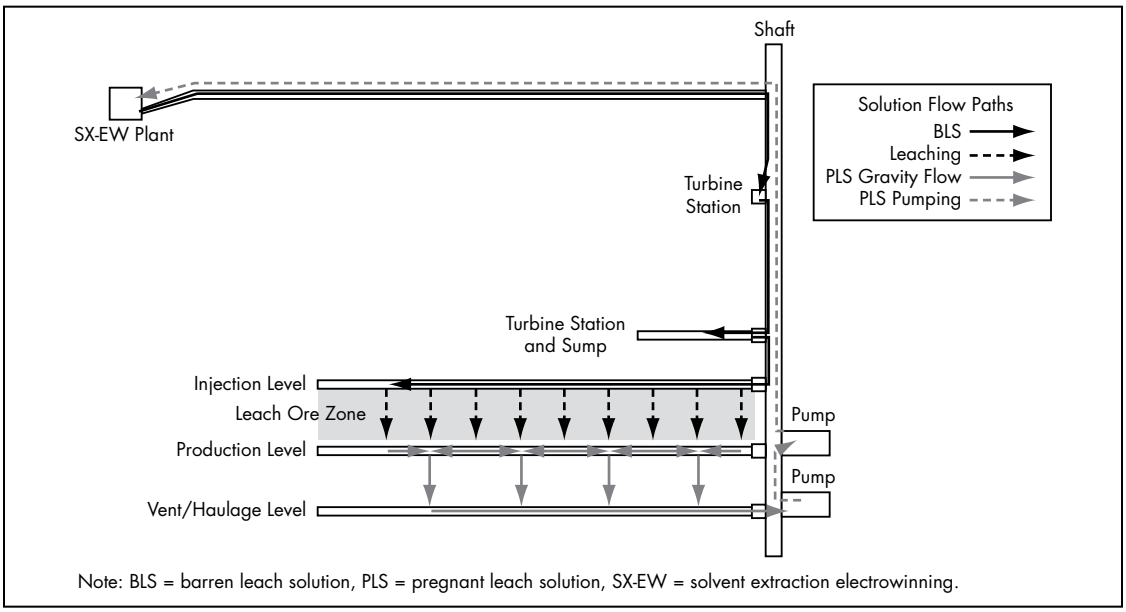
Heap and In-situ Leaching

The ultimate extension of reducing material movement is to leach the ore in the host rock (in place) and not take any waste material to the surface. This technology would be applied to deep ore bodies that are initially developed for caving using fully automated methods to ensure high health and safety standards. The mineralized material is leached in place using acids or solvents chosen according to the metal to be extracted. The dissolved metals are then pumped aboveground and extracted. The acid/solvent works in a closed loop, and the system would be designed in a way that prevents escape from the mining zone. A conceptual approach to in-situ leaching is shown in Figure 1.3-3. This method is expected to have much lower capital and operating costs and use significantly less energy. It would also allow for minerals to be extracted from harder to reach places and would eliminate the need for people to enter the mine altogether, dramatically increasing operational safety.

In-situ leaching is already used to extract water-soluble salts such as sylvite and halite. The application of commercial scale in-situ leaching to sedimentary uranium deposits has also been around since the 1960s. Effectively, the in-situ leach process leaves the ore in the ground and recovers minerals by pumping a leachate solution into boreholes drilled into the deposit; the pregnant solution from the dissolved minerals is then pumped to the surface. The key to successful leaching of uranium is the identification of suitable, below-water-table sedimentary deposits in which uranium is confined in permeable rock by impermeable layers.

In the future, it is expected that the uranium industry's experience will lead to technology developments to enable extraction of other metals—for example, copper—in this way. For copper, however, the nature of deposits poses a significant target, because a key requirement is for the ore body to be permeable to the liquids used. Because porphyry copper deposits have low permeability, future challenges include economic mine development and sufficient initial fragmentation, as well as subsurface control of the leach solution.

In-situ processes could potentially deliver the highest goal: a zero environmental footprint. They would enable land close to or even under cities or in environmentally sensitive areas to be mined without any adverse impact. In the case of copper, 99% of the rock mass is left intact and only the valuable material is transported to the surface.



Source: Rio Tinto 2003.

Figure 1.3-3 In-situ leaching

Leaching technology also lends itself to the extraction of minerals from heaps that, with low head grades, have previously been seen as uneconomic to process. Purpose built, fully automated plants would allow extraction rates and yields to be optimized. Solvents would be within closed loops, and heaps monitored and managed with advanced sensor systems.

Energy Supply

Energy issues discussed more fully elsewhere in this chapter apply equally here. Low-emission energy sources must be pursued and regenerative technologies utilized where possible. Of particular relevance for deep underground mines, geothermal energy sourced in situ may be used to power all mining processes.

In summary, the growth in demand for all minerals will continue for the foreseeable future. If the industry is to keep pace with this growth, improved mineral-processing techniques must be developed in parallel with improved mining processes. This demand, together with cost, sustainability, and skills issues combine to drive toward ever larger, automated mining operations. Mineral processing will be altered by the change in scale, particularly the use of ore sorting and advanced comminution technologies. However, the growing scarcity of new high-quality surface deposits is pushing the industry toward a greater dependence on underground mining. Here, underground sorting and comminution will reduce the energy consumed in transporting waste from the mine. Alternatively, in-situ leaching will lead to the elimination of waste movement and allow extraction to occur with almost zero environmental and community footprint.

SUSTAINABILITY AND ENERGY

According to the United Nations Brundtland Commission, sustainable development "meets the needs of the present without compromising the ability of future generations to meet their own needs" and covers a diversity of issues that continue to evolve (Skinner 2008).

Good management is managing a business with an embedded sustainability culture delivered through senior management commitment and documented strategies, procedures, and goals, with benefits far outweighing the costs (Skinner 2008). These benefits include:

- Reputation,
- Access to resources,
- Access to talent, and
- Access to capital.

Mining companies must put sustainable development at the forefront of their operations and future developments. They must work closely with host countries and communities, respecting their laws and customs. It is important that the environmental effects of their activities are kept to a minimum and that local communities benefit as much as possible from these operations through employment, capacity building, personal development, and poverty reduction (Lenegan 2007). Higher local employment reduces risk to the business. The mining industry often operates in remote locations, so it makes great business sense to increase the availability of local goods and services.

Society's expectations of mining companies include reducing the footprint of activities so that habitat and species conservation is compromised as little as possible. This means leaving as much natural variety in place after operations finish as existed before (Slaney 2008). The discipline and management tools that underpin sustainable development provide a mechanism for continually increasing efficiency and productivity in the business, generating long-term returns to shareholders. It is this willingness to think in terms of economic, social, and environmental sustainability that separates us from the past and gives us a pointer to the future.

Social License to Operate

Working closely with local communities and indigenous groups to understand and respond to their concerns and

aspirations develops the social license to operate that is essential to successfully developing and managing a long-term mining operation. Mutual respect depends on our understanding the issues important to our neighbors and on our neighbors' understanding what is important to us. Wherever we operate, we must do our best to accommodate the different cultures, lifestyles, heritage, and preferences of our neighbors, particularly in areas where industrial development is little known. Our community and environmental work is closely coordinated and takes account of peoples' perceptions of the effects and consequences of our activities. Obtaining the social license to operate is a key requirement for mining companies in the future.

Acid Rock Drainage and Waste Disposal

Determining the inherent acid rock drainage (ARD) potential of solid and liquid samples will continue to gain significance in the mining industry as more complex ore bodies are discovered and more complicated processing methods are employed. The key elements for future predictive classification based on ARD will include mineralogy (i.e., characterization of acid-generating vs. acid-neutralizing minerals), mineral surface analysis (i.e., availability of reactive mineral surfaces to water and/or the atmosphere), and mitigation strategies. Trade-offs between these elements will also be relevant throughout the life of the mine.

The potential effects of ARD will impact brownfield, but possibly even more so greenfield, exploration in the future. Indeed, exploration targets with high ARD potential or likely waste treatment issues may be classed as nonprospective. For example, it is well known that tailings from sulfide mineral extraction processes are likely to have high ARD generation capacity. Deposits of this type will therefore require active management in tropical environments but less so in less reactive terrestrial settings. For future exploration targets in reactive environments to be prospective therefore, demonstration of ARD prevention through treatment and disposal or by successful containment through the application of barrier technologies will be necessary.

Current minerals processing methods often include fine grinding, which generates fine-grained acid-generating minerals and leads to potential ARD issues. The large amounts of tailings, albeit of low grade, that are generated by these processes are also likely to become less environmentally acceptable. These environmental concerns will make in-situ processes to reduce mining waste and contain acid-generating minerals in-ground more attractive in the future.

Clearly, there are some obvious environmental advantages of in-situ mining, including a much smaller footprint, significantly reduced amounts of waste generated during mine life, simpler closure and rehabilitation procedures, more effective water treatment, and ARD prevention. Significant challenges, however, remain for in-situ processes to become a reality—preventing leakage of the leach liquor into the groundwater and isolating the system from water ingress through the development of suitable barriers, improving geomechanics to induce fragmentation, and controlling the chemical environment at depth.

The ARD potential of metalliferous deposits opens significant challenges for managing terrestrial waste rock dumps, including long-term geotechnical stability, long-term drainage and hydrology (particularly in high-rainfall areas), locating appropriate sites to place the waste dumps, and placement methods such as riverine and marine tailings.

Product Stewardship

Rio Tinto defines product stewardship as an action program that recognizes the need to ensure that products are produced, used, and managed at end of life in a socially and environmentally responsible manner in order to support societal goals of sustainable development and commercial goals of sustainable markets. A coherent and comprehensive approach to product stewardship contains the following elements:

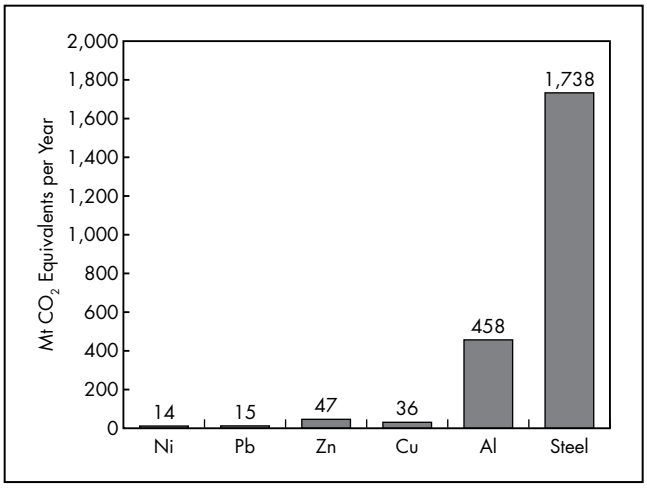
- **Life-cycle assessment:** using life-cycle methodologies to gain value and to understand the benefits and impacts of products along the full value chain and to explore shared responsibility beyond the production gate
- **Eco-efficiency:** ensuring that processes are as eco-efficient as possible and the mineral and metal resource is used wisely
- **Product disclosure:** disclosing information on product health and environmental effects as well as providing information on safe methods of handling and disposal
- **Customer/supplier engagement:** engaging with customers and suppliers to identify opportunities and to assess and manage risks. This knowledge can be used to better meet the needs of customers and (where the risks are justified) protect existing markets, grow sales, develop new markets, and also to leverage supply arrangements.
- **Market protection:** participating in scientific, regulatory, and political arenas to influence policy and regulation that have the potential to limit market access or restrict product uses in ways that constrain sustainable development
- **Research:** identification and filling data gaps on issues related to product and process health and environmental effects

In our changing world, new paradigms are emerging about the sale of products so that continued access to markets cannot be assumed. A timely and proactive strategy designed to address issues that threaten both our license to mine and market our new and existing products will be required. Additionally, it is evident that product stewardship will help identify and manage the safety, health, environmental, social, and economic risks and benefits of our products across the value chain (i.e., from mineral extraction to end of life).

Energy and Climate Change

Of greatest importance today is the issue of climate change and global warming, which will remain center stage for decades. Even with immediate action, it will take generations to reverse a trend that has been gathering strength since the industrial revolution. This global issue requires a decisive global response through coordinated local actions. While the developed countries must show leadership through action, there is no global solution without the participation of India and China.

Climate change will mean many things depending on the extent of global temperature rise and the extent to which the ice caps thaw, but for the mining industry, a reduction in available water and a greater incidence of extreme weather events could impact mine planning and operations.



Source: Norgate and Jahanshahi 2006. © CSIRO Australia 2006.

Figure 1.3-4 Global greenhouse gas emissions for selected metals

The activities of human beings and companies are contributing to climate change through GHG emissions, particularly carbon dioxide (CO_2) (Chiaro 2007). Most analysts predict that world energy demand will increase significantly over time and that coal will continue to be an important contributor in the energy mix, particularly as the emerging economies rely heavily on coal-fired generators. The International Energy Agency predicts that by 2030 world energy demand will increase by 66% and fossil fuels will remain the primary energy source.

Given growing energy demand, stabilizing CO_2 levels to 550 ppm in the atmosphere (to limit global temperature rise to 2°C) will require a significant reduction in GHG emissions. A "business as usual" approach to energy generation and consumption will likely result in accelerating and unsustainable levels of CO_2 in the atmosphere. Technology gaps must be filled by the development and adoption of low or zero CO_2 generation technologies and step-change improvements in energy efficiency.

Emissions from the mining, refining, and smelting of metals are a major contribution to global CO_2 emissions. Global emissions for a number of metals are shown in Figure 1.3-4 (Norgate and Jahanshahi 2006). Global trending of GHG emissions are not showing any signs of improvement, and climate-change challenges represent a significant threat to the global minerals industry. Because GHG emissions are dependent on the management of energy supply and use, energy audits are a useful tool for identifying energy-saving opportunities.

Energy Consumption in the Minerals Industry

Energy costs are a significant proportion of total cost inputs for the global mining industry. The Mining Association of Canada completed a series of mine benchmarking studies (Mining Association of Canada 2005a, 2005b) to determine the energy consumption for both surface and underground mining operations. Extracting the raw data from the Canadian studies and averaging them across mines and commodities, a general picture of energy consumption can be formed for different operations, as shown in Figure 1.3-5 (Batterham and Goodes 2007). Not surprisingly, underground mining operations are significantly more energy intensive. As surface ore deposits become more difficult to find and the requirement for more and deeper underground operations increases, energy consumption is therefore expected to increase.

A number of opportunities for reducing energy consumption are evident:

- New technologies for reducing grinding and comminution energy, switching from processes that are known to be inefficient
- More efficient conveyor and transportation systems, particularly in automated mines
- Improved ventilation systems for underground mining (totally automated underground mining operations could negate the need for any underground ventilation)
- Advances in mineral flotation and concentration
- Development of in-situ leaching processes for minimizing or eliminating the environmental footprint
- Improved on-line analysis to minimize the amount of gangue material that is processed
- Underground or in-pit sorting to reduce the amount of material moved

In addition, technologies aimed at reducing energy consumption in the production of metals, for example, iron and aluminum, will be critical.

The second crucial element in reducing GHG emissions relates to electricity supply. The emissions arising from the generation of electricity need to be minimized to reduce the overall footprint. A range of carbon-free alternatives exist today and a number of advanced energy technologies for electricity generation are being developed.

Existing alternate power generation technologies include

- Nuclear,
- Wind,
- Solar,
- Solar thermal, and
- Hydroelectric.

New power generation technologies include

- "Clean coal" technology involving carbon capture (as CO_2) and storage (CCS),
- Energy from advanced biotechnology and biomass,
- Geothermal power, and
- Hydrogen-based transportation and electricity systems.

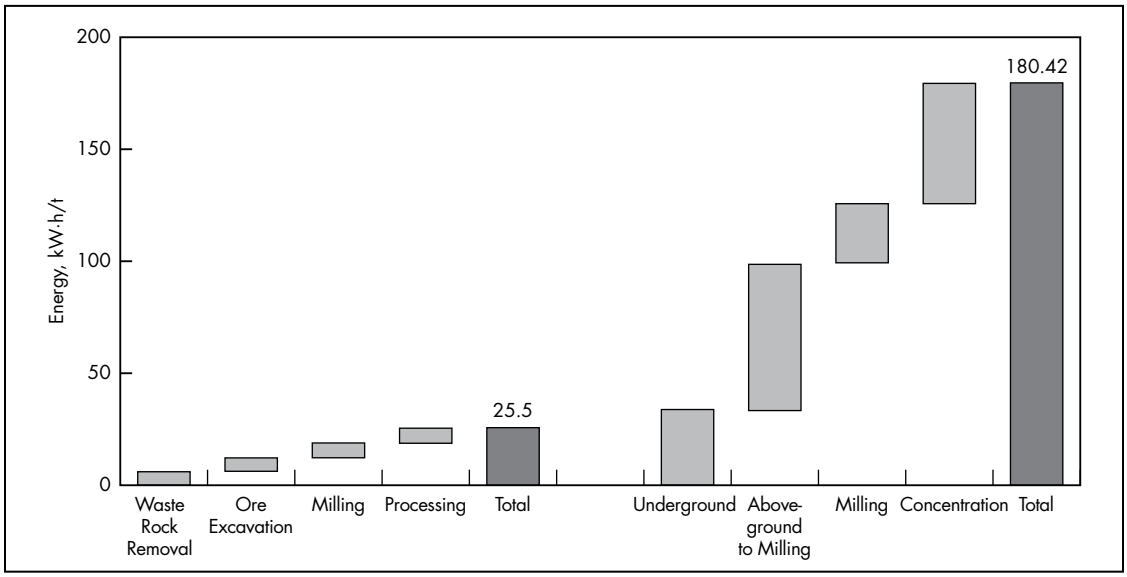
The application of these alternative energy technologies can have a significant impact on reducing GHG emissions. More remote mining operations operating on discrete electricity grids can employ a range of these energy types optimized for local circumstances. End-use technologies to improve energy efficiency and reduce energy demand must also be developed.

Examples of Emerging Energy Innovations

Many large mining companies are actively involved both in identifying and implementing short-term energy efficiency improvements and in developing step-change technologies to significantly reduce energy consumption. Both activities are critical. The following step-change energy opportunities are given as examples of industry developments.

Novel Comminution Approaches

Comminution energy efficiency is known to be low, and often less than 1% of the energy consumed goes into the breakage process. However, the understanding of comminution processes has markedly increased with recent computer



Source: Batterham and Goodes 2007.

Figure 1.3-5 Energy consumption in surface and underground mining operations

modeling. The results point to significant improvements being possible, in the order of 10%–15% energy consumption and corresponding increases in mill throughput.

One opportunity for improving energy efficiency lies in the pretreatment of the ore, with the aim of introducing microcracks into the system. The development of next-generation industrial microwave delivery systems offers a pathway to implement this approach. With these technologies, it may be possible to deliver microwaves to tonnage quantities of ore at economic rates.

HIsmelt Process

HIsmelt is a new technology developed by Rio Tinto to enable the direct smelting of fine iron ore and coal into molten iron. By avoiding the coking process, it offers significant technical and environmental advantages over existing iron-making techniques. Construction of the first 800,000-t/yr HIsmelt plant, owned by a joint venture company, was completed at Kwinana in Western Australia in 2005 and has been progressively commissioned since then. Iron ore from Western Australia and low-volatile coal are injected as fines into the molten bath of the smelt reduction vessel where they are directly smelted to molten iron.

Coal Initiatives

FutureGen is a U.S.-sponsored research project that aims to install a large gasification power station with integrated hydrogen production, carbon capture, and storage as a prototype demonstration of zero emission coal-fired technologies.

For amenable coal seams, underground coal gasification (UCG) is an energy technology likely to feature prominently in the future. By initiating and then controlling combustion autonomously within the coal seam, UCG produces a syngas (typically H_2 [hydrogen], CO_2 [carbon dioxide], CO [carbon monoxide], CH_4 [methane] and H_2S [hydrogen sulfide]-dry basis) that can be further refined aboveground to produce a relatively clean, affordable, and versatile source of energy. Although not a new concept, products from UCG compare favorably with alternatives in the context of today's more mature whole of life oriented assessment criteria, culminating in its recent rediscovery.

Clean coal initiatives are also being promoted by regionally based programs such as Coal21. This Australian-based program is a collaborative partnership between federal and state governments, the coal and electricity generation industries, the research community, and unions, that aims to promote and facilitate the demonstration, commercialization, and early uptake of clean coal technologies in Australia.

Hydrogen Energy

Hydrogen Energy is a joint venture between Rio Tinto and BP, created to further develop the hydrogen economy. Hydrogen-fueled power plants with CCS combine a number of existing technologies in a unique way to create low-carbon energy. It works by "decarbonizing" a primary fuel such as coal, oil, or natural gas. This decarbonization technique separates the hydrogen and captures the carbon from the fossil fuel as CO_2. The clean hydrogen is then burned in a specially modified gas turbine to produce clean electricity, and the CO_2 is stored securely deep underground in depleted oil and gas oil fields or natural saline formations. CCS technologies will be a key component in the fight against climate change.

Each of the component CCS technologies is proven and has been practiced within the oil and gas industries for decades. At this scale, their combination and integration is innovative, providing Hydrogen Energy with a real opportunity to generate large-scale clean electricity using existing fossil fuels.

Emissions trading is key to the development of a market-based carbon price that will help drive the lowest-cost pathway to a low-emission future. The target must be a zero net energy mine, and a technical pathway to achieve that objective can already be envisioned.

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PART 2

Market Economics

CHAPTER 2.1

Economics of the Minerals Industry

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Mining is like any other economic activity in that its contribution to the gross domestic product (GDP) is measured by its value added—that is, its worth based on the sum of wages, salaries, rent, royalties, direct taxes, interest payments, and gross profits involved in producing its final products. However, its modest share of the global gross product greatly understates its importance. Like other primary industries, it produces the raw materials that form the essential basis of all other economic activities. Mineral products perform the same function for the global economy as the vitamins and trace elements in the human diet. Without their continuing supply, economic activity would gradually wither and die.

BOUNDARIES

The minerals industry embraces a wide range of different products with differing methods and structures of production and with diverse markets. Its boundaries are conventionally defined to exclude petroleum, natural gas, and water, although hard-rock mineral fuels such as coal and uranium are included. Distinctions are also drawn between mining and quarrying for sand, gravel, and construction materials, although those share the characteristics of other mineral products. The downstream boundaries of the industry are imprecise as it merges, in many instances, into processing and manufacturing industry. That is especially true of some industrial minerals, particularly those used in chemical manufacture, but it also applies to metallic minerals.

Where the division is made is largely one of convention. The production of pig iron and steel, for example, is regarded as manufacturing, so the boundary with mining is the output of iron ore. For copper the division is less clear-cut, with some mines producing copper metal on-site and others selling ores and concentrates. Conventionally primary nonferrous metals are grouped as mineral products, but their semifabricated shapes and wrought products are grouped as manufactures. That raises questions about a metal like aluminum, for which the major input is not the mineral raw material but energy.

Mines produce ores rather than finished salable products. With few exceptions, most run-of-mine output requires some degree of processing before it can be sold. In some instances that may be no more than simple washing and sorting, but, at the other extreme, complex metallurgical or chemical processing may be necessary. Mineral producers aim to maximize the profitability of their operations and will balance the costs of additional processing against any advantages gained from selling higher-value products.

DEMAND

Few mineral products are demanded for their intrinsic worth but instead for their various properties. Most are intermediate goods used as raw materials or processing aids for more finished products. The main exceptions are precious gems and precious metals in some but not all of their uses. Whereas each mineral has a specific set of properties, whether physical or chemical, few of those properties are unique to any one mineral product. End users of the finished product may be completely unaware of what minerals are used in its manufacture and are merely interested in whether the product meets their needs satisfactorily. In short, demand for minerals is derived from demand for finished products, and in many instances the minerals used account for a relatively small proportion of the product's total cost.

This derived nature of demand means that it is difficult for producers of many mineral products to differentiate their material from that of other suppliers, except concerning its inherent properties. While the properties of a specific mineral product may make it ideally suited for some end uses, most uses will normally have substitutes of varying degrees of effectiveness. The fertilizer minerals are a partial exception in that regard, because growing plants require potash, phosphate, and nitrogen in varying combinations, but even the consumption of fertilizers is dictated by the demand for agricultural products and by farmers' cost structures. Demand for mineral products is always at the mercy of fashion, of technological changes in end-use markets, of governmental regulations, and, above all, of relative prices. These characteristics are not unique to mineral products but set them apart from the general run of manufactures.

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Substitution and Prices

That substitutes exist for most mineral products in many of their uses circumscribes their producers' market power. A marked shift in a product's relative price will impact the size of its market. Some substitution may be straightforward, as in the use of different minerals as fillers or extenders in the plastics industry, or of different materials in the manufacture of refractories. Sometimes the most effective substitutes are abstinence, as in a farmer's decision not to apply phosphate to his fields one year, or a change in processing technology, such as heat treatment to replace the addition of alloying elements. It will often involve process adaptations or investment in new technology and equipment. In those cases substitution will be spread over a period, rather than instantaneously, but when it has occurred, it is unlikely to be reversed quickly. Changes in relative prices often drive irreversible technological change.

Often demand for a specific mineral product will be affected not by changes in its own relative price but by the prices of the products in which it is incorporated or used in processing. For example, the demand for fillers and coating clays is driven by changing demands for paper, and for zinc in galvanizing by markets for steel products. Changes in end uses and in consumers' spending patterns, driven by demographic and technical change, and sometimes merely by fashion, can open up new markets for individual minerals and wipe out existing markets.

Markets for Mineral Products

Most types of economic activity, including trade and services, depend on mineral products, but their use is biased toward the goods-producing sectors. Demand for many mineral products, and for metals in particular, is heavily reliant on construction, capital goods industries, and the manufacture of vehicles and consumer durable goods. Because these markets are responsive to changes in expectations and to movements in interest rates, they are typically the most volatile segments of overall expenditure. Even where mineral products are used for processing or in goods for immediate consumption, their demand can fluctuate by far more than overall economic activity and is at the mercy of variations in work in progress and in the holding of inventories of all types throughout the production chain—from mine to final consumer. In general, therefore, demand for mineral products is more volatile than economic activity, as measured by changes in industrial production or GDP.

Demand, Incomes, and Intensity of Use

Typically an economy's demand for goods rises with per capita incomes. When these are near subsistence levels, and only the most basic needs can be satisfied, the usage of minerals per unit of output tends to be low. Initially, many of the needs for mineral products will be met by imports of finished goods rather than by domestic manufacture. As per capita incomes rise, the demand for more sophisticated products will also expand, thereby raising the economy's intensity of mineral use per unit of output. Rapidly growing demands for investment in modern infrastructure of all types will further boost the intensity of use (IU). The demand for mineral products, and especially for the building blocks of steel, cement, and construction materials, will tend to increase far more rapidly than the GDP. Eventually, the nation's basic infrastructure will be developed and individuals will devote an ever increasing proportion of their growing incomes to services of all types rather than to products. The goods that are purchased will tend to be less materials intensive than those purchased at lower levels of income. Consequently, the economy's IU of mineral product per unit of GDP will eventually peak and then fall back. Although its absolute demand for mineral products may continue growing with expanding per capita incomes, a point may be reached where even that ceases to rise and even weakens.

Smaller economies with limited populations and/or inherent geographical disadvantages may never develop their own materials-based manufacturing industries but continue to rely on imports. This means their IU of many mineral products will never rise substantially, regardless of the levels of their per capita incomes. By contrast, some economies may develop large export-oriented manufacturing capabilities so that their apparent IU not only rises rapidly but also reaches atypically high levels. Densely populated countries with limited land area will have different consumption patterns than more sparsely populated bigger countries, even at similar levels of per capita income. In short, each country's IU of mineral products depends on far more than per capita incomes alone, but those provide the main impetus for change.

The basic pattern of changing intensities over time as incomes rise can be traced in the historical experience of the mature industrial economies and is still being written in the newly industrializing and developing countries. Continuing technological change, both in products and production processes, and shifts in relative prices, especially of energy, mean that newly industrializing countries do not slavishly follow the path of their predecessors. Each country will naturally adopt the cheapest and most effective set of technological options available, which will probably, but not necessarily, be less materials intensive. Global demand reflects both the sum total of the component economies' varying patterns of IU and their GDP rates of growth. To the extent that some of the final output of the rapidly growing economies, such as China and India, is based on exports to the mature economies, their rising demands for materials will be partly at the expense of the mature economies rather than merely additional to them.

Demand, Supply, and Prices

Since production is relatively slow to react to swings in demand and tends to be more stable in the short run, supply and demand are seldom in precise balance. There is normally a dynamic equilibrium over time, with capacity moving in step with demand, but market balances can quickly veer from shortage to excess with little warning. The consequence is volatile prices for most mineral products. In some instances, a buildup or reduction of inventories can cushion the immediate impact, but many products are not amenable to large-scale stockpiling. Producers are therefore forced to reduce their output when demand falls, but they can seldom react quickly enough when demand surges. Even the mere existence of accumulated inventories can adversely affect expectations and depress prices.

Demand and Recycling

Quite apart from changes in inventories, supplies of many mineral products are not met by newly mined material. Even some of the minerals used as processing aids may be recyclable, although most are dissipated in use. Where mineral products are processed into capital goods and consumer durables they become part of the global capital stock and potentially available for recycling when the goods are scrapped. Products

used in packaging can also be recycled. The extent to which recycling supplements primary production depends on the nature of the original end uses, the lives of the finished goods, the available technology, and governmental regulation. It is also governed by the rate of demand growth. Even in mature markets, recycling alone cannot meet new demand, but its ability to satisfy part of that demand does constrain the commercial freedom of the mining industry. Much recycled material will be supplied for environmental reasons as a result of governmental regulation or fiscal incentives. Its supply is thus fairly insensitive to price, a contrast to the supply of newly mined material.

SUPPLY

Attention is normally focused on minerals supply, and in particular on its nonrenewable nature and site specificity. These are extremely important but over a rather longer time span than the immediate characteristics of supply and demand. It is often overlooked that the mining industry exists solely because there is a demand for its products rather than because ore deposits exist. Unless there are potential end uses for the products of their exploitation, ore deposits would be not much more than interesting geological phenomena.

Geographical Location

It is a truism that viable deposits of most minerals are not scattered evenly across the globe but that their geographical distribution is uneven. The lower a mineral element's crustal abundance, and the higher the concentration factor required for economic mining, the scarcer and less evenly spread geographically it becomes. In sharp contrast to manufacturing and to mineral processing, the location of mines is dictated by a combination of geology and geography. Yet the contrast can be overstated because the existence of known ore deposits does not necessarily guarantee that they will be mined. That will depend on a wide range of technical, economic, social, environmental, and political factors.

Mining is not the only type of economic activity whose location is geographically predetermined. Others include, for example, certain forms of agriculture, the generation of hydroelectric power, and even tourism. It is, however, unquestionable that mining can only occur where mineral deposits exist, often in places that are remote and inhospitable. Historically, mining was regarded in nearly all locations as having a prior claim on land use and taking precedence over alternative uses, almost regardless of where the mineral deposits were located. That is increasingly less likely today, with mining having to compete with other potential uses. This applies not just near urban centers in densely populated countries but also in populated regions where the preservation of the natural landscape may take precedence over any form of development.

All forms of mining involve some, often irreversible, disturbance to the physical landscape. When today's advanced economy countries were industrializing and needed minerals both as raw materials for industry and to create infrastructure, such disturbance was accepted as an unavoidable consequence of accessing mineral wealth. Moreover, mines tended to be relatively small and not too obtrusive on the landscape. Metallic minerals were largely extracted from underground workings, which produced limited amounts of waste. By contrast, much modern mining is carried out on a large scale, often through open-pit workings that both scar the landscape and create substantial volumes of waste rock.

Ore and Waste

The product of all mines is ore. Even where it is mined underground from rich veins or seams, its extraction will necessarily require the removal of waste rock. Historically, miners followed the often narrow veins using muscle power to extract the ore with pick and shovel, and the amount of waste was kept to an absolute minimum. With the introduction of ever more complex machinery and equipment, mines produce a greater proportion of waste. The underground workings now have to be large enough to provide access for and to accommodate modern equipment. Open-pit mines have to remove any overburden and sufficient surrounding rock to ensure the stability of the pit walls. The deeper the mine, the greater the volume of waste removed.

In most instances, the run-of-mine product requires treatment and upgrading before it can be profitably sold. Even coal may require washing. Most industrial minerals are subjected to a variety of physical or chemical processes in order to maximize their market value. Metallic ores usually contain only modest proportions of commercially valuable products whose extraction generates large amounts of tailings. Today a typical mine covers a much larger land area than it did in the 19th or early 20th century, and it alters the landscape irreversibly. That makes the mine a much less desirable neighbor than in earlier times, to the extent that local communities are often unwilling to trade the environmental and social costs for the benefits. Even so, mining of all forms accounts for a modest proportion of the earth's surface.

Depletion and Transport

The mining industry's substantial need for supplies of energy, power, and water—both in the extraction of raw ore and in its subsequent processing—partly governs its location. These requirements certainly help dictate the form in which mineral products are transported and the nature and extent of upgrading on the mine site. So too, crucially, does the availability of transport. Indeed, there has been an almost symbiotic relationship between the development of transport systems and the mining industry.

That relationship has been fostered by the main characteristic of the mining industry: its dependence on the extraction of nonrenewable resources. Individual ore deposits are strictly finite, either because they are physically bounded or because the costs of extracting the remaining ore become prohibitive. Naturally, any society will first exploit the most accessible and easily worked ore deposits. As these become worked out, it will develop technologies to extract more ore from the existing workings, whether by going deeper or by process innovations. These productivity improvements will be accompanied by a search for new ore deposits, which will probably be increasingly distant from population centers and their markets. Hence, the imperative is to improve transport links and reduce transport costs in order to enable the development of more distant mines at acceptable costs.

Mineral deposits will only be exploited if the mines based on them can profitably ship their products to the marketplace. With relatively few exceptions, mines are primarily developed with the expectation of earning profits, over and above their operators' capital costs. If they are persistently unprofitable, they will close down, even if the underlying mineral deposits are not depleted. The emphasis is on *persistently*, as explained later. The cost of transport relative to the price of the final product dictates the geographical reach of each mine's market

and strongly influences the nature and degree of processing at the mine site. Those products with high value-to-volume ratios, such as gemstones and precious metals, can be transported long distances easily and will have global markets. At the other extreme, products with high volume-to-value ratios, such as sand and gravel and construction materials, will secure only local markets. Marked reductions in the costs of deep-sea shipping since World War II have widened the geographical markets for bulk products, such as iron ore and coal, from the national to the regional and even to the global.

Where mineral deposits are located deeply inland, remote from major markets for their products, they can only compete if their products are upgraded locally in order to minimize transport costs. Such upgrading will, in turn, rely on the proximity of sufficient competitive sources of energy. Whereas Chile's copper mines are fortunate in being close to the sea, which can be used to profitably ship concentrates, those of central Africa depend on local smelting and refining to minimize the volumes being transported to ports.

Competitive Influences

A mine's ability to compete mainly depends not on relative transport costs, important though those are, but on the characteristics of its mineral deposit, on the mining method used, and on the nature of its processing plant. These inherent influences will inevitably be modified by the political, social, and economic conditions of the host countries, including such factors as wage inflation, energy costs, and exchange rates, which are outside the mining company's control. Such factors tend to be relatively more important for manufacturing industries, including mineral processing plants such as smelters and refineries, than for mines.

The nature of each ore deposit normally dominates the inherent influences on relative costs. Since few mineral deposits are identical in all respects, their costs, and hence their profitability, will tend to vary. Some will be close to ample supplies of water and energy, and easily accessible, while others will be more remote. Some may outcrop at or near the surface and be amenable to surface mining through open-cast methods, whereas others may be so deep as to require extraction through shafts or adits. The nature of the host rocks and of the minerals themselves will vary from the easily worked to the physically demanding. In some instances, the commercially valuable components may be easily liberated, whereas in others, complex physical and chemical processing may be needed. Perhaps the major factor is the grade of a deposit—the proportion of salable materials it contains. Other things being equal, which they seldom are, the higher the grade of a mineral deposit, the lower its relative costs compared with a similar but lower-grade deposit.

For many raw minerals, the average grade is less important than the extent to which a mine can maximize its output of the higher-priced types of the basic product without expensive processing. Typically the higher-value uses offer sizable premiums over the more common uses. Metallic ores are seldom pure, but they contain a variety of different elements and compounds. Obviously, the higher the content of the main metal, the more attractive the deposit, but the nature and importance of co- and by-products are also relevant. Some are highly desirable and raise the potential value of the ore, whereas others are toxic and reduce the marketability of the mine's products. Sometimes the costs of their removal and safe storage outweigh any mining benefits. Where mines yield a range of by-products, their production decisions can become complex, especially where, as is usually the case, the grades of the constituent elements vary throughout the ore body. Sometimes those decisions may appear perverse, as when maximizing profitability may involve concentrating on by-product recovery at the expense of the main product's output, irrespective of shifts in its demand and prices. This merely emphasizes that the underlying objective of mine managements is usually creating value for shareholders rather than ensuring supplies of raw materials.

Ore Grades and Exploration

The relative grades of ore of different deposits, even allowing for the contributions of all the salable products, are not necessarily reflected in the relative costs per unit of product. The higher the grade, the more a deposit is able to support deep underground mining, which usually costs far more per ton of ore than open-pit mining. Also higher grades may be able to offset the additional costs of complex treatment and processing.

In theory, the higher-grade, more easily processed, and more accessible deposits are mined first. Certainly, the average grades mined of some metallic ores such as copper have tended to decline over time, being substantially lower today than in the 19th century. The tendency toward declining grades has in many cases been countered by falling transport costs and by improvements in the technology both of extraction and processing and of exploration. The average iron content of ores mined in Western Australia and Brazil today is much higher than the typical grades of iron minerals earlier processed in most of Western Europe. The uranium deposits of Ontario exploited in the 1950s had much lower average grades than the latest generation of Canadian mines.

Ore Depletion and Technological Change

In essence, each generation will tend to exploit the best deposits of which they are aware with the technologies at their disposal. As geological knowledge increases, both through a better understanding of the nature and genesis of mineral deposits and through continued exploration with ever more sophisticated techniques, the number of known mineral deposits also rises. Some may have higher ore grades than the deposits already being worked but in many cases will suffer from offsetting disadvantages, such as remoteness from markets or poor accessibility. Simultaneously, the technology of ore extraction and processing is continuously changing and enabling the economic development of previously sterile mineral deposits. Such technological changes can be driven both by end-use requirements and by a need to remove existing constraints and bottlenecks in supply.

Over time, changes in the costs of producing mineral products result from a continuing tug of war between the cost-raising effects of ore depletion and the cost-reducing impact of technological change and rising productivity. That leaves aside such general economic influences as movements in general price levels, wage rates, energy costs, and exchange rates. The result is never a foregone conclusion. There have been periods where the forces of depletion have prevailed and inflation-adjusted costs of production have tended to rise, as in the 1950s and 1960s. There have also been lengthy periods, as in the 1980s and 1990s, where those forces have been outweighed by productivity improvements and real costs have tended to fall. That the real prices of most mineral products have not shown any marked trends over long periods suggests

that the opposing forces have been fairly evenly balanced. In that regard, however, the past is not necessarily a good guide to the future, if only because there have been wide and prolonged fluctuations around the long-term averages. While all mineral products are subject to some common influences, the historical behavior of their prices has differed widely, reflecting both their different end uses and the varying influences on the nature and locations of their production.

Capital Expenditure Requirements

The industry's requirements for capital spending vary with the location and nature of each mineral deposit. In addition to construction of the mine itself and of the associated processing plant necessary to produce a marketable product, usually an associated infrastructure is needed. This will naturally be less when the mine is situated near an established town or an existing mining district, with existing supplies of water and power and established transport links, than when located in virgin territory. Even in the former instance, existing facilities will probably need supplementing and upgrading. Although mining may not be the most capital intensive of industries, it is one of the leaders.

The lead times before a mine produces any income can be considerable, especially if full account is taken of the period from initial exploration to commercial production. Even the lead time from first discovery to a decision to invest will normally exceed a decade for a large mineral deposit. The expenses of prefeasibility and feasibility studies in fully delineating the deposit, developing a viable extraction and treatment process, and carrying out the requisite environmental assessments will normally be capitalized. Construction itself is likely to be spread over several years, and it normally takes about 2 years from start-up for a large mine to reach and sustain its design capacity. There has, however, been a near-universal tendency for the scale of mines to increase over time.

Unlike many productive enterprises, the mining industry not only has a heavy burden of capital expenditure before starting production but also large continuing needs over the life of its mines. This is an inevitable feature of the depleting nature of mineral deposits. In addition to normal capital spending on replacement equipment and maintenance, the industry has to meet the capital costs involved in extracting ore from increasing depths and more remote sections of the deposit, and in maintaining production in the face of declining ore grades. Without such continuing capital expenditure, an individual mine's output would soon tail off and decline.

So, too, replacement investment is needed in each mineral-producing sector. Although mine closures are likely to be bunched in periods of weak market conditions, some take place even when markets are buoyant. When total demand is static or even falling, new capacity is needed to offset losses from mine closures and from reductions in the output of some continuing operations. As the overall demand for most mineral products rises over time, the gross annual additions to capacity normally need to exceed the net additions—and in many instances by a substantial margin. The balance depends on the typical life and size of mines, which are a function of the nature of the underlying deposits, and on the rate of demand growth.

Additional capacity can come from expansions of existing facilities on known mineral deposits or their extensions, or from the exploitation of previously undeveloped deposits. Usually mining companies only prove sufficient ore to justify their initial capital expenditure. As mining progresses and the operators obtain a better understanding of their ore deposits' characteristics, they will often delineate additional material that can justify mine expansion. The expansion of existing mines on known ore deposits usually accounts for a substantial share of annual changes in net capacity. At one extreme, such expansions may involve no new capital spending, perhaps by the use of different explosives in the mine or a new reagent in the processing plant, while, at the other extreme, substantial capital expenditure is needed to deepen or extend the mine. A common feature is the introduction of the latest available technology, allowing improved productivity and offsetting underlying cost increases.

Economies of Scale

As mines expand and become larger, they can exploit economies of scale. The larger the mine, the more it can reduce its fixed costs per unit by spreading them over an increased output. It can justify investment in larger items of equipment, such as shovels and trucks, or in more capital-intensive mining methods than smaller mines. It can also support the mining of much-lower-grade ores than smaller operations. Indeed, it is only by operating on an ever-increasing scale that the mining industry has been able to offset the costs of extracting and processing leaner ores in more remote locations. From time immemorial there has been an inexorable trend toward more capital-intensive and larger-scale mining methods. This trend accelerated across the board from the late 1980s, resulting in reductions in the number of mines for each product and a strong rise in their average size. Open-pit mines tend to operate on a much larger scale than underground mines, and their relative expansion was favored for many years because of their ability to exploit technical economies of scale to the utmost.

That ability was enhanced by the growth of debt-based project finance in the post-war decades. Historically, the risks involved in mining inhibited companies from relying unduly on debt finance. That necessarily meant a dependence on equity markets and internally generated funds that tended to constrain companies' abilities to optimize the scale of their mines. The scope of the local market was often a further limitation when transport costs acted as a constraint. Thus mines tended to start off relatively small and expand when market conditions and finances permitted. In the past two decades, however, the availability of project finance on a large scale and the existence of a global market enabled companies to develop at a technically optimum scale from the outset.

Increasing scale is a mixed blessing. The larger the mine, especially if open pit, the greater its environmental and social impact on the surrounding neighborhood. Moreover, the larger mines' needs to minimize their fixed costs per unit of output reduce their flexibility to respond to changing market conditions. That can potentially lead to a greater volatility of prices than in markets with a larger number of smaller operations.

In practice, most mines tend to maximize their throughput during periods of low prices, often by raising the average grade of ore mined. If mining companies aim to maximize the net present value (NPV) of their ore deposits, they might logically reduce their cutoff grades when prices weaken, but their objectives are more complex. They also need to watch the profitability of their capital investment, which is different from the implied value of the mineral deposit. Corporate survival will tend to take precedence over theoretical maximization of NPV.

Closures and Cutbacks

Mines tend to remain in production as long as possible, even when prices fail to cover their costs. Experience shows that downturns do not last indefinitely, and companies hang on in the hope of improved conditions, as long as variable costs are covered and some contribution is made to fixed costs. Banks may support a loss-making mine's continued operation to ensure the eventual repayment of their loans. In many cases, the costs of closure may exceed the costs of continued production because of the need for rehabilitating the site. The continuing expenses of care and maintenance need to be taken into account. If mines are crucial contributors to the local economy, governments may be prepared to underwrite their losses to maintain a district's economic and social stability. In some cases, the mine's product may be essential for downstream operations, and the need for a continued supply of raw materials counterbalances any cash losses.

The rigidity of production in the face of falling prices is greatest in terminal markets of last resort or where the costs of stock building are low relative to the value of output. By-products have to be stored in one form or another when their market conditions weaken differentially to those of the main product. Thus the sour gas producers of Canada have periodically vatted large quantities of by-product sulfur, whose removal is essential for the sale of gas. The costs of mining and stockpiling most industrial and bulk minerals are prohibitive, and their mines will normally have to reduce their output when demand falls and prices weaken. By contrast, producers of products with terminal markets, namely the major nonferrous and precious metals, can always sell their entire output as long as they are prepared to accept the going prices. Even when users are unable or unwilling to purchase, they can place their product in warehouses.

Pricing and Barriers to Entry

The ease or difficulty of building up inventories is only one of the factors explaining the pricing of mineral products. The most important are the responsiveness of supply to changing demand and prices, and the level and nature of barriers to entry. These factors are closely intertwined. Where the barriers are high, the existing producers will enjoy some monopoly power and will be able to influence prices. In the short term, that power will be circumscribed only by the availability and effectiveness of substitutes, including recycled materials; by any countervailing power of purchasers; and by the extent to which the existing producers share similar objectives and philosophies. In the longer term, the existence of apparently attractive profits will attract new entrants, whose successful penetration of the market will depend on the causes of high barriers to entry. Where the barriers to the entry of new suppliers are low and easily surmountable, markets will tend to be competitive, with prices more closely reflecting the interplay of supply and demand.

Easily the most important constraint on new production is the availability of untapped mineral deposits. That is not just a function of a mineral's crustal abundance, vital though that is, but also depends on the number and nature of the potentially mineable deposits. The crustal abundance of platinum is slightly higher than that of gold, but the commercial deposits are few and geographically concentrated, whereas deposits of gold are much more widely dispersed. Therefore, platinum's barriers to entry are considerably higher than those of gold. Even where unexploited mineral deposits are known to exist, they may not be commercially viable with the existing technology. In some instances, the possession of proprietary technology by the existing producers sets a high hurdle for new entrants. Elsewhere the limited size of the total market and its concentration may restrict the scope for new suppliers to secure sufficient sales to justify investment.

Most of the barriers except geological shortage can eventually be surmounted or circumvented. Even that encourages exploration, which may result in the discovery of new deposits that, in some instances, may be superior in many respects to the deposits already being mined. The existence of processing constraints fosters technical research and development that often results in less expensive or more efficient processes, thereby enabling the new entrant to leapfrog over the constraint. Even the small size of an existing market is not an absolute barrier for innovative new entrants that search out and develop new uses. In order to keep prices in check, most purchasers will be keen to encourage additional suppliers. Users naturally favor a degree of excess capacity among their suppliers. Over time the role of falling transport costs in lowering the locational barriers to entry and expanding the geographical reach of markets has greatly reduced the abilities of local and regional suppliers to control their prices and market conditions.

Frequently, artificial impediments to new entrants can be as effective as technical limitations. For national markets these include subsidies and trade barriers of all types. Shielded from the full ravages of foreign competition, local suppliers can wield considerable control over their local markets. Most tariffs and similar obstacles to trade in mineral products have been eliminated or greatly reduced over the years in successive rounds of international tariff negotiations. Nevertheless, some tariff barriers still remain, mainly affecting countries that want to do more than simply extract and export their minerals without further processing. Imports of basic mineral products may be duty free, but there may be seemingly modest tariffs on downstream products, including semimanufactures. In such instances, there may be fairly heavy effective protection on the value added in the downstream products.

POLITICAL FACTORS

Trade barriers aside, most administrative and political fetters on supply are in mineral-rich rather than importing countries. Even where many known mineral deposits are awaiting development, and seemingly able to produce profitably with existing technology, their exploitation depends on a benign, or at least neutral, social, political, and economic environment. In many mineral-rich countries, the environment is anything but benign. As noted earlier, mineral enterprises commit considerable resources to developing mines over many years before commercial production starts. The prospective lives of their investments stretch many more years from start-up. At each stage, the technical and economic risks are high, with an ever-present possibility that the investment will fail. After the investor has committed resources to developing a deposit, it is a captive hostage to fortune. The company's capital is well and truly sunk, and it is impossible to transfer the investment to a more accommodating environment. Thus a mining company's bargaining power with the host country weakens considerably after its capital is invested.

This change in relative bargaining strength is not necessarily reflected in any change in the terms under which a

mine is developed. It does, however, focus exploration and mine development on countries with relatively stable political, social, legal, and economic frameworks. Where these essential preconditions are not met, many companies will tend to shun investment, no matter how geologically prospective the country. Investors in riskier countries will seek commensurately high returns and will tend to concentrate on developing mines with limited needs for expensive infrastructure and with rapid payback periods.

By no means are all the administrative and political barriers to entry confined to developing countries. For example, Australia restricted the development of new uranium mines for several decades because of opposition to nuclear power. Concern over potential environmental damage from mineral development has also led to specific prohibitions on mining in, or near to, national parks and areas of natural beauty in North America, Australia, and Western Europe.

Such environmental concerns are but a part of the strong atavistic feelings aroused by mining throughout the world. No matter that mineral deposits have little or no value until their contents are extracted by mining, there are widespread and deeply held beliefs that its rewards should accrue not to the investor but primarily to the state as the custodian of the public interest. With but a few exceptions, most mineral rights throughout the world are vested in the state. Where undeveloped mineral deposits are sold or auctioned, their prices reflect a mixture of the capital already expended in their exploration and discovery, and the expected rent accruing from their future development and exploitation.

Mineral Exploration

Like any other form of investment, mineral exploration is made in the expectation of yielding a return over and above the capital invested. Although the risks of failure are high, companies do not explore in the pursuit of average rates of return, but in the hope that they will, so to speak, hit the jackpot with a highly attractive discovery. Their return may come either from their own development of mines to exploit any discoveries or from selling the discovered deposits to other companies. Thus, guaranteed property rights that can be legally enforced are as essential for exploration as for mine development. The exploring company needs an assurance that it can profit from any discovery it makes.

Mineral Rents

The mining industry's economic contribution is measured by its value added: the sum of the rewards paid to the primary factors of production, namely land, labor, and capital. For all but the marginal producer, a proportion of the value added reflects rents accruing to the resource. In that regard, mining is like other primary industries such as agriculture and forestry where part of the annual income springs from the characteristics of the natural resources used. Rent is the difference between the value of the product and the sum of the opportunity costs of all the resources involved in making the product, including the minimum rate of return to capital required to make an investor commit funds in the first place. In the short run, it can accrue to any of the factors involved, including labor and management, but such *quasi-rents* are normally competed away in the longer term, unless there are artificially high barriers to competition for these factors. By contrast, the rent accruing to the resource reflects its inherent nature with prevailing technologies and geological knowledge.

Resource rents are only mobilized when a mineral deposit is mined. They require the application of the other production factors, notably labor and capital, before they are valorized. When undeveloped mineral deposits are sold or auctioned, those involved are making implicit assumptions about the NPV of the future rents that their exploitation will yield. Those assumptions may prove wide of the mark, whether unduly optimistic or far too cautious.

The concept of mineral rents may be straightforward, but their estimation is complex. Even where prices and demand are stable, costs and incomes will vary from one year to the next. Heavy capital expenditure is incurred before commercial production and at periodic intervals during subsequent years, and capital costs are also associated with final closure. Given the costs involved in the discovery and proving of ore, the life of the mine is unlikely to be known with certainty. Typically, companies only prove sufficient mineral content to justify their initial investment, no matter how large the underlying deposit. As mines near the end of their productive lives, their operating costs tend to rise as their grades decline and they go deeper. In practice, demand and the prices of inputs and products can fluctuate considerably, even where their underlying trend is stable. All too often it is not. For these varying reasons, the amount of rent will vary from one year to the next. That means that all incomes and expenditures should be converted to NPVs in order to ascertain a mineral deposit's rents.

This raises further complications. Should future values be discounted at the investor's opportunity cost of capital or at some other rate? In any case, what is the opportunity cost of capital? It is likely to vary among different categories of potential investor and among different countries. The minimum required rate of return will be lower for a short-lived operation requiring minimal facilities in a developed country than for a long-lived large-scale mine needing massive infrastructure in an unstable developing country. A large diversified mining company may be able to raise funds far more cheaply than a much smaller company without an established track record, but the cost of finance to governmental agencies may well be lower still. Since mineral rents ultimately arise from the nature of the underlying resource, these opportunity costs are in any case less relevant for discounting future values than the national social discount rates of the host countries. These are also likely to differ, probably being much lower in a rich advanced economy than in a poor developing country.

Rent and Depletion

No matter how future values are discounted to present values, part of the rent more properly represents the cost of the depletion of the resource rather than a genuine surplus. As such, this user cost element of the rent should naturally go to the owner of the resource, usually the state, as recompense for its exhaustion. This *user cost* is specific to nonrenewable resources, as the productivity of most agricultural land can be maintained or restored by appropriate husbandry. The precise magnitude of user costs varies from mineral to mineral and over time. Theoretically, its annual incidence should in some manner reflect the rate of net depletion of each deposit. As already noted, however, the ultimate size of most mineral deposits and the lives of the mines they support can normally be assessed only retrospectively rather than in advance.

Rent and Taxation

There is no theoretical basis for sharing among the various stakeholders any remaining rents beyond user costs. In practice, the owners of the mineral deposits, usually the state, make strong claims for retaining the lion's share through taxation. That presupposes that the potential rents of each deposit are not captured by purchasers of mineral products through prices that fail to cover full costs or are prematurely wiped out by new entrants or technological developments. Changes in geological knowledge, in mining and processing technology, and in end uses can affect the inherent profitability of operating mines and even lead to their early closure before they have extracted the full potential of their mineral deposits. In such instances, some of the potential rent will remain sterilized.

That mineral rents are both unpredictable and unstable and can only properly be assessed retrospectively make them an unsatisfactory basis for taxation. Governments want and need some degree of certainty in their fiscal receipts. Typically, royalties that are linked in some manner to output provide such certainty and are relatively easy to compute. That is a further advantage for many host nations that have limited administrative capacity and shortages of skilled labor. Companies are liable to pay such royalties as long as they produce, even when they make no profits, so that governments are guaranteed an income. Ad valorem percentage royalties based on turnover are more flexible than fixed royalties based solely on volumes of production. The great disadvantage of any output-related royalty, no matter how it is calculated, is that it is a fixed cost and consequently raises the economic cutoff grade for mining, potentially sterilizing marginal parts of a mineral deposit that might be extracted and processed in its absence. Nonetheless, governments regard that as a price worth paying for royalties that typically average in the 1%–3% range. Higher royalty rates are often applied to precious metals and gemstones. In all cases they are seen as payments to the owners of the resource and may be conceptually regarded as covering user costs.

Most governments also levy income taxes of varying coverage and complexity. Attitudes to appropriate rates and allowances change over the course of the economic cycle with shifts in product prices and profitability. During the 1980s and 1990s, there was a tendency for the overall tax burden on mining projects to fall. Competition between putative host countries to attract new investment led to a narrow range of tax incidence. Those countries with tax burdens significantly above the prevailing norm were considered unlikely to capture foreign investors. In reality, tax is only one factor among many in influencing investment decisions and is by no means the most important. The boom of the mid-2000s caused host nations to reassess their tax regimes and to seek additional revenues. Some countries raised or imposed royalties and others reexamined their income/profits taxes in order to capture greater shares of increased profits. The sharp drop in the prices of many minerals in the second half of 2008 demonstrated the volatility of the industry's profits and that host countries were chasing an elusive moving target.

Mining and Economic Management

Volatile and unpredictable earnings because of fluctuating demand and prices can be readily absorbed by a broad-based and widely diversified economy, even if the local impact may be great. They are much less manageable for developing countries with few or no alternative sources of income and employment. The problems of economic management for such countries were compounded when mineral prices appeared to be on a long-term declining trend in real terms. They were then faced with falling terms of trade so that it became progressively harder to earn sufficient foreign exchange to finance imports. Although the mid-2000s' boom apparently disrupted previous trends, it might be premature to announce their death.

For many economies, the start-up of new mining projects can cause sudden spurts in incomes, employment, and foreign exchange earnings. This lumpiness accentuates the problems caused by the underlying volatility of most sectors of the minerals industry. Unless it is appropriately managed, a surge in activity arising from new mine development can unbalance the host economy. The effects tend to be much more pronounced in the oil and gas sector than in nonfuel minerals, but they are nonetheless observable. A strong rise in exports affects the balance of payments and tends to cause the currency to appreciate. At the time, resources, especially labor, are attracted away from their existing uses into the newly expanded mining sector, and costs are bid up. The competitiveness of the existing sectors declines, leading to reduced sales and rising unemployment, so that booming mineral exports are accompanied by a shrinking economic base in the rest of the economy. These are the symptoms of *Dutch disease*, named after the experiences of the Netherlands following large natural gas discoveries in the 1960s. It is not an inevitable accompaniment of mineral development and can be countered by the host government taking timely and appropriate budgetary and fiscal action.

Institutional Capacity and Mining

Unfortunately, some host countries lack the necessary institutional capacity and administrative competence to adopt and pursue appropriate fiscal policies. Nor are they always capable of installing and operating systems for offsetting volatile tax receipts. Such systems include stabilization funds that receive all mineral tax revenues as they are earned but only release them for government spending over a period based on an estimate of likely average earnings over the business cycle. Effective stabilization funds can insulate governmental spending from extreme fluctuations in tax receipts, but they are not a universal panacea. Aside from the inherent difficulties involved in assessing future trend prices and revenues are the problems of ensuring the independence of a fund's management and operations from political and bureaucratic interference. Those are issues for mature democracies, let alone for most developing countries.

Geology knows few (if any) political or social boundaries, and mineral deposits are often located in unstable countries riven by tribal and social tensions. That their exploitation might finance civil wars, exacerbate preexisting tensions, or involve corruption is not the fault of mining in and of itself. With proper management, mineral wealth can be used to ensure economic and social development and eliminate such tensions. All too often, however, powerful interest groups seek to capture the potential benefits of mineral development for their own ends rather than for the national good. At the extreme, such rent-seeking behavior might become kleptocracy by the ruling elite.

When the obvious environmental impacts of mineral development, no matter how well and sensitively those are managed, are added to its potential adverse economic and social consequences in poorly managed countries, it is understandable that the mining industry's critics talk in terms of an

inevitable resource curse. In many instances, however, host countries have few, if any, alternative sources of potential wealth creation. They lack the luxury of choosing between mining and other economic activities. Furthermore, weak institutions and poor governance are the major causes of most of the ills that have been ascribed to dependence on mineral development. The appropriate response is to tackle those issues directly rather than abjure mining on the grounds that it might make them worse rather than help solve them.

Resource Depletion and Sustainability

Regardless of the effectiveness and competence of host governments, all mineral-rich countries have to cope with the consequences of reserve depletion. No matter how extensive a country's mineral deposits, they will eventually be exhausted. In this narrow sense, mining is not sustainable. Often the minerals extracted will have been exported rather than retained within the country and hence will not be potentially available for recycling locally. Unless new discoveries and technological developments unlock additional mineral resources, alternative means of wealth creation will be required to maintain existing living standards, let alone meet the rising needs of an expanding population. Investment of some of the proceeds of today's mining in other means of generating wealth may be economically and socially desirable, but the imperative of satisfying immediate needs may be difficult to resist, even in well-governed nations. In weakly governed states they may be irresistible.

Historically, mining has often provided the springboard for sustained economic development, both through its requirements for infrastructure, its demands for goods and services, and downstream linkages into further processing. As well as their direct employment and purchases, mining developments can create indirect demands from their workers and suppliers. The size of such multiplier effects depends on the location and nature of the initial projects, and on the extent to which those of the mine itself or all the facilities associated with the project are taken into account. Often the main, indirect benefits may be realized far from the mine itself and for many countries and regions may indeed leak overseas. Similarly, there may be little prospect of economic downstream processing in many countries that lack appropriate infrastructure or suitable labor. Compared with earlier times, changes in communications and technology have greatly reduced the potential for local multiplier effects.

Even in advanced countries, mine closures raise economic and social problems. There is a perennial debate between those who would create alternative employment opportunities near worked-out mines and those who would completely abandon remote mine-based settlements and move their inhabitants elsewhere. If alternatives can flourish without artificial support such as subsidies, the former may be the more sustainable approach. In other cases, it is best to tackle the social dislocations involved head on and accept that townships orphaned by mine closures are unlikely to be sustainable.

The paradox is that mineral development mobilizes wealth by using up the potential wealth inherent in natural resources. Provided that some of the wealth thus mobilized is reinvested in providing for future needs, then mining makes a proper contribution to sustainable development. An unexploited mineral deposit is completely sterile, making no contribution toward meeting either present or future needs. There is never any guarantee that its contents will be needed in the future.

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CHAPTER 2.2

Pricing and Trading in Metals and Minerals

David Humphreys

PROJECT PRICE ASSUMPTIONS

When assessing the feasibility of developing a mining project, assumptions need to be made about the price which the product to be mined will bring when it is sold. Indeed, it is often the case that the price of the product has a bigger influence on the outcome of the assessment than any other single variable. Because forecasting prices is not a simple matter, this is a challenge. Commodity prices are subject to significant short-term volatility as well as longer-term cycles. Moreover, the challenge is compounded by the fact that the mine undergoing evaluation may have an expected life of 20 or more years and may not be scheduled to come into production for several years ahead.

Nevertheless, in order for the feasibility of project development to be determined, a view has to be taken on the likely price of the product over the project's life. This central case assumption will be the expected annual average price to be achieved for the mined product during each year of the project's life. No less important, those responsible for setting the price assumptions will need to understand the risks inherent in price forecasting and properly assess the range of possible price outcomes around their central case assumption in order to determine whether the project can live with the price risk it faces. This might be done by examining past patterns of price volatility and the length of previous price cycles.

Failure to do such a risk analysis potentially means committing to a project that comes on stream at a point in the cycle where it cannot cover its operating costs or that it never achieves a return for the investor. At the same time, it needs to be borne in mind that a policy of being overly conservative with one's price assumptions brings with it a contrary risk—that of missing out on a viable business opportunity. Therefore, to do their job, price assumptions for mining projects have to balance both upside and downside risks.

UNITS OF MEASUREMENT

Considerable care needs to be exercised when working with metal and mineral prices to ensure that the units for pricing have been correctly specified and apply to the product to be produced. Prices are generally set according to a unit weight of product, although for some commodities other considerations such as purity play a part. Gem diamonds are famously priced according to the four "C's"—carat, clarity, color, and cut. Talc prices take account of whiteness and grain size, while prices of coal consider calorific content and the quantity of ash present.

Prices for metals are commonly set according to the weight of a standard unit of refined product. Increasingly, the International System of Units (SI) is employed, with the metric ton (t), the kilogram (kg), and the gram (g) as key units of measurement. However, for reasons of industry convention as well as history, the use of SI continues to live alongside other older systems of measurement, such as U.S. or imperial. Since much of the world quotes metal prices in terms of U.S. dollars per metric ton, the use of U.S. dollars or U.S. cents per pound is widely used, particularly in North America. (For purposes of conversion, 2,204.622 lb = 1 metric ton.) Also in North America, one still encounters the use of the 2,000-lb short ton. Prices of precious metals are universally quoted in terms of troy ounces (oz), 1 troy oz equating to 31.1042 g.

Moving away from refined metals, things become rather more complicated. Metal concentrates and other intermediate metallurgical products, such as copper–nickel matte, are typically priced on the basis of the weight of recoverable metal they contain, with premiums earned for recoverable by-products such as gold and discounts imposed for unwanted impurities such as arsenic. Some mineral raw materials are priced according to the metal compounds they contain rather than the metallic element alone. Thus, chrome ore is typically priced according to its Cr_2O_3 (chromium oxide) content while bauxite is priced on, among other things, its Al_2O_3 (aluminum oxide) content. The pricing of iron ore is based on the weight of contained Fe. However, because these ores often contain significant amounts of water, which the buyer does not want to pay for, prices of iron ore are typically quoted on a dewatered basis in dry metric tons (dmt) or dry metric ton units (a dry metric ton unit representing one hundredth of a dry metric ton).

Because transport costs are an important component in the pricing of these bulkier, less-processed products, their prices often include reference to where the price has been

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established. Thus products quoted "FOB" (free on board) represent the price of a product delivered on board a ship. The cost of transport and insurance beyond this point is the responsibility of the buyer. By contrast, prices quoted "CIF" include the cost, insurance, and freight to the destination port. Prices quoted "EXW" (ex-works) signify that all costs for transport and insurance beyond the seller's gate must be met by the buyer. Occasionally, prices are quoted for a specific geographic location, usually a major port such as Rotterdam or Vancouver.

The pricing of industrial minerals represents another level of complexity because these are commonly valued as much for their physical and chemical properties as for the elements they contain. Minerals such as bauxite, magnesite, and zircon used for refractory purposes are valued and priced according to their heat-resisting qualities. Minerals used as fillers and coaters in the manufacture of paper, paints, and plastics, such as talc, titanium dioxide, kaolin, and calcium carbonates, are valued and priced according to their brightness, whiteness, and qualities as an opacifier, among other things. The fertilizer minerals, which include potash and phosphate rock, are closer to metals in that they are priced according to the elements they contain that contribute to plant growth, namely phosphorus, potassium, or nitrogen. Some industrial minerals are priced according to the form in which they are to be delivered, for example, bagged or bulk.

Because of this relative complexity and the lesser transparency of the markets for industrial minerals, the business of determining likely prices for these commodities generally involves a great deal more detailed market research, including talking to potential customers, than is the case with metals. Reference prices for industrial minerals quoted in trade journals may not be directly applicable to the particular specification of the product that the project being evaluated is going to produce, and assuming that they are could prove costly.

PRICES AND INFLATION

Several other issues must be addressed when setting the prices of mineral commodities for project evaluation. The first is the awkward fact that while physical units of measurement do not vary over time, the value of money does. Thus its use in establishing prices means that one is in effect employing a variable yardstick.

The terms used to describe changes in the value of money are *inflation* and *deflation*. Inflation signifies that the value of money in a country is declining. A 10% rate of inflation effectively signifies that money is losing a tenth of its value every year and, by implication, that one needs 10% more of it to buy a given quantity of any commodity. Conversely, deflation signifies that the value of money is increasing over time.

Inflation has been the most common experience of the global economy during the course of the last half century, although prior to this there were extended periods of deflation. In more recent times, Japan experienced deflation for many years following the bursting of its bubble economy in 1989. Since 1970, inflation in the United States, under the broadest definition of the term, has averaged around 4% a year. However, there have been some extreme cases, such as in Brazil at the start of the 1990s or Zimbabwe in the 2000s, when the rate of inflation has run at thousands, or indeed millions, of percent a year.

Market prices are what customers have to pay for a commodity on the basis of prevailing money values. Prices quoted on this basis are thus referred to as prices in money of the day or prices in current money terms. Another label sometimes applied is *nominal terms* (implying the pricing terms named). Investors, however, are generally interested in the real returns they stand to get for their investment, which is to say, returns after the effects of inflation or deflation have been taken into account. It is simply not attractive for an investor to receive a nominal 10% return on his or her investment in a year when the value of money has declined by 20%.

For this reason it is common for feasibility studies to be conducted in real or constant money terms, with the effects of inflation or deflation taken out, and all cash flows into and out of a project brought back to current money values. This is achieved by adjusting nominal prices and costs by a general index of inflation, such as the consumer price index, the producer price index, or (the broadest index of all) the gross domestic product (GDP) deflator.

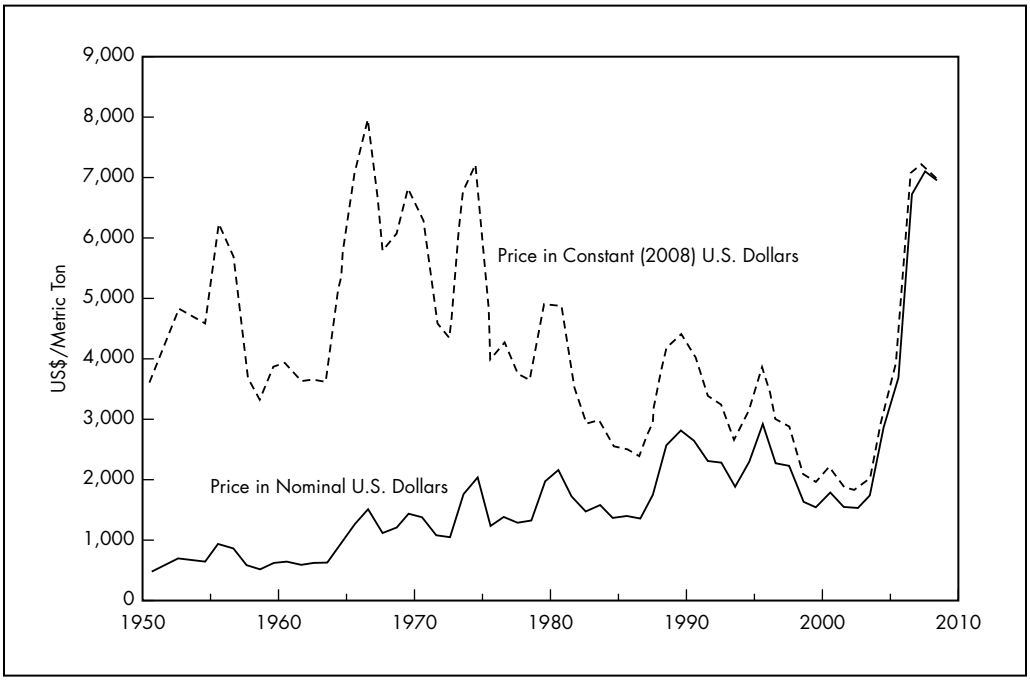
Figure 2.2-1 shows the trend in the price of copper on the London Metal Exchange (LME), quoted in nominal dollar terms between 1950 and 2008. It also shows the effects of applying to these nominal prices the U.S. GDP deflator to produce a constant dollar price series. The dollars used for this conversion are 2008 U.S. dollars. As is apparent, the trends described by these two price series over the years since 1950 are dramatically different.

PRICES AND EXCHANGE RATES

The second issue to be addressed is the currency in which the commodity price is to be quoted. Most internationally traded commodities are conventionally quoted in U.S. dollars. This was not always the case. In fact, it was only in 1992 that the LME, the world's largest metal exchange, went fully over to the dollar. Up to that point, contracts had been traded in a mixture of U.S. dollars and pounds sterling.

Commodities that are not much traded internationally—for example, bulk low-value minerals such as construction aggregates—will more generally be quoted in the currency of the producing country. The principle also applies where a market has a strong regional component. Thus, mineral sands containing titanium and zirconium are often quoted in Australian dollars, barite used as drilling mud in the North Sea oil industry is quoted in pounds sterling, asbestos ex-mine from Canada is quoted in Canadian dollars, and magnesite from Greece is quoted in euros.

For projects in the United States, the use of the U.S. dollar as the basis for pricing commodities poses relatively few problems because mining companies will typically be assessing their costs and prices using the same currency. For projects outside the United States, however, where costs may be largely incurred in another currency, dollar-based price forecasts need to be made in association with explicit assumptions about the exchange rate between that currency and the U.S. dollar. This is partly because in global commodity markets, price levels are a product of the interplay of the currency exchange rates of producing and consuming countries, just as they are a function of supply and demand. (See section on "Prices in the Near Term.") However, more importantly in a practical context, it is because exchange rate movements can have important implications for project revenues—and thus margins—in local currency terms at any given U.S. dollar price.



Source: Data from LME 2009 and BEA 2009 (for U.S. GDP deflator).

Figure 2.2-1 LME copper price, 1950–2008, in nominal and constant dollar terms

Consider what happens when the currency of the country in which a project is located declines in value against the U.S. dollar. In effect, the cost of production in dollar terms declines or (effectively the same thing) the revenues in local currency terms increase. Either way, the project margins improve. By the same token, if the currency of the country in which a project is located increases in value against the U.S. dollar, the cost of production in dollar terms also increases (the revenues in local currency terms decline). Understanding the interplay between prices and exchange rates can therefore be crucial to the assessment of a project.

The point can be illustrated by looking at what has happened to gold prices quoted in different currencies. Setting the U.S. dollar price of gold at an index of 100 in January 2000, one can see from Figure 2.2-2 that the price by the end of 2008 had risen almost 200%. By contrast, the price of gold quoted in euros had increased only 100%. Australia and South Africa, two important gold-producing countries, show rather different trends. While the price of gold in South African rand increased some 370% over the period, the price of gold quoted in Australian dollars increased only half this amount, implying, all other things being equal, that Australian producers felt the benefit of rising gold prices rather less than those in South Africa.

In reality, of course, other things are seldom equal. For one thing, exchange rates between currencies partly reflect differences in the rates of inflation in the countries concerned. Thus, if the weakening of a currency against the U.S. dollar solely reflects the fact that the country using that currency is experiencing a higher rate of domestic price inflation than is the United States, it may not be the case that a fall in the value of the currency signifies that the dollar costs of production are declining at all. For another, in countries where the export of a mined product is a major component of its total exports, there is likely to be a strong interrelationship between the price of the product and the country's exchange rate. This is the case, for example, with the copper price and the exchange rates of Zambia and Chile.

These are not easy issues to deal with, but analysts evaluating projects need to recognize their importance and address them explicitly when determining prices for projects. Because exchange rates are an integral part of the pricing equation, feasibility studies can be as easily flawed by a failure to understand the interplay between prices and exchange rates as they can by choosing the wrong price.

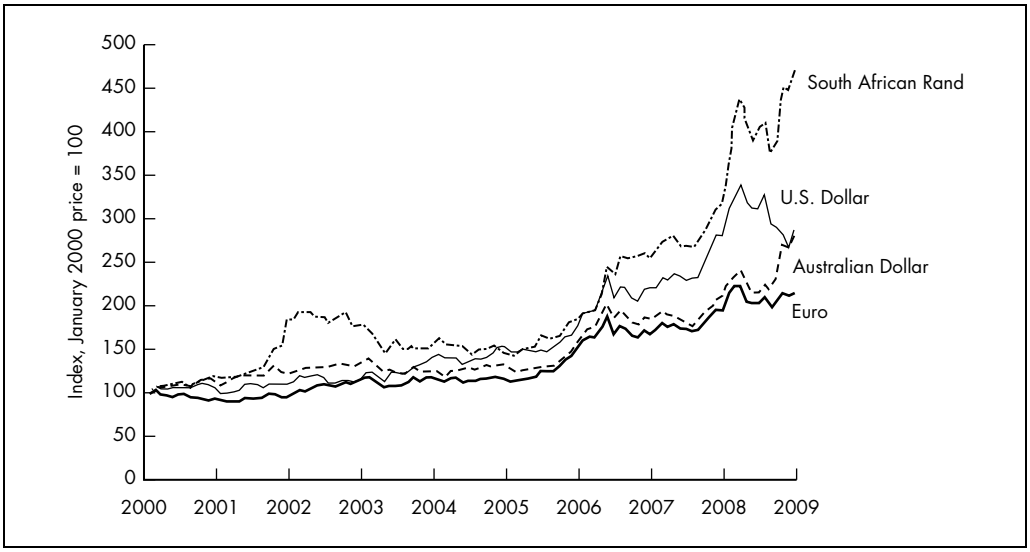
WHAT DETERMINES PRICE

Subject to the nature of the project being assessed and when it is expected to generate revenues, assumptions will need to be made about both near-term product prices—that is, prices over the next 3 to 5 years; and prices over the longer term—those beyond this period.

Prices in the Near Term

The volatility of metal and mineral prices makes assessments of the expected future level of a mineral product's price in the near term somewhat challenging. How important this is depends on the status of the mineral project being evaluated. For a start-up mining project, which is not due in production for several years, it is obviously not so important. But in the evaluation of a merger and acquisition target, where the cash flows from the acquisition over the next few years are expected to contribute to funding its purchase, it may be absolutely critical.

There are broadly two approaches to forecasting prices in the near term, commonly referred to as technical and fundamental. Technical analysis focuses exclusively on the price variable itself and on price behavior. It is based on the proposition that prices tend to follow identifiable patterns, which, if one analyzes price history sufficiently rigorously, can be



Source: Data from World Gold Council 2009.

Figure 2.2-2 Gold price in different currencies, 2000–2009

used to predict how prices will behave in the future. Technical analysis originated in the close observation of price charts—for which reason technical analysts are often still referred to as chartists—but today tends to be more computer based. The relatively short time scales over which this approach has the power to forecast (typically measured in weeks or even days) makes it popular among traders and speculators but rather less relevant to mining companies, which tend to think in years rather than in weeks or months. Such companies will generally favor a fundamental approach to price forecasting—that is, one focused on prospective developments in supply and demand for the mineral product concerned.

Because the demand for mineral commodities is driven by developments in the broader economy, an assessment of the economic growth environment is typically the starting point for demand analysis. However, because different minerals are used in different sectors of the economy, the structure of economic growth can be important, too. Metals such as steel, copper, and aluminum have a heavy concentration of use in investment goods such as buildings and machinery and in big-ticket consumer items like cars and refrigerators, which tend to be quite cyclical. This explains why demand for these metals tends to be more volatile than economic activity overall. Other mineral products such as titanium dioxide, which is extensively used in paper, plastics, and paints, have their principal applications in lower-value consumer goods, and therefore their demand tends to be more stable. Regional factors can also be important. China's rapid economic growth in recent years has had a huge impact on the global demand for mineral commodities.

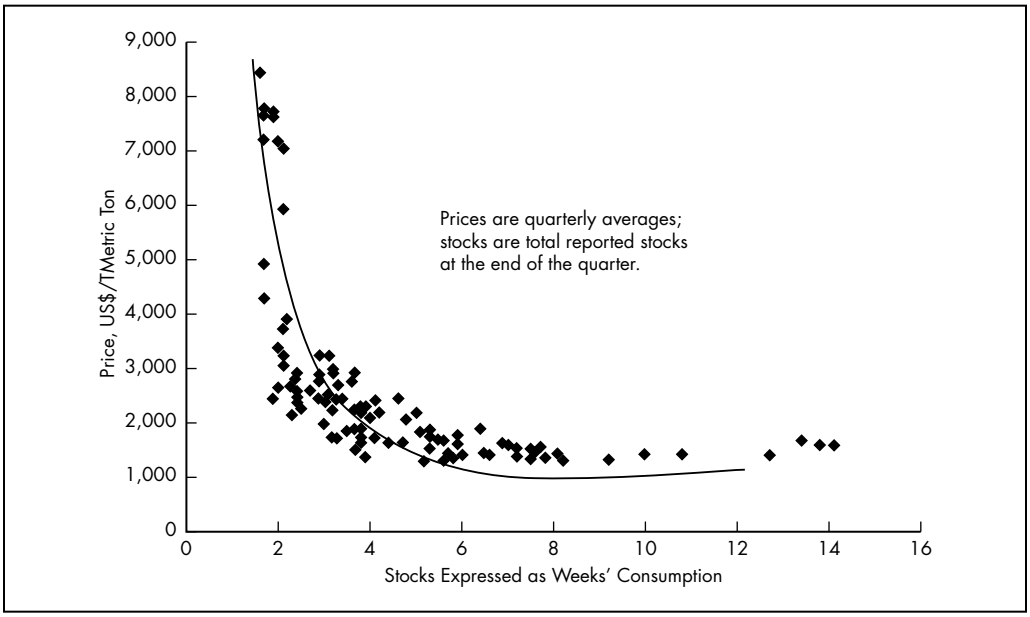
On the face of it, supply ought to be easier to forecast because companies often make public their production plans, while the length of time it takes to construct new capacity means that production changes are usually signaled well in advance. In practice, however, producers' plans get disrupted as a result of geological discontinuities, mining problems such as pit wall failures, equipment breakdowns, labor disputes, and power shortages. Because undershooting production targets is fairly common, analysts will often factor in a disruption allowance to their projections of future supply. However, the scale of such losses can vary substantially from year to year. Moreover, production sometimes actually exceeds plans, particularly during cyclical downturns when producers are able to bring forward development projects or are boosting their productivity to reduce unit costs of production.

The idea here is that balances derived from such supply–demand analysis should provide guidance on whether markets are likely to be in surplus or deficit in the next few years and whether prices are therefore likely to fall or to rise over this period. Combined with information on existing stock levels (an indicator that market analysts watch closely), supply–demand analysis should give an idea of the scale of likely future market imbalances and thus the possible extent of price movements.

Some modest science can be brought to the exercise by plotting past imbalances against prices in a so-called pinch point chart, using the resulting relationships to predict likely future prices. Such a chart, plotting quarterly copper prices against quarterly reported copper stocks expressed in terms of weeks of consumption for the period 1983–2008, is shown in Figure 2.2-3. Indeed, one can go further and combine these prospective market imbalances (or the stock changes implied by them) into a statistical model along with indicators of economic activity, such as industrial production and the strength of the currency in which the price is quoted, to capture other price drivers, to get a still more precise fix on likely future prices.

Unfortunately, projections of prices flowing from these models are only as good as the forecast data on which they are based and, regrettably, these are generally not very good. Moreover, one has little prospect of being able to incorporate extraneous factors affecting prices into such models, such as unanticipated economic or political shocks and investor sentiment.

It is one of the characteristics of metal markets that they often attract significant interest from investors, in addition to those engaged in the physical trade. These investors include hedge funds, technical traders, the proprietary trading desks of investment banks, and commodity index funds. These last are funds designed to give commodity price exposure to the likes of pension funds and insurance companies, either because they believe that commodity prices are likely to go up in the



Source: Data from LME 2008 and Brook Hunt 2008.

Figure 2.2-3 Copper price and stocks, 1983–2008

future or else (and more commonly) because they believe that returns from commodity investments are inversely correlated with those from securities and bonds and thus are likely to hold them for portfolio diversification.

Quite where the line should be drawn between investors and speculators is far from obvious, since both are essentially in the business of trying to make a profit by buying and selling metals in anticipation of future price movements. Typically, investors are deemed to have a positive influence on the process of price formation because, by acting as counterparties in deals involving trade participants, they provide liquidity to the market and accept price risk. Speculators, by contrast, tend to be viewed much more negatively on the grounds that their buying and selling creates self-reinforcing upward and downward price spirals and drives prices to extremes that are unwarranted by market fundamentals.

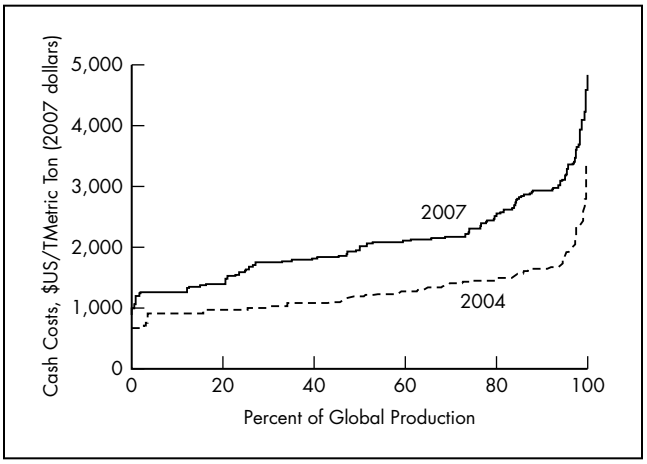
In the absence of any objective and reliable means of distinguishing between these sets of actors, the application of the terms *investor* and *speculator* often comes down to a question of the commentator's preconceptions or the underlying conditions of the markets in which the investors and speculators happen to be operating. The matter is not, however, entirely academic. The sharp and sustained increase in prices, which began for many metals in 2005, coincided with a large increase in investor/speculative interest in metals, which produced large profits for the mining sector but also led to substitution and demand destruction for some metals. In practice, while the involvement of nontrade players can undoubtedly be an important influence on prices in the short term, most economists would take the view that over the long term—which is to say, over the life of a mine—the effect of investor and speculator behavior tends to balance out, making their net influence relatively marginal.

Prices in the Long Term

While assessments of supply and demand are a useful price forecasting tool for 3 to 5 years into the future, mining investments typically span several commodity cycles and therefore need to be based on prices that stretch well beyond the horizon of a single cycle, perhaps to 20 or 30 years. The longer-run prices used for this purpose need to be grounded in the underlying economic realities of the minerals industry and the assumption of a balanced market (because, by definition, in the long run, markets must balance). Because such underlying economic realities change only slowly with time, long-run prices for project analysis will typically be assumed either to be flat in real terms or else to trend gently—which is to say, they will not incorporate any assumptions about economic cycles.

There is no single or simple way of evaluating these longer-run economic realities, and the approach used needs to be adapted to the characteristics of the commodity under analysis. However, for many commodities, including most of the major nonferrous metals, long-run price forecasting is commonly based on an analysis of production costs. The primary tool for the analysis of production costs is the industry cost curve. This is essentially a compilation of production costs for all the producers in an industry, ordered from lowest to highest cost, and organized to create what is in effect a proxy for the industry supply curve. (It is not strictly a supply curve because the basic unit of supply used in the construction of the curve is a producer rather than a single unit of output.) As an example, cash cost curves for copper in 2004 and 2007, both of them expressed in 2007 U.S. dollar terms, are shown in Figure 2.2-4. By convention, revenues from by-products recovered from a mine are treated as negative costs.

The construction of cost curves is extremely data intensive, requiring as it does detailed information on the costs of every mining operation worldwide. Such information can be assembled from published financial statements, where these exist, or else can be generated using technical engineering models of individual operations together with generic assumptions about the costs of inputs, such as power, labor, flocculants, and parts. In practice, because of the extent of these data



Source: Data from Brook Hunt 2008.

Figure 2.2-4 Cash operating costs for global copper mining industry

requirements, most mining companies will buy such data from consultants specializing in this sort of work.

The rationale for using costs as the basis for long-run price determination is to be found in the economic theory that the price of any commodity in a competitive market will, over the long run, tend toward the costs of the marginal producer and in the presumption that commodity markets come about as close as possible to perfect markets. The marginal producer is defined as the last, or highest-cost, producer who is needed in production to meet prevailing demand. Any tendency of prices to fall below this level will result in some producers being forced to shut down. Any tendency for prices to rise above this level will be a signal for producers to bring back into production higher-cost capacity or else to invest in new capacity. Precisely where the marginal producer is to be found on the cost curve over the long run (i.e., over the cycle as whole) is a matter of judgment and may vary according to commodity but is typically a point some 90% to 95% up the cost curve.

It needs first, however, to be determined exactly what costs are relevant to the behavior of the marginal producer. In the short run, the relevant costs are generally considered to be cash costs. So long as producers can cover their cash outgoings, they will remain in production. Over the long run, however, economic theory suggests that the costs of the marginal producer will incorporate an incentive element representing a return to investors over and above cash costs, without which they would be better off withdrawing from the sector and investing their money elsewhere.

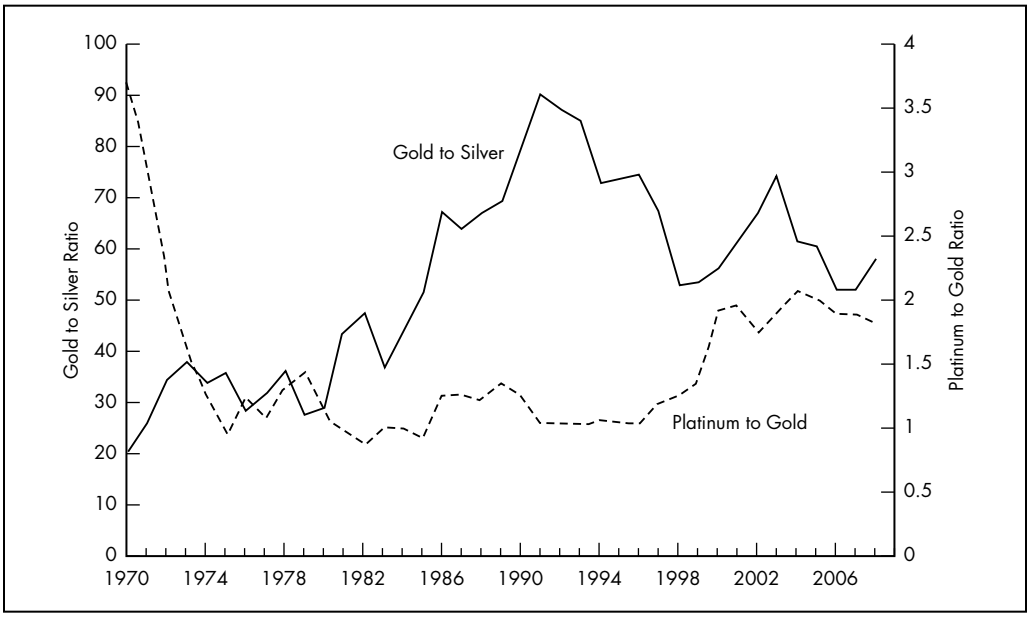
The experience of the 1980s and 1990s, unfortunately, did not appear to conform to this theory. For much of the period, prices seemed to reflect cash costs—that is, short-run marginal costs. There was little or no incentive element, a fact borne out by the generally poor returns achieved by investors in the mining sector during these years. The implication seemed to be that the long run was little more than a series of short runs strung together. It was also generally the experience of these years that prices of many commodities appeared to be trending downward in real terms. (See, for example, the price of copper in Figure 2.2-1.)

In retrospect, it would appear that the 1980s and 1990s constituted a rather special set of conditions. Following the two oil shocks of the 1970s, demand for many commodities was suppressed, both by cyclical influences as well as by a shift to service sector activities in many developed economy countries. Supply, at the same time, was getting a boost from the opening of several important mineral-rich parts of the world (notably Latin America and Southeast Asia), in some instances from direct or indirect state subsidization, and, in the 1990s, from the flood of metals coming onto global markets following the economic collapse of the Soviet Union.

The combination of these factors produced conditions of almost permanent oversupply, making it difficult for prices to lift themselves off the short-run marginal cost curve for any sustained period. Most likely these special conditions will not apply with the same force over the next 20 years and metal markets will generally be in better balance. This being the case, prices will be able to climb above the short-run marginal cost curve for more sustained periods than during the 1980s and 1990s and, in accordance with what economic theory asserts, incorporate more of an incentive element. There is no common view on how big this incentive element is likely to be, with opinions ranging from those who use historical margins as the basis for their projections to those who believe that the incentive element will need to be substantially higher in the future than during the 1980s and 1990s in order to ensure that incomers to the industry earn a competitive market return on their capital and that sufficient investment in production capacity is forthcoming to balance the market.

For precious metals and metals produced as a by-product of other metals, such as cobalt, cadmium, or indium, the costs of production are a poor guide to long-run prices (even where it is possible to compute them, which often it is not). Therefore, the setting of long-run prices for these metals has to rest more heavily on an analysis of historical price information. This might include the analysis of averages and trends during the period over which price data are available, together with an examination of relationships with prices of other metals, especially those that have common markets or serve as substitutes. The analysis of price relativities is particularly popular for forecasting prices of, and for developing strategies for trading in, precious metals. Figure 2.2-5 shows the relationship between annual gold and silver prices and between annual gold and platinum prices between 1970 and 2008.

This analysis of historical trends needs, of course, to be supplemented with more qualitative judgments about the likelihood of the factors that have shaped prices in the past (e.g., the structure of production or developments in key markets) continuing to persist in the future. Structural breaks in price series are unusual, but they can and do occur from time to time. The price of molybdenum became a lot more volatile as a result of the breakup of its concentrated supply structure in the 1970s. (See section on "Producer Pricing.") Zircon sands, which were historically used mostly in relatively low-value foundry and refractory applications, have, in recent years, seen more and more of their market demand accounted for by the ceramics industry, in which customers are both willing and able to pay higher prices for premium-quality raw materials. This, combined with pressures on available supply, has meant that zircon prices appear to have moved onto a sustainably higher plane. Banking on structural change of this sort to generate a higher price for a project's output and thereby justify the investment in it may conceivably pay off but, in light of how rare these events are, is a high-risk strategy.



Source: Data from *Metal Bulletin* 2009.

Figure 2.2-5 Precious metals price ratios, 1970–2008

Price Trends

A key problem in using production costs to determine long-run prices is that, while the physical parameters of production may change only slowly over time, the financial parameters can change quickly and sometimes significantly. Accordingly, the cost curve shifts through time—rising and falling, steepening and flattening—in response to broader economic and industry developments such as fuel prices, wage increases, and exchange rates, as well as the closures of old mines and the start-up of new ones. Ideally, for the determination of long-run prices, analysts would like to have a decycled or normalized cost curve incorporating long-run estimates for all the key input variables. In practice, however, one is always looking at these things through the lens of the current cyclical phase and having to make judgments about which changes are temporary and which are likely to persist on the basis of imperfect information.

As a result of the sustained commodity price boom which followed 2003–2004, there have been major upward revisions in mining analysts' expectations of long-run real dollar prices of mineral commodities across the board. The reasons given for these revisions generally reflect a combination of claims that global demand for commodities has shifted onto a steeper trajectory because of the industrialization of Asia and because the cost of both producing metals and of creating new production capacity has moved higher and is destined to remain higher for the foreseeable future. An additional reason is the decline in the value of the U.S. dollar, which occurred over the same period and which on its own would lead one to expect future metal prices to be higher in U.S. dollar terms.

In assessing the longer-term outlook for prices, it is critical—albeit quite difficult—to try to separate two distinct factors: (1) the effects of the business cycle and (2) a structural shift in the long-run supply curve (the underlying economic realities previously discussed).

Between 2002 and 2007, the global economy experienced an unusually strong cyclical upswing. Annual growth during this period averaged almost 5%, compared with 3.5% for the previous decade. A major role in this cyclical upswing was played by China, which was both growing extremely fast and was highly materials intensive, being heavily concentrated on investment (notably residential and infrastructure construction) and export manufactures. This resulted in an unusually long cyclical upswing, with metal demand staying stronger for longer and metal supply being slower than would normally be the case to catch up. It also resulted in widespread investor interest in metals, as previously discussed. As a consequence of these factors, metal prices stayed higher for longer than in previous cycles while costs escalated as producers were forced to absorb higher raw material costs (e.g., steel and energy) and as the prices of equipment and services (e.g., trucks, tires, and skilled labor) were bid up. While these factors are undoubtedly real and take time to work themselves through, they are, for the most part, reversible and have few implications for the long term.

The structural factors that shape the long-run supply curve for the mining industry are of a rather different nature. They include physical factors such as resource quality and institutional factors such as access to minerals, as well as the effects of improving technology and management. For 30 years, starting from the mid-1970s, these factors combined to keep production costs in the mining industry moving steadily downward. New mineral provinces were opened up, mining regimes were liberalized, the discovery and exploitation of larger deposits permitted the achievement of substantial economies of scale through the use of ever larger equipment, and energy costs generally trended downward, while labor reforms and the introduction of information technologies led to substantial increases in workplace efficiency. In short, productivity growth in the industry consistently outran the effects of resource depletion. The implications of these developments for mining industry cost curves was to depress and flatten them.

There are strong grounds for believing that this phase has come to an end. Mineral resources are not running out, but the

real cost of developing and managing them appears to be on the increase for a whole raft of reasons, including declining ore grades, more complex mineralogy, smaller deposit size, more difficult (e.g., deeper) mining conditions, and increasing geographic remoteness. Also of relevance to an energy-intensive industry, it is highly probable that the lengthy decline in real energy prices has come to an end. Indeed, in some parts of the world where mining is important (e.g., southern Africa and Chile), power shortages have become a serious problem for the industry. Similarly, in some parts of the world, water, which is typically required in large quantities by mining operations, is becoming scarcer and more expensive.

The prospective shift in the emphasis of nickel production away from ores in which the nickel occurs in the form of sulfides toward those in which it occurs in the form of oxides is part of this general shift toward the exploitation of higher-cost resources, as is the development of the lower-grade, harder-to-mine, and harder-to-process platinum–palladium deposits on the eastern limb of the Bushveld in South Africa. For copper, a high proportion of recent production growth has come from smaller deposits, which have not benefited from the economies of scale enjoyed by some of their predecessors. In the future, much more of the world's copper will have to come from higher-cost underground mines than is presently the case. Similar challenges are evident in the production of many other metals and minerals.

Structural change is also manifest in the rising barriers to entry in the mining industry. Partly this is because many of the remaining large, undeveloped mineral deposits are located in developing countries, which have poor infrastructures and where mine development implies an associated—and invariably costly—requirement to build ports, power stations, townships, and railways. This is arguably why capital costs per unit of installed production capacity in the industry have increased more in recent years than can readily be explained by purely cyclical influences. In terms of the implications for the cost curve, if the effect of rising operating costs tends to be to shift the cost curve along its entire length, the effect of increased costs of entry is to tilt the cost curve—that is, make it steeper. As illustrated schematically in Figure 2.2-6, both factors serve to raise the point of intersection on the supply curve (and thus the implied equilibrium price) for any given demand function. Specifically, as rising costs push the curve describing the amount of product that producers are willing to offer consumers at different prices from S (supply) to S^1, so the market-clearing price (P) of the product rises from P to P^1.

No less important as a barrier to entry—and having a similar impact on the cost curve—are the growing political obstacles to mine development. Recent years have seen a major revival of resource nationalism worldwide, a tendency undoubtedly aggravated by high commodity prices. This has led to the blocking of possible mine projects by governments keen to extract the maximum benefits from resource developments on their soils (or else ensure that it is done by a domestic producer) as well as to widespread increases in mineral taxation and mineral royalties. The large Oyu Tolgoi copper mine project in Mongolia, for example, was stalled for a long time because of the sponsors' inability to agree on terms for development with the host government.

Elsewhere, mining companies face tightening government regulation, often for environmental reasons, and challenges to their activities from growing local activism. All of this adds to the costs of mine development and increases the prices that mine producers and their bankers require if they are to accept the implied risks and commit to investments. Although there is a good prospect that real prices will be sustained in the future above the levels experienced in the 1990s, the extent of the price increases are far from clear. Investors' need for higher prices does not necessarily deliver higher prices, and it is important in this context to note that, while a lifting and tilting cost curve may result in higher long-run prices for the industry as a whole, the benefits of this will only flow to the lower-cost producers. Marginal, high-cost producers will remain just that—marginal.

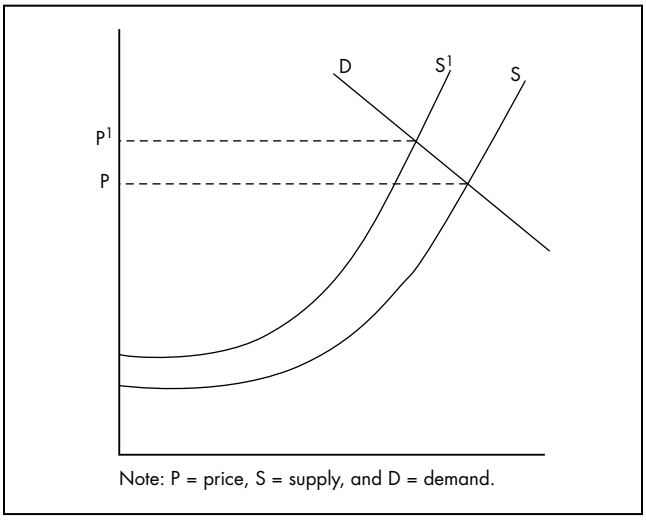


Figure 2.2-6 Price implications for increased cost of entry to industry

WHERE AND HOW PRICES ARE DETERMINED

Many factors influence the manner in which metals and minerals are traded and priced. Among these are the number of buyers and sellers in the market, the distance between them, the nature and complexity of the products being traded, and whether or not they can be stored for long periods without deteriorating. Two of the more critical dimensions are transportability of the product and its degree of standardization (homogeneity). These are used in Figure 2.2-7 to provide a rough schematic categorization of market types.

The transportability of a product refers not just to the bulk and weight of a product and the distances to be carried but to its ability to bear the cost of transport. In other words, it is about its value-to-weight ratio. Transportability is important in determining the reach of a product's market. Greater transportability will generally imply a more global market and greater potential for the involvement of many players. Gold and diamonds, because of their high value-to-weight ratio, are obviously among the most transportable commodities. Bulk minerals, such as iron ore and coal, which have a low value-to-weight ratio, have markets that are more regionally defined, with producers being primarily focused on either the Atlantic or Pacific basins. However, most iron ore and coal producers are able to supply both markets, even if only to a limited degree, while South Africa, an important iron ore and coal producer, sits equidistant between the two markets. As a result, the prices in the two markets tend to move in line with each other. Some commodities, such as construction aggregates, have scarcely any international trade at all.

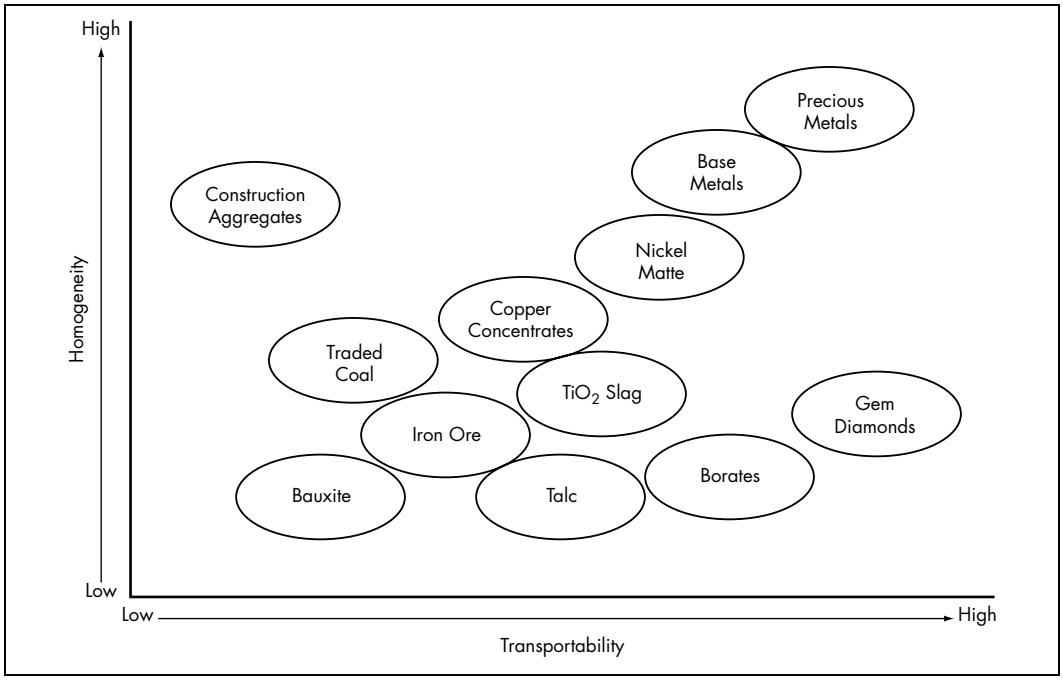


Figure 2.2-7 Dimensions of market structure: schematic representation

The second key dimension is that of product homogeneity. At one pole are commodities that meet a globally defined standard—gold bullion, for example, or copper of 99.99% purity. Such products are known by economists as fungible—that is, any one metric ton of the product is fully substitutable for any other metric ton. Such substitutability creates the potential for swaps in different parts of the world, thereby diminishing the need for transportation. A high level of homogeneity also permits paper- or computer-based trade in warrants for a commodity and makes possible the participation of traders and speculators who do not produce or use the commodity as such but who play a role in matching producers and consumers in different parts of the market and in oiling the wheels of the price formation process by trading among themselves.

Differentiated (heterogeneous) products lack fungibility. Consumers buy these products for the specific properties they possess. This requires producers and consumers of the product to negotiate directly with each other and set their prices in relation to the particular characteristics of the product being traded. For some industrial minerals these properties can be highly specific and more or less totally non-substitutable. Talc, for example, is produced to a wide range of specifications with different colors, different particle sizing, and even different mineral constituents. For other commodities, such as iron ore or coal, there may be a degree of substitutability. All iron ore is ultimately valued for its iron content just as all thermal coal is ultimately valued for its heat-generating qualities, but consumers may nevertheless have preferences for certain types of material because they have other positive attributes or are low in certain impurities. Some steelmakers are prepared to pay a premium for iron ore in lump form because it saves them the cost and effort of agglomerating iron ore fines. Other steel mills may have a preference for iron ore low in alumina. Similarly, with thermal coal, different electrical power plants have different tolerances for the ash content of coals, and some are prepared to pay a premium for a low ash product.

These factors also help determine how big a role traders play in the market. Products that are highly specific to a particular market or a particular plant will generally need to be traded directly between producers and consumers of the product. The more homogeneous a product, the greater the scope for participation in the market by traders. Distance to market is also a factor, with traders traditionally playing a more important part in markets where supply comes from producers in remote regions (particularly in developing countries) or who have a relatively small production and for whom it is more economic, or more practical, to employ the services of traders to market their products than familiarize themselves with the market or develop a marketing network of their own.

These intermediaries (as traders are sometimes called) are generally headquartered in metropolitan centers such as New York, Tokyo, Singapore, London, Frankfurt, and Zug (in Switzerland) and maintain offices or agents in the main producing and consuming regions. They generally deal with a whole range of minerals and metals and develop detailed knowledge about producers' products and consumer requirements so as to be able to match them. They will generally also be adept at organizing financing for trade transactions (a significant challenge in certain difficult countries) and managing price and currency risk. The key to their success is their access to market information and their skills in using this information to find and structure profitable deals.

Metal Exchanges

The most economically important nonferrous metals are traded and priced on metal exchanges. The two most widely known and quoted are the LME and the New York Mercantile Exchange (NYMEX). As of 2008, it was possible to trade aluminum, aluminum alloy, copper, lead, nickel, tin, zinc, and steel billet on the LME, as well as certain plastics and indexes. From early 2010, it will be possible to trade cobalt

and molybdenum. NYMEX offers contacts in copper, aluminum, gold, silver, platinum, palladium, coal, uranium, and hot rolled steel coil.

Many other exchanges exist, and this number is growing. The Shanghai Metal Exchange offers trading in aluminum, copper, lead, nickel, tin, and zinc; the Tokyo Commodity Exchange trades gold, silver, platinum, palladium, and aluminum, while on the Chicago Board of Trade it is possible to trade gold and silver. There are also metal exchanges in Moscow, Mumbai, Kuala Lumpur, and Dubai (Radetzki 2008).

Although the precise manner of trading varies from one exchange to another, in essence, metal exchanges are places for staging auctions at which contracts for a given commodity are offered for sale during prescribed hours of business each day. On the LME, these prescribed times are 5-minute rings during which trading is conducted verbally across the floor of the exchange, a process known as open outcry. Trading on an exchange is open to anyone with an appropriate credit rating, but actual transactions must be conducted through a member of the exchange. Trading can generally also take place outside prescribed hours using interoffice computer-based systems.

Exchange contracts specify a standard quantity of the metal being traded. For example, the LME aluminum contract is for 25 t (metric tons) while the NYMEX contract is for 44,000 lb. The quality of the metal constituting acceptable delivery is precisely specified. Brands produced by smelters and refiners must be approved for listing by an exchange before they can be traded.

Offerings made on the exchange floor are for delivery at a specific time. LME contracts are traded for the prompt or cash position, requiring immediate delivery of metal, or for future delivery on a specified date, while NYMEX contracts specify the month of delivery. Aluminum and copper can be traded out to 10 years on the LME, while zinc and nickel can be traded out to 5 years, although trading can be quite thin for the far forward dates. Where future prices are higher than the spot prices, the market is said to be in *contango*; where it is below future price (usually a sign of tightness in the market for immediately available metal), it is said to be in *backwardation*. Some metal exchanges also permit the trading of options—the right to buy or sell metal at a given (*strike*) price at a specified point in the future—for the payment of a stipulated premium.

Under standard exchange contracts, sellers may deliver to buyers any acceptable brand the sellers choose at any warehouse approved by the exchange. This latter option is important because the LME has approved warehouses not only in the United Kingdom but also at major ports in continental Europe, Asia, and the United States. (They are typically located near major centers of metals consumption to ensure that producers have to bear most of the costs of getting the metal to market.) NYMEX has approved warehouses scattered throughout the continental United States. If the buyer takes delivery at a warehouse location that is inconvenient, or if the brand tendered is not one that the buyer would normally choose to use, then as a rule it is possible to arrange to make an exchange for a more suitable warehouse location or a more desirable brand through an exchange member, although this is likely to involve paying a premium.

In practice, for the vast majority of exchange transactions, there is no physical delivery of metal. Instead, the buyer of a contract will sell the contract, or the seller of a contract will buy back the quantities involved in the original transaction prior to the effective date of the contract. This is because one of the central functions of metal exchanges is to enable buyers and sellers of metal to lock in—or *hedge*—prices to cover their raw material costs or their plant output.

By way of illustration, assume a secondary copper smelter buys scrap material to be delivered to its plant in 30 days. After delivery, an additional 45 days may be required for processing. The price of copper may rise or fall substantially between the date of the original purchase and the date, 75 days forward when the refined metal is available for sale. By selling the equivalent amount of copper on the exchange the day the scrap material is purchased for delivery 90 days forward, the secondary smelter has fixed the price it will receive based on the conditions in the market at the time it bought its raw material. When the refined metal is available, it is sold to the consumer at the then prevailing price and the outstanding exchange contract is bought back. These two transactions offset each other, so the net outcome is based on the original exchange sale, even though no delivery was made to the exchange. Producers of mined products can similarly use a metal exchange to lock in prices if they want the security of knowing their selling price or if they have concerns that the price of their products may be about to fall.

In addition to the large volume of exchange transactions based on this type of hedging, trades are also made by investors who believe they can profit by purchasing when a rise in price is probable or by selling when a fall in price looks imminent. (Rather confusingly, this kind of activity is sometimes also referred to as hedging. It is really the opposite of hedging because hedging is about reducing risk, whereas investing or speculating is about deliberately taking on risk in the hope of earning a commensurately higher return. Such activity is more accurately described as taking a position in the market or, perhaps, making a strategic hedge.) Such investors need not actually be involved in the metals business at all or have any interest in holding supplies of metal. In the great majority of instances, therefore, they either liquidate their positions in the market prior to maturity or, alternatively, liquidate nearby positions and roll the contracts forward to more distant dates.

Because of the large scale of these paper trades, the total amount of metal traded through an exchange is generally many times the amount of metal physically passing through it or indeed traded in the market as a whole. The amount of aluminum, copper, and zinc traded through the LME in 2007 was more than 30 times the annual consumption of those metals. For nickel it was around 20 times. Nevertheless, exchanges remain important to the physical market for metals as the place where producers who have no customers can deliver their metal and where consumers who do not have an alternative supplier can buy it. It is effectively the physical market of last resort, for which reason exchanges are sometimes referred to as terminal markets, and the stocks lodged in the warehouses and registered with an exchange are accordingly watched closely by analysts for evidence of emerging imbalances in supply and demand.

Metal exchanges are also crucial to physical producers and users of metals because their products, even if they go nowhere near an exchange, will typically be based on prices determined on an exchange. The central pivot of price determination on an exchange is the spot, or cash, price, which is the price of metal for immediate delivery. The three most important spot prices on the LME are (1) the *official* price, which is the final price from the last ring session of the morning;

(2) the *unofficial* price, which is the final price from the last ring of the day; and (3) the *closing* price, which is the price of the day's last recorded LME trade. These are rapidly and widely communicated throughout the world and used for off-exchange trading. Although differences can exist temporarily between prices on exchanges in different parts of the world, such differences provide an opportunity for traders to buy in one market and sell in another and so are quickly traded away in a process known as arbitrage, ensuring that exchange prices everywhere converge.

Trades made between buyers and sellers off the exchange will normally take place at a premium to quoted exchange prices. This premium reflects the cost to the producer of storing, transporting, and insuring the metal on its way to the consumer. Thus, for example, a metric ton of nickel might be contracted to be sold to a customer for delivery in 3 months' time, at an average of today's exchange settlement price and the price prevailing on the date of delivery, plus a premium of $150/t. The transport element in premiums means that the level of premiums in different markets often reflects imbalances in regional supply and demand. Movements in premiums sometimes also act as a leading indicator of price movements. Deliveries of metal to exchange-registered warehouses do not earn producers a premium, which explains why they are generally reluctant to deliver metal there if they can avoid it.

Bullion Markets

Although precious metals are traded on exchanges, the greater part of the trade in these metals is conducted through bullion markets. Participants in such markets include members of the London Bullion Market Association (LBMA), which covers gold and silver; and the London Platinum and Palladium Market (LPPM), which covers the metals in its title.

In contrast to the metal exchanges, where metals are traded and cleared anonymously through a common open channel and which provide clearing and guarantees for participating members, trading in bullion markets takes place directly between the members of the market and their clients on a principal-to-principal basis, which means that all risks, including credit risk, are between the two counterparties to a transaction. This is termed an over-the-counter (OTC) market. Participants in OTC markets are not necessarily obliged to report their trading volumes or their stock levels, so markets in which metals are traded in this fashion tend to lack the transparency of metal exchanges.

The London bullion market is a wholesale market where the minimum traded amounts for clients are generally 1,000 oz of gold and 50,000 oz of silver. Good delivery bars (i.e., bars that meet prescribed criteria for weight, purity, and other physical characteristics set out by the bullion market and which come from certified refiners) are 350–400 oz for gold and 750–1,100 oz for silver, while platinum and palladium are normally traded in multiples of 500–1,000 oz. Unlike a metal exchange, bullion markets permit flexibility with respect to contract units, settlement dates, and delivery specifications. And, unlike the typical contract for a nonferrous metal, which will commonly use a price formula linked directly to published LME prices, the price for a precious metal traded OTC is in principle negotiable along with everything else.

Public price quotations for metals traded on bullion markets are established in two ways. The first is through daily price fixes, which offer a snapshot of market conditions at particular points in time. The process for determining prices under this arrangement starts with a small group of member banks assembling in London each business day—twice a day in the cases of gold, platinum, and palladium; once a day for silver. The chairman of the meeting suggests an opening price, which is relayed to members' dealing rooms and then to the customers of the members. On the basis of orders received, members declare as a buyer or a seller. Provided both buying and selling interests are declared, then members are asked to state the amount they wish to trade. If the amounts of buying and selling do not match, the same procedure is followed again at higher or lower prices until a balance is achieved, at which point the price is declared fixed. The price fix is communicated to the market and used as a benchmark for trading worldwide and for the pricing of financial instruments such as swaps and options. In addition to the fix, the LBMA offers forward quotations for gold and silver, these being the rates at which member banks are prepared to lend these metals on a swap against U.S. dollars for up to a year ahead.

While the London fixes are widely used as the guide to pricing precious metals, the amounts of metal involved in the fixing processes are small compared to the total volume of precious metals traded OTC by bullion banks and dealers worldwide. The greater part of this trade is done at prices quoted with delivery in London or Zurich, referred to as Loco London and Loco Zurich, respectively. Prices quoted by dealers for these locations are updated effectively 24 hours a day and can be adjusted to cover the costs of producing smaller, exact weight bars or bars of fineness in excess of the defined minimum level of purity. Surveys of these dealer prices are collected and published on a regular basis for reference purposes by trade journals such as *Platts Metals Week*, *Metal Bulletin*, and *American Metal Markets.* More up-to-the-minute indications of price are provided by the wire services, most notably Reuters, which continually survey dealers and banks around the world on their trades and, on the basis of this information, compile reference prices, which they make available to service subscribers. The silver refiner and fabricator, Handy & Harman (part of the WHX Corporation), has traditionally published a reference price for silver in North America. With a shrinking proportion of the global use of metals taking place in Europe and North America, an increasing role is being played in the precious metals trade by centers outside these regions, notably Tokyo, Shanghai, Hong Kong, Mumbai, and Dubai.

Producer Pricing

Under the producer pricing system, the producer of a metal or mineral informs the market of the price at which it is prepared to sell its product. It is typically found in markets where production is concentrated in the hands of relatively few firms and where demand is robust. These factors combine to focus market power in the hands of sellers and permit them to charge a price that reflects their cost of production plus a suitable margin. Proponents of producer pricing have argued that the system gives stability and predictability to prices, which industrial customers like; provides incentives for undertaking research and development; and promotes the development of constructive long-term business relationships between buyers and sellers.

Producer pricing has historically been widely employed in the metals industry. For many years it was the norm for aluminum, nickel, and molybdenum, although at various times copper, lead, and zinc have also been subject to producer pricing. In most cases, however, this form of pricing

was killed off during the 1970s and 1980s when a combination of slower demand growth and a wave of new market entrants and the breakdown of protected regional markets halted the power of producers to set prices and facilitated the growth of metal exchange trading. The LME contracts for aluminum and nickel both date from this period.

Variants of producer pricing nevertheless continue to exist, generally in sectors where the range of suppliers is relatively limited or where markets are small and products have a high degree of specificity. Sometimes prices issued by producers are described, more neutrally, as list prices rather than producer prices to avoid the suggestion that producers are somehow imposing prices (and in practice it may well be the case that there is scope for negotiation around the prices quoted). In the market for gem diamonds, the Diamond Trading Corporation (DTC, part of the De Beers group of companies), which handles almost half of all the world's gem diamonds by value, sells uncut diamonds to authorized buyers from a list of prices laid out in the DTC price book. Johnson Matthey and Engelhard publish daily the prices at which they are prepared to sell platinum group metals. BHP Billiton and Norilsk Nickel have for several years posted on their Web sites the prices at which they are prepared to sell cobalt, along with prices realized in recent sales. For more complex products, such as borates, for example, producers make available to their customers a list of the different chemicals they can offer with associated prices. In all these cases, of course, while it may serve everyone's purposes to have a list to work from in what otherwise can be a rather nontransparent markets, producers cannot escape the laws of supply and demand and inevitably have to adjust their prices from time to time in light of actual or prospective market imbalances.

Negotiated Pricing

The term *negotiated pricing* is used here to describe the various pricing systems that lack a formal institutional focus and are typically the product of bilateral contracts made directly between buyers and sellers (i.e., principal to principal). It includes a wide range of trading and pricing types, with variable degrees of sophistication, and tends to apply to products that are either too differentiated in their nature to permit exchange trading, too bulky, or too unstable chemically to be stored for long periods in warehouses, or to products for which the markets are too small scale and fragmented to warrant exchange trading.

At one pole are the bulk commodities iron ore and coal, the prices of which are the subject of annual, formal, high-profile negotiations between buyers and sellers. The scale of the trade in these commodities and its importance to the economies of the participating countries make these negotiations almost political in nature. After several months of preparatory meetings, conferences, and subtle public maneuvering, negotiations on seaborne iron ore prices typically begin in earnest early in the calendar year. For historical reasons, contracts in Europe tend to run over the calendar year and those in Asia over the financial year starting in April, though both tend to be negotiated simultaneously. Although most producers and buyers of seaborne iron ore participate in this process, attention inevitably focuses on the larger producers (currently BHP Billiton and Rio Tinto out of Australia and Vale from Brazil) and the larger steel mills in Europe, Japan, and China. Although negotiations can often drag on for months, after a deal has been struck between any two companies, the convention is that all participants in the market automatically adopt this as the "benchmark" for pricing during the remainder of the year, with discounts and premiums to the benchmark grade of ore applied according to the precise form in which the ore is to be sold (lump, fines, or pellets), the mineral type, and the iron content and impurities (e.g., sulfur and phosphorus). Allowance for freight costs between seller and buyer may be treated as part of the discount structure or else may be subject to direct negotiation, depending on whether the product is being traded FOB or CIF. With FOB, the buyer pays for the transport; with CIF, the seller pays.

Seaborne trade in coking coal and steam coal is subject to a similar process of negotiation but with a slightly different cast of players. While the steel mills are the primary purchasers of coking coal, seaborne steam coal is brought by electrical power companies. In common with iron ore and coking coal, the buyers are heavily concentrated in Europe and Asia. Prices for iron ore and coal in North America are settled independently of these processes but nevertheless track those in the seaborne market quite closely because North American companies are active participants (albeit not generally large ones) in these seaborne markets.

Although this process of annual negotiation has undergone severe strains over the years, notably during periods of substantial shortage or surplus, it has generally served the interests of producers and buyers to persist with it and to lock in multiyear contracts with only the price to be negotiated annually. The operators of large continuous-process plants such as steel mills and power stations like to have secure and consistent supplies of known-quality raw materials and to know what they expect to pay for them. For the producers, the same predictability is useful for planning production and evaluating growth opportunities. Indeed, without the certainty of long-term contracts, mining companies have often in the past been reluctant to commit to major investment in the expansion of capacity for bulk commodities, given the capital-intensive nature of their production.

However, change is beginning to come to the trade in bulk seaborne commodities. In the important Asian market, the rapid growth of China's steel production has been shifting the focus from Japan, the traditional leader of negotiations in the Asian theater. The severe shortages of iron ore experienced in the first decade of the century by the rapid growth of China led to the emergence and then rapid growth of a spot market for iron ore, based on production from small producers in China and India. In turn, this gave rise to proposals for the development of an official index of spot trades to provide a more sensitive and timely indicator of market developments. The intention was that this should inform negotiations on longer-term contracts and permit some movement away from pricing as a once-for-all annual event. At the end of 2008, after months of discussions with industry participants on methodology, the journal *Metal Bulletin* launched a price index for iron ore fines imported into China.

Trade in steam coal has gone further down this path. The growing spot market in steam coal led a number of large coal producers and power utilities in 2001 to set up an Internet system for trading cargoes of steam coal, called globalCOAL. Based on trades among companies using its Internet platform, globalCOAL generates indicators of spot and forward prices of coal, which can be used to benchmark trades outside the market as well as permitting hedging and price risk management in the industry. Although annual contracts are still

widely used in the industry, there is a close interplay between spot and contract prices.

Raw materials used for the production of refined nonferrous metals are often traded on a hybrid basis, containing elements of both terminal market pricing and negotiated pricing. When a copper mine contracts to sell copper concentrates to a custom (i.e., third-party) smelter or refinery, it will typically do so based on the LME price in the 3 months after delivery, applied to the copper content of the concentrate being sold, less a deduction intended to reflect the negotiated cost of smelting and refining. Most such concentrates are sold on multiyear frame contracts with the terms for smelting and refining negotiated annually or biannually according to market conditions. These negotiations are quaintly referred to in the trade as the "mating season." Additionally, a smaller spot market acts as an indicator of the near-term balance of supply and demand.

The deduction from the copper price, which the smelter refinery must accept in order to acquire a supply of concentrate and which is the focus of the negotiations with the miners, is based off the percentage of metal in the concentrate that is considered recoverable (known as payable metal) and consists of two principal elements: (1) the treatment (or smelter) charge (TC), which is applied per metric ton of dried concentrate (dmt) on payable metal and (2) a refining charge (RC), which is applied per pound of payable copper. Collectively these are referred to as TC/RCs and are sometimes expressed as a percentage of the refined metal price. The level of TC/RCs is determined by the balance of supply and demand in the concentrates market (which may or may not be in step with the balance of supply and demand in the market for refined metal); the tighter the market for concentrates, the lower the TC/RCs.

By way of a worked example, take the case of a 30% copper concentrate in which the payable metal content is deemed to be 29%, the TC is $70/dmt, the RC is 7¢/lb and the LME price is $4,410/t ($2/lb). On the basis of these assumptions, each metric ton of concentrate contains 639 lb of copper (29% of 2,204.6 lb). The cost of the TC in this case will equate to 70 (the rate of the TC) times 200 (the LME price of copper quoted in U.S. cents) divided by 639, which is 21.9 ¢/lb. The RC has already been given as 7¢/lb so the miner will receive 171.1¢/lb of the payable metal in the concentrate (200 – 21.9 – 7), or 85.5% of the prevailing copper price. Other adjustments will be made to reflect unwanted impurities in the concentrates (e.g., arsenic and mercury) and recoverable by-products such as gold and molybdenum.

Similar arrangements to those for copper concentrates apply also to lead and zinc concentrates, though with different formulas. For aluminum, a common practice is for smelters to purchase their alumina on the basis of contracts specifying a given percentage of the prevailing LME aluminum metal price. The percentage is fixed through negotiation and generally falls into the 10%–20% range. The consequence of this arrangement is that aluminum and alumina prices tend to move broadly in step with each other, helping to reduce smelter costs in adverse market conditions and spread the benefits when prices are high.

Negotiated pricing is the norm for finished metals and alloys, which are not traded on metal exchanges. These include bulk ferroalloys such as ferromanganese and ferrochrome, special steel additives such as molybdenum and tungsten, minor metals such as cadmium and bismuth, and a variety of other specialist mineral-based products such as titanium-containing beach sands and uranium. The pricing of many of these is a function of direct negotiation between producers and consumers of the metals. However, because of the fragmented nature of some of these markets and some of the producers' remoteness from markets, it is typically the case that traders play a bigger part in these markets than with the major nonferrous metals or the bulk mineral commodities. These traders seek to make a profit by intermediating between buyers and sellers and by using their knowledge of these markets. Based usually near major demand centers where they can closely monitor developments in the market, trading companies can provide a useful service to producers who lack such knowledge or else who view marketing as outside their core competence. Historically, traders have accounted for a large part of the supply for these metals coming out of regions such as China, the former Soviet Union, and Africa.

By the nature of the processes involved, pricing in these markets is much less transparent than is the case with the other metals already discussed, and quite often this suits the participants in these markets. Reference prices for many of these commodities, derived from telephone surveys and national customs data, are regularly published by the trade press, and these provide an indication of trends in the various markets, although the reliability of these quotations varies. In some cases and at regular intervals, traders and consultants in the business make the prices at which trades have been conducted publicly available. In the case of uranium, for example, such a service is provided by TradeTech and Ux Consulting Company.

The pricing of industrial minerals is broadly similar to that for these more minor metals; that is, prices are normally established on the basis of negotiation between producers and consumers. The need for this is enhanced in the case of industrial minerals by the wide range of forms in which these products are sold and the specificity of the characteristics that consumers of these products are looking for. Thus, price variations for an industrial mineral may follow, among other things, chemical composition, grade, color, form of packaging, and delivery location (O'Driscoll 2006). To take but one example, prices of barium minerals will vary according to whether the product is deemed paint, drilling, or chemical grade, the amount of Ba_2SO_4 (barite) contained, the mesh size of the micronized product (if being used in paint production), the size of the bags in which it is being delivered (or if it is being sold in bulk), whether it is being sold FOB or CIF, and to which port the FOB or CIF pricing applies.

Prices for a wide range of industrial mineral products are published in trade journals, most notably *Industrial Minerals* and *Mineral PriceWatch*. These products range from clays such as bentonite and kaolin; inputs to chemical production such as borates, sulfur, and salt; refractory products such as chromite, magnesite, and bauxite; glass-making products such as lithium minerals, feldspar, and silica sands; coaters and fillers such as talc, wollastonite, and calcium carbonates; and fertilizer minerals such as potash and phosphate rock. Because of the wide range of different forms, published prices are typically quoted in ranges, and care needs to be taken in using the prices because some are more reliable than others. Because industrial minerals are among the most challenging products to price when evaluating mineral deposits, there is no substitute for talking with potential customers about the salability of the specific mineral under evaluation.

Price Behavior

The cyclical nature of mineral demand, discussed earlier, and the inflexibility of its supply mean that all mineral commodities show some degree of cyclicality in their price behavior. Market structure and the pricing systems employed are also important factors in determining price behavior. Markets that permit the involvement of speculators are more prone to large and rapid price movements than those that do not. Markets for minor metals, which lack transparency and have a large trader presence, are also susceptible to high price volatility, especially when they are produced as a by-product of another more economically important metal and where supply is accordingly inflexible. Conversely, in markets where supply is concentrated and producer pricing is employed, or where prices are fixed on an annual basis, it is likely that the price variability will be less.

Figure 2.2-8 shows an index of price variability for a variety of metals and minerals between 1970 and 2007. In line with what one might expect, the bottom part of the chart (indicating least-price volatility) is dominated by industrial minerals and bulk commodities while the top of the chart is dominated by minor metals and by-products. The major nonferrous metals fall around the middle of the table, as do gold and platinum. (The uranium price used is the spot price; contract prices would show less volatility.)

This comparison points up some important differences in the adjustment process of different markets. In markets where there is a buyer of last resort to deliver to, such as the LME, producers can pretty much always sell their product—at a price. The consequence of doing so, however, is that stock accumulates and price likely declines. Eventually prices will reach a point where marginal producers are losing money on a scale where they are forced into closure. Lower-cost producers, meanwhile, will almost certainly not be cutting back and indeed could even be seeking to increase their output as a means to reduce their unit operating costs. Under these sorts of conditions, prices have an inevitable tendency to overshoot.

For the many commodities that do not have a market of last resort, the pressure to respond to market imbalances is more immediate. If producers cannot find a buyer then, after they have filled their own storage space, they will generally be forced to cut their output to what the market will bear. In other words, while for base and precious metals the strain of market adjustment is principally taken on price and on a few higher-cost producers, for bulk and industrial minerals the strain is taken more on volume reductions across the industry and through the reduced efficiency of plants operating at less than full design capacity (Humphreys 1991). In the case of a number of internationally traded industrial minerals, this adjustment through volume modulation rather than through sharp price swings is facilitated by the relatively small number of suppliers and by the importance to them of maintaining good long-term relationships with customers. For these reasons, and because of general inertia, prices fixed on an annual basis often respond to market changes with a lag. A good example of this is provided by the iron ore market, where prices typically move in line with changes in steel production in the major ore-consuming areas in the previous year.

In conclusion, and returning to the point made at the beginning of this chapter, it is important for those undertaking the analysis of mineral projects not only to have a good sense of the product's likely average price over the life of a project but also to understand the associated risk of price volatility. For this purpose they will need a clear understanding of the product's price history and an appreciation of the market structure and the specifics of the processes of price formation.

SOURCES OF PRICE INFORMATION

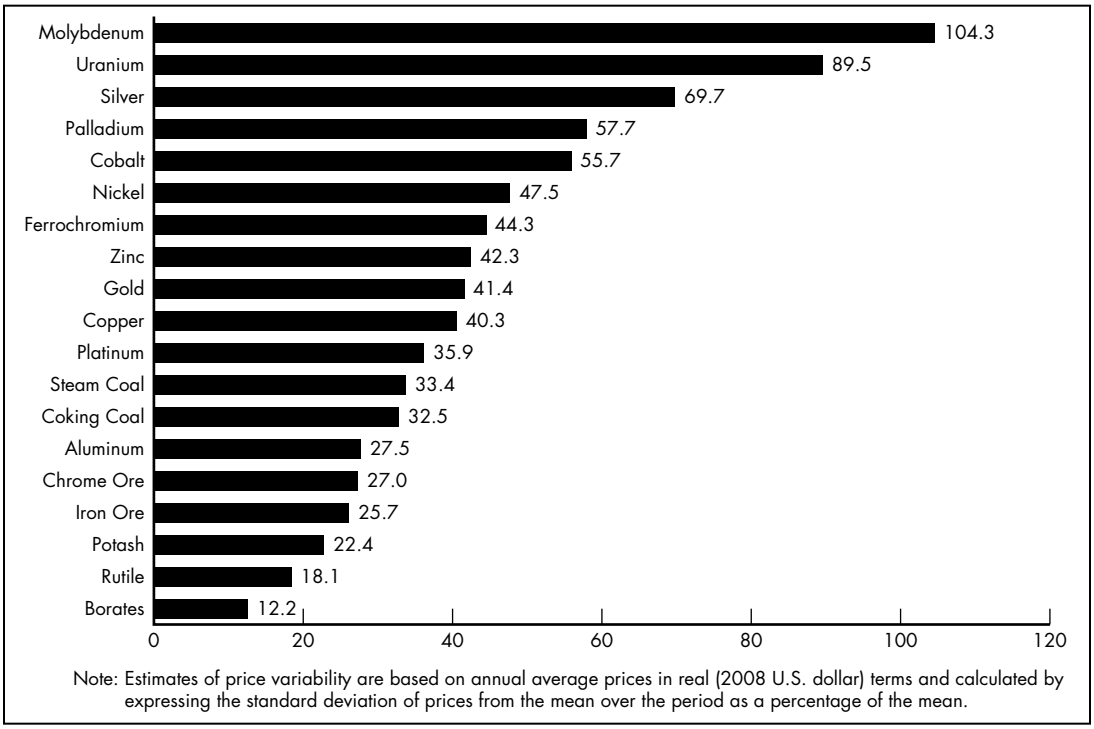
The development of electronic information systems means that price data on metals and minerals are much more readily accessible than in the past. However, it does not necessarily mean that the quality of this data has improved nor that the particular data needed can be obtained free.

Price data for metals traded on metal exchanges are generally available through the exchanges' Web sites. Thus, data on LME prices can be found at www.lme.com, while that for NYMEX can be found at www.nymex.com. Historical price data can be found in a variety of statistical publications, such as those of the World Bureau of Metal Statistics (www.world-bureau.com) and the American Bureau of Metal Statistics (www.abms.com), and are available on line from a number of commercial data suppliers, such as Reuters and Bloomberg. These metals are also well covered by research analysts at the major banks, whose publications are often a good source of information on the nonferrous metals and other metal and mineral products.

Price fixes for the precious metals can be found on the Web sites of the LBMA and LPPM, at www.lbma.org.uk and www.lppm.org.uk, respectively. The World Gold Council Web site, www.gold.org, is a good source for gold prices as well as a range of other gold-related information. Johnson Matthey posts the prices of the platinum group metals it sells on its site at www.platinum.matthey.com.

Commercial publications, such as *Metal Bulletin* in the United Kingdom (www.metalbulletin.com) and *Platts Metals Week* and *American Metal Markets* in the United States (respectively, www.platts.com/metals and www.amm.com) are important sources of information on the prices of major metals as well as on a range of minor metals, ferroalloys, and mineral raw materials. Some of their price series are based on their own regular surveys and are copyrighted. Prices of mineral raw materials such as base metal concentrates or alumina are generally hard to obtain publicly on a consistent long-term basis and generally have to be obtained from proprietary sources such as the publications of CRU (www.crugroup.com), Brook Hunt (www.brookhunt.com), or AME Mineral Economics (www.ame.com.au). Proprietary research on the precious metals is available from GFMS (www.gfms.co.uk).

The trading and pricing of bulk commodities are fairly transparent affairs and well covered in the trade press. An important source of information on the seaborne trade in iron ore and coal is *The Tex Report*, published daily in Japan. The Tex Report Ltd. (www.texreport.co.jp/xenglish) also publishes annually manuals on iron ore, coal, and ferroalloys, which contain a wealth of detailed historical information. The most authoritative source for the North American iron ore industry is the *Skillings Mining Review* (www.skillings.net). Prices for steam coal trades are posted on globalCOAL's Web site at www.globalcoal.com. The Organisation for Economic Co-operation and Development's International Energy Agency (www.iea.org) is a rich source of information on the trade and



Source: Data from ABARE 2008, LBMA 2008, LME 2008, LPPM 2008, *Metal Bulletin* 2008, USGS 2008, and Ux Consulting 2008.

Figure 2.2-8 Variability of metal and mineral prices, 1970–2007

pricing of energy products generally and publishes an annual report on coal called *Coal Information*.

As discussed previously, price data on industrial minerals are commonly hard to obtain and difficult to interpret. Many producers like the lack of transparency because knowledge of what is going on in their market is part of their competitive advantage. The publication *Industrial Minerals* (www.mineralnet.co.uk) is one of the few sources of information on these minerals and routinely publishes price data on such minerals in its flagship journal as well as in its specialist publication *Mineral PriceWatch*. However, while these are adequate for getting a general indication of price levels and of price trends, evaluators needing a detailed assessment of the likely prices for a mineral from a particular deposit will generally need the assistance of a consultant specializing in the commodity concerned.

Finally, there are a few government-sponsored sources of information on metal and mineral prices. The best known of these in the U.S. Geological Survey, which, building on the work formerly carried out by the U.S. Bureau of Mines, has a large database of mineral prices going back many years. These are easily accessible at http://minerals.usgs.gov/minerals. Although this covers metals and minerals of all types worldwide, it is particularly useful as a source of information on the smaller and more obscure mineral commodities and on prices within the United States. The Australian Bureau of Agricultural and Resource Economics (ABARE) and its Web site at www.abare.gov.au are also valuable sources of price information on commodities, especially those for which Australia is an important global producer. Similar information is available from Natural Resources Canada at its Web site, www.nrcan-rncan.gc.ca.

ACKNOWLEDGMENTS

The "Metal Exchanges" section borrows heavily from the corresponding chapter in the previous edition of this handbook (Strauss 1992).

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CHAPTER 2.3

Market Capitalization

Fiona Perrott-Humphrey

Market capitalization is the value that the equity market places on a company or a group of companies. Over the bulk of 2003 to 2008, the FTSE 350 Mining Index (the market capitalization of the world's most important mining equity market) rose strongly in relative terms against that of the FTSE Med 100 (the top 100 companies listed in London) (see Figure 2.3-1). This relative rise reflected a belief that the structural growth of industrial giant China had created a *supercycle* of demand for commodities to fuel its development that would last decades. This investment thesis saw the risk premium of historically cyclical mining companies reduced as they were treated more like growth stocks with strong underlying cash flows.

Figure 2.3-1 also shows, however, that since mid-2008, these same shares have seen a pullback of more than 50% from their high point, based on fears that the turmoil in financial markets is feeding into the real economy and that even the powerful growth engine of China would not offset demand falls for commodities elsewhere.

These dramatic moves in market capitalization are no mere lottery. The fundamental driver of valuation is the multiple the equity market places on the net present value (NPV) of future cash flows coming from mining projects. To arrive at the NPV, we need to understand factors affecting both

- Projected cash flows and the discount rate applied to them, and
- The multiple that the equity market places on that NPV.

By their very nature, mining companies cannot easily be valued on measures used in industrial and financial sectors—that is, a straightforward price/earning ratio; EV/EBITDA multiples (equity plus debt divided by earnings before interest, tax, depreciation, and amortization); and cash flow multiple or dividend yield of current year's results, because

- In the early stages of development of a mining resource, all potential earnings are in future years;
- Each single resource within a mining group's portfolio has a finite life and is a wasting asset; and
- Variations of grade within each mineral deposit lead to a volatile production profile over the life of the mine and associated differing unit costs.

In addition to these factors, the prices of the commodities produced by mining companies can vary dramatically over an economic cycle, affected as they are by movements in demand, supply, and inventories. At differing points in the economic cycle, therefore, the multiple that the equity market places on future projected earnings also varies. Figure 2.3-2 attempts to encapsulate the influences on the market capitalization of the miners through an economic cycle.

The bulk of this chapter will expand on the factors that affect projected cash flows and the discount rate for mining groups. However, it is worth noting at this point that stock markets contain a wide range of mining companies. The spectrum includes

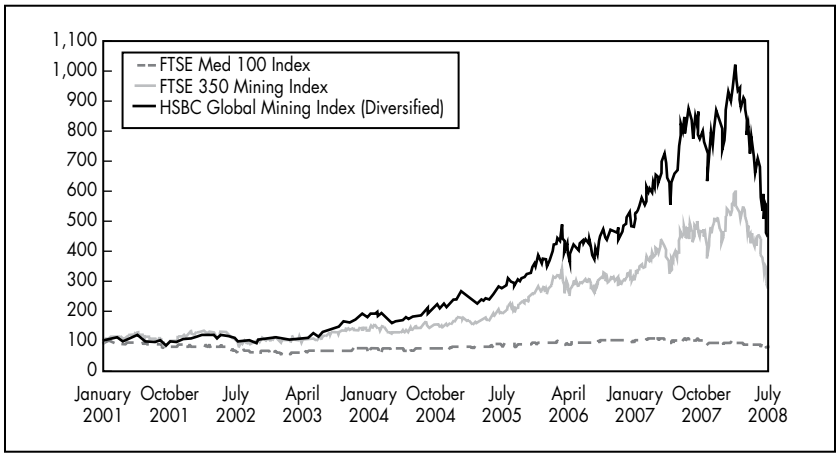
- Junior exploration companies (ranging from owners of a resource that shows potential, but is still being proven by drilling, to companies where a project is under construction and metal production proposed within a relatively short-term time period);
- Single commodity companies that nevertheless have a range of both producing, developing, and exploration projects, and can spread across more than one geographic region (e.g., Antofagasta plc; Impala Platinum Ltd.); and
- Large multinational diversified mining groups, spread across the whole spectrum of project life cycles, but which also produce a wide range of different commodities and operate around the globe (e.g., Rio Tinto and BHP Billiton).

Figure 2.3-3 shows a graphic representation of theoretical valuation of a *single* mining project. For the investor, his investment risk (the y-axis) reduces as the knowledge of the mineral asset is expanded and as better data on the "modifying factors" that will affect the economic mining of the mineral asset become more reliable (the x-axis).

VALUATION OF JUNIOR MINERS

Figure 2.3-3 certainly applies to the valuation of junior mining groups, which are frequently listed on separate stock exchanges (e.g., the Toronto Stock Exchange in Canada and the Alternative Investment Market [AIM] in London). The regulatory requirements of these exchanges reflect the reality that such start-up companies have no history of profit

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Source: Bloomberg 2008.

Figure 2.3-1 Performance of mining indices versus FTSE Med 100, 2001–2008

or cash flows such as those required by the main board exchanges. Because some are not yet producing cash flows or earnings, valuation methods applied to their theoretical future earnings potential are different from those applied to producing miners.

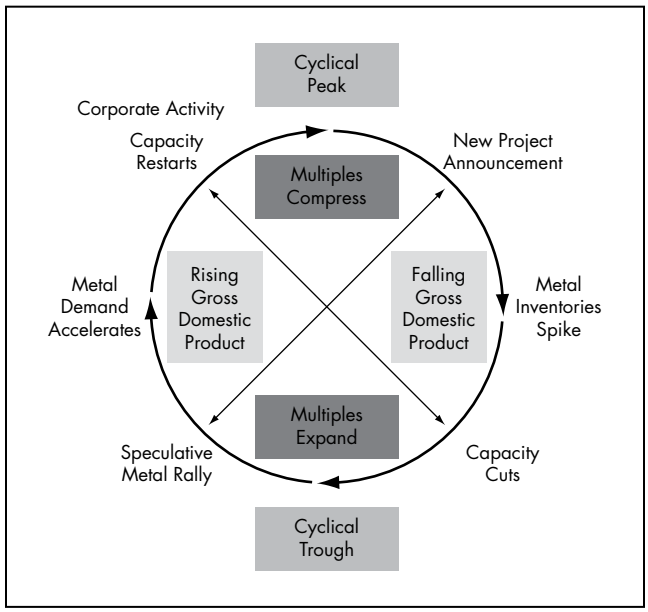
Valuing exploration and development companies in the resource sector is fraught with challenges because of the variability of the information that becomes available through the exploration and development phases, and the fact that each mineral deposit is in some ways unique in its characteristics. For investors with relatively limited access to internal company information and usually no opportunity for a site visit, the process is made even more challenging.

There is an oft quoted, slightly tongue-in-cheek maxim from the mining world that says: "Why spoil a good deposit by mining it?" The implication is that in many ways an exploration junior is like a journey and the stock market believes it is better to travel hopefully than to arrive. When you arrive, after all, and start producing an output that is sold at a fixed price, your company can eventually be valued on the metrics applied across most other sectors of the stock market—that is, price/earning ratio; cash flow multiple, and dividend yield. And the eventual outcome may well be less exciting than the projections built off limited data through the exploration and development phases (e.g., Figure 2.3-3 shows that the theoretical market capitalization of a project is likely to take a dip when the project is commissioned), and probable disappointments occur in the timing/scale of execution of the project versus targets in the feasibility study (termed *preproduction gap*). This fall can then be reversed once revised production targets start to be met.

In that sense, some commentators have compared analysis of early-stage mining juniors to that of biotech companies—that is, not only are the data patchy in the early days of the companies' undertakings, but there also exists a risk that the final outcome could be a complete failure—either no economic mineral body to exploit or no successful biotech product. This is clearly a very different framework from valuation of most other small companies, and the common methods used to value juniors are listed next.

Net Asset Value

In theory, the value of a company equates to the net asset value stated in its report and account. In practice, for junior mining companies in particular, this can be a meaningless figure, for in the early stages of exploration, it reflects no more than the money already spent on the project. This could be overstating its worth (if eventually no economically viable mineralization is found) or severely understating it (if the deposit is proved up according to early projections or even contains further unexpected *blue sky potential*).



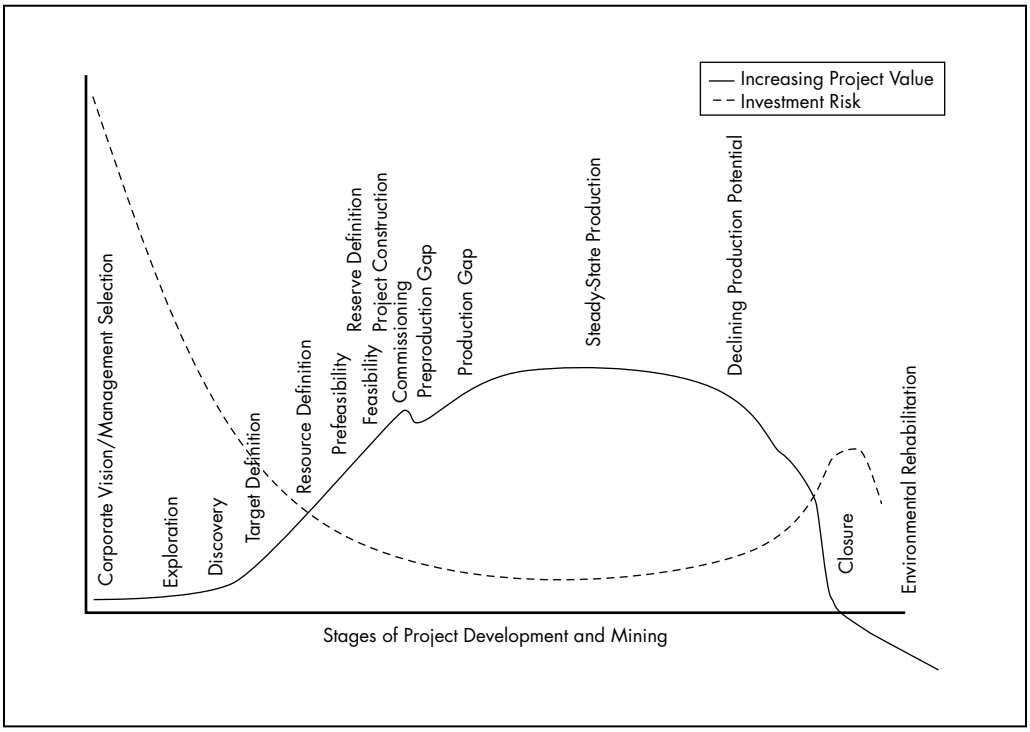
Source: N M Rothschild 2008.

Figure 2.3-2 Valuation drivers through the economic cycle

Ore in the Ground

In the gold industry, a very basic guideline that is used for valuations when little real data are available from the property is: "gold in the ground." This valuation will obviously shift along with the ruling commodity price and average industry costs (e.g., in 2007, analysts typically used US$30/oz of reserves for an exploration property but moved up to US$90–US$120 when the company proved that it could actually produce metal).

Companies can frequently trade at premia to the underlying value as calculated on this basis, if investors perceive upside in



Source: Diamond Core Resources 2004.

Figure 2.3-3 Value of mining project versus investment risk

the probability of resources being converted into reserves in the future. There is thus a case for including some kind of weighted value of measured resources as well as reserves, guided by the conversion ratio that the company provides.

There are, of course, significant weaknesses with this valuation tool, primarily the following:

- The wide variances in the *actual cost* of extracting the metal from the wide range of different types of deposit. In addition to the mining costs, there are the logistical and political risk costs associated with operating in a number of far-flung emerging market regions. Finally, processing costs vary dramatically according to the metallurgy required, particularly in the case of platinum group metals, which occur in a combination of five or six elements in differing ratios.
- The *currency-adjusted* cost, which can move significantly as the U.S. dollar moves against the so-called resource currencies, such as the Australian dollar and the South African rand, and possibly even the Chilean peso.
- The variance in regional categorization of resources and reserves. On AIM, for example, a range of standards is used to classify reserves and resources, with the Russian gold producers and a number of companies in the former Soviet Union using the Russian categories of reserves and resources. No formal publication has been released to enable investors to translate the more complex Russian categories into the equivalents provided by the Joint Ore Reserves Committee (Australia), the 43-101 system (Canada), or the South African Mineral Resource Committee.
- The assumed operating margin obviously moves with significant shifts in the commodity prices.

Probability-Weighted NPV

NPV is the most widespread valuation method for resource stocks and in theory is best suited to nonproducing resource companies, given the nature of the development timeline—that is, upfront exploration and development spending before the medium- or long-term benefits flow from producing metal from the mineral asset. The latest guidance note for mining, oil, and gas companies listed on the AIM (London Stock Exchange 2006), issued by the London Stock Exchange, states that the Competent Persons Report should include (among other things): "An estimate of net present value (post-tax) at a discount rate of 10% of reserves (or equivalent depending on Standard used) analysed separately and the principal assumptions (including costs assumptions, effective date, constant or forecast prices, forex [foreign exchange] rates) on which valuation is based together with a sensitivities analysis."

Because of the nature of commodity markets, the scope for variances around many of the items previously listed is significant even when the data available on the mineral asset are becoming relatively well fleshed out. This is clearly so much more the case in the early stages of development, when the geological, and hence all the mining, parameters are prone to major revisions as more data become available.

In the biotech industry, one of the analytical approaches is to develop a discounted cash flow based on the assumption that the products will be successfully developed and sold in a way that is comparable to established industry peers, and then to weight that outcome based on the probabilities of different levels of success in order to come to a final valuation. This approach is also being adopted in different formats in the junior mining sector. The argument for assigning weightings to the four key risk factors in the sector's life cycle based on

probabilities is a compelling one. The risk factors represent four possible downsides:

1. Geological and exploration risk (versus early estimates of the nature of the ore body)
2. Development risks
3. Operational risks (versus feasibility study projections)
4. Financial risks

Offset against these factors would be a weighting for a fifth factor—possible blue sky potential not included in an NPV based only on proven reserves.

VALUATION OF PRODUCING MINING COMPANIES: EXTERNAL FACTORS AFFECTING CASH FLOWS

Typically, investors looking at industrial companies have only one element to consider in interrogating management's projected revenue figures: projected sales volumes at a fixed price. Mining companies provide significantly more of a challenge in this sphere in which, of the three key drivers of the revenue line, two are completely outside of management's control. (Hedging does not allow one to control pricing, only to protect against movements in them.) The three key drivers are

1. Commodity prices;
2. Currency trends, which will determine the actual revenue received in local currency terms translated from U.S. dollar prices; and
3. Production capacity (discussed in the "Valuation of Producing Mining Companies: Internal Factors Affecting Cash Flows" section).

Commodity Prices

Quoted in U.S. dollars, commodity prices are typically determined on terminal markets, depending on cyclical supply and demand factors. The latter were historically driven by industrial consumers of the metals, but since the 1990s, financial investors' influence at the margin has been growing. Table 2.3-1 shows the categories of metals and minerals and how their prices are set. The terminal markets provide the most visibility on future price trends, whereas those not traded on terminal markets tend to be relatively opaque.

In order to assess how realistic the management assumptions are for future revenue streams, investors are unlikely to become experts in all of the commodity and currency markets, but there are numerous sources of external metal price and currency forecasts, including (but not limited to) most stockbroking research departments plus independent metal consultancies such as CRU, Bloomsbury Minerals Economics, and GFMS. The platinum group metals market is covered by Johnson Matthey in a biannual publication, and diamond prices for different ranges of gems are available from industry sources, the most well known of which are Rapaport and Lenco (www.roughdiamondprices.com).

These external sources are useful not only for the absolute price forecasts that they may provide but also for their information on key aspects of the *structure* of each specific commodity market. That is, prices on terminal markets are influenced not only by overall supply and demand factors but by shorter-term levels of inventory metal held either by end consumers or in market warehouses. In bulk and industrial minerals markets, on the other hand, pricing negotiations are influenced by the level of concentration of industry production (e.g., iron ore) or consumption among that market's players.

Table 2.3-1 Price setting for metals and minerals

Category	How Prices Are Set
Base metals (Cu, Ni, Al, Zn)	Terminal markets such as the London Metal Exchange (LME)
Precious metals (Au, Pt, Pd, Rh, Ag)	Terminal and bullion markets
Diamonds	Influenced by the prices set by largest distributor, De Beers' Diamond Trading Company
Bulk minerals (iron ore, coal, bauxite, chrome, manganese, cobalt, uranium)	Contract prices set by negotiations with large consumers, either annually or more frequently, while spot prices reflect more frequent fluctuations driven by short-term demand and supply factors (and coal markets starting to reflect some of the features of terminal markets)
Industrial minerals (titanium dioxide, borax)	Influenced by individual contracts with major consumers

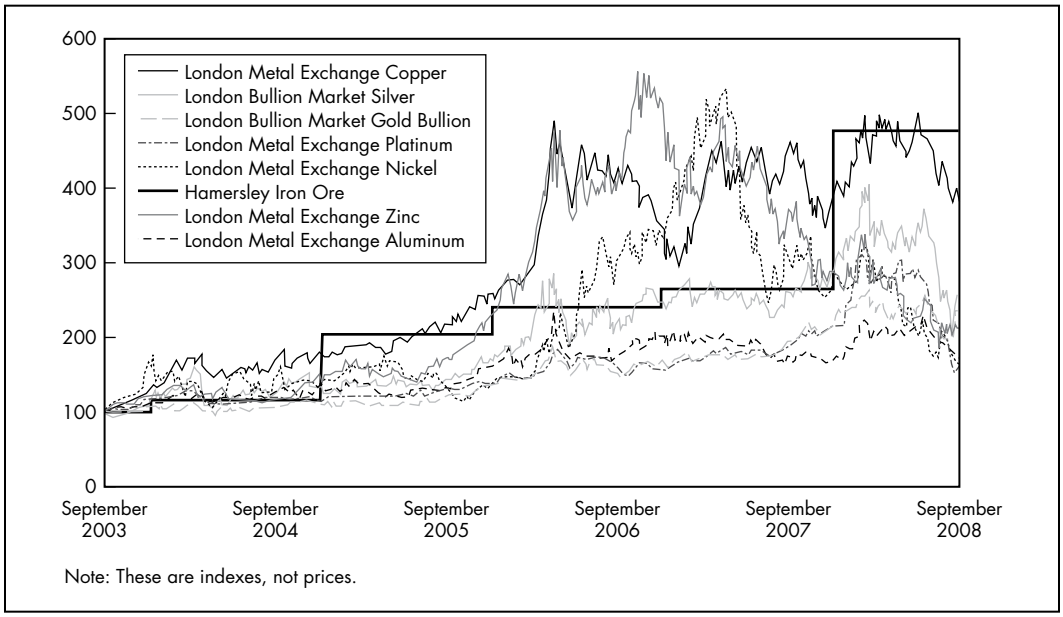
To assist with the projection of future mining revenues, many of the major mining companies provide some kind of earnings/NPV sensitivities with respect to possible moves in either currencies or commodity prices. Rio Tinto, for example, in its investor presentations will state that, all other things being equal, a 10% move in the Cu price or in the Australian dollar would have a consequent percentage impact on net earnings. These are key in running what-if scenarios on future cash flows at inflection points in the commodity cycles, in view of the scale of the moves in some commodity prices from the bottom of the cycle to the peak (see Figure 2.3-4).

Spot or Long-Term Prices

Given the extent to which commodity prices can move over a cycle, revenue forecasts should be clear as to whether they are based on spot or average long-term prices. At any juncture, spot prices should be treated with some caution, as individual mining companies may have off-take agreements that are tied into contract prices (as well as volume parameters). This is particularly true of smaller, more opaque markets such as uranium, cobalt, and molybdenum, where spot prices can vary significantly from long-term contract prices at some points in the cycle. In most commodity markets, there is also a sliding scale of prices where premia to quoted benchmark prices may be available for better-quality or more highly beneficiated products.

Finally, it is worth checking whether the *theoretical price* will always be available to a mining company operating in a new frontier region. Three particular issues which need to be clarified in this regard are

1. Whether there are restrictions on the potential buyers of the end product, which could influence the price paid (e.g., gold in Zimbabwe);
2. Whether revenue proceeds are able to be remitted freely out of the country where the mining operation is located; and
3. Whether the company is being required by any of its financiers to hedge (sell forward at a fixed price) any element of its future production in order to mitigate political risk considerations.



Source: Bloomberg 2009.

Figure 2.3-4 Rebased commodity prices, 5-year view

VALUATION OF PRODUCING MINING COMPANIES: INTERNAL FACTORS AFFECTING CASH FLOWS

The graphic representation of the life cycle of a mining project (Figure 2.3-3) points to the factors that affect cash flows in a single-project company. Any larger mining group would simply be a portfolio of such projects, often at very different stages of development. Those mining groups typically rated most highly by the equity investors have the right balance of such projects to ensure that, as a major producing asset starts to tail off in terms of grade and production life, others are waiting to replace it, thus achieving a relatively stable or growing production profile. This is the part of the revenue line that management teams are expected to influence, even as sales and earnings remain vulnerable at another level to the vagaries of the commodity price cycle.

Two key internal components will affect cash flows from any mining project:

1. The *size and quality* of the mineral deposit, which will determine the production potential. In the early stages of exploration, when confidence is low for geological information, the potential is expressed as resources. At later stages, when drilling and geological modeling increases levels of confidence, the potential is expressed as reserves. The size of the economic deposit will determine the life of the mine and the economies of scale to be achieved in mining it, whereas the quality refers to the grade of metal found in the surrounding "waste" material (i.e., the higher the ratio of metal to waste in the deposit, the more economic the project).
2. The *cost of extracting and treating* the ore to turn it into a salable product (either an intermediate concentrate that requires refining by an independent refiner or as a fully refined end product—for example, a gold bullion bar). For a given commodity price, the cost of production can differ substantially between projects (and sometimes countries), thus affecting the profit margin earned.

Table 2.3-2 Production cost standard

Guideline	Features
Cash operating costs	All direct and indirect operating cash costs related directly to the physical activities of producing metals, including mining, processing, and other plant costs; third-party refining and marketing expense; on-site general and administrative costs; and net of by-product revenues earned from all metals other than the primary metal produced at each unit
Total cash costs	Same as cash operating costs plus royalties and mine production taxes
Total production costs	Same as total cash costs plus depreciation, depletion, amortization, and accretion of asset retirement

Equity investors thus rate mining companies according to their relative position on the industry cost curve. Due to the lack of consistency across the industry in reporting cash costs of current production, it is worth clarifying what items are included in various levels of cost categories. Table 2.3-2 contains broad guidelines that are followed by conservative investment analysts.

In addition to the cost items listed in Table 2.3-2, a number of other (sometimes indirect) expenses should be built into any projections that cover the life of the mine:

- **Exploration and mine development expenses, which are typically capitalized.** While not related to the current production, hence the exclusion from the current cost per ounce, this expenditure needs to be incurred to ensure that there are ounces to mine in the future. A key element of this is stripping. The correct measure of the minimum cash expenditure required should thus be "total cash costs plus the essential capital expenditure required for ongoing mining."

- **Financing costs.** Although not included in Table 2.3-2, these costs are a crucial element for junior miners in particular (because they lack cash flow from producing assets), both in ongoing total cash costs and in NPV terms. During the mining boom of 2003–2008, junior miners had relatively easy access to equity financing, usually at rising share prices each time, and this element of costs could thus have been glossed over. Because of the cyclical nature of the industry, however, all aspects of the company's financing issues are also cyclical—access to debt financing alternatives; current cost of debt alternatives and whether this is a fixed or variable element of costs; nature of debt covenants; whether the lenders demand an element of price hedging, which would cap the upside of the project in markets of rising prices; whether the presence of quasi-government lenders on the book imposes any related constraints on the company (e.g., International Finance Corporation); whether the company has fully covered the risks of borrowing heavily from local banks in some of the emerging markets areas; and whether companies have raised sufficient capital to fund their projects to completion.
- **Cost escalation over life of mine.** Because of the variable nature of each geological deposit, today's snapshot of a cost profile day is likely to be subject to significant change over the life of mine. Factors that can cause quantum moves to the cost profile include shifts in grade or stripping ratios, or the need to move from open-pit to underground operations. In addition, the global mining industry is currently experiencing cyclical cost escalation on the back of what has been (up to early 2008) the longest commodity price boom since World War II. It has driven up both the cost and availability of energy, skills, contractors, spares (e.g., tires), and materials. Longer lead times combining with cost escalation in new projects has seen capital expenditure estimates revised upward at almost every mining company reporting period, with implications for both NPV values and financing costs.
- **Smelting and refining costs.** These can also vary significantly over the cycle for a miner that does not have a fully integrated operation (toll smelting agreements are fixed or variable). The efficiency of the metallurgical process in treating more complex metals such as platinum group metals is a key competitive element among producers, as is the cost of energy in highly energy-intensive treatment processes such as aluminum smelting.
- **Royalties and taxes.** Although in theory these should be steady factors over the life of mine, in practice, commodity-related charges tend to be increased by governments as they watch the products' prices rise through the cycle or as new regimes take control in resource-rich nations. This trend of so-called economic nationalism has been evident on almost every continent and resulted in a rise in the risk premium assigned to many companies operating in emerging markets some 6 months before concerns about a global economic downturn emerged.

DISCOUNT RATES APPLIED TO PROJECTED CASH FLOWS

Even though this is a key factor in driving the end result of the NPV, there are no guidelines that hold across the board. The major areas of difference rest on assessments of cost of capital, political risk, and project development risk.

Cost of Capital

Although this should reflect a company's weighted average cost of capital (equity and debt), estimates have historically varied between key resources markets, with the North American analysts tending to use lower figures, ranging from 0% to 5%, in contrast to their counterparts elsewhere who use numbers ranging from 5% to 10% (and interestingly AIM's guideline number of 10% after tax). This whole spectrum of cost of capital is likely to shift as the long period of easy and generally low-cost finance of 2003–2008 ends in a cyclical economic downturn. Within these average statistics, however, the capital structure of individual companies and the cost of the financing available will also vary widely.

Political Risk

These discount factors are even more subjective (usually biased by proximity), primarily because each company is unique in its aggregate exposure to generic (or sovereign) country risk (depending on where its many projects are and to what extent it can benefit from the portfolio effect of one type of risk offsetting another) and to more specific project-related risk (still within the context of political risk assessment). In addition, most investors will be pricing political risk in different contexts, depending on their own portfolio spreads, ability to hedge, and even the requirement for a particular country exposure (where a mining company's discount rate is then priced relative to other sectors in that country, rather than against other geographic regions).

Project Development Risk

The key elements that equity markets consider most important in determining the success or failure of the vital development phase—turning a geological resource into ounces of metal sold—are the following:

- Security of tenure, given the need for most natural resource companies to operate in emerging market regions
- Key characteristics of the ore body, particularly grade
- Operating parameters of open-pit versus underground mine development
- Plant and metallurgical efficiencies
- Logistics, particularly when a mine is located in a remote region
- Management record in developing projects to production stage
- Environmental issues that demand increasing amounts of company time and resources, covering not only the physical environment (where actual legislation will govern company requirements) but also the socio-cultural component (where local communities are capable of stalling potentially viable ventures if their concerns are not addressed)

MARKET CAPITALIZATION: THE FINAL ELEMENT

Much of this chapter has covered the issues surrounding projected cash flows for mining companies and determination of the discount rate used to convert them into an NPV. The final element of a mining company's market capitalization is the multiple which the equity market places on this theoretical NPV. A number of factors affect it:

- **Company profile in terms of the diversity of the asset portfolio by geography and by product.** Typically,

the large diversified groups such as Rio Tinto and BHP Billiton trade at a premium to their peers because of the lower risk to cash flows if one commodity or region is adversely affected. However, it is also true that at certain points in a rising commodity cycle, investors seek to play more highly geared or single-commodity companies that could see higher earnings growth from a lower base. At these junctures, the diversifieds can be de-rated as "dull."

- **Growth potential.** Although large producers may be less risky, investors also worry that it becomes increasingly harder for them to achieve earnings growth from a high base. In a world in which it is a challenge to find new, large, low-cost resources to replace wasting assets, the market will frequently rate small or mid-cap producers more highly than their major peers. This can be particularly true in the gold sector, where the shorter average life of mine means that wasting reserves need to be replaced more frequently than in other sectors.
- **Point in the economic-and-commodity cycle.** Equity markets discount future earnings as far ahead as 18 months to 2 years. Figure 2.3-2 illustrates that (counter-intuitively) multiples compress as the economy heats up, and commodity prices/earnings top out and expand in the depth of a recession as profitability declines.
- **Merger and acquisition activity.** Historically, major upsurges in merger and acquisition activity have occurred at low points in the economic/commodity cycle. Companies with stronger balance sheets have sought to pick up assets at low prices from distressed sellers in order to cut costs and undertake consolidation/capacity closures. In the recent upturn, however, mining companies who believed in the supercycle were worried about a shortage of new high-quality assets to fuel their future growth in supply. A rash of activity thus occurred at a time of historically high metal prices. Either way, a belief that corporate buyers are seeking to add to their asset portfolios can distort valuation multiples away from those based on fundamental analysis.

MARKET CAPITALIZATION: DIFFERENCES WITH JUNIORS

As indicated earlier in this chapter, there can be key differences in valuing early-stage mining companies and major producers. Equally, it is the case that at times during the commodity cycle, indices of junior shares (such as those listed on AIM in London) can tend to show relatively high levels of volatility, not necessarily correlated to the performance of the FTSE Mining Sector Index. Reasons can include the following:

- Internal flow of information is the primary share price driver for early-stage miners, as mispricing frequently occurs on extrapolation of the earliest data. This is either followed by disappointments (e.g., Tran-Siberian Gold, which in 2006 announced that one of its three key deposits would not be economically viable on the technology originally assumed) or upgrades of blue sky potential (e.g., Peter Hambro Mining, which acquired new licenses in auctions during 2006, and, as it met development and production targets on current assets, the market priced in the upside on a similar basis for newly acquired development assets over the following year).
- Liquidity is lower than on the main stock exchange boards, so when new information emerges or when a small circle of original backers looks to take profits, the moves either way can be exacerbated.
- Funds flowing into the sector in an up-cycle can exacerbate price moves, due to the previous point, but when the cycle turns decisively or even has a short-term correction, lack of liquidity can also exacerbate downward price adjustments.
- Political risk shocks have a much more powerful impact, given the exposure of many AIM mining companies to no more than two or three major projects and their current propensity to explore in the new frontier regions without the portfolio insurance effect of the majors.
- Investment by a major in a junior can underpin the share price, but that is not a guarantee that the share price will be immune to either internal news flows or political risk shocks.
- Interest rate moves tend to affect those shares and sectors most reliant on future discounted cash flows for their valuations (rather than more defensive sectors valued on near-term cash multiples). Exploration and development mining companies fall into that category.
- Scarcity value can sustain juniors at what seem to be very high underlying valuations, when there is a perceived shortage of large new discoveries of a particular commodity. This was the case with most diamond juniors during 2006–2007, given the dearth of new supply sources announced in this sector for some years now.
- Primary commodity exposure will also be key over time, as price rises across the spectrum have not been of the same scale. In 2005, for example, iron ore price increases dwarfed those in any other commodities, seeing majors Rio Tinto and BHP Billiton outperform peers for some months. This was clearly a commodity in short supply among the juniors at that time.
- Dividend distribution potential is in most cases a distant prospect for most early-stage juniors, whereas substantial capital returns/special dividends have been a major driver of the main board mining sector during 2006–2008. When the market is focusing on dividends and immediate cash returns (usually in bear market phases), this becomes another potential reason for the two markets to diverge in their performance trends.

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CHAPTER 2.4

Investment Analysis

Michael Rawlinson

Investment analysis is the study of the likely return from a proposed investment with the objective of evaluating the amount an investor pays for it, the investment's suitability for that investor, or the feasibility of a proposed development. The result of the analysis would typically be a "buy," "sell," or "hold" action for a listed mining equity or "go" or "don't go" decision on a new investment in, for example, a new mine. Crucially, investment analysis is *not* the exclusive study of whether one company or asset is high quality or not. Good assets can be overpriced and poor ones too cheap; the investment analyst's job is to make money from his or her investments, not find great, well-run companies.

For the purposes of this chapter, the analysis behind making investments in listed equities from the standpoint of a sell-side investment analyst is looked at. (*Sell-side* refers to those in the investment community who make a living selling advice to investors. Their adjunct, the *buy-side*, are their investing clients.) In reality, the process of investment analysis is similar for the buyers of the assets (typically at fund management institutions such as pension funds, insurance firms, or hedge funds).

INTRINSIC VALUE VERSUS "CASTLES IN THE SKY"

Investment analysis is more than a straight analysis of the intrinsic value of an entity. In his book *A Random Walk Down Wall Street*, Burton Malkiel expands on this theme by suggesting at one extreme there is a "firm-foundation theory" behind investing—where intrinsic value should drive investment decisions—and at the other end is the "castles in the sky" theory of investing (Malkiel 1973).

The firm-foundation theory argues that each investment instrument, be it a common stock or a piece of real estate, has a firm anchor of something called intrinsic value, which can be determined by careful analysis of present conditions and future prospects. When market prices fall below (or rise above) this firm foundation, a buying (or selling) opportunity arises, because this fluctuation will eventually be corrected—or so the theory goes. Investing then becomes a dull but straightforward matter of comparing something's actual price with its firm foundation of value. In mining, the net present value, or NPV (see Chapter 2.3), is the primary measure of intrinsic value upon which rational investment decisions ought to be made.

In deriving a company's NPV, the mining equity analyst must estimate key drivers such as production, grade, and costs but also potential future growth and key macroeconomic variables such as commodity prices. Views on such factors—in particular, growth rates and key macroeconomic variables such as commodity prices—are extraordinarily cyclical, so the firm-foundation theory relies on some tricky forecasts that themselves move with market sentiment.

Malkiel's castles-in-the-sky theory of investing concentrates on the psychology of investing. John Maynard Keynes, the famous economist and successful investor, enunciated the theory most lucidly in 1936. It was his opinion that professional investors prefer to devote their energies not to estimating intrinsic values but rather to analyzing how the crowd of investors is likely to behave in the future and how during periods of optimism they tend to build their hopes into castles in the sky. The successful investor tries to "beat the gun" by estimating what investment situations are most susceptible to public castle-building and then buying before the crowd.

WHY INVESTMENT ANALYSIS IS IMPORTANT

Good investment analysis can make or save investors huge amounts of money. Backing Xstrata at its float in 2002 and selling in May 2008 would have made investors a return of 756% versus what many thought was a safer, better investment in the blue chip Anglo American (which was still up an impressive 200% over the period) (Figure 2.4-1). At the smaller end of the scale, picking winners in the juniors can be much more rewarding. Fortescue Metals Group had a similar market capitalization to iron ore miner Mount Gibson Iron in 2002, but an investor backing Fortescue over Mount Gibson could have made 316,986% more (Figure 2.4-2).

By the same token, a good analyst knows when to avoid certain investments—identifying bad companies with problems or perhaps when it is an inappropriate time to buy. The brave call of May 2008 was to sell, and a small handful called that right. Aside from making money for investors,

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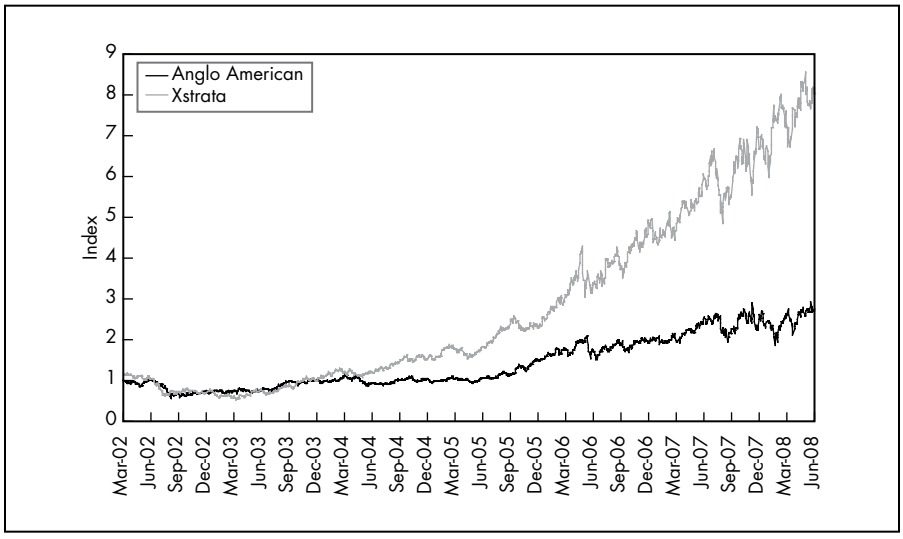


Figure 2.4-1 Relative share performance between Xstrata and Anglo American, March 2002 to May 2008 (indexed March 1, 2002 = 1)

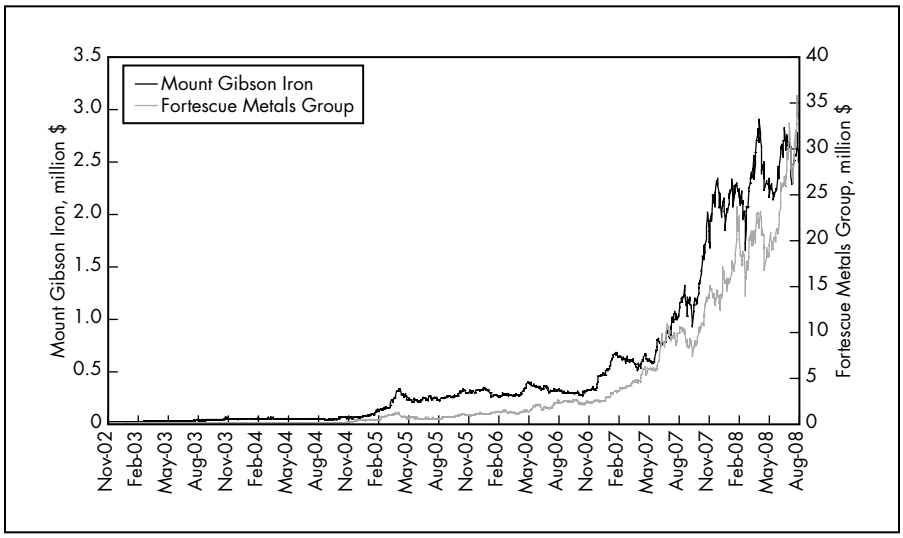


Figure 2.4-2 Relative market cap performance between Fortescue Metals and Mount Gibson, November 2002 to June 2008

analysts play an important part in the capital markets that allocate debt and equity to companies. Analysts are an integral part of the feedback loops that guide management of publicly listed companies—providing insights and observations about the strategic, operating, and financing practices of each company. Companies that have supportive analysts typically enjoy a heightened profile among financial institutions and also rival corporations. A strong and positive analytical following means companies are often better able to raise equity and debt capital when they need to and attract partners/bidders in mergers and acquisitions work. Although this applies more to smaller and mid-capitalization companies, even large companies need to be sure they are seen (at worst) fairly by the analytical community. Companies that are loathed by analysts often end up with low-rated shares and find it more difficult to raise money cheaply.

JOB OF AN EQUITY INVESTMENT ANALYST

The day-to-day job of a mining investment analyst is split between researching, modeling, and writing about companies; visiting and talking to the companies and industry contacts; and marketing to investors.

Financial Model

Having a financial model on the company is a starting point for an analyst but, importantly, cannot provide the answer as to whether the company is a sound investment proposition. The financial model should have key operational metrics (tonnage, grade, production, and cash costs) with macroeconomic variables such as commodity prices and exchange rates. In larger banks, some of the macroeconomic assumptions are determined by in-house economists, so the analyst's view is necessarily colored by that financial institution's house view. The operational model then creates the traditional outputs of

Table 2.4-1 Valuation as of March 10, 2009

Mining Company	Share Price, £/share	EV	EBITDA		Earnings per Share (US¢/share)		EV/EBITDA		Price/ Earnings per Share	
			2009E*	2010E	2009E	2010E	2009E	2010E	2009E	2010E
Anglo American	9.14	27,460	6,421	5,364	177	118	4.3	5.1	7.3	10.9
BHP Billiton	10.34	89,728	19,896	13,638	192	119	4.5	6.6	7.5	12.1
Rio Tinto	18.23	71,467	16,542	13,655	507	372	3.8	4.5	5.0	6.9
Xstrata	3.36	26,683	5,529	4,795	56	35	4.8	5.6	8.4	13.3

*E = estimated.

the profit-and-loss, cash-flow, and balance sheet statements for the company.

The purpose of a robust but simple model is twofold:

1. It generates the key valuation outputs with which investors compare companies across sectors. The most commonly used metrics in mining remain the traditional ones: price/earnings (P/E) multiples (PERs or price/earnings per share), EV/EBITDA (enterprise value/earnings before interest, taxes, depreciation, and amortization), dividend yields (price/dividend per share), leverage metrics such as net debt/EBITDA and net debt/market capitalization, and the net debt and price/NPV. (See the glossary at end of this chapter for definitions of financial terms.) An example of the valuation metrics for the big four U.K. mining companies is set out in Table 2.4-1.
2. The spreadsheet facilitates the modeling of various scenarios (macroeconomic, production, and costs) that allow investors to understand the company's key characteristics such as operational and financial leverage (earnings and cash-flow sensitivities to operational drivers such as grade, volumes, capital expenditure, unit costs; and macro drivers such as currencies, commodity price, interest rates, and key cost inputs).

Written Product

To the external world, the product of research is the research note—typically a 10- to 50-page document describing the company, its financials, valuation, and investment thesis, with some pointers to catalysts for share price moves. (See Figure 2.4-3 for examples.) Although the note has a shelf life, in reality, more informal and frequently written communication forms are the main outputs from a research analyst. These tend to be PowerPoint presentations and chart books, e-mails, and/or Bloomberg messages. Indeed, the more influential investment analysts tend to write less and communicate more by way of short update communications (typically e-mails, telephone calls, Bloomberg chat messages).

Client Contact: Analysis as a Contact Sport

Analysts need to be good communicators, because it is their business to gather and trade in information. They build their investment theories via cross-checking information with industry and other market participants and, once done, must market their own research products through direct contact to portfolio managers and buy-side analysts.

INDEPENDENT RESEARCH AND LINK TO INVESTMENT BANKING

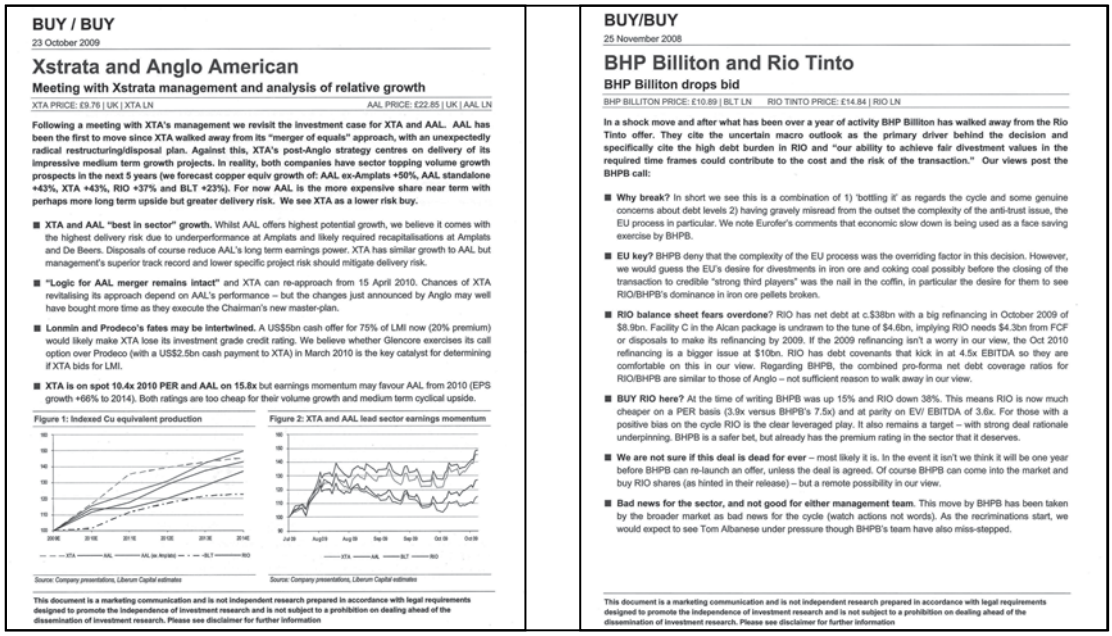
When considering the quality and fairness of any investment research, it is worth knowing the context in which it has been written: Is it genuinely independent or was it influenced by some broader investment banking prerogative? Investment banks, the employers of the investment analysts, can make money from mining companies in a number of ways. Their corporate finance/capital markets departments can make lucrative advisory and underwriting fees from equity and debt securities (rule of thumb 2% to 5% of any money raised in a financing), while the securities businesses earn income from providing sales, trading, and research advice to institutions about shares, including mining equities. Traditional payment terms for this kind of secondary agency trading are around a tenth of that for primary (new equity) offerings. In addition, securities firms can make money from trading in the securities themselves (proprietary trading).

In the small-capitalization arena especially, returns from writing primary-oriented research (i.e., favoring corporate clients to win potentially lucrative corporate business for the broader firm) can potentially dwarf the income from secondary commissions. As such, firms without proper, trusted, and transparent policies that address the potential conflicts of interests surrounding research are often not regarded as highly valued providers of investment research. Also, it is more likely at the small-cap end of the market that the existence of truly unbiased, independent research on corporations is scarce, because the economics of secondary commissions do not make coverage worthwhile.

SKILLS FOR THE SUCCESSFUL INVESTOR

The following skills are needed to be a successful investor:

- **Recognize history and investment cycles.** The four most dangerous words in investing are "this time it's different"; therefore, a knowledge of business and investment cycles is especially important for a cyclical sector such as mining. The same can be applied to cycles in market sentiment to a sector or stock: The time of maximum pessimism is the best time to buy, and the time of maximum optimism is the best time to sell.
- **Be contrarian and be patient.** If the goal is to buy low and sell high, it is worth observing the words of Sir John Templeton (an investor of world renown) that "it is impossible to produce superior performance unless you do something different from the majority" (Templeton 1997). Timing of the buying is key, and it is worth being careful to select the right time to buy an industry, stock, or management team. U.S. businessman Warren Buffett draws parallels with baseball in which he observes those batsmen with the best batting averages don't swing at every ball but wait for a "fat pitch" when the right stock can be acquired at an attractive price.
- **Be skeptical and think both top down and bottom up.** Leaving market assumptions and company managements unquestioned will not unearth profitable investment



BUY / BUY
23 October 2009

Xstrata and Anglo American

Meeting with Xstrata management and analysis of relative growth

XTA PRICE: £9.76 | UK | XTA LN AAL PRICE: £22.85 | UK | AAL LN

Following a meeting with XTA's management we revisit the investment case for XTA and AAL. AAL has been the first to move since XTA walked away from its "merger of equals" approach, with an unexpectedly radical restructuring/disposal plan. Against this, XTA's post-Anglo strategy centres on delivery of its impressive medium term growth projects. In reality, both companies have sector topping volume growth prospects in the next 5 years (we forecast copper equiv growth of: AAL ex-Amplats +50%, AAL standalone +43%, XTA +43%, RIO +37% and BLT +23%). For now AAL is the more expensive share near term with perhaps more long term upside but greater delivery risk. We see XTA as a lower risk buy.

- **XTA and AAL "best in sector" growth.** Whilst AAL offers highest potential growth, we believe it comes with the highest delivery risk due to underperformance at Amplats and likely required recapitalisations at Amplats and De Beers. Disposals of course reduce AAL's long term earnings power. XTA has similar growth to AAL but management's superior track record and lower specific project risk should mitigate delivery risk.
- **"Logic for AAL merger remains intact"** and XTA can re-approach from 15 April 2010. Chances of XTA revitalising its approach depend on AAL's performance – but the changes just announced by Anglo may well have bought more time as they execute the Chairman's new master-plan.
- **Lonmin and Prodeco's fates may be intertwined.** A US$5bn cash offer for 75% of LMI now (20% premium) would likely make XTA lose its investment grade credit rating. We believe whether Glencore exercises its call option over Prodeco (with a US$2.5bn cash payment to XTA) in March 2010 is the key catalyst for determining if XTA bids for LMI.
- **XTA is on spot 10.4x 2010 PER and AAL on 15.8x** but earnings momentum may favour AAL from 2010 (EPS growth +66% to 2014). Both ratings are too cheap for their volume growth and medium term cyclical upside.

Figure 1: Indexed Cu equivalent production

Figure 2: XTA and AAL lead sector earnings momentum

Source: Company presentations, Liberum Capital estimates

Source: Company presentations, Liberum Capital estimates

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BUY/BUY
25 November 2008

BHP Billiton and Rio Tinto

BHP Billiton drops bid

BHP BILLITON PRICE: £10.89 | BLT LN RIO TINTO PRICE: £14.84 | RIO LN

In a shock move and after what has been over a year of activity BHP Billiton has walked away from the Rio Tinto offer. They cite the uncertain macro outlook as the primary driver behind the decision and specifically cite the high debt burden in RIO and "our ability to achieve fair divestment values in the required time frames could contribute to the cost and the risk of the transaction." Our views post the BHPB call:

- **Why break?** In short we see this is a combination of 1) 'bottling it' as regards the cycle and some genuine concerns about debt levels 2) having gravely misread from the outset the complexity of the anti-trust issue, the EU process in particular. We note Eurofer's comments that economic slow down is being used as a face saving exercise by BHPB.
- **EU key?** BHPB deny that the complexity of the EU process was the overriding factor in this decision. However, we would guess the EU's desire for divestments in iron ore and coking coal possibly before the closing of the transaction to credible "strong third players" was the nail in the coffin, in particular the desire for them to see RIO/BHPB's dominance in iron ore pellets broken.
- **RIO balance sheet fears overdone?** RIO has net debt at c.$38bn with a big refinancing in October 2009 of $8.9bn. Facility C in the Alcan package is undrawn to the tune of $4.6bn, implying RIO needs $4.3bn from FCF or disposals to make its refinancing by 2009. If the 2009 refinancing isn't a worry in our view, the Oct 2010 refinancing is a bigger issue at $10bn. RIO has debt covenants that kick in at 4.5x EBITDA so they are comfortable on this in our view. Regarding BHPB, the combined pro-forma net debt coverage ratios for RIO/BHPB are similar to those of Anglo – not sufficient reason to walk away in our view.
- **BUY RIO here?** At the time of writing BHPB was up 15% and RIO down 38%. This means RIO is now much cheaper on a PER basis (3.9x versus BHPB's 7.5x) and at parity on EV/ EBITDA of 3.6x. For those with a positive bias on the cycle RIO is the clear leveraged play. It also remains a target – with strong deal rationale underpinning. BHPB is a safer bet, but already has the premium rating in the sector that it deserves.
- **We are not sure if this deal is dead for ever** – most likely it is. In the event it isn't we think it will be one year before BHPB can re-launch an offer, unless the deal is agreed. Of course BHPB can come into the market and buy RIO shares (as hinted in their release) – but a remote possibility in our view.
- **Bad news for the sector, and not good for either management team**. This move by BHPB has been taken by the broader market as bad news for the cycle (watch actions not words). As the recriminations start, we would expect to see Tom Albanese under pressure though BHPB's team have also miss-stepped.

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Figure 2.4-3 Examples of research notes

opportunities. In the same way, there is no substitute for long hours of forensic research in understanding the detail behind specific companies and markets. To a degree, a rising tide lifts all boats, but, as the 2008 downturn has shown, large dislocations in markets can trigger some widely differing market outcomes for stocks in a similar sector.

SKILLS FOR THE MINING ANALYST

The coverage of small-cap/development mining companies (i.e., less than $200 million to $500 million) requires a different skill set than that required for large-cap companies. In the small-cap world, some technical background (geology, mining engineering) is a likely prerequisite so as to be credible in ascertaining whether a project will be successful. Analysts covering the large diversified mining companies are probably better served if they have a background in business or economics, because understanding the implications of news in the global macro-economy or capital markets is likely to be a far more significant driver to equity performance for a company such as BHP Billiton than knowing the technical viability of a new project.

SKILLS FOR THE SELL-SIDE ANALYST

On a personal level, success can be assisted by several factors:

- Being energetic and committed, following hunches rather than waiting and not acting
- Being first to elucidate an investment idea: identifying then talking about the one thing that matters to a company's share price before it happens
- Taking the time to have better and deeper relationships with company management, industry players, and key institutions

Success is *not* about writing a definitive 200-page report on a single company or producing a multi-megabyte Excel model that no one can understand.

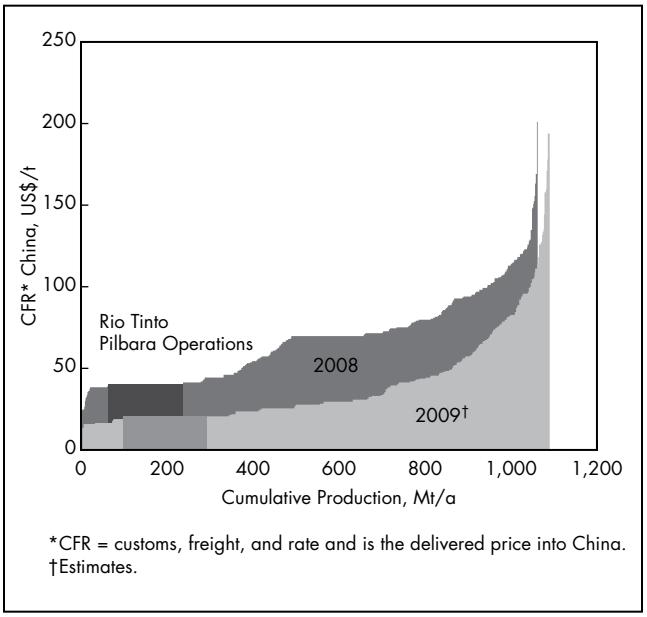
INSIDE THE ANALYST'S TOOLBOX

A mining analyst needs to understand how mining is different from other industries: It is commonly but not always characterized as large-scale, capital-intensive, driven by asset locations, and with low pricing power and high barriers to entry. Long-term returns of a mining company over several investment cycles are driven by the quality of the assets and the management teams that run them. However, for shorter investment horizons, a panoply of other factors can become more important to the performance of a mining share. The full range of drivers is discussed in the following paragraphs.

Assets: Quality, Optionality, and Operational Leverage

The Holy Grail in the world of mining is a large, low-cost, and long-life asset. Superior long-term returns can be enjoyed by companies that own these assets; indeed, Rio Tinto has estimated that the best 30% of the world's mines generated 70% of the investment returns. Companies that have assets at the bottom end of a cash cost curve are likely to generate cash flows throughout the cycle, allowing the owner to survive and prosper in even the worst macroeconomic climate (Figure 2.4-4). Analysts judge assets by their position on the cash cost curve and the capital costs per metric ton (t) of the capacity to build it. Large, long-life assets can be company makers in the longer term, because they have expansion optionality at low capital costs.

Capital intensity and operating costs drive a company's operational leverage—that is, the sensitivity of financial returns (cash flows) to changes in price, cost, and volume throughputs. High-quality assets typically have low operational leverage, which gives the investment relatively stable earnings and cash flows—something rated highly by both debt and equity investors. Of course, there are times in the business cycle when investors seek out operational leverage; recovering commodity prices can transform high-cost producers' earnings, making them massively outperform higher quality peers, in any cyclical recovery.



Source: Rio Tinto 2009.

Figure 2.4-4 Iron ore cost curve (delivered into China at spot freight prices)

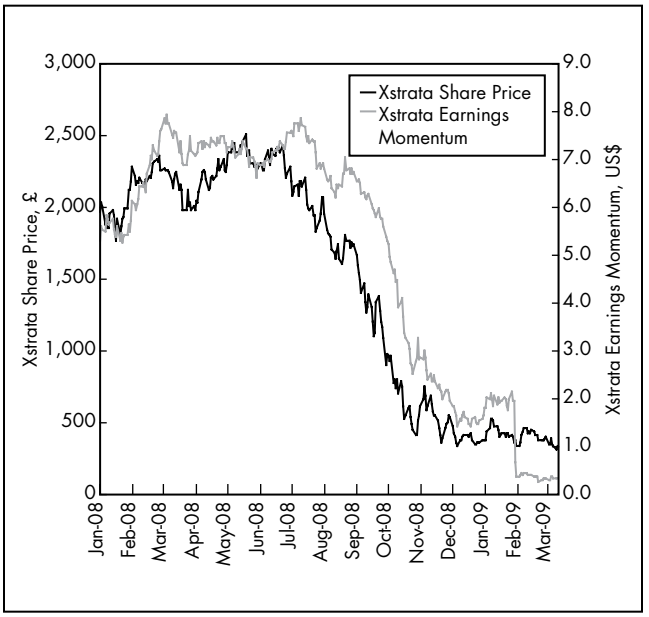


Figure 2.4-5 Xstrata share price versus Xstrata earnings-per-share momentum

Earnings Momentum and the Cycle

Equity markets are often criticized for being short-sighted—with cyclical valuations that fail to look through economic cycles. At the time of writing (early 2009), the mining sector is trading at less than half the level that it was under a year earlier, yet it is not clear that the intrinsic long-term value of the assets has halved (Have asset lives changed? Have long-term pricing and margins for the assets changed?). What has happened in the credit crunch hit of 2009 is that mining company earnings have collapsed and the price of alternative investments has also fallen dramatically. Figures 2.4-5 and 2.4-6 illustrate how the earnings momentum of a stock (here described as the estimated earnings per share of Xstrata on a rolling 1-year forward basis of the spot price of its commodities on that day) is highly correlated with its equity price performance. The resulting PER trades in a tighter range through the cycle, with peak multiples achieved when equity prices (and earnings are troughing) and valuation troughs at the top. Getting the investment calls right then is mostly driven by calling the price cycle for the basket of commodities.

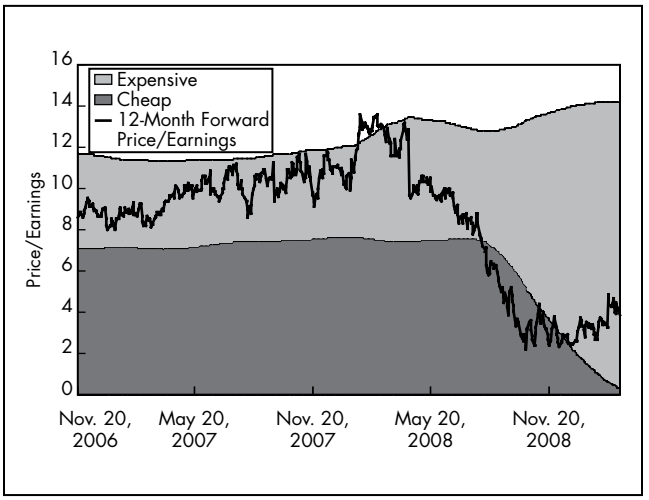


Figure 2.4-6 Xstrata PER range

Calling commodity prices is a dark art in itself—often aided by supply/demand balance models. Analysis of the supply side involves modeling the existing production of a commodity and making educated guesses as to the timing of any capacity additions and/or curtailments and having a view on the all-important stock position. As a general rule, supply additions are usually known well in advance (i.e., there is always a low risk of a surprise price event from new supply), but negative supply shocks (i.e., mine closures) can trigger terrific price movements when markets have no inventory. This occurred in 2008 in ferrochrome and platinum markets when South African electricity supply problems led to supply disruptions and a subsequent spike in pricing.

The demand side of commodity forecasting is heavily influenced by global macroeconomics. The key market for demand has now become China for most major commodities, though the OECD (Organisation for Economic Co-operation and Development) countries account for a greater overall share of demand (Figures 2.4-7 and 2.4-8).

Currencies also have an influence on commodity prices. Typically, a weak dollar (i.e., the dollar is weak relative to the basket of other major currencies) is good for dollar pricing (more purchasing power for a mainly nondollar-denominated consumer). Currencies can be an important driver of unit costs too—the commodity currencies of the rand, Australian dollar, and Canadian dollar tend to be negatively correlated with commodity prices—giving mining companies in those countries somewhat of a natural hedge against the commodity cycle.

Figure 2.4-9 shows the relationship between the Commodity Research Bureau (CRB) Index—an index of key commodity prices including those in energy, agriculture, and metals—and the Australian dollar.

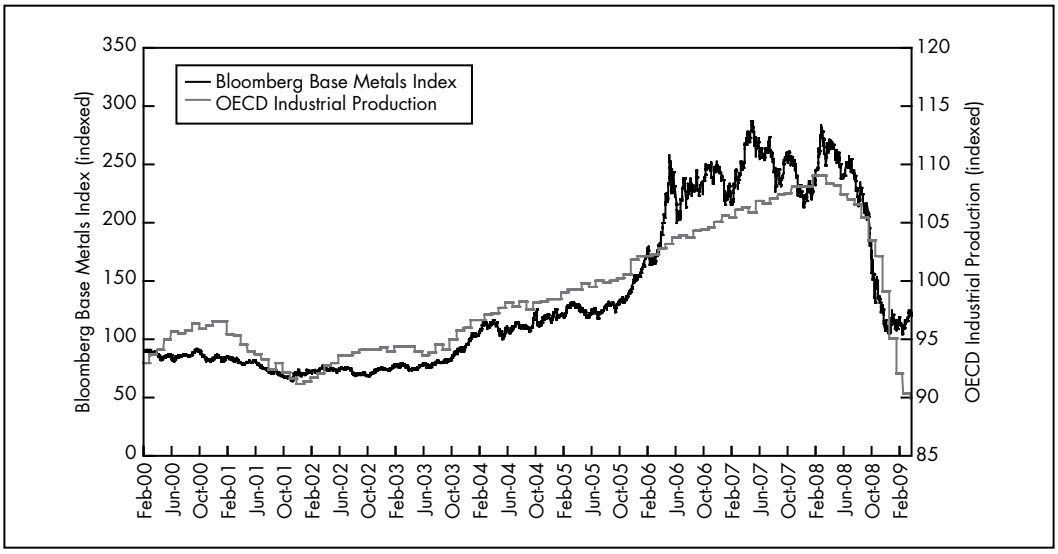


Figure 2.4-7 Base Metals Index versus OECD total industrial production (indexed Feb. 1, 2000 = 100)

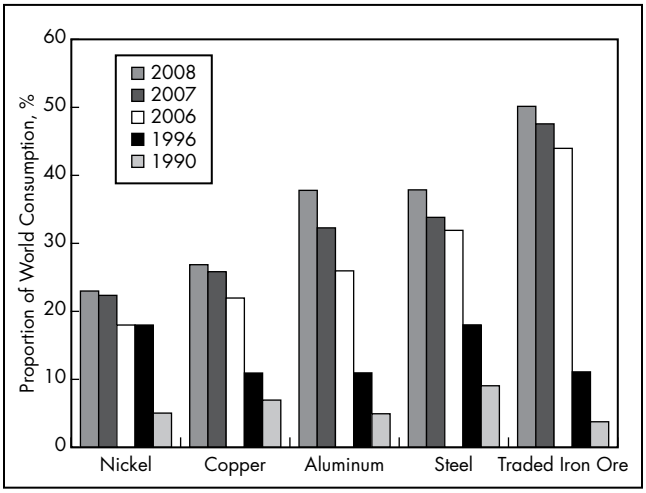
Financial Leverage and Liquidity

Debt and the ability to service debt becomes a major issue in any cyclical downturn. Companies with high levels of debt have more leverage to the cycle, which increases their riskiness to investors and limits their ability to raise capital. Modeling the ability of companies to service debt obligations in a downturn can be crucial in saving investors' money. Things to look for in a balance sheet and cash-flow statement are debt/EBITDA ratios (anything above 2× can be dangerous), interest coverage (interest charge/EBITDA), free cash-flow numbers post-capex (capital expenditure), and other obligations and major debt refinancing dates. Figure 2.4-10 illustrates that highly leveraged companies had poor equity performance in the 2008 downturn relative to their less leveraged peers.

Management Skills: Execution, Mergers and Acquisitions, Governance

It is probably true that a mining company's most important resource is its people and not its ore deposits. A solid management team is one that can run its asset base efficiently through the cycle. Xstrata's management team acquired what at the time was regarded as tier 2 assets, then lowered their unit costs and extended their lives. Similarly, fantastic ore bodies can be ruined by imprudent mine designs and poor operational practices such as high-grading.

Because mining is such a capital-intensive industry, the best returns and most expensive mistakes by a management team are made at the point of approving major capital decisions. These choices define a management team and the shareholder returns of a company, and range from the decision to develop a major project or embark on a major merger and acquisitions (M&A) transaction. Famously sound decisions to develop projects are made in the teeth of bear markets when contracting and construction costs are low and consensus outlooks for commodity prices are so conservative that other producers dare not expand production (thereby setting the scene for a price recovery). Similarly, projects launched at the peak of the boom suffer from high capex costs and poor ability to execute through staffing and other shortages,



Courtesy of Rio Tinto and Liberum Capital.

Figure 2.4-8 China: the world's number one consumer of every major mined commodity

and are typically founded on overoptimistic commodity price decks.

It is perhaps in the world of mergers, disposals, and acquisitions where the most value can be created and destroyed in a cycle. Since about 2000, the sector heroes are those that bought large assets in the early years of the decade: Rio Tinto's acquisition of North Ltd., Anglo American's acquisition of the Disputada copper mine in Chile, and Xstrata's acquisition of MIM Holdings, for example. Interestingly, these same companies, some under different managements, were seen by the market to have made expensive errors of judgement at the tail end of the decade (2007–2009) with the acquisitions of Alcan by Rio Tinto, Falconbridge by Xstrata, and MMX (Mineracao's Minas Rio mine) by Anglo American. A keen sense of the cycle (which can be as much luck as judgment) as well as a robust capital allocation committee can be a key attribute of management.

Aside from strong technical, operating, and capital allocation skills, investors look for management that can deftly

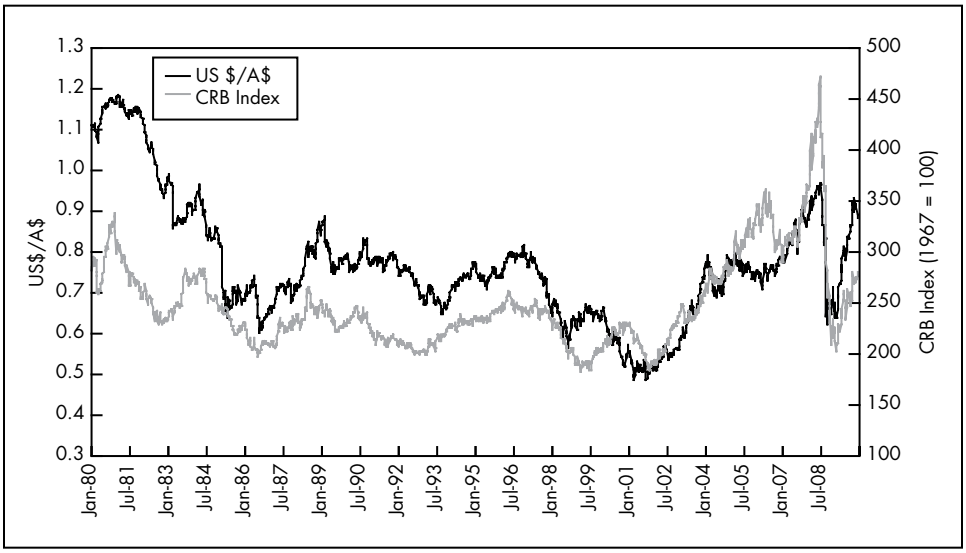


Figure 2.4-9 Relative performance of Australian dollar versus CRB Index

handle political risk; listen to minority shareholders' views; and take social, environmental, and safety risks seriously. In short, investors are prepared to pay a premium for a company whose management they can trust. In the 2008 market meltdown, stretched balance sheets or M&A deals that have gone bad have led to a reassessment of the perceived quality of management teams. In bull markets, most get the benefit of the doubt, but in bear markets, few are trusted fully.

External Environment: Geopolitical, Tax, Financing, Legal

When evaluating a company, the location of its exploration assets or mines is integral in determining its risk profile. Riskier companies often require a higher rate of return from investors and debt holders because of greater uncertainty about the future cash flows of its operations. This is especially true for mining companies, which can be higher-risk companies because of the assets' location.

Generally, companies with operations in countries with established mining industries and laws, such as Australia and Canada, are deemed to be lower risk, owing to low sovereign risk, strong legal systems, stable tax laws and royalty arrangements, and lower probabilities of government intervention.

In high-risk countries, including many parts of Africa and other developing nations, a number of uncertainties may arise, the first of these being sovereign risk. Higher sovereign risk causes debt providers to require a higher rate of return to compensate for the greater uncertainty of developing a project. This may lead to financing becoming unobtainable for some exploration projects, no matter what the grade or resource. Additionally, the legal system in these countries may be weak or ad hoc, providing no protection from government intervention or challenges to mineral and property rights. There is also greater risk that mining laws or royalties may change, resulting in uncertainty of future cash flows or contributions from operations. This has been evident in Zambia, where new windfall taxes were proposed for copper producers.

In developing nations, it is often the case that mines provide a large source of employment and income to the government through royalties and corporate tax. In the absence of other major industries, mining may also provide a country

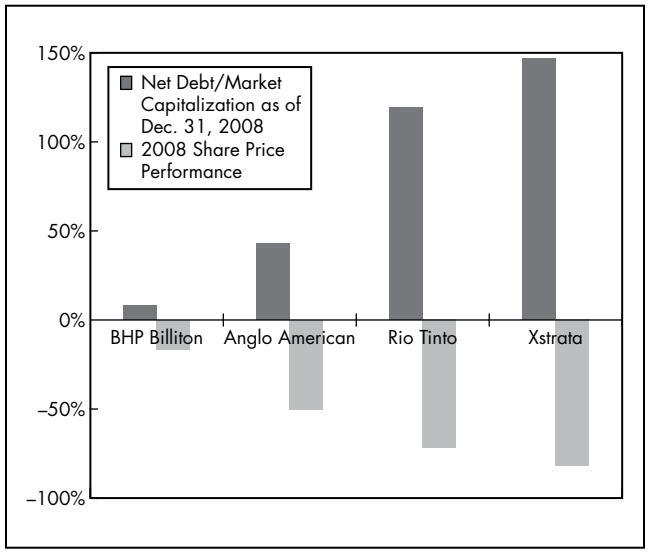


Figure 2.4-10 Net debt/market cap for the "Big 4" mining companies versus 2008 share price performance

with the majority of its exports and therefore foreign currency. For example, in Zambia, mining accounts for 90% of the country's exports. Therefore, governments have a vested interest in keeping mining projects in operation, which may limit the scale of any downsizing or cuts in production that may be needed in a downturn.

Spotting step changes in the external environment can be a key to anticipating large changes in the value of listed mining companies. How will a global credit crunch differentially affect various listed mining companies? How will a banking crisis in Kazakhstan affect a mining company with assets there? (Clue: What will happen to exchange rates and exchange controls?) How does equity issuance across all sectors affect the returns for liquid and nonliquid mining companies?

CONCLUSIONS

This chapter conveys two points. First, the usefulness of the analytical community goes beyond making money for a

potential investor but, in fact, collectively shapes and informs corporate management decision-making processes and, by identifying winners and losers, makes the capital markets more efficient.

Second, investment analysis is both empirical and a black art. Understanding and modeling financial statements are important elements for success, but the ability to see through management and interpret macroecomomic and political trends can be even more important in the art of investing.

Understanding how mining share prices have moved can be fairly obvious after the event, so the trick for any investor is to sift through the myriad of daily data and decide what is significant and important and will have a lasting effect on the company's value. Distinguishing early enough what is important from what is not generates the differentiated investment returns and by association defines the great investment analyst.

ACKNOWLEDGMENTS

Special thanks are due to Liberum Capital for providing most of the figures and tables in this chapter.

GLOSSARY OF FINANCIAL TERMS

- **Balance sheet:** One of the main components of a company's report and accounts, which provides a snapshot of everything the company owes and owns at the end of the financial year in question. On a specific date, the balance sheet lists tangible assets, intangible assets, stock, debtors, cash, bank creditors, trade creditors, share capital, and reserves. The profit-and-loss account reveals how the company has performed in the previous year, while the balance sheet is more revealing about its fundamental health, indicating whether it can pay its debts and how good its cash management is. A strong balance sheet is where liabilities (including borrowings) are considerably outweighed by assets (including cash). If a company is having problems, the balance sheet (together with the cash-flow statement) will reveal whether it can stand the strain.
- **Cash flow:** The amount of money that flows in and out of a business, the difference between the two being the important number. If more money flows into a business than out, it is cash positive. If more money flows out than in, it is cash negative. Cash flow is regarded by many as the ultimate test of financial health. Seasoned analysts do not entirely trust the stated net-profits-after-tax figure, because profits can be massaged, whereas cash is more difficult to manipulate. Profit, as they say, is a matter of opinion. Cash is a matter of fact. The best way to check the cash-flow position of a company is to scrutinize the cash-flow statement in its annual report and accounts. It provides facts on whether a company has generated or consumed cash in the year and how. It can be used in conjunction with the profit-and-loss statement to assess the trading results or with the balance sheet to assess liquidity, solvency, and financial flexibility.
- **Dividend yield:** The annual dividend income per share received from a company divided by its current share price. This is simply how much income a shareholder is getting out of the company for the capital locked up in it. Dividend yields are calculated on the net dividend paid over the course of a year divided by the stock's price. For example, if a company declares a net dividend of 2.1 pence (p) per share and has a share price of 150p, the dividend yield is 2.1/150 = 1.4%. The higher the share price, the lower the dividend yield.
- **EBITDA (earnings before interest, taxes, depreciation, and amortization):** A commonly used way of measuring the profitability of a company and an approximation for pretax operating cash flows.
- **Enterprise value (EV):** A measure of the worth of a company's ongoing operations. This is typically the market capitalization of the company plus any debt and minorities to fund it.
- **EV/EBITDA:** A valuation multiple that is an alternative valuation metric to the price/earnings ratio. This methodology is useful for comparing companies that have very different capital structures (e.g., companies that may have lots of debt versus those that don't).
- **Market capitalization (market cap):** The market value of a quoted company, calculated by multiplying its share price as quoted on an exchange by the number of shares in issue.
- **Price/earnings (P/E) ratio:** P/E is the current share price of a company divided by its earnings per share. A company with a share price of 100p and earnings per share of 5p has a P/E ratio of 100/5 = 20. A company's P/E (also known as its multiple) shows how high its shares are priced in relation to its historical earnings. Although mathematically it relates share price to past performance, the reality is that P/Es are more about forward expectations than past. A high P/E indicates that the investment community expects the company's earnings to grow fast in the future.
- **Profit-and-loss (P&L) statement:** A set of accounts, usually prepared annually, that depict a company's trading performance and are normally read in conjunction with the balance sheet and cash-flow data. The P&L account can broadly be shown as follows:
  - Turnover (sales) less manufacturing costs (or cost of sales if, for example, a retailing company) = gross profit or loss
  - Gross profit plus any nontrading income less operating costs = operating minus profit
  - Operating profit less interest payments on bank loans or loan stock = profit before tax (pretax profit)
  - Pretax profit less tax = net profit after tax

Part of the net profit after tax may be used to pay a dividend, with the balance being retained within the business for future investment.

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PART 3

Exploration

CHAPTER 3.1

Geological Features and Genetic Models of Mineral Deposits

Richard Herrington

INTRODUCTION

Mineral deposits are natural concentrations of one or more mineral commodities that may be termed *ore deposits* if they reach levels where economic extraction is viable. They form as a result of many geological processes, but they form within specified geological settings. Similar genetic processes may operate to form deposits that have similar characteristics and, thus, may define a "class" of deposits. The geological processes that may concentrate minerals include fractional crystallization of magmas, release of volatiles from a crystallizing magma, magmatic interaction with host rocks, metamorphic reactions producing fluids, chemical changes in circulating heated groundwater, organic interactions, and weathering, among others. Eckstrand et al. (1995) neatly summarize that "mineral deposit types" are mineral deposits that share a set of geological attributes and contain a particular commodity (or commodities) that collectively distinguish them from other types.

Ideally, mineral deposit types should reflect how the mineral deposit was actually formed, and in this case, one might be able to use the term *genetic model*. However, in many cases there is considerable debate among geologists as to how a specific deposit was formed, and thus classifications based purely on a given genetic model will encounter problems. An alternative is to classify deposits based on empirical features (minerals, host-rock association, etc.), which will lead to the unique fingerprint of a particular deposit (i.e., a descriptive model). Even though no two mineral deposits are identical, empirical descriptions of deposits tend to show natural groupings into a small number of loosely definable categories or types. In turn, these categories tend to coincide with genetically derived models; so even by using purely physically descriptive classifications, there is often a close coincidence between these and models defined using genetic criteria.

Prior to the 20th century, models for the formation of mineral deposits were subject to the often-polarized views of either plutonist (all deep igneous origins) or neptunist (all sedimentary origins) theories for the origin of rocks, and it was really only in the 20th century that modern views of mineral deposit formation emerged.

From the late 1960s, plate tectonic theory has revolutionized geological thinking, and in the mid-1980s, Guilbert and Park (1986) devised a mineral deposit classification scheme based on plate tectonics.

Figure 3.1-1 is a very broad attempt to show an interpretation of the environment of formation of key mineral hypogene deposit types with respect to plate tectonic settings in their broadest sense.

ORE GENESIS (FORMATION PROCESSES)

Linking deposit types directly to ore-forming processes is certainly the preferred way to classify, as it will provide better criteria for the understanding of the deposit with respect to associated features (such as how its formation relates to associated igneous rock suites, alteration patterns, etc.). In turn, this will lead to better exploration models for their discovery and evaluation. Nevertheless, descriptive models are needed in practical terms to aid engineers in the evaluation of particular deposits (choice of exploration tool, elements to analyze in geochemical exploration, etc.).

Evans (1993) divides ore-forming processes into the four main categories of internal processes, hydrothermal processes, metamorphic processes, and superficial processes. The former three processes relate to subsurface phenomena, while the last of these obviously covers those processes occurring at the earth's surface. In this chapter it is preferred to use the three main divisions of orthomagmatic, hydrothermal, and surficial deposits in line with Einaudi (2000) and broadly encompassing the range of Evans' criteria (Evans 1993). Hydrothermal should be further subdivided into magmatic hydrothermal, metamorphic hydrothermal, diagenetic hydrothermal, and surface hydrothermal to refine the nature of the hydrothermal process.

Further qualifying terms such as syngenetic and epigenetic are useful in discussing the genesis of ore deposits. Syngenetic is used for mineral deposits that are interpreted to have formed at the same time as the enclosing rocks. Syngenetic deposits are bounded by the host rocks themselves as in the case of a deposit formed by crystallization of magma, the result of precipitation within sediments on the seafloor from a hydrothermal vent, or perhaps by the physical sedimentation of heavy mineral sand. Conversely, epigenetic is used for mineral deposits that formed later than the enclosing rocks, and as a result the deposits are found as masses, layers,

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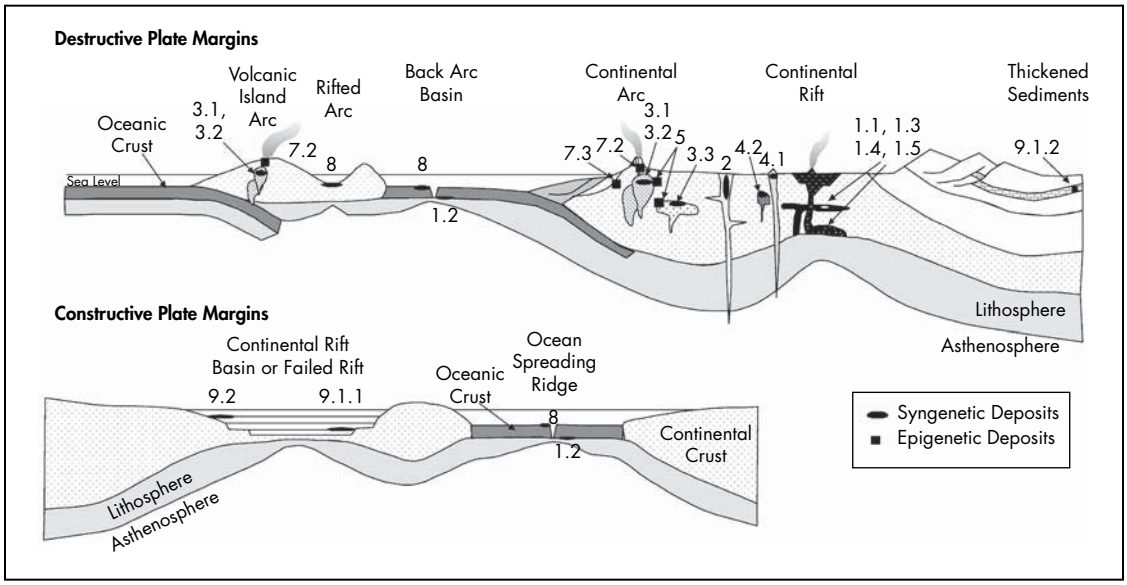


Figure 3.1-1 Plate tectonic settings for key magmatic and hydrothermal deposit types (numbers refer to deposit classes and types shown in Table 3.1-1)

or disseminations that are seen to cut or overprint the original rock. An example of this might be a gold-quartz vein crossing a deformed greenstone.

Hypogene is another useful term used to describe mineralization formed by processes deep in the earth, whereas *supergene* refers to processes superimposed on original mineralization. Often these terms are synonymous in their use with the terms *primary* and *secondary*, respectively. In the context of a sulfide deposit formed, for instance, as a magmatic segregation of chalcopyrite that is then partly oxidized by weathering to, for example, chalcocite, the chalcopyrite is considered to be hypogene (or primary), while the weathering product chalcocite would be termed supergene (or secondary).

KEY DEPOSIT-FORMING PROCESSES

Orthomagmatic Deposits

Orthomagmatic deposits are those that form from primary magmatic processes (i.e., cooling and crystallizing magmas). They are hosted within the rocks from which they formed. Therefore, they are by definition also syngenetic. Deposits may form as a result of

- Solid phases crystallizing as a differentiate as the magma cools;
- Minerals crystallizing from the enriched residual fluids formed as magma cools and crystallizes;
- The formation of a sulfide melt that developed by immiscibility from a coexisting silicate melt; or
- Where a magma transports xenolithic or xenocrystic phases that it has picked up on its passage through the earth's crust.

Hydrothermal Deposits

Hydrothermal deposits are those that literally form from hot water (hydrothermal fluids) circulated through the earth's crust. Direct evidence for the presence of these fluids in the earth's crust are surface manifestations such as hot springs and fumaroles, but indirect evidence is provided by the presence of hydrous minerals in even crystalline rocks. Hydrothermal fluids may have their origin in igneous plutons or from metamorphic reactions deep in the crust, but they can also be the result of deeply circulated and heated meteoric or seawater, or they may be the release of trapped water from sedimentary basins undergoing diagenetic change.

Tracking the original source for this range of fluids is now largely possible using studies of fluid inclusions trapped in minerals and the use of a range of isotopic markers. At one end of the range of hydrothermal deposits there are fluids directly related to or exolving from cooling igneous intrusions, and at the other end there are surface waters heated as they have been circulated into deeper, hotter parts of the earth's crust. Technically speaking, the term *hydrothermal* should only be used for water-dominated fluids, whereas fluids in the gaseous state should be referred to as *pneumatolytic*. Hydrothermal deposits can be either syngenetic or epigenetic, depending on where and when the deposit formed.

Magmatic Hydrothermal

Magmatic-hydrothermal fluids form as a body of magma cools and crystallizes. In some cases the magmatic system may simply be a passive source of heat that drives the circulation of fluids exotic to the magma through adjacent fractured crust into which the magma is intruding. In many other cases the magma, particularly felsic magmas such as those that form granitic rocks, contain very significant amounts of miscible water, which is carried in the magma itself. As this magma cools and crystallizes it becomes more concentrated and eventually forms an immiscible fluid phase, which in the process collects other components that prefer to partition from a silicate melt into a hydrous-fluid phase. Gases are also miscible in silicate melts at higher temperatures and pressures. But again, as the magma cools and is intruded into shallower portions of the crust, the gases become immiscible and may partition into a fluid phase or may escape entirely as a vapor (pneumatolytic fluids). Components concentrating in a fluid phase that may exsolve from magma include chloride ions, which leads

Table 3.1-1 Major classes of economically important mineral deposits

Class	Type/Subtype			Formation
1 Deposits in mafic magmas	1.1 Layered chromite deposits			Syngenetic
	1.2 Podiform chromite deposits			Syngenetic
	1.3 Titanomagnetite deposits			Syngenetic
	1.4 Magmatic platinum group metal deposits			Syngenetic
	1.5 Nickel sulfide deposits	1.5.1 Sudbury		Syngenetic
		1.5.2 Flood basalt association		Syngenetic
		1.5.3 Ultramafic volcanic association		Syngenetic
		1.5.4 Other mafic and ultramafic intrusive associations		Syngenetic
2 Magmatic diamond deposits	Kimberlites and lamproites			Syngenetic (xenocrystic)
3 Deposits associated with felsic magmas	3.1 Porphyry Cu-Mo-Au deposits			Epigenetic (with respect to host rocks), contemporaneous with magmatic activity
	3.2 Porphyry Mo (W) deposits			Epigenetic (with respect to host rocks), contemporaneous with magmatic activity
	3.3 Granite-hosted Sn-W deposits			Epigenetic (with respect to host rocks), contemporaneous with magmatic activity
	3.4 Intrusion-related gold deposits			Epigenetic (with respect to host rocks), contemporaneous with magmatic activity
4 Deposits associated with peralkaline and carbonatite magmas	4.1 Peralkaline Ta-Nb, rare earth element deposits			Syngenetic with late magmatic activity and epigenetic
	4.2 Carbonatite Cu, rare earth element, Nd, Fe, P deposits			Syngenetic and epigenetic
5 Skarn and carbonate replacement deposits				Epigenetic
6 Iron oxide copper-gold deposits				Epigenetic
7 Hydrothermal gold and silver deposits	7.1 Sediment-hosted gold deposits			Epigenetic
	7.2 Epithermal gold and silver deposits	High-sulfidation epithermal		Epigenetic
		Low-sulfidation epithermal		Epigenetic
	7.3 Lode (or orogenic) gold deposits			Epigenetic
8 Volcanic-hosted or volcanogenic massive sulfide deposits	Mafic			Syngenetic
	Bimodal mafic			Syngenetic
	Pelitic mafic			Syngenetic
	Bimodal felsic			Syngenetic
	Siliciclastic felsic			Syngenetic
9 Sediment-hosted deposits	9.1 Sediment-hosted sulfide deposits	9.1.1 Sedimentary exhalative Pb-Zn (Cu) in clastic sediments (*+Broken-Hill type deposits*)		Syngenetic (*Epigentic*)
		9.1.2 Mississippi Valley type (MVT) Pb-Zn		Epigenetic
		9.1.3 "Irish" type Pb-Zn (Cu)		Syngenetic-epigenetic
		9.1.4 Clastic sediment-hosted Cu		Epigenetic
	9.2 Sediment-hosted iron and manganese deposits	9.2.1 Ironstones		Syngenetic
		9.2.2 Banded iron formation (BIF)	9.2.2.1 Algoma BIF	Syngenetic (+epigenetic)
			9.2.2.2 Superior BIF	
			9.2.2.3 Rapitan BIF	
		9.2.3 Manganese ore		Syngenetic
	9.3 Sedimentary uranium deposits	9.3.1 Unconformity vein type uranium		Epigenetic
		9.3.2 Sandstone-hosted uranium		Epigenetic
	9.4 Gold and uranium in conglomerates			Syngenetic (+epigenetic?)
	9.5 Chemical sediments	9.5.1 Evaporites		Epigenetic
		9.5.2 Manganese nodules		Syngenetic
10 Ores related to weathering	10.1 Laterites	10.1.1 Bauxite		Epigenetic
		10.1.2 Nickel (cobalt) laterite		Epigenetic
		10.1.3 Lateritic gold		Epigenetic
	10.2 Supergene weathering	10.2.1 Secondary copper		Epigenetic
		10.2.2 Secondary zinc		Epigenetic
11 Placer deposits				Syngenetic

to the formation of high-temperature brines that are effective solvents for many metals such as iron, copper, lead, and zinc (Brimhall and Crerar 1987). These metal-rich fluid phases can then migrate away from the magma and can begin to interact with minerals and fluids in previously crystallized magma or outside rocks, which can cause these to become altered by chemical reaction and lead to precipitation of new mineral phases, including the ore minerals. These fluids can also concentrate miscible gases, for example, carbon dioxide (CO_2), sulfur dioxide (SO_2), and hydrogen sulfide (H_2S), which can also aid in the transport of dissolved components with vapor-dominated fluids derived from magmas implicated in the transport of some metal deposits (Williams-Jones et al. 2002).

All mineral deposits that formed by replacement or in fractures around the cooled magma chamber are by definition epigenetic because they overprint the host rocks, although they are contemporaneous with associated magmatic activity. Other rarer metals may remain miscible in the silicate magmas much longer but may become concentrated in the final hydrous fluid phase, trapped at the very final moment within the cooling magma chamber itself. In this case, the deposits are defined as syngenetic, because the deposit fills interstices in the magma as it crystallizes in a similar fashion to orthomagmatic deposits. Depending on the crystallization path for the magma and its interaction with external components, a range of deposits may form from these processes.

Metamorphic Hydrothermal

Metamorphic-hydrothermal fluids form as metamorphism results in mineral-chemical processes that may release volatiles, often dominated by water but which may include gases such as CO_2. Metamorphism is induced in rocks by external heat or pressure or (usually) by a combination of both. Heat may be provided by the deep burial of a rock mass through time or alternatively by the intrusion of a magma body nearby. Pressure to cause metamorphism may be provided again during deep burial or else by tectonic processes. By definition, all of these deposit types are epigenetic because the effects are seen to overprint rocks that have already formed.

Diagenetic Hydrothermal

Diagenetic-hydrothermal fluids are formed as pore waters trapped during sedimentation, then loosely bonded to clays and other minerals, which are released during compaction and lithification. This process may develop on a large scale in a sedimentary basin undergoing burial and lithification and is a related process to hydrocarbon generation. The released water can pick up dissolved salts (becoming a brine), which then has a greater ability to transport many cations and ligands to a point of deposition to form an ore deposit (Brimhall and Crerar 1987). In these sedimentary basins, evaporite beds may be a specific source of salts that can be dissolved by the basinal water.

Basins undergoing diagenesis become heated, and thus the basinal brine may become a highly effective solvent for dissolving large quantities of metals. These basinal brines may then migrate via crustal faults and permeable horizons to depositional environments. Syngenetic deposits can form when these fluids are expelled in a submarine or lacustrine environment, where chemical precipitation can result in layered deposits of minerals. Epigenetic deposits can also be formed where the fluid interacts with already lithified rocks causing alteration and the precipitation of minerals.

Surface or Seafloor Hydrothermal

Surface or seafloor hydrothermal fluids are generated as deeply penetrating meteoric- or seawater-derived waters descend and become heated deeper in the crust. This process is particularly apparent in regions where there is elevated crustal heat flow, often where the earth's crust is being thinned. In the case of the seafloor, this phenomenon is common where a new ocean is formed by the seafloor spreading through the formation of submarine volcanoes. On land, such hydrothermal fluids can be generated in zones of crustal attenuation, often associated with subaerial volcanism. These phenomena may not directly relate to igneous activity, however, as fluids may be simply interacting with residual and convecting heat in the deep crust.

Surface manifestations of this process may be the presence of hot springs on land or seafloor hydrothermal vents. Deposits formed by these processes can by epigenetic, where they replace rocks already formed, or they may be syngenetic, for example, in the case of seafloor hydrothermal vents where exhaled fluids may result in layers and masses of chemically precipitated ore minerals. Syngenetic deposits can also form on land where hot springs vent to the surface and minerals are deposited in layered masses.

SURFACE UPGRADE (LOW TEMPERATURE)

Low-temperature (<50°C) surface processes can also be responsible for the formation of economic ore deposits. Physical processes such as physical erosion, transportation, and deposition lead directly to the redistribution and accumulation of specific minerals. Such deposits are formed as a result of the differing physical and chemical behavior of the minerals forming the original rock. These physical processes can be either hydraulic (water) or aeolian (wind).

Weathering is also a very important ore-forming process resulting in chemical change and redistribution of components in surface rocks by migrating solutions. The differential chemical properties of minerals at the earth's surface and within the surface-crustal interface can lead to residual upgrades or chemical dissolution and reprecipitation mechanisms to concentrate the metal/mineral of interest. In such cases, ore formation is driven by the circulation of largely meteorically derived water at the earth's surface, although similar analogous processes can take place on the seafloor. These subsurface waters can dissolve components, reprecipitating them at favorable mineral sites or surface interfaces. Another process important for ore formation that is a surface phenomenon is the process of evaporation. Dissolved salts precipitate as water is lost in an evaporating basin for example, or by the evaporation of water from the ground's surface due to heat energy from the sun.

SPECIFIC DEPOSIT TYPES

Deposits have been classified in a number of ways by different authors. Table 3.1-1 shows the classification scheme preferred here, which is largely based on empirical features of the deposits but which happily coincides with genetic classification in many cases. A number of less-important deposit types are not covered in this section. Their omission (e.g., vein-style base metal deposits) is largely on the grounds that current economic considerations do not make these deposits attractive targets for mining operations. References for this section are in the "Source References" section at the end of the chapter.

CLASS 1: DEPOSITS IN MAFIC MAGMAS

Type 1.1: Layered Chromite Deposits

- Bushveldt, South Africa
  1,100 Mt (million metric tons) at 42% to 45% Cr_2O_3 (chromium oxide)
- Great Dyke, Zimbabwe
  113 Mt at 26% to 51% Cr_2O_3
- Kemi, Finland
  59 Mt at 26% Cr_2O_3
- Campo Formoso, Brazil
  17 Mt at 17% to 21% Cr_2O_3
- Bird River, Canada
  60 Mt at 4.6% Cr_2O_3

Age and Distribution

Most significant deposits of this type are Achaean or Early Proterozoic in age and are known in cratonic regions of Canada and the United States, South Africa, South America, and Finland.

General Characteristics

These deposits occur in large layered intrusions that show a differentiation into a lower ultramafic zone and an upper mafic zone. They generally occur in two forms: the first are emplaced like tabular sills where igneous layering is parallel to the intrusion floor (or roof), whereas the second are funnel-shaped, with the layering dipping into the center like a saucer shape. Bodies can be enormous (e.g., Bushveldt, which is 480 × 380 km; Great Dyke, which is 530 × 6 km) or they can be quite small (e.g., Kemi, which is 15 × 2 km). The intrusions are developed in different tectonic settings and can relate to cratonic rifts or be synvolcanic intrusions.

The deposits themselves generally take the form of laterally continuous chromite-rich layers that vary from one centimeter to several meters thick. Laterally, they can extend for many hundreds of kilometers. The chromite may be interlayered with other rock types such as dunite, peridotite, pyroxenite, anorthosite, or norite, but those with the best grades are usually associated with primitive peridotites.

Type 1.2: Podiform Chromite Deposits

- Kempirsai, Kazakhstan
  >90 Mt at 50% Cr_2O_3
- Kavak, Turkey
  2 Mt at 20% to 28% Cr_2O_3

Age and Distribution

Podiform chromite deposits are all typically Phanerozoic in age, occurring in the tectonized mantle parts of ophiolite complexes. Significant deposits are recorded in the Philippines (Tertiary), Albania (Jurassic), Turkey (Jurassic-Cretaceous), Kazakhstan (Silurian), and many small occurrences in the Caledonian-Appalachian orogen.

General Characteristics

The chromite is found in the basal part of an ophiolite sequence, namely, the ultramafic tectonite (usually a harzburgite). Ophiolites are the obducted fragments of oceanic crust usually preserved in orogenic belts, where thrusting has exposed the basal mantle sections of the sequence. These ultramafics are likely to be the residual mantle, and the upper contact of this unit with overlying cumulate ultramafic dunites is normally taken to be the petrological Moho.

The deposits take the form of lens-like bodies of massive to heavily disseminated chromite, although often the bodies are highly discordant and rod-like. The variable morphology undoubtedly reflects the tectonized nature of these upper mantle rocks. Ores are dominated by chromite that occurs with interstitial olivine, orthopyroxene, clinopyroxene, and plagioclase. This assemblage is often partially serpentinized or altered to more hydrous minerals such as chlorite, amphibole, talc, and carbonate. Textures may appear to be magmatic or clearly overprinted by deformation.

Type 1.3: Titanomagnetite Deposits

- Lac Tio, Canada
  125 Mt at 32% to 35% TiO_2 (titanium oxide)
- Tellnes, Norway
  >380 Mt at 18% TiO_2
- Kachkanar, Russia
  6,937 Mt at 53% Fe, 5.3% TiO_2, 0.2% V

Age and Distribution

Large anorthosite intrusions of Mesoproterozoic age hosting titanium, vanadium, and iron extend from North America to the Sveconorwegian province of Scandinavia. Similar deposits are found in many tectonic belts around the world, such as in the Ural Mountains in Russia.

General Characteristics

Deposits relate to large differentiated intrusive complexes composed largely of anorthosite, gabbro, norite, and monzonite. Concentrations of metallic oxide minerals occur in the following styles: disseminated oxides in the host rocks, conformable lenses with sharp to indistinct boundaries with earlier crystallized phases of anorthosite and gabbro, late-stage dykes rich in oxide minerals cutting host gabbro and anorthosite, and as replacements linked to alteration zones on the margins of host intrusions. Ore minerals may be disseminated and interstitial to silicate minerals or else form massive aggregates separated from the silicates. The giant Lac Tio deposit in Canada is an irregular, tabular, 1,100-m-long, 1,000-m-wide intrusion, whereas the Tellnes deposit in Norway takes the form of a 14-km-long dyke.

Type 1.4: Magmatic Platinum Group Metal (PGM) Deposits

- Bushveldt Merensky Reef, South Africa
  2,160 Mt at 8.1 g/t PGM + Au
- Bushveldt UG2, South Africa
  3,700 Mt at 8.7 g/t PGM + Au
- Bushveldt Platreef, South Africa
  1,700 Mt at 7.3 g/t PGM + Au
- Stillwater (J-M Reef), United States
  421 Mt at 18.8 g/t PGM + Au
- Great Dyke, Zimbabwe
  1,680 Mt at 4.7 g/t PGM + Au
- Lac des Iles, Canada
  6.7 Mt at 5.4 g/t PGM + Au

Age and Distribution

The bulk of the intrusions hosting these deposits are Archaean to Proterozoic. Stillwater and a group of smaller Canadian

examples are Achaean, whereas the Great Dyke and Bushveldt are early and mid-Proterozoic, respectively. The Skaergaard intrusion in Greenland is Tertiary in age and is the youngest known. The deposits occur in large intra-continental rifts or at the margins of rifted continents, and thus they are usually hosted in older continental crust.

General Characteristics

The mineralization in all cases is largely stratiform, but there are some differences in the degree of conformity to enclosing host rock. Broad common features occur in all deposits. For example, PGMs are almost entirely found within sulfides and specific PGMs. The morphologies and mineral associations are diverse, however.

The Proterozoic-age Bushveldt-layered intrusion is the world's largest PGM repository. The intrusion forms a cloverleaf-shaped outcrop measuring 400 × 240 km, and it comprises mafic and ultramafic cumulate rocks located around a granitic core. The lower cumulate rocks, the Rustenburg Layered Suite, is up to 7 km thick, with the lowermost Critical zone the main host to PGM deposits in the Merensky Reef and UG2 layers. Mineralization is also developed in a unit known as the Platreef, which forms the base of the Critical zone in the northeastern lobe of the Bushveldt.

Type 1.5: Nickel Sulfide Deposits

The four categories of nickel sulfide deposits are Sudbury, flood basalt association, ultramafic volcanic association, and other mafic intrusive associations. These are discussed as follows.

Subtype 1.5.1: Sudbury

- Sudbury, Canada (global)
  1,648 Mt at 1.2% Ni, 1.03% Cu (+ PGMs and Au)

Age and Distribution

The Sudbury Igneous Complex formed during the Proterozoic age as the likely result of a meteorite impact. This impact ruptured the earth's crust and allowed the intrusion of a mantle-derived mafic magma body, which differentiated to form the igneous complex and its nickle-copper sulfide deposits.

General Characteristics

The rather unique Sudbury nickel-copper sulfide deposits are largely confined to the so-called "sublayer" unit of the Sudbury Igneous Complex at its contact with the rocks into which it was intruded. The complex is a 65-km-long, 25-km-wide oval-shaped outcrop. In section across the narrow dimension, it forms a funnel-shaped body widening at the surface. Two types of sublayer are host to the nickel-copper deposits in two settings. The contact sublayer is a discontinuous gabbronorite at the basal contact with nickel-copper sulfides and xenoliths of footwall and exotic ultramafic units. The offset sublayer comprises apophyses of quartz-diorite, which crosscut into the footwall. The sublayer has a distinctive mineralogy of quartz, plagioclase, biotite, hornblende, and copper-nickel sulfides and may contain exotic xenoliths of the wall rocks.

The nickel sulfide ores in both cases are dominated by pyrrhotite-pentlandite-chalcopyrite with pyrite and magnetite. Bornite is common in the copper-rich zones. Arsenic-bearing sulfides are common in places. Copper-rich vein ores are known in the footwall of the complex and are dominated by chalcopyrite and cubanite. PGMs include complex tellurides and arsenides.

Subtype 1.5.2: Flood Basalt Association

- Norilsk, Russia
  900 Mt at average grade of 2.70% nickel
- Jinchuan, China
  515 Mt at average grade of 1.06% nickel

Age and Distribution

Flood basalt provinces are fairly widespread through the geological record, but not all have nickel sulfide deposits associated with them. Special conditions appear necessary to produce the nickel sulfides and then to trap and retain them in ore deposits.

General Characteristics

The deposits can comprise both massive sulfide and disseminated sulfides. The relative proportions of the two styles of mineralization vary widely, with massive mineralization dominant at Norilsk but minor mineralization at Jinchuan. The principal minerals are pyrrhotite (FeS), pentlandite (Fe,NiS), and chalcopyrite ($CuFeS_2$), though there are minor amounts of many other minerals.

The igneous provinces that host these deposits are typically very large, commonly more than 1,000 km in diameter. They are dominantly tholeiitic in composition although alkaline volcanic rocks are often associated, and the occurrence of olivine-rich rocks indicates they formed from high-temperature melts.

Subtype 1.5.3: Ultramafic Volcanic Association

- Kambalda District, Australia
  48 Mt at 3.6% Ni, 0.25% Cu
- Thompson Belt, Canada
  69 Mt at 2.5% Ni, 0.13% Cu
- Raglan, Canada
  18.5 Mt at 3.13% Ni, 0.88% Cu
- Shangani, Zimbabwe
  22 Mt at 0.71% Ni

Age and Distribution

Komatiites are mainly restricted to Archaean sequences, and Kambalda-type deposits are almost exclusively developed in Precambrian sequences. In addition to the type locality in Western Australia, good examples occur in similar rocks in Zimbabwe and Canada, and deposits are recorded in Brazil and Russia. Most deposits occur in mid-Archaean to late-Archaean rocks, with only a few minor deposits developed in the oldest greenstone sequences of southern Africa or the Pilbara in Australia. Younger komatiite-like rocks are barren of significant nickel sulfide deposits.

General Characteristics

The nickel sulfide ore bodies in this class can be stratiform or stratabound. Stratiform deposits typically form tabular or ribbon-shaped bodies situated at the base of the host ultramafic units. Such deposits have dimensions from a few hundred meters to 5 km in length, but mostly the stratigraphic thickness of the sulfide zone is less than 10 m, with distinct lower and upper contacts. In some cases they show marked

elongation parallel to the regional volcanic or tectonic grain, with length-to-width aspect ratios of up to 20 at Kambalda.

Stratabound massive and disseminated mineralization is not confined to the base of ultramafic units but can be distributed through the lower to central portion of the host cumulate unit. The shape and size of stratabound deposits are more difficult to quantify because of their diffuse, gradational character; the margins are defined by arbitrary economic cutoff grades. Broadly, they have oblate to elongate lens shapes, depending on the shape and size of the host unit. Dimensions are generally much larger than equivalent stratiform deposits, particularly stratigraphic thickness, which may be up to several hundred meters.

Ore minerals are pentlandite, present as granular aggregates, and, to a much lesser extent, locked in pyrrhotite. Chalcopyrite, pyrite, and ferrochromite are the other principal minerals present. Nickel tenors within the sulfide component (as opposed to the Ni grade of the whole rock) can vary from less than 5% to more than 20%, but they are usually constant in an ore lens. Copper, cobalt, and the platinum-group elements are minor components of the mineralization. Arsenic is present and can occur in significant concentrations, resulting in adverse environmental impact when nickel concentrates are sent to a smelter.

Subtype 1.5.4: Other Mafic and Ultramafic Intrusive Associations

- Voisey's Bay, Canada
  136 Mt at 1.59% Ni, 0.85% Cu
- Selebi-Phikwe, Botswana
  110 Mt at 0.75% Ni, 1% Cu
- Lynn Lake, Canada
  20 Mt at 1.02% Ni, 0.54% Cu

Age and Distribution

This type is generally found in areas of thickened crust in late Achaean to Proterozoic terranes, usually in highly deformed regions where reconstruction of the tectonic setting is difficult. Key examples are Lynn Lake, Canada; Selebi-Phikwe, Botswana; Monchegorsk, Russia; and Kotalahti, Finland. Voisey's Bay, Canada, is possibly also in this type but shows many features similar to the Norilsk deposit (see "Subtype 1.5.2: Flood Basalt Association" section).

General Characteristics

The host mafic/ultramafic intrusions associated with these Ni-Cu sulfide deposits include a variety of types: multiphase stocks, multiphase chonoliths, multiphase sills, and highly deformed sills. The styles of mineralization are also varied, including massive sulfides, breccia sulfides, stringers and veins, and disseminated sulfides. The sulfide minerals are usually pyrrhotite-pentlandite-chalcopyrite-pyrite assemblages and may form massive, matrix, or disseminated types. Pyrrhotite is often dominant, and as a result of this, nickel tenors are generally lower than other magmatic deposits. Nickel–copper ratios are typically between 3:1 and 1:1.

CLASS 2: MAGMATIC DIAMOND DEPOSITS

- Argyle, Australia
  83 Mt at 2.7 ct/t (carat per metric ton)
- Ekati, Canada
  60 Mt at 0.9 ct/t
- Jwaneng, Botswana
  287 Mt at 1.44 ct/t

Age and Distribution

Economic diamond deposits occur with volcanic centers of unusual, highly alkaline volcanic and subvolcanic intrusive rocks, which are known as kimberlites and lamproites.

Diamond-bearing kimberlites and lamproites only occur around the margins of or within stable cratonic blocks that have not undergone major deformation or metamorphism. Productive kimberlites vary widely in age from Precambrian to Mesozoic. Some of the lamproites that host diamond deposits are as young as 55 Ma (million years). Thus, it is likely that the formation of primary diamond deposits has gone on throughout geological time.

General Characteristics

Both kimberlites and lamproites are formed from very volatile-rich magmas that are generated at great depth (about 200 km deep for lamproites and between 200 and 300 km deep for kimberlites). Kimberlites are more gas-charged (with carbon dioxide) than lamproites, so they are more likely to form large diatreme structures close to the surface. In both cases they occur as dykes, sills, and minor plugs formed at depths of several kilometers. However, these are generally small, and so they cannot provide the tonnage required for an economic deposit.

Kimberlite magmas tend to form a deep conical explosion crater or diatreme (pipe). These so-called pipes typically range from 50 to 500 m in diameter with four contrasting rock facies. The diatreme facies consists of fragmented wall rocks and material brought up from deep below. The maar facies contains bedded material that fell back into the crater after an eruption. The crater facies includes finely bedded ash and crater-lake sediments, possibly containing fossils. The pyroclastic facies forms as a tuff ring around the vent. All facies may be diamondiferous, but because diamonds are not actually formed as a part of the magmatic process but are xenocrysts simply carried by the magma from the mantle where they formed, their grades can be highly variable.

The shape of kimberlite pipes is typically steeply conical, rather like a carrot. Lamproites, by contrast, form shallower conical to irregular bodies that commonly do not extend to depth.

CLASS 3: DEPOSITS ASSOCIATED WITH FELSIC MAGMAS

Felsic ("granitic") magmas are often generated in subduction-zone settings, and within this setting, different types of granitic magmas are linked to different metal associations. Subduction-related ore-forming magmas have been classified into two broad chemistries of granitoids: S-type and I-type granites, reflecting contrasting sources for the magmas deep in the crust. It is apparent that Sn-W deposits are associated with S-type granites, whereas Cu-Mo-Au mineralization is associated with I-type granites. A further A-type of granite has been discriminated, and these are associated with Mo-W mineralization.

Type 3.1: Porphyry Cu-Mo-Au Deposits

- Chuquicamata, Chile
  15,000 Mt at 0.77% Cu, 0.01% Mo, 0.0133 g/t Au
- El Teniente, Chile
  11,850 Mt at 0.92% Cu, 0.02% Mo, 0.0236 g/t Au

- Kalmakyr, Uzbekistan
  5,625 Mt at 0.4% Cu, 0.509 g/t Au
- Morenci, Arizona, United States
  4,693 Mt at 0.52% Cu, 0.00639 g/t Au
- Grasberg, Indonesia
  3,409 Mt at 1.12% Cu, 1.07 g/t Au
- Oyu Tolgoi, Mongolia
  2,700 Mt at 1.15% Cu, 0.4 g/t Au
- Sar Cheshmeh, Iran
  1,200 Mt at 1.02% Cu, 0.03% Mo, 0.27 g/t Au
- Madjanpek, Serbia
  1,000 Mt at 0.6% Cu, 0.3 g/t Au

Age and Distribution

Most porphyry deposits are Mesozoic or younger, but deposits are known from rocks older than 3,000 Ma. Porphyry copper deposits have been directly related to the process of subduction and formation of volcanic arcs, and this accounts for their distribution along active subduction margins of the Cordillera of the Americas and western Pacific. Porphyry copper deposits are also found distributed along ancient volcanic arcs, now preserved in such structures as the Tethyan orogenic belt and the Central Asian orogenic collage.

General Characteristics

Typically porphyry copper deposits take the form of low-grade disseminations and fracture-hosted sulfide mineralization usually developed within or at the margin of high-level felsic intrusive rocks. The deposits may take the form of irregular to ovoid bodies in plan and from solid to cup or hollow cylindrical shapes in section. Individual ore bodies may be discrete, they may overlap or even be stacked, and they may have dimensions up to thousands of meters in each direction, such as in the case of the "supergiant" deposits at El Teniente. Ore bodies are characteristically zoned according to ore minerals and alteration assemblages with several "classic" models proposed to describe the distribution of such zonation. In reality, these zones are often far more complex and overlapping, commonly with the superimposition of alteration zones.

Principal ore minerals are chalcopyrite, bornite, chalcocite, tennantite, enargite, molybdenite, and electrum. Associated minerals include pyrite, magnetite, quartz, biotite, potassium-feldspar, anhydrite, muscovite, epidote, chlorite, and clay minerals. In gold-rich porphyries, tellurides and arsenopyrite may be part of the ore assemblage, while accessories such as scapolite, calcite, albite, fluorite, and even garnet may be present.

Hydrothermal alteration is extensive and zoned, and at a deposit scale, it ideally comprises an inner potassic zone characterized by the presence of biotite and/or K-feldspar often accompanied by one or more of magnetite, amphibole, and anhydrite. Outside this is a propylitic zone comprising quartz, chlorite, epidote, calcite, and sometimes albite, with local development of pyrite. A further alteration type, phyllic, may be developed between the potassic and propylitic zones, but it may be more irregular or tabular and show evidence for being often "changed" later than the initial alteration stages. The economic sulfides are usually closely associated with the potassic alteration, although in some cases the presence of strong sodic alteration is important.

Type 3.2: Porphyry Mo (W) Deposits

- Climax, Colorado, United States
  907 Mt at 0.24% Mo, 0.02% W
- Henderson, Colorado, United States
  727 Mt at 0.171% Mo
- Quartz Hill, Alaska, United States
  1,700 Mt at 0.136% Mo
- Malmbjerg, Greenland
  136 Mt at 0.138% Mo, 0.016% W

Age and Distribution

Most porphyry molybdenum deposits are Mesozoic or younger. Located in continental arc terranes, the deposits are hosted in A-type (or anorogenic) granites normally related to a rift or extensional setting related to a mature subduction-zone setting. Most economic deposits are located in the Cordillera of the Americas, with a few in the Pacific region, and there is an important prospect in Greenland.

General Characteristics

The key examples of the type, Climax and Henderson, are genetically related to a small cupola of a regional intrusive batholith emplaced during regional extension. The batholith is an anorogenic or A-type pluton. Host intrusions vary from quartz monzodiorite through to granite. Mineralization is more likely to form as a stockwork than as disseminations. The intrusive bodies at Climax are highly evolved granite magmas rich in tin, tungsten, fluorite, and rubidium, while other bodies are adamellite in their composition.

Ore bodies can be associated with simple, composite, or multiple intrusions associated with dykes and breccia pipes. Multiple intrusions are common, which has given rise to multiple, often stacked, ore bodies (known as "shells") in some of the deposits. The shapes of the ore bodies often form inverted cups, shallow bowls, or hollow cylinders mantling an intrusive stock.

The principal ore minerals are molybdenite, scheelite, wolframite, and sometimes associated cassiterite and bismuthinite. Fluorite and topaz are common accessories not seen in typical porphyry copper deposits. Molybdenite occurs in quartz veinlets that carry other sulfides and gangue, in small fissure veins, in fracture coatings, in breccia matrix, and very rarely as disseminations.

Alteration patterns are very similar to porphyry copper deposits (mentioned earlier), with potassic alteration and silicification particularly important in the cores of the deposits.

Type 3.3: Granite-Hosted Sn-W Deposits

- East Kemptville, Nova Scotia
  56 Mt at 0.165% Sn (+ Cu, Zn)
- Ardlethan, Australia
  9 Mt at 0.5% Sn
- Wheal Jane, Cornwall, United Kingdom
  5 Mt at 1.2% Sn
- Hemerdon, Cornwall, United Kingdom
  42 Mt at 0.18% WO_3 (tungsten trioxide), 0.025% Sn
- Panasqueira, Portugal
  31 Mt at 0.3% WO_3, 0.02% Sn
- Cerro Rico, Bolivia
  828 Mt at 0.35% Sn, 200 g/t Ag

Age and Distribution

The age of these deposits ranges from Archaean to Tertiary, with the bulk of the deposits of Mesozoic age. The Hercynian orogenic belt of Europe (Cornwall to Central Europe) is host to a significant proportion of these deposits, and important deposits are associated with granites in the Appalachian-Caledonide belt of North America. The common feature to all these belts is the association with late-stage magmatism in collisional orogens, where the deposits group in distinct metallogenic provinces.

Well-known locations of Sn-W deposits occur in Cornwall (England), Portugal, Erzgebirge of Central Europe, Malaysia, Bolivia, Tasmania (Australia), and Brazil. The deposits are all related to granitoids but show a range of styles from porphyry Sn (Bolivia), tin greisen deposits, to lode-type Sn-Cu (Cornwall and Tasmania). Tin skarns are often developed in the same terranes (see next section).

General Characteristics

Primary tin deposits can occur within granites or within pegmatites or aplites (dyke-like rocks) associated with the granite. They also occur in rocks surrounding the margins of the intrusive rocks as veins, disseminations, skarns (discussed later), or more distal carbonate replacements (generated by magmatic fluids). Pegmatites, fault-controlled veins, stockworks, and breccias are common, as well as the development of distinctive altered granite contact zones known as greisens. Greisen is a metasomatic-hydrothermal rock that results from high T-alteration of granite by fluorine-rich fluids that exsolved from enriched magma during the last stages of solidification. Greisens are associated with silicification and areas where alkali feldspar is replaced by quartz and muscovite. Wolframite- and cassiterite-bearing quartz veins often form swarms within the alteration, and tourmaline is a common accessory silicate. The outer margins of greisen tend to be marked by zones of intense kaolinization. Mineralization tends to form near the top of granite intrusions (often within the margin of a cupola), but also occurs in large quartz-cassiterite-sulfide veins that may pass from the granite into the metamorphosed margin to the granite.

Type 3.4: Intrusion-Related Gold Deposits

- Fort Knox, Alaska, United States
  220 Mt at 0.9 g/t Au
- Salave, Spain
  15 Mt at 2.99 g/t Au
- Mokrsko, Czech Republic
  57 Mt at 1.9 g/t Au
- Vasilkovskoe, Kazakhstan
  100 Mt at 2.4 g/t Au
- Kidston, Australia
  94 Mt at 1.48 g/t Au

Age and Distribution

Most intrusion-related gold (IRG) deposits so far identified are Phanerozoic, particularly the Cretaceous and mid-Paleozoic. Key belts are the mid-Cretaceous Tintina Gold Province in western Canada and Alaska, the New England-Lachlan Fold Belt of eastern Australia, parts of the Paleo-Tethys belts of Europe, and regions of the Tien Shan in Central Eurasia. Further Paleozoic deposits are recorded in New Brunswick, Canada.

General Characteristics

The deposits are highly variable in precise features, but they generally can comprise a range of mineralization styles reflecting their location with respect to distance from a felsic, ilmenite-series (reduced) pluton. A general zonation of the mineralization is seen with distance away from any associated intrusion, with characteristically a more proximal W-Mo-bearing assemblage passing laterally to a more distal Au-As-Bi assemblage.

Intrusion-proximal deposits may comprise sheeted veins associated with thin aplite and pegmatite dykes developed in the brittle carapace of the reduced pluton itself. In the best example of this type, Fort Knox, gold relates to both higher-temperature magmatic features and more distal hydrothermal veins. The ore assemblage contains gold typically intergrown with bismuth- and tellurium-bearing phases and commonly molybdenum or tungsten minerals. There is generally a low sulfide content, but where present, the more common associated sulfide phases are consistent with a reduced magmatic source often comprising pyrrhotite, pyrite loellingite, and arsenopyrite. Some of the deposits are like skarns, where mineralization replaces adjacent host rocks. The more distal examples of this deposit type share many features found in deposits classed as lode or orogenic gold deposits in metamorphic rocks, where causal intrusions have not been unequivocally identified (see "Type 7.3: Lode Gold Deposits" section). Because of this, discrimination between gold deposits in metamorphic rocks (lode gold deposits) and IRG remains controversial.

CLASS 4: DEPOSITS ASSOCIATED WITH PERALKALINE/CARBONATITE MAGMAS

Type 4.1: Peralkaline Ta-Nb Rare Earth Element Deposits

- Lovozero, Russia
  >1,000 Mt at 0.3% Nb_2O_5, >1% REE

General Characteristics

Peralkaline rocks are important repositories of rare metals such as zirconium, niobium, beryllium, uranium, thorium, tantalum, rare earth elements (REE), yttrium, and gallium. Currently, only deposits in Russia are being mined for niobium. Peralkaline rocks form plutons in stable cratonic, anorogenic regions. Their occurrences are controlled by regional fault systems, and they may form ring complexes. The concentrations of rare metals occur as a closed magmatic system and slowly crystallize with rare metals concentrated in the latest cooling phases. High fluorine in the magma lowers the freezing point and the viscosity, allowing the complexes to fractionate to extreme compositions, concentrating rare metals in the residuum.

Type 4.2: Carbonatite Cu, Rare Earth Element, Nd, Fe, P Deposits

- Mountain Pass, California, United States
  36.3 Mt at 7.67% REE
- Palabora, South Africa
  286 Mt at 0.69% Cu
- Aley, Canada
  20 Mt at 0.7% Nb_2O_5 (niobium pentoxide)

Age and Distribution

Four recognized frequency highs of carbonatite magmatism related to tectonic activity are as follows:

- Mid-Proterozoic: 1,800 to 1,550 million years ago (coincident with the Hudson-Svecokarelian orogeny)
- Mid- to late-Proterozoic: 1,100 million years ago (Grenville orogeny)
- Late-Proterozoic to Paleozoic: 750 to 500 million years ago (Caledonide orogeny)
- Breakup of Pangea: 200 million years ago

The major hard rock niobium deposits are in Brazil, Canada, Palabora, and the major Kovdor apatite deposit in northern Russia.

General Characteristics

Carbonatites are igneous rocks that contain at least 50% (modal) carbonate minerals. These comprise calcite, dolomite, and ankerite, as well as sodium- or potassium-bearing carbonates along with diopside, sodic pyroxenes, or amphibole; phlogopite; apatite; and olivine. They are generally found in stable intraplate areas but are located along major tectonic features, usually associated with topographic doming constrained by crustal-scale faults.

Carbonatites form modest-sized intrusive bodies, flows, or tuffs, and they are closely associated with a range of alkali silicate rocks such as syenites. Carbonatites are usually surrounded by an aureole of metasomatic rock called fenite. Mineralization associated with carbonatites can be either magmatic or metasomatic, where mineralization occurs in the altered assemblage.

Mineralization relates to magmatic layering and flow structures, with the mineralization in the case of the niobium deposits taking the form of irregular lenses. Ore minerals include pyrochlore (niobium), rare earth fluorocarbonates or phosphates, perovskite-magnetite, and apatite. Host rocks also contain many unusual accessory minerals. Barite is common in almost all carbonatites. Phosphorus-rich deposits contain apatite with olivine and magnetite.

CLASS 5: SKARN AND CARBONATE REPLACEMENT DEPOSITS

Examples of Skarns

- Hedley District, Canada
  7.0 Mt at 1.7 g/t Au, 17.1 g/t Ag
- Fortitude, Nevada, United States
  5.1 Mt at 10.45g/t Au, 27.8 g/t Ag, 0.2% Cu
- Mactung, Canada
  33 Mt at 0.88% WO_3
- Antamina, Peru
  745 Mt at 1.24% Cu, 1.03% Zn
- Moina, Tasmania
  30 Mt at > 0.15% Sn
- Las Bambas, Peru
  339 Mt at 1.3% Cu, 0.15g/t Au

Examples of Carbonate Replacements

- Cerro de Pasco, Peru
  175 Mt at 7% Zn, 2% Pb, 103 g/t Ag and
  50 Mt at 2% Cu (+ Au, Ag)
- Colquijirca, Peru
  25 Mt at 7% Zn

General Characteristics

A skarn is defined as an assemblage of zoned, coarse-grained, generally iron-rich mixture of calcium or magnesium silicates resulting from the alteration of an original rock-type. Main types of skarn are those resulting from the following:

- Metamorphic recrystallization of silica-carbonate rocks
- Local exchange of components between lithologies during high-grade metamorphism
- Local exchange of components between magmas and carbonate rocks at high temperatures
- Large-scale transfer of components over a broad range of conditions between magmas and predominantly carbonate rocks

Replacement skarns can form a carapace of altered intrusive rock (endoskarn) surrounded by a zone of alteration in the wall rock (exoskarn). Skarns may be associated with both I-type and S-type magmas (as discussed previously), and the type of magma associated may in part dictate the metal budget of the skarn. Carbonate replacement deposits may lack the typical skarn alteration assemblages.

Because of the complications of describing skarns based on the alteration minerals (which are a combined function of wall-rock chemistry and the superimposed system), mineralized skarns are best classified in terms of component of interest. Seven major types are recognized: iron, gold, tungsten, copper, zinc, molybdenum, and tin.

Carbonate replacement deposits, often termed *mantos* in the literature, tend to be hosted in carbonates or calcareous sediments and are largely polymetallic zinc-lead rich, often with copper and silver. Alteration of the host rocks in this deposit type can be quite cryptic and may be represented by minor calcite and dolomite and quartz, developed close to the ore body itself.

CLASS 6: IRON OXIDE COPPER-GOLD (WITH OR WITHOUT URANIUM) DEPOSITS

- Olympic Dam, Australia
  8,300 Mt at 0.8% Cu, 280g/t U_3O_8 (uranium oxide concentrate), 0.76g/t Au, 3.95g/t Ag
- Candelaria, Chile
  600 Mt at 0.95% Cu, 0.2g/t Au, 3.1g/t Ag
- Ernest Henry, Australia
  122 Mt at 1.18% Cu, 0.55g/t Au
- Salobo, Brazil
  986 Mt at 0.82% Cu, 0.49g/t Au

Age and Distribution

Within Australia and parts of South America, where the largest of these deposits is found, the deposits tend to be hosted in Mesoproterozoic to Neoproterozoic rocks (1,800 to 850 Ma). However, many deposits are found in the Phanerozoic, with the important Chilean iron oxide copper-gold (IOCG) belt hosting the Mesozoic Candelaria deposit. Worldwide, recorded ages of IOCG deposits range from 1,800 to 15 Ma for the deposit group.

General Characteristics

The deposits included in this class range from around 10 Mt to more than 8,000 Mt of contained ore at grades around 1% Cu plus gold, which brings them into the size of porphyry copper deposits. In the case of the giant Olympic Dam deposit, the

setting is interpreted as a diatreme breccia within a caldera, and many authors regard the presence of breccias as a key diagnostic for this deposit type. Other deposits clearly relate to major crustal structures or else have a strong association with the aureole of a distinctive intrusive suite. Kiruna-type iron ore deposits have been proposed by some to represent the copper-poor end members to this deposit class.

The deposits are characterized by the development of large volumes of low-titanium magnetite and/or hematite with a simple assemblage of copper, usually as chalcopyrite and associated gold and sometimes uranium. In the copper-rich deposits, paragenetically late chalcopyrite and bornite occurs within or near the iron oxide phases. The oxides are often zoned from an earlier magnetite to a later or more distal hematite. Other associated minerals are phosphates, dominated by apatite and usually highly enriched REE and sometimes a cobalt and silver enrichment. Silicate gangue minerals include actinolite, pyroxene, epidote, chlorite, and tourmaline. Apatite and other phosphate minerals are common, as well as the rare earth-rich epidote mineral allanite.

Wall-rock alteration is a characteristic of the deposit type. This is often developed on a regional scale (>100 km^2) and may comprise regionally extensive sodic and calcic alteration (up to tens of kilometers away from mineralized districts) and more focused, proximal potassic alteration more closely associated with mineralization. In some cases, the alteration resembles skarn seen associated with magmatic intrusive suites.

CLASS 7: HYDROTHERMAL GOLD AND SILVER DEPOSITS

All the deposits in this class are epigenetic and are subdivided into three broad groups based on host-rock association and depth of formation within the earth's crust.

Type 7.1: Sediment-Hosted Gold Deposits

- North Carlin trend, Nevada, United States
  491 Mt at 5.86 g/t Au
- Twin Creeks, Nevada, United States
  90 Mt at 2.5 g/t Au
- Jerritt Canyon, Nevada, United States
  33 Mt at 6.9 g/t Au
- Bau, Malaysia
  ca. 40 Mt at ~2 g/t Au
- Barneys Canyon, Utah, United States
  2.8 Mt at 2.4 g/t Au
- Zarshuran, Iran
  2.5 Mt at 10 g/t Au

Age and Distribution

Deposits with common features in this class are generally Mesozoic or younger. The type locality is the Carlin District of the United States, which has produced more than 50 million ounces (1,600 t) of gold up to 2002. Here, the deposits are hosted in Paleozoic rocks (largely Devonian) although the mineralization is dated at 42 to 36 Ma. The most comparable deposits are found in the West Qinling and Dian-Qian-Gui areas of southern China, while other analogs are found in Utah, Chile, Malaysia, Indonesia, and Iran. Nevada is by far the most productive district for this deposit type to date.

General Characteristics

Carlin-type deposits are characterized by auriferous pyrite associated with a range of anomalous trace elements occurring as replacements in carbonate-rich host rocks. The deposits may take a variety of forms (e.g., tabular, branched, carrot-like, and irregular), and they may be stratabound or clearly crosscutting. The mineralization appears to be hosted in locally porous lithologies, particularly developed where such lithologies intersect high- and low-angle faults related to regional thrust faults.

Deposits are up to 3 km long, with mineralization occurring in a depth range of up to 1,000 m, although this may comprise several stacked ore bodies. Barren zones between ore bodies are generally formed of less-permeable horizons. Impermeable units generally cap ore zones, and the best mineralization is developed in structural-lithological settings similar to favorable traps for oil reservoirs.

At Carlin, gold is associated with As, Sb, Tl, and Hg and was deposited in main-stage arsenian pyrite and marcasite. Quartz, kaolinite, dickite, and illite are minor gangue phases.

Type 7.2: Epithermal Gold and Silver Deposits

Examples of High Sulfidation

- El Indio, Chile
  23.2 Mt at 4.6 g/t Au, 10 g/t Ag
- La Coipa, Chile
  52.1 Mt at 1.58 g/t Au, 50 g/t Ag
- Lepanto, Philippines
  38 Mt at 3.53 g/t Au
- Pueblo Viejo, Dominican Republic
  150 Mt at 7 g/t Au

Examples of Low Sulfidation

- Martha Hill, New Zealand
  83.3 Mt at 3.2 g/t Au, 33 g/t Ag
- McLaughlin, Nevada, United States
  17.5 Mt at 5.2 g/t Au
- Hishikari, Japan
  3.2 Mt at 63 g/t Au
- Gunung Pongkor, Indonesia
  6 Mt at 17 g/t Au, 162 g/t Ag

Age and Distribution

Epithermal and related deposits occur in modern volcanic arcs such as the circum-Pacific "Rim of Fire" of active volcanism. Other deposits are found in similar volcanic arc settings in other parts of the world, where processes similar to those active around the Pacific Ocean occurred in the past. Ancient volcanic arcs such as parts of the Tethys and the arcs of Central Asia also yield such deposits.

Many of the epithermal deposits occur in regions where copper-gold porphyries and skarns form (see previous sections), and there is sometimes a direct genetic relationship.

General Characteristics

Epithermal deposits formed at shallow crustal levels (typically from the water table to about 1,000 m below at the time of formation). The temperatures that correspond to this depth range (controlled by the boiling point of water with depth) are from less than 100°C to about 350°C.

A major difference is recognized between two contrasting classes of epithermal deposit usually referred to as low sulfidation and high sulfidation, which reflect the nature of the hydrothermal fluid. Low sulfidation means an association of the minerals quartz ± calcite ± adularia ± illite, while in high-sulfidation systems, alteration is dominated by the minerals quartz ± alunite ± pyrophyllite ± dickite ± kaolinite.

Both deposit types can occur in subaerial volcanic settings of calc-alkaline to alkalic and intermediate to acid composition. High-sulfidation deposits tend to occur in proximal volcanic settings, whereas low-sulfidation deposits can occur in any setting and may be hosted in basement rocks. Both deposit types are localized by structures, but any suitable structure may be used including favorable volcanic features such as calderas.

For low-sulfidation deposits, the most important features are their typically open-space vein-fill character, their diverse range of distinctive textures, and the characteristic alteration pattern. Silica (quartz) veins and the presence of surface sinters (hot-spring deposits) are typical in low-sulfidation systems. For high-sulfidation deposits, the most important features are their typically massive, disseminated character; their mineralogy (especially the ore mineral enargite and gangue mineral alunite); and the characteristic alteration pattern. Quartz is uncommon as a vein mineral in high-sulfidation systems but may be present as distinctive vuggy or sometimes massive-quartz alteration.

Hydrothermal alteration around low-sulfidation deposits is characteristic of that produced by neutral-pH fluids. The characteristic ore minerals in low-sulfidation deposits are pyrite, gold (often electrum), other base-metal sulfides (chalcopyrite, high-Fe sphalerite and galena), and often selenides along with gangue minerals quartz, chalcedony, calcite (and other carbonates), adularia, and illite. Hydrothermal alteration in high-sulfidation deposits results from very acid fluids, and the zonation results from their progressive neutralization. The characteristic ore mineral phases of this deposit type are pyrite, enargite (copper arsenic sulfide), pyrite, chalcopyrite, tennantite, covellite, and gold. Tellurides are uncommon but present in some deposits.

Type 7.3: Lode (or Orogenic) Gold Deposits

- Muruntau, Uzbekistan
  1,320 Mt at 4.0 g/t Au
- Ashanti, Ghana
  430 Mt at 4.73 g/t Au
- Golden Mile, Australia
  1,000 Mt at 1.98 g/t Au
- Homestake, South Dakota, United States
  148 Mt at 8.34 g/t Au
- Sukhoi Log, Russia
  388 Mt at 2.7 g/t Au
- McIntyre-Hollinger, Canada
  104 Mt at 9.47 g/t Au

Age and Distribution

These deposits are widespread in metamorphic, orogenic terranes around the world, and settings include Precambrian shields, more specifically Archaean greenstone belts, Paleoproterozoic fold belts, late Proterozoic, and younger orogens. These younger orogens are located along the former margins of the supercontinents of Gondwana and Laurentia, and there are more recent belts related to orogens at the margins of the Pacific. Deposit frequency highs occur in the periods 2,800 to 2,550 million years ago, 2,100 to 1,800 million years ago, and 600 to 50 million years ago.

General Characteristics

Lode gold deposits are normally located adjacent to regionally extensive, often broad first-order deep crustal structures that have a complex geological history. The gold deposits themselves are usually located along second- or third-order structures linked to the regional first-order features, often in sites where the structures bend, change direction, or branch.

Mineralization styles vary from quartz-sulfide stockworks and breccias to quartz dominated vein deposits, showing simple to complex banded textures to disseminations without discrete vein development in sheared rocks. Mineralization is most commonly developed in rocks of greenschist metamorphic grade, although important deposits are found in both lower- and higher-grade rocks.

The wall rocks around the mineralization are often altered, with the commonest type of alteration dominated by sericite, carbonate minerals, and sulfides. Silicification is also common as are tourmaline, K-feldspar, biotite, and albite in some cases. Alteration intensity and width vary with the style of deposit and the nature of the host rocks. In deposits developed in higher metamorphic grades, the alteration can resemble skarn-like assemblages.

The largest of the deposits are typically 2 to 10 km in their longest dimension and up to 1 km wide. The deposits are characterized by generally low amounts of contained sulfide (typically 2% to 5%), with gold–silver ratios generally between 1:5 and 1:10. Pyrite and arsenopyrite are dominant sulfides, although pyrrhotite is common in deposits in higher-grade metamorphic rocks. Tungsten, bismuth, and tellurium are often geochemically anomalous in the deposits, with discrete minerals containing these elements often present.

CLASS 8: VOLCANIC-HOSTED OR VOLCANOGENIC MASSIVE SULFIDE DEPOSITS

- Rio Tinto, Spain
  500 Mt at 2.5% Zn, 0.8% Pb, 0.7% Cu, 0.8 g/t Au, 30 g/t Ag
- Gai, Russia
  380 Mt at 1.57% Cu, 0.74% Zn, 0.06% Pb
- Kidd Creek, Canada
  300 Mt at 9.50% Zn, 1.5% Cu, 0.4% Pb, 130 g/t Ag
- Bathurst, Canada
  110 Mt at 7.3% Zn, 2.4 % Pb, 1% Cu, 65 g/t Ag
- Rosebery, Australia
  9.4 Mt at 16.2% Zn, 5% Pb, 0.74% Cu, 2.9 g/t Au, 155 g/t Ag

Age and Distribution

Volcanogenic massive sulfide (VMS) deposits are one of the few mineral deposit styles that occur throughout the geological record up to the present day. The oldest known sulfide deposits of this type are located in the pre-3,700 million year old Isua group in Greenland, and the youngest are actively forming today in the deep oceans and are the subject of active mineral exploration. Much is understood about the deposits, as there have been many detailed studies of modern seafloor hydrothermal systems where these deposits are seen to form in generally deep ocean areas of active volcanism, either

at mid-ocean ridges or in volcanic back-arc basins. Rare examples are known from the relatively shallow water of the Mediterranean. In the geological record, 80% of VMS deposits are hosted in former volcanic-arc sequences.

General Characteristics

In broad terms, the deposits ideally comprise a broadly stratiform, stratabound mass or lens of sulfide that most likely is hosted in volcanic rocks or sediments. The sulfide lens may vary from less than a meter to several tens of meters thick and have highly variable length and widths. The deposit may thicken in the center, and below this a stockwork zone of altered and mineralized rock may be developed. Above the massive sulfide lens, the deposit may be covered by volcanic rocks or sediments with commonly a very distinct change in the nature of the footwall and hanging wall rocks. Individual deposits vary widely in size and shape. Multiple sulfide lenses are common. Most mines consist of several closely spaced ore bodies.

Plots of the dominant metals in VMS deposits define three major groups of deposits, namely, a copper-zinc, a zinc-lead-copper, and a lead-zinc group. The metal contents of the deposits relate to the nature of the host-rock package, and thus a fivefold classification of mafic, bimodal-mafic, mafic-siliciclastic, bimodal-felsic, and felsic-siliciclastic deposits is proposed. In the broadest sense, the first three groups are dominated by deposits where iron and copper sulfides dominate, while in the last two groups, more polymetallic sulfide assemblages dominate.

In the copper-zinc deposits, the massive sulfide lenses are usually dominated by pyrite, with pyrrhotite, chalcopyrite, and sphalerite normally forming the other major sulfide minerals. Magnetite is a common accessory mineral. Quartz, chlorite, barite, gypsum, and carbonates are the main gangue minerals. The ore bodies are commonly strongly zoned with copper enriched at the base of the deposit with zinc (and lead) at the top. This is usually marked by a mineralogical zonation, particularly in the sulfides. The regional ore-equivalent stratigraphy is sometimes marked by a metalliferous sediment. In the more polymetallic deposits, galena, sulfosalts, and barite are common.

Gold and silver are common credits in VMS deposits, particularly the more polymetallic types. The stockwork feeder zone to the deposits and the peripheral margin to the massive sulfide lenses are likewise commonly gold enriched.

CLASS 9: SEDIMENT-HOSTED DEPOSITS

Examples of this deposit type can be broadly subdivided into sulfide deposits, iron and manganese oxide deposits, sedimentary uranium deposits, chemical sediments, and gold- and uranium-bearing conglomerates.

Type 9.1: Sediment-Hosted Sulfide Deposits

This group of deposits ranges from clastic-hosted copper deposits, such as the Kupferschiefer and the African copper belt, to the clastic-hosted sedimentary exhalative (SEDEX) Zn-Pb (Cu) deposits, where a syngenetic, seafloor origin is proposed, and the clearly epigenetic carbonate, or rarely sandstone replacement Mississippi Valley type (MVT) Pb-Zn deposits. The "Irish type" carbonate-hosted deposits are a subgroup with much in common with SEDEX Zn-Pb deposits, but they also have a few aspects of MVT deposits making them a distinctive group.

Subtype 9.1.1: Sedimentary Exhalative Pb-Zn (Cu) in Clastic Sediments

- Red Dog, Alaska, United States
  150 Mt at 5% Pb, 17.1% Zn, 82.3 g/t Ag
- Gamsburg, South Africa
  150 Mt at 0.5% Pb, 7.1% Zn, 6 g/t Ag
- McArthur River, Australia
  227 Mt at 4.1% Pb, 9.5% Zn, 40 g/t Ag
- Rammelsburg, Germany
  27.2 Mt at 1.0% Cu, 7.8% Pb, 16.4% Zn, 103 g/t Ag

Age and Distribution

These deposits are found in large sedimentary basinal sequences from the mid-Proterozoic to the present. Although SEDEX deposits are probably found in rocks from the Archaean to the present, there are two frequency peaks in the mid-Proterozoic and Palaeozoic. The Proterozoic is host to a series of deposits in Australia (e.g., Mt. Isa), Canada (Sullivan), and South Africa (Gamsburg). The Paleozoic of western Canada and Alaska is host to a number of deposits, including Cirque, Howards Pass, and Red Dog. The Paleozoic of Western Europe hosts the Meggen and Rammelsburg deposits.

The common theme of association is the formation of deposits during periods of major continental rifting. In the case of the Proterozoic, there is a link to the breakup of a large supercontinent to separate Australia, Antarctica, North America, and South Africa. The deposits in the Rhenish of Western Europe and Selwyn Basin of Canada and Alaska are linked to major continental rifting in the Devonian to Carboniferous.

General Characteristics

The deposits form largely conformable to semiconformable stratiform lenses of sulfides and associated hydrothermal products. Deposits typically have a length-to-thickness ratio in the order of 1:20, and many are underlain by some form of vein system that can be interpreted as a feeder. The deposits are usually compositionally layered to some degree on a millimeter-to-meter scale. Breccias, irregular veining, and disseminated textures are common.

Mineralogically, the deposits comprise sulfides together with carbonates and very commonly barite. Pyrite often dominates as the major sulfide, although in many of the Proterozoic deposits, pyrrhotite is the main phase. Sphalerite and galena are usually the main economic sulfides. Siderite and ankerite are common gangue minerals.

Zonation in the deposits is often both lateral and vertical, with transitions from sulfides to oxides common from the core to periphery, both laterally and vertically. Carbonate alteration haloes are common in these deposits, and very commonly the Zn–Ba and Zn–Mn ratios decrease in carbonate with distance from the "feeder" system or core of the deposit. Silicate alteration and silicification may occur in some deposits.

Subtype 9.1.2: Mississippi Valley Type Pb-Zn

- Polaris, Canada
  22 Mt at 4% Pb and 14% Zn
- Pine Point, Canada
  76.1 Mt at 2.9% Pb and 6.5% Zn
- Tri-State District, United States
  500 Mt at 0.6% Pb and 2.4% Zn

- Viburnum Trend, Missouri, United States
  111 Mt at 0.8% Zn and 5.8% Pb
- Jinding, China
  90 Mt at 7.8% Zn, 1.6% Pb
- Angouran, Iran
  14.5 Mt at 26.6% Zn, 4.6% Pb
- Silesia, Poland
  200 Mt at 4% to 5% Zn, 1% to 2% Pb

Age and Distribution

Mississippi Valley type (MVT) deposits are generally restricted to post-Proterozoic rocks, with the exception of the Pering deposit in South Africa. They are known in Paleozoic to recent carbonate-rich sedimentary packages with important deposits in Canada at Polaris, Pine Point, Gays River, and Newfoundland Zinc. The platform carbonate sequences of the United States host these deposits, including the district that gives the deposit its name (Mississippi Valley), Tennessee, Missouri, and the Tri-State District. Major deposits of this type are also found in the Lennard Shelf of Australia, Silesia in Poland, and the Cevennes of France.

General Characteristics

The deposits are generally carbonate-hosted, usually in a dolostone or limestone, although, rarely, they can be sandstone-hosted. The deposits occur mostly in relatively undeformed platform carbonate sequences at the flanks of large sedimentary basin sequences. Districts are often large (up to thousands of square kilometers), but they may comprise clustered small (<10 Mt) individual deposits.

The deposits are normally stratabound, but they are sometimes stratiform where they can resemble SEDEX deposits. Mineralization generally cuts across the stratigraphy but often forms bodies within specific horizons or beds. Ore minerals are dominated by sphalerite and galena with an iron sulfide (pyrite or marcasite). Ore grades in deposits rarely exceed 15% combined Pb-Zn, and mineralization is often accompanied by abundant gangue minerals, dominated by carbonates (calcite, dolomite, siderite, etc.). Fluorite and barite are common; the former mineral seems to be absent from SEDEX deposits, which may be a key discriminating feature.

The deposits range from bedded types to discordant bodies commonly taking the form of mineralized breccias. There are often strong physical controls to the development of mineralization, including the location of basement highs, faults, fractures, and the fault-controlled development of solution collapse breccias.

Subtype 9.1.3: Irish-Type Pb-Zn (Cu)

- Navan, Ireland
  70 Mt at 10.1 % Zn, 2.6% Pb (+14 Mt at 9.85% Zn)
- Lisheen, Ireland
  23 Mt at 13.1% Zn, 2.2% Pb
- Reocin, Spain
  80 Mt at 10% Zn, 1% Pb

General Characteristics

The type locality for this deposit subtype is Ireland, although other deposits are known at Reocin and the Kootenay arc in Canada. Deposits are generally stratabound but sometimes occur stratiform, typically wedge-shaped lenses abutting a fault. The mineralized lenses may be up to 30 m thick and extend 200 m from bounding fault. Larger deposits may comprise individual or stacked lenses that are roughly concordant to bedding.

The mineralization often comprises massive sulfides, but in some deposits layered textures are present. Mineralization comprises sphalerite and galena, the principal ore minerals with iron sulfides (generally pyrite and marcasite) variable from being the dominant sulfide to being a minor constituent. Barite is common in most of the deposits forming massive lenses in some cases.

Subtype 9.1.4: Clastic Sediment-Hosted Copper

- Lubin, Poland
  >3,400 Mt at 2.0% Cu, 40 g/t Ag, 0.2% Pb, 0.1% Zn
- Kolwezi, Democratic Republic of Congo
  880 Mt at 4.5% Cu, 0.4% Co
- Konkola, Zambia
  1,000 Mt at 2.5% Cu
- Nchanga, Zambia
  710 Mt at 2.5% Cu
- Udokan, Russia
  1,310 Mt at 1.51% Cu, 9.6 g/t Ag, 0.5 g/t Au
- Dzhezkazgan, Kazakhstan
  2,000 Mt at 1.02% Cu, 0.82% Zn, 0.8% Pb, 13 g/t Ag
- Aynak, Afghanistan
  705 Mt at 1.56% Cu
- White Pine, Michigan, United States
  >150 Mt at 1.2% Cu

Age and Distribution

These deposits are found in rocks ranging in age from early Proterozoic to Tertiary but are most frequent in late Mesoproterozoic to late Neoproterozoic rocks and in late Paleozoic sequences. These were all periods of major continental breakup, glaciation events, and times when there is evidence for the dissolution of thick evaporite sequences. The following six giant districts are known around the world:

- Neoproterozoic Katangan Basin of Central Africa (Central African copper belt)
- Permian basin of Europe (Kupferschiefer)
- Paleoproterozoic Kodara-Udokan Basin of Siberia (Udokan)
- Paleozoic Chu-Sarysu basin of Kazakhstan (Dzhezkazgan)
- Middle Proterozoic mid-continent rift of the United States (White Pine)
- Neoproterozoic-Cambrian Basin in Afghanistan (Aynak)

Other smaller districts are known in Australia and North Africa, among others.

General Characteristics

Two associations are generally involved for this deposit type. One type (Kupfershiefer) is hosted in reduced, organic-rich sedimentary rocks formed in shallow marine or lacustrine environments comprised of silts, shales, sandstones, and dolomites. These units are typically overlain by oxidized coarser units of oxidized red-bed type sequences. The second type of deposit ("red-bed copper") is hosted within continental red-bed sequences where mineralization occurs at the interface between gray (relatively reduced, often pyrite-bearing) and red (oxidized, hematite-bearing) sandstones, arkose, or

conglomerates. Some deposits in the Central African copper belt are found in highly metamorphosed rocks that appear to be the basement sequences to the basinal deposits seen elsewhere hosting copper mineralization. In some cases, such as the Kupfershiefer, thick sequences of oxidized volcanic-rich sediments are present.

In most cases, deposits are generally thin (usually less than 3 m and never greater than 30 m thick), stratiform, sediment-hosted mineralization comprising Cu (±Co, Ag, Pb, Zn, Pt, Pd). The grade of the mineralization is typically 1.5% to 3% Cu, but mineralization is laterally very continuous with huge tonnages developed (the Central African copper belt contained more than 3 Gt of proven ore grading close to 3% Cu). The deposits are dominated by fine-grained disseminated sulfides, comprising largely chalcocite (Cu_2S), digenite (Cu_9S_5), bornite, chalcopyrite, carrolite ($CuCo_2S_4$), galena, sphalerite, and pyrite.

Type 9.2: Sediment-Hosted Iron and Manganese Deposits

Subtype 9.2.1 Ironstones

These deposits include the Minette-type iron ores of Lorraine and Luxembourg; the Northamptonshire ironstone in England; and the Clinton-type ironstones of Newfoundland, Alabama, and the Appalachians. The giant Lisakov deposit in the southern Urals is an oolitic ironstone. The Minette-type oolitic deposits have been major sources of iron in the past, but their low grades (typically 25% to 35% Fe) have made them increasingly uneconomic as a source of iron.

Age and Distribution

Both Clinton-type and Minette-type iron ores are restricted to the Phanerozoic. Clinton-types are developed in Cambro-Ordovician sequences of North America. Minette-type ironstones are common throughout the Mesozoic of Europe, in western Kazakhstan, and in parts of northern Africa. Major Oligocene ironstones are present at Kerch in Ukraine and at Lisakovsk in Kazakhstan, among others.

General Characteristics

Minette ores are characterized by regionally extensive horizons of siderite and chamosite within dominantly limestone sequences. In Europe they are found in the Jurassic, extending from the United Kingdom through to central Germany. The chamosite is often distinctive, as it usually takes the form of oolites in a mixed silicate and carbonate matrix. The ores are thus lime-rich (up to 20%), which leads to these ores being generally self-fluxing. Clinton ores extend for at least 1,000 km along strike across the Appalachians. They are frequently associated with black shales and have been deposited in continental shelf and estuarine basins. Beds in the ores are generally less than 1 m thick, separated by ferruginous shales. The ores are typified by oolitic hematite-rich beds, with features suggesting a shallow depositional environment although minor sulfide-rich (pyrite) facies are developed.

Subtype 9.2.2: Banded Iron Formation

Three types of banded iron formation (BIF) are recognized that broadly equate to three major time periods in the earth's history, from the Archaean to Proterozoic, and three distinctive tectonic settings. Algoma-type deposits formed in rocks 3,500 and 3,000 million years old, Lake Superior (Superior)-type deposits occur in rocks 2,500 to 2,000 million years old, and the smaller group of Rapitan-type deposits are found in rocks 1,000 to 500 million years old.

Sub-subtype 9.2.2.1: Algoma BIF

- Cerro Bolivar, Venezuela
  500 Mt at >60% Fe
- Koolyanobbing, Australia
  116 Mt at 62% Fe

Age and Distribution

Algoma-type deposits are typically found in Archaean greenstone belts with the type locality in the Abitibi greenstone belt of Canada. Other deposits include Kudremuk, India; Cerro Bolivar, Venezuela; and Koolyanobbing, Australia. Similar deposits are known in the Paleozoic of the Bathurst District, Canada, and modern analogs may be present on the current seafloor.

General Characteristics

Algoma-type iron deposits are formed as part of sedimentary sequences, but they show a close relationship to volcanic rocks of mafic to felsic composition. The deposits are geographically related to proximal and distal volcanic centers. Mineralization comprises micro- to macro-interlayered iron mineral (oxides, sulfides, or silicates) and silica bands. These silica-iron bands are typically interbedded with clastic sediments and volcanics. The host horizons are regionally extensive features that are often laterally zoned into silicate, oxide, carbonate, and sulfide-dominated facies. Deposits become mineable usually where metamorphosed to a coarser grade or where thickened by tectonic events, or else where they are residually enriched by secondary processes such as weathering.

Sub-subtype 9.2.2.2: Superior BIF

- Hammersley Province, Australia
  25,000 Mt at >55% Fe
- Quadrilatero Ferrifero, Brazil
  6,000 Mt at >50% Fe
- Serra Norte, Brazil
  17,300 Mt at 66% Fe

Age and Distribution

Superior-type deposits are generally Proterozoic in age, largely in the time period 2,500 to 2,000 million years old, and examples are found distributed around the world. Examples include the following: Transvaal Basin, South Africa; Labrador, Canada; Minnesota, United States; Damara, South Africa; and Krivoy Rog in the Kursk region of Ukraine and Russia.

General Characteristics

Mineable deposits of Superior-type BIF comprise discrete stratigraphic units of oxide lithofacies. Stratigraphic thicknesses may be 30 to 300 m, with strike lengths in the order of kilometers. Folding and faulting may result in favorably thickened packages.

BIFs are characterized by alternating bands of silica and iron minerals that when unmetamorphosed are very fine grained. The primary minerals are largely magnetite, hematite, quartz and minor iron silicates, carbonates, and occasionally

sulfides. In manganiferous facies, pyrolusite, manganite, and hollandite may be present. Primary sedimentary features may be present, such as oolites in a cherty or carbonate matrix, rare cross-bedding features, intraformational breccias, slumping, compaction structures, and stromatolite textures. Nonmetamorphosed BIFs are referred to as taconite, whereas metamorphosed BIFs generally become coarser grained and are known as metataconite or itabirite.

Taconite rarely makes mineable ore unless upgraded by secondary processes. Itabirite is much coarser grained and can be upgraded by physical sorting, although in most cases both types need to be upgraded by secondary processes to produce mineable ore. Secondary upgrading serves to remove silica and to cause iron enrichment by oxidation.

Sub-subtype 9.2.2.3: Rapitan BIF

- Crest, Yukon, Canada
  18,600 Mt at 43.8% Fe

Age and Distribution

Rapitan deposits are known from the Mackenzie Mountains in Canada; Jacadigo, Brazil; and the Adelaide Geosyncline, Australia. They are late Proterozoic to early Paleozoic in age.

General Characteristics

The Rapitan BIFs are generally formed of bedded chert-hematite, much like other Superior-type iron formations. However, their associations are significantly different as they are generally associated with diamictites (tillites), dropstones, sandstones, conglomerates, and argillites. The tectonic setting is one of graben and fault scarps located on the margins of continents or ancient cratons.

Subtype 9.2.3: Manganese Ore

- Mamatwan, South Africa
  13,600 Mt at >20% Mn

Age and Distribution

The largest single manganese resource is the Kalahari manganese field of South Africa, which is Proterozoic and related to an iron-rich BIF. Manganese facies of iron-formation ores are also developed in the Lake Superior region and in the Devonian sequences of Kazakhstan and Russia. Manganese is produced from BIF in Gabon. Other important deposits occur in the Oligocene Nikopol Basin in Ukraine and in the Chiatura Basin in Georgia. Important Cretaceous deposits are found in northern Australia (Groote Eylandt) and in the Molango District of Mexico.

General Characteristics

In BIF-type associations, the manganese-rich bodies lie stratigraphically above the iron-rich facies with manganese minerals such as pyrolusite and hollandite present instead of the iron oxides. Textures in the ores are similar to the BIFs described in the previous sections.

In the case of the important Oligocene deposits of the Ukraine and Georgia, the ores consist of irregular concretions, nodules, and masses of manganese oxides with or without carbonates in a silt to clay matrix. The manganiferous bed averages 2 to 3.5 m thick, runs 15% to 30% Mn, and is traceable for at least 250 km along the strike, although postdeposition erosion frequently disrupts the bed. The main minerals in the ore zone are pyrolusite and psilomelane, together with manganocalcite and rhodochrosite. The deposits are laterally zoned from oxides to carbonate facies.

Type 9.3: Sedimentary Uranium Deposits

Sedimentary uranium deposits can be classified in two catagories: unconformity vein-type deposits and sandstone-hosted uranium deposits.

Subtype 9.3.1: Unconformity Vein-Type Uranium

- Cigar Lake, Canada
  160,000 t at 19% U_3O_8
- McArthur River, Canada
  150,000 t at 21% U_3O_8
- Jabiluka No. 2, Australia
  163,000 t at 0.55% U_3O_8
- Crouzille, France
  13,400 t at 0.25% U_3O_8

Age and Distribution

These deposits are generally Proterozoic in age (2,500 to 600 Ma) and are found in a number of countries around the world, The most famous are the deposits in the Proterozoic Athabasca and Thelon basins of Canada, but other important fields are found in Australia (East Alligator, Jabiluka) and formerly the Crouzille region of France.

General Characteristics

Deposits are generally small but can range up to 50 Mt in size. The deposits can also be very high grade. Cigar Lake and McArthur River in Canada have the highest grades of any uranium deposits, averaging 19% and 24% U_3O_8, respectively. Deposits often produce important by-products, and in the case of Jabiluka in Australia, the deposit carries 15 g/t Au. Key Lake in Canada has a nickel resource but is too radioactive to exploit.

Deposits are largely hosted in sandstone units overlying older, metamorphosed, generally Proterozoic basements (Paleozoic in the case of the French deposits). Mineralization is developed at and below the unconformity, especially where the sequence is intersected by faults passing down into the basement. The ore bodies are generally tubular to flattened cigar-shaped high-grade zones passing into a more disseminated style mineralization. Mineralization is strongly controlled by subvertical faults, shear and fracture zones, as well as the subhorizontal plane of unconformity.

Mineralization appears as either polymetallic mineralization containing uranium, nickel, cobalt, and arsenic, or it is monomineralic containing uranium only. Monomineralic-type mineralization is usually formed of euhedral grains of uraninite and pitchblende within the host rock, which may be arkose, semipelite, graphitic schist, pelite, or in some cases mylonite. Polymetallic mineralization is typical for sub-unconformity mineralization in the Athabasca Basin. At the high-grade Cigar Lake deposit, this consists of pods and lenses of pitchblende and coffinite with arsenides and sulfoarsenides of nickel and cobalt, together with nickel, cobalt, lead, molybdenum, iron, and zinc sulfides. Gold and platinum group metals (PGMs) can also be present. Monomineralic mineralization is commonly hosted in lenses around structures, or as thin stockwork veinlet zones.

Alteration is common and may take the form of chloritization, argillization, carbonation (dolomite), silicification, pyritization, and tourmaline alteration.

Subtype 9.3.2: Sandstone-Hosted Uranium

- Chu-Sarysu and Syrdarya, Kazakhstan
  1.3 Mt contained U_3O_8
- Callabonna, Australia
  41.2 kt contained U_3O_8
- Wyoming Basin, United States
  320 kt contained U_3O_8
- Southern Texas, United States
  ~80 kt contained U_3O_8
- Franceville Basin, Gabon
  ~38 kt contained U_3O_8

Age and Distribution

Sandstone-hosted deposits span a range of host rock ages and are distributed globally with principal deposits found in the Wyoming Basin, Kazakhstan, New Mexico, and Australia.

General Characteristics

These deposits are generally hosted within sandstones as the name suggests, and they are generally medium- to coarse-grained, continental fluvial to marginal marine sediments and are ideal for in-situ leach mining. Foreland basin settings are the most prospective for this deposit type as tectonic activity is a key component to promote fluid flow through the sediments to precipitate uranium. A key feature is also the presence of impermeable units, either a shale or mudstone, interbedded in the host stratigraphy, which helped to focus and contain fluid flow into favorable trap sites.

The deposits are subdivided into the following three main types based on morphology: tabular, roll-front, or basal channel. The tabular-type deposits form elongate lenses of mineralization within sediments that show selective reduction. This reduction may be caused by the presence of organic matter in the host sediments. The mineralization develops parallel to the postulated groundwater flow through the host rocks. Roll-front deposits form disseminations in permeable and porous sandstone beds. The mineralization is developed within the sandstone at an interpreted redox front. Roll-front deposits usually form the largest deposits, and the fact they are hosted in porous sandstone makes them ideal for in-situ leaching. Basal channel deposits are similar to roll-front deposits, but they are specifically hosted in the coarse channel gravels of ancient fluviatile sediments.

The primary mineralization is pitchblende and coffinite, but secondary minerals (e.g., autunite, uranophane, carnotite, and tobernite) are often present due to later weathering events. Grades in the deposits are usually quite low (commonly $<0.4\%$ U_3O_8), but they can usually be mined by low-cost leach technology, making them economic targets for extraction and processing.

Type 9.4: Gold and Uranium in Conglomerates

- Witwatersrand Basin, South Africa
  4,000 Mt at 9.2g/t Au > 200 g/t U
- Elliot Lake, Canada
  140 Mt at 0.09% U

Age and Distribution

These deposits are found in rocks as old as 3.0 billion years in southern Africa and India. Gold-bearing conglomerates in South Africa are 3,070 million years old, whereas the bulk of the Witwatersrand deposits, which have accounted for more than 35% of all gold mine production, are hosted in rocks between 2,950 and 2,710 million years old. The Elliot Lake conglomerates in Canada are 2,450 million years old and the Tarkwian gold-bearing conglomerates in West Africa are 2,130 million years old. Tertiary-aged gold-bearing pebble conglomerates are known in New Zealand with many features similar to the more ancient examples. The bulk of the economic deposits are Archaean to Proterozoic, although if they represent paleoplacer deposits (which is the traditional model for their formation), then modern analogs could be the gold-bearing alluvial gravels found in many goldfields still actively forming today.

General Characteristics

The most significant deposits of this group are the gold-uranium deposits of the Witwatersrand Basin in South Africa and the uranium-bearing conglomerates of Elliot Lake in Canada. The following observations are based on these deposits.

In the Witwatersrand, gold is hosted within a sequence of quartz arenite-dominated rocks that are chemically, mineralogically, and texturally submature to supermature. Quartz pebble rocks dominate, but feldspathic quartzites are also relatively common, suggesting a granitic source for much of the detritus. At Elliot Lake, the mineralization is hosted in similar quartz-rich pebble conglomerates where the detritus appears to have been derived from the reworking of pegmatitic granites.

In the Witwatersrand, a number of host rock types are associated with the gold-bearing conglomerates. At the base of the section, submature arenites are developed in a stream system on granitic basement rocks. Further up, the West Rand Group contains shale, sandstone, and iron formation. The main host part of the package, the Central Rand Group, formed as part of major braided stream systems, clearly developed in broad valleys as parts of alluvial fans. Unidirectional and herringbone cross bedding is recorded, indicative of both single flow directions and a degree of reworking in shallow intertidal conditions. Argillites, siltstones, and minor limestones are testament to standing water during parts of the sedimentation, and in places, glaciogenic deposits and iron formations are developed. Finally, the uppermost part of the sequence comprises a unit called the Black Reef, which contains gold where it has reworked the Central Rand Group rocks.

The ores in the Witwatersrand are typically free-milling, with gold largely present as native grains containing 7% to 14% Ag. The ore material is made up of the following three types of ores: coarse pebble conglomerates ("blanket reef"), carbon seams, and pyritic quartzites. Gold may take one of five different forms: free gold particles with minor sulfides (e.g., sphalerite, linneite, gersdorffite, cobaltite, and chalcopyrite), gold associated with carbonaceous matter, redistributed gold along fractures or on the surface of other grains, gold in pyrite or arsenopyrite, or gold in later secondary quartz veins.

At Elliot Lake, a somewhat similar package of fluviatile sediments hosts mineralization where uraninite occurs as poorly rounded to euhedral grains up to 0.2 mm across within the matrix of a quartz-pebble conglomerate.

Type 9.5: Chemical Sediments

Chemical sediment deposits are divided into evaporites and manganese nodules. These are discussed as follows.

Subtype 9.5.1: Evaporites

Evaporites are the main source of rock salt, potash, borates, nitrates, agricultural fertilizers, and lithium.

Age and Distribution

Evaporites are only known to have formed from the Proterozoic onwards, although in the bulk of cases in the Proterozoic, the evaporite horizons are now no longer present. Basins hosting major exploited evaporite deposits include the Permian Zechstein Formation, present through most of northern Europe and the eastern United States. Other basins are the modern intracontinental lacustrine basins of Chile, Bolivia, Argentina, California, and Utah, where borates, nitrates, and lithium deposits are mined. One of the world's largest borate-producing regions is located in western Turkey.

Subtype 9.5.2: Manganese Nodules

This deposit type is unique, as it occurs largely in international waters of the ocean. The deposits have never been mined owing to the combination of the water depths and the lack of a comprehensive legal framework for the exploitation of the nodules.

Age and Distribution

The nodules occur in the deep pelagic parts of all major modern oceans, but they especially occur in the Pacific Ocean, where exploration indicates there may be up to 100 nodules per square meter. They represent an enormous future resource of nickel, copper, cobalt, and manganese.

General Characteristics

The nodules occur in the deep oceans where normal clastic sedimentation is low to absent. In the Pacific, where the nodules are best developed, they contain up to 2% combined Ni and Cu with substantial Co and Zn. Deposits are very extensive; the most attractive resource is in the so-called Clarion-Clipperton region of the Eastern Pacific, in a belt bounded by longitude 60°W to 180°W and latitude 15°S and 15°N.

CLASS 10: ORES RELATED TO WEATHERING

Type 10.1: Laterites

Lateritization is a weathering process occurring under humid tropical conditions leading to the dissolution of most soluble elements, such as K, Na, Mg, Ca, and some Si, and accumulating less soluble elements, such as Al and Fe. Principal deposits formed by this process are nickel-cobalt, bauxite (Al), and lateritic gold. Lateritization of existing lower-grade iron ores can result in significant upgrades to the deposits, largely as a result of silica removal. Important phosphate and niobium deposits have been upgraded in this way at Mabounie in Brazil, in Gabon, and at Lueshe in the Democratic Republic of Congo.

Subtype 10.1.1: Bauxite

Two main types of bauxite are karstic and lateritic. Karstic bauxites form by the dissolution of carbonate sequences with concomitant weathering of associated aluminosilicates. Lateritic bauxites are developed over a variety of aluminum-bearing rocks and form a characteristic zoned profile. Karstic bauxites are commonly developed in carbonate sequences in the Mediterranean region, for example, at Les Baux in France, from where the deposit type gets its name. Lateritic bauxites are developed in five major regions of the world: Guiana and the Brazilian shield in South America, a province in West Africa, India, Southeast Asia, and Australia. These lateritic bauxites account for around 90% of world's bauxite resources.

General Characteristics

In a typical lateritic bauxite, the fresh parent rock is overlain by a saprolite composed largely of kaolinite with some iron hydroxides and residual parent minerals (e.g., quartz, zircon, and magnetite). Saprolite is where the texture of the original rock is still discernable and where the volume is preserved despite removal of soluble components. This saprolite varies in thickness up to 100 m in deeply weathered terrains. Above the kaolinitic saprolite, gibbsite replaces kaolinite, and this is the bauxite horizon. This horizon, which constitutes the ore zone, is typically a few meters thick and usually has very high aluminum grades and low iron values. Above this, the profile is capped by either a duricrust (which may be bauxitic or ferruginous) or a clay-rich zone. Any duricrust is likely to be composed of oxyhydroxide minerals and kaolinite.

Three types of lateritic bauxite are recognized: orthobauxite, metabauxite, and cryptobauxite. Orthobauxites are those interpreted to have formed in situ by a single-stage weathering process. Metabauxites are more aluminum-rich, having developed on quartz-poor rocks as an upgrade reworking under drier conditions of an already formed orthobauxite. Cryptobauxites are used to describe bauxites buried under clay cover, again resulting from the reworking of an orthobauxite under probably more humid conditions.

Subtype 10.1.2: Nickel (Cobalt) Laterite

- Koniambo, New Caledonia (silicate)
  150 Mt at 2.57% Ni
- Cerro Matoso, Colombia (silicate)
  40 Mt at 2.5% Ni
- Murrin Murrin, Australia (clay-silicate)
  125 Mt at 1.02% Ni
- San Felipe, Cuba (clay-silicate)
  230 Mt at 1.33% Ni
- Euboea, Greece (oxide)
  185 Mt at 1.3% Ni
- Goro, New Caledonia (oxide)
  200 Mt at 1.57% Ni
- Buruktal, Russia (oxide, clay-silicate)
  110 Mt at 1.2% Ni
- Vermelho, Brazil (oxide, silicate)
  410 Mt at 1.23% Ni

Age and Distribution

A number of known nickel laterites occur in the currently tropical zones of the earth, and it is clear that many are actively forming today. Major deposits occur throughout Central America, Africa, Southeast Asia, and the Pacific, and they can be attributed to recent weathering periods. Southern Europe (Balkans and Greece) through to Turkey and many parts of Russia and adjacent states host nickel laterites, which are the result of late

Mesozoic and Tertiary weathering periods. In addition, many deposits are located on ancient crust (e.g., Australia and Brazil), where the weathering history is likely to have been very protracted and difficult to ascribe to one particular time period.

General Characteristics

In broad terms, in-situ nickel laterites form simple zoned profiles over unweathered parent rock. In a number of cases the laterite may be transported to some degree where the nickeliferous unit is no longer authochthonous on its parent. In all cases, the initial parent rock was an ultramafic, and in a rare case, mafic igneous parent rock was largely composed of Mg- and Fe-bearing silicates where nickel substituted for Mg to some degree. Typically, nickel values in unweathered parent rock are 0.2% (by weight) Ni. Apparently there is no necessity for the parent rock to be unusually enriched in nickel. Ultramafic rocks suitable for forming nickel laterites include dunites, peridotites, and serpentinized equivalents, where nickel resides in olivine, pyroxene, and serpentine-group minerals.

A typical laterite profile comprises an unweathered ultramafic parent rock passing up into a saprolite zone (weathered but still retaining a fair degree of competence, original texture, and volume). Above the saprolite zone is an oxide zone that may have a variable component of clay (usually mixed-layer and iron-rich). This oxide zone is often zoned into a lower limonitic (goethite-rich) and upper hematitic facies, which is likely to have silica present as aggregates or distinct ledges. The whole profile is capped by a ferruginous or siliceous duricrust where present.

Nickel is usually enriched in the saprolite zone where Ni substitutes for Mg in a range of hydrous layer silicate minerals (such as serpentine, talc, sepiolite, and chlorite). Nickel is also enriched in the oxide zone, where nickel may be hosted in mixed-layer clays, more commonly in goethite, and also in a range of manganese oxides, including asbolane. Cobalt is an important associated metal in these deposits, and this is strongly fixed by the manganese oxide phases. Hematitic parts of the profile are generally nickel-poor and silicification decreases grade.

Subtype 10.1.3: Lateritic Gold

Artisanal miners in West Africa and Brazil have mined this deposit type for centuries. In these deposits weathering has both liberated gold and also resulted in the physical concentration of gold particles. Major gold resources of this type have been identified at Ity in Cote d'Ivoire, Ashanti in Ghana, and Igarape Bahia in Brazil.

In parts of Australia, nugget-bearing weathering zones have produced spectacular numbers of gold nuggets, and bauxitic deposits in Queensland carry gold where no mineralization is detected in the basement rocks. A major gold resource of 45 Mt at 1.8 g/t Au was identified at the Boddington bauxite mine, Western Australia, in 1988 after material was finally sent for assay.

Type 10.2: Supergene Weathering

Supergene weathering deposits are divided into secondary copper deposits and secondary zinc deposits.

Subtype 10.2.1: Secondary Copper

- Bingham Canyon, Utah, United States (porphyry Cu)
  338 Mt at 1.5% Cu
- Morenci, Arizona, United States (porphyry Cu)
  379.4 Mt at 1.02% Cu
- Potrerillos, Chile (porphyry Cu)
  137.4 Mt at 1.51% Cu
- Chuquicamata, Chile (porphyry Cu)
  684.3 Mt at 2.12% Cu
- El Teniente, Chile (porphyry Cu)
  234.8 Mt at 2.18% Cu
- Phoenix, Cyprus (VMS)
  9.2 MT at 0.78% Cu + 6 Mt at 0.31% Cu

Age and Distribution

Many major porphyry copper camps around the world have a developed supergene zone that makes them attractive targets for exploitation. Many large VMS deposits also have supergene blankets (e.g., Paleozoic of the Urals), although these have not been exploited to date. The Phoenix ore body in Cyprus is an example of a supergene-enriched VMS that has been exploited for the oxidized ores.

General Characteristics

At the base of a typical supergene profile is the primary hypogene ore, which in the case of a porphyry copper deposit is typically comprised of chalcopyrite and pyrite with various silicate phases (e.g., K-feldspar, muscovite, epidote, biotite, and quartz). Above this lies the more-reduced part of the supergene blanket, which may contain secondary sulfide copper minerals such as the more common chalcocite, covellite, and bornite. Above this is the more oxidized supergene assemblage, including copper carbonates and oxides. Above the supergene zone are the leached zones, where sulfide minerals are generally destroyed, and finally, a full profile would be capped by a zone rich in hydrated iron oxides (gossan or iron hat).

The main zone of economic interest is the secondary sulfide zone containing bornite, covellite, and chalcocite. These sulfides are copper-enriched compared to the primary ore, and thus the grades are often significantly higher than the primary ore. In addition, the mineralogy of the ore is more favorable for low-cost solvent-extraction methods of treatment. Gold is generally conserved during the supergene enrichment process, but its fineness and grain size may increase. In lateritic weathering terranes, however, gold grades may be significantly upgraded.

Subtype 10.2.2: Secondary Zinc

- La Calamine, Belgium
  1.9 Mt at 40% Zn
- Jabali, Yemen
  9.4 Mt at 10.8% Zn, 2.3% Pb, 77 g/t Ag
- Angouran, Iran
  3.3 Mt at 38% Zn, 2% Pb
- Jinding, China
  50 Mt at 8% Zn, 1% Pb
- Silesia, Poland
  17.3 Mt at 17% Zn
- Skorpion, Namibia
  24.6 Mt at 10.6% Zn

General Characteristics

Three types of secondary zinc deposits are defined: direct replacement, wall-rock replacement, and residual and karst

fill deposits. Direct replacement deposits are effectively zinc-rich gossans where the sulfide body is weathered. In this case, sphalerite is replaced by smithsonite and hemimorphite. Usually, deposits are complex mixtures of sulfides (galena, pyrite, chalcopyrite, etc.), so the secondary assemblage is usually a more complex mixture of oxides, carbonates, and silicates. Deposits formed from a weathered MVT tend to be the most mineralogically simple, dominated by smithsonite, hemimorphite, and hydrozincite. In more arid weathering profiles, willemite (silicate) is formed.

CLASS 11: PLACER DEPOSITS

Placers have been a source of metals from early times, as they are easily worked by manual methods, processing by washing or winnowing techniques. Most major gold rushes were based on the discoveries of alluvial gold, which may have later led to hard rock discovery. Very significant placer diamond deposits are worked on the west coast of southern Africa. Beach sands are mined in various regions of Australia, eastern Africa, and the Americas.

Deposit types range from various beach sands containing heavy minerals, such as rutile, ilmenite, zircon, and monazite, to deposits of gold, platinum, tin (cassiterite), and diamonds. In all these cases, the mineral of interest becomes concentrated in either river drainages or in active marine environments by the action of moving water. The minerals of interest are generally both obdurate and of higher density than other minerals, so they tend to become concentrated in higher-energy environments in marine or river systems.

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CHAPTER 3.2

Minerals Prospecting and Exploration

José L. Lee-Moreno

INTRODUCTION

Minerals prospecting and exploration cover a wide range of earth science disciplines including geology, geochemistry, geophysics, and remote sensing (which incorporates satellite imagery and multispectral data interpretation). New laboratory techniques and computer aids are used for modeling and understanding the genesis of ore deposits. These issues will be discussed in detail in this chapter.

Prospecting

Although *prospecting* and *minerals exploration* are widely considered synonymous, there are still some regions in the world where prospecting is considered the initial, more-primitive phase of exploration, reminiscent of the old gold-panning prospectors, but with the addition of some modern technology. Exploration refers to semidetailed or more-advanced detailed studies. Prospecting can help locate sites that, after a formal exploration program, may result in an ore deposit discovery. These terms will be used indistinctively here.

Ancient prospecting began with panning in creeks and riverbeds and continued with the physical exploration of surrounding hills. These areas were quickly mined out and abandoned but more recently have provided excellent guides for the discovery of source deposits.

In modern prospecting, metal detectors and artisanal heavy mineral concentrators are still used. In more-advanced placer mining operations, draglines and high-capacity gravity, magnetic, and centrifugal classification and concentration equipment are used successfully. A new generation of optical and laser mineral separators have also been introduced.

Minerals Exploration

The main objective of minerals exploration is to locate ore deposits, which are anomalous accumulations of one or more minerals that can be mined at a profit. Drilling is the most reliable, more-expensive technique used to confirm or deny the existence of an ore deposit, but new methods and technologies are still being developed.

Most outcropping ore deposits have already been discovered, so the modern mineral prospector must use more-subtle markers to recognize the presence of an anomalous mineral occurrence. New mining and metallurgical techniques allow today's mining industry to evolve beyond limited high-grade production to take advantage of lower-grade, large-volume extraction and processing. The days of the small mine are practically over.

An additional objective of mineral exploration today is to find significant extensions of preexisting deposits associated with operating or abandoned mines. Some well-known mining districts, under changing economic conditions and the application of modern exploration techniques, may present new opportunities to extract remaining minerals from them.

Increased understanding of ore-deposit modeling using statistical analyses, computer technology, and new laboratory methods provides the explorationist with powerful tools for discovering new deposits in the subsurface including deposits extending from previously known or partially mined ore bodies. Broadly classified, new deposits may be considered either as outcropping (surface) or concealed (subsurface).

There is little new ground to be explored in more-developed countries, except in less-accessible, remote areas. The search for outcropping deposits is carried out primarily in developing countries. In some cases, outcropping deposits may have been buried by tropical soils or by some other post-mineralization event, thereby eluding discovery.

Discovery of concealed deposits at shallow depths requires the integration of various advanced technological methods for preliminary assessment. Sound exploration methods are the most valued tools for the exploration geologist, whose primary objective is to identify the geological characteristics of a mineralized system that can lead to the discovery of an ore body. These characteristics must include geological, lithogeochemical, isotopic, and petrochemical features; physical properties of both altered and unaltered host rocks; structural and tectonic framework; and any other relevant relationships of the mineralizing events in terms of time and space.

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Compiling and analyzing preexisting data is indispensable. The exploration geologist must identify specific factors that controlled the mineralization in an area, district, or province and, using multidisciplinary methods, "see through" any post-mineralization cover to determine the possibility of a concealed deposit.

Geological surveys, varying in detail and scale, are fundamental to all exploration programs. It is important to accurately identify any geological features of nearby deposits and the general characteristics of their position or emplacement. The features of previously outlined metallogenic provinces are very useful at this stage. Among the most important features considered during this entire exploration process are geochemical, geophysical, and remote sensing signatures; depositional models; mineral alteration; zoning; and other mineralogical guides. The details may be modified and redefined more accurately as the program advances and new indicators are discovered.

Geological interpretation must rely on additional techniques and disciplines such as field mapping, structural geology and tectonics, geomorphology, petrology and petrography, sedimentology and stratigraphy, mineralogy, geostatistics, geotechnical engineering, hydrogeology, metallurgy, geothermometry, topography, and geochemistry. The three major methods that integrate all geological observations in formal exploration programs will be reviewed briefly here: exploration geochemistry, geophysical exploration, and ore deposits. Resource and reserve estimations and definitions constitute one of the final stages before entering into production. These methodologies are discussed in detail in other chapters.

In regional exploration, geochemistry begins with stream sediment surveys followed by detailed sampling of soil and rock. Hydrogeochemistry, biogeochemistry, and geobotany also should be considered and applied when relevant.

Regional aerial geophysics such as magnetic, radioactive, and electromagnetic methods have proven very useful in helping to delineate major structures and in identifying lithology, overburden, and concealed mineralization. In some specific cases, aerial gravimetric technology has been applied, but not with great success. Together with electrical methods, these offer follow-up support to ground geophysical exploration. Some geologists include remote sensing in the area of geophysical exploration, but it is more correctly categorized as a separate method.

Aerial photography continues to be extremely useful, and the Internet allows the downloading of mono- and polychromatic images of nearly every part of the earth's surface, often free of charge. Older methods of low- and high-flight aerial photography have practically disappeared and now are applied only in very specific cases. High-resolution satellite imagery has become the preferred method of aerial photographic imaging.

Multispectral data is also available to the public from some government agencies that possess special satellites. Rock alteration and zoning features can be detected easily by manipulating multispectral data. The technology is still under investigation by universities, federal and state geological surveys, and earth science research organizations; a few private firms offer commercial services.

In the second phase of an exploration program, analysis of survey data and methodologies are selected and applied. The depth of subsurface exploration depends on the mineral commodity. Only a very few minerals such as diamonds and high-grade gold- and platinum-group metals have sufficient unit value to support deep exploration.

In a subsequent stage, local geological studies collect structural, stratigraphic, lithologic, and petrochemical data. All spatial and temporal relations between different rock types must be defined. Of great significance is the relative time of emplacement of volcanic and/or intrusive rocks in the entire local rock package. Each type of mineral deposit will require specific exploration methods. Porphyry coppers, precious metal epithermal veins, skarns, sediment-hosted deposits, and polymetallics should be studied using different combinations of methodologies.

Post-Mineralization Cover

Normal weathering and erosion, as well as tectonic processes, often make direct observation of mineral deposits difficult; however, weathered materials can be excellent indicators of mineralization. Argillization, oxidation, propylitization, and silicification are major alteration varieties that produce significant color and textural changes in outcroppings. Rock alteration therefore is very important at this stage. Detailed geochemistry and geophysics are widely applied in these cases to investigate post-mineralization cover. Remote sensing helps to detect these surface manifestations of altered rock.

Computer databases, geographic information systems (GISs), and statistical analytical methods are used to organize and study the geological data. Commercial software packages are available and useful for these purposes.

The next step typically is to sample and test the potential target directly by diamond core and reverse circulation drilling. Core samples allow the direct observation of the rock for more-detailed geological, geotechnical, and structural data. Reverse circulation provides a more general idea of lithological variation and permits faster sampling for grade determinations.

The primary problem for the exploration geologist is finding anomalous mineral accumulations in accessible locations with grades that will make their extraction, processing, and marketing profitable. Exploration concessions are registered regularly by the thousands, primarily in developing countries. Most are simple "land play" by junior companies or the hope of an old-time prospector to own a mine. Prospect submittals must be selected carefully and technically reviewed before investing any time and capital in further examining such concessions.

Modern exploration involves an initial reconnaissance visit, followed by a more-intensive exploration program if warranted. Very few ore bodies are discovered today as a result of only one geological visit. The high cost of exploration can be reduced by applying additional methods of scientific evaluation rationally. World metal prices are the primary factor controlling mineral production; however, complex global economic factors such as politics and armed conflict may cause unexpected and unpredictable price fluctuations.

The search for concealed ore deposits is ongoing and conducted in areas hitherto considered prohibitive. These include deeper subsurface ore bodies and those under the oceans. Industrialized countries will continue to be the major consumers of mineral raw materials, while developing countries will continue to be the main providers. Talented exploration geologists will be more in demand, along with the use of

technologies, some of which still need defining, refining, and even discovery.

EXPLORATION GEOCHEMISTRY

Exploration geochemistry (EGCH) is an exploration method based on systematic measurements of the amounts and distribution of chemical elements along with their migration in the earth's crust. The exploration of minerals is based on the premise that elements or traces of geochemical "pathfinders" migrate out from the original ore body by various means to form anomalous areas or dispersion aureoles that can be detected by highly sensitive methods of analysis (see Table 3.2-1 for some of these elements). Further details are discussed in Chapter 3.4.

In general, the larger the anomaly, the larger the source. Also, the intensity of the anomaly is usually proportional to the grade of mineralization; however, the interpretation of a potential ore body depends on the different mobilization capacities of pathfinder elements that occur in various environments.

EGCH has important applications in all stages of mineral exploration. In large areas such as metallogenic provinces, regional low-density stream sediment surveys are initially conducted (one sample every 100 km^2) followed by more-detailed stream sediment sampling intervals (1–3 km/sample as determined by local topography). Soil and rock geochemical sampling are planned and conducted after an approximate perimeter of the anomaly is delineated.

Geochemical data is stored using orthogonal coordinates of sample locations as well as the values of the elements analyzed. Statistical methods are then applied to assist in interpreting the results. Normal (background) and anomalous (above threshold) values within a given geochemical population must be estimated in the initial phase of an investigation. Contrast, which consists of the ratio between a sample and background values, is also commonly used. Profiles of distance versus element value are very useful. Frequency and cumulative frequency distribution plots thus are produced in combination with other basic statistical analyses. Isograd or isopach maps are drafted by contouring selected elements mostly from stream sediment surveys; circles of various sizes represent relative element ranges. Element ratios have proven highly successful in geochemical interpretation. Noneconomic accumulations of metals and human contamination often produce insignificant or spurious geochemical anomalies. The exploration geologist must be alert to these potentialities and filter them out.

Although a relatively young technique, EGCH's economic utility in the discovery of mineralization has a proven track record. EGCH is now universally accepted as the most important and effective exploration method for the discovery of precious metal deposits.

Dispersion Models

A dispersion model illustrates the abundance and distribution as well as the stability of a chemical in the local environment. Element dispersion is the foundation of EGCH. Dispersion halos are of a primary or secondary nature. Primary dispersion halos are directly related to the place of mineralization, and many sometimes occur as a reflection of the deposit's alteration halos, but they can be of a much larger extension. Secondary dispersion halos are formed by later migration of traceable guide elements. In large mineral districts, dispersion halos form "geochemical provinces" with higher than normal background values. The specific applicable methods of EGCH and the associated sampling materials depend on these dispersion halos. These are briefly discussed in the following sections; however, one should always run orientation surveys to determine which group of elements is best to analyze for, along with the types of sampling and sample spacing.

Table 3.2-1 Elements acting as pathfinders of ore deposits

	Ore Deposits												
Pathfinders	Ag	Au	Be	Co	Cu	Li	Mo	Ni	Pb	Sn	U	W	Zn
Ag		x		x	x				x				x
As	x	x		x								x	
Ba	x								x				x
Cd													x
Co	x							x					
Cu		x		x			x	x	x				x
H_2S					x			x	x				x
Mn					x								
Mo					x							x	
Ni						x	x						
Pb				x	x								x
Re							x	x					
Sb	x	x											
Se							x		x		x		x
Sn										x		x	
SO_4					x		x	x	x		x		x
Te	x	x					x						
W							x						
Z n	x	x	x		x	x		x	x				

Primary Dispersion Halos

There are three main types of primary dispersion halos:

1. *Geochemical provinces* are specific zones in which the chemical composition is suspiciously different in one or various elements from adjacent zones or from average crustal values.
2. *Aqueous fluid dispersions* are produced along paths of hydrothermal mineralizing solutions. This results in disseminations on wall rocks and seepage halos along broken ground around the deposit.
3. *Gaseous dispersions* are produced by high-temperature gaseous fluids that invade adjacent ground. These can be detected later in their gaseous forms or as elemental condensates. Mercury, some radioactive gases, and carbon dioxide or sulfur dioxideare used in the study of gaseous halos.

Secondary Dispersion Halos

The physical and chemical properties of the rock-forming minerals and the types of erosion agents control distribution of the disintegrated products of the parent rock. These processes are responsible for forming secondary dispersion halos. The patterns that result are very useful in geochemical exploration and are discussed in the following paragraphs.

Weathering is the process whereby rocks are broken down by the action of chemical, physical, and/or mechanical agents.

The following main products of weathering are important in EGCH:

- Primary residual products are more stable, only partially disintegrated, and help in defining the parent rock.
- Secondary residual products occur close to the source; hydrolysates such as clays and iron oxides may carry tracks of nearby mineralization.
- Soluble products are unstable products that travel considerable distances from their source(s) as dissolved solids (e.g., calcium and magnesium carbonates, sodium and potassium salts, manganese and iron oxides, and other basic metal salts).

Surface waters carry visible to microscopic particles of organic and inorganic compounds with traces of chemical elements incorporated along the way. Element mobility is the main factor controlling this type of dispersion. It is the basis for hydrogeochemical prospecting.

Groundwater involves the same principles as in surface waters. Shallow and deeper water wells and spring waters are used for sampling.

Stream sediments are clastic materials derived from ore deposits and are often transported large distances by fluvial water, then deposited by mechanical or chemical effects, particularly near confluences. Stream sediment geochemistry has been a very effective method in exploration. Anomalous values are followed upstream to discover the source.

Soils retain chemical and mineralogical characteristics of their parent material and often capture elements that travel through them. Most soils have a great capacity for absorption and so they become useful in EGCH. One should recognize whether soils are transported or residual in nature to correctly interpret anomalous patterns and possible sources. In soil sampling, always sample from the same soil horizon; however, a new methodology analyzes soil samples at the same depth regardless of horizon. This topic is examined further in the following section.

Vegetation may indicate soil chemistry because some plants grow only where certain nutrients occur. Others are natural absorbers and accumulators of specific elements. These travel through soils and fractured rocks and are taken up by vegetation. In some cases, plants grow long, deep roots that may reach subsurface dispersion halos and take up some of their chemical components. In other cases, deleterious elements and compounds may result in detectable atrophies or hypertrophies in plants that can lead to discovering mineralization.

Organic accumulations resulting from some organic materials that possess a high ionic exchange capacity can cause them to act as traps for certain cations. This is usually observed in swampy areas near water feeder channels.

Animal actions are less important in EGCH, and only in a few isolated cases have they been helpful. Burrowing animals such as foxes and ants, for instance, or mound builders such as termites may bring subsurface metal-bearing minerals or precious-gem indicators to the surface where they can be sampled and observed easily.

Glacier sediments may be explored similarly to stream sediments. Terminal and lateral moraines may contain mineral fragments from upstream. Also useful are fine clays, which should be collected and analyzed. If an anomaly is detected, any additional geochemical sampling would be difficult because of the glaciated terrain; therefore, indirect geophysical methods can be used to investigate further.

Meltwater actions may carry traces of certain anomalous elements trapped in the pore spaces in rocks. Freezing produces upward flows of some groundwater that could be helpful in locating concealed deposits. This method is only of limited advantage.

Geochemical Analytical Methods

Although geochemical methods are routinely updated and redefined, the exploration geologist must keep three main factors in mind:

1. Sensitivity
2. Precision
3. Cost

All samples from a particular area must be analyzed by the same technique and preferably by the same laboratory for consistency. In some cases, incorrect results have caused additional unnecessary expenses, or worse, unjustified project abandonment. Duplicate sets of samples should be prepared and delivered to the lab, with one blank and one known standard provided for each set of twenty samples. Double-checking the analyses at different laboratories is highly recommended.

EGCH is based on the detection of very small amounts of certain elements, making use of many modern methods of analysis. The most common are the following:

- Regular fire assay remains a preferred method for gold analysis.
- Spectrometric methods are mostly applied for multi-elemental analyses:
  - Atomic absorption, developed in the 1950s, is still widely used in EGCH when a single or small number of elements need to be determined. In atomic absorption the electrons of an element are promoted to higher orbitals for a short time by absorbing a quantity of energy specific to that particular element. This gives the technique its elemental selectivity. The amount of energy is measured and is directly related to the concentration of the element in the prepared sample.
  - Inductively coupled plasma–mass spectrometry (ICP-MS) is the preferred method in modern geochemistry that allows for quick and precise multi-elemental analysis. ICP-MS determines the elemental composition of samples by counting the number of ions at a certain mass of the element.
  - Inductively coupled plasma–optical emission spectrometry (ICP-OES), a complement to the mass spectrometer, is also used in detecting lighter elements of the periodic table.
- Mobile Metal Ion is the low-level chemical analysis of soils by use of special extractant solutions and later concentration determination by ICP-MS. Soil samples are taken from equal depths regardless of soil horizon. Initially applied by the Geochemistry Research Centre of Australia, this method is now widely used.
- X-ray fluorescence handheld analyzers allow for fairly accurate detection of low elemental concentrations of most types of solid geochemical samples in the field.
- The colorimetric method was one of the pioneering methods of analysis several years ago. It was replaced

Table 3.2-2 Main applications of geophysical methods of exploration

Method	Application Media	Main Exploration Applications
Gravimetric	Ground, marine	Heavy minerals deposits, iron ores, pyrite, chalcopyrite, chromite, salt domes, intrusive vs. volcanic or sedimentary rocks, structural mapping
Magnetic	Ground, marine, airborne, drill-hole logging	Iron ores, magnetite, pyrrhotite, black sands, kimberlites, chromite ores, mafic intrusives, basement irregularities, geological and structural mapping
Electrical: Resistivity	Ground, marine, drill-hole logging	Sulfide deposits, conductive vs. resistive rocks, massive sulfides, base metals, graphite, quartz and calcite veins, salt domes, coal beds, underground conductive fluids, tectonics
Electrical: Induced polarization	Ground, drill-hole logging	Nonconductive polarizable mineralization; disseminated and massive sulfides; porphyry copper and gold, and silver deposits; tin; zinc; stockworks
Electrical: Self-potential	Ground, drill-hole logging	Sulfide ores; tin, cobalt, nickel, gold and silver deposits; massive sulfides
Electromagnetic: Coil sensors	Ground, airborne, drill-hole logging	Sulfides and oxides, magnetite, graphite, base metals, kimberlites, shear zones, geological and hydrological mapping
Electromagnetic: Superconductive quantum interference device (SQUID) sensors	Ground, airborne, drill-hole logging	Deep conductive deposits, massive sulfides
Radioactive	Ground, airborne, drill-hole logging	Radioactive minerals: uranium, thorium, potassium; coal; phosphates; monazite; structural mapping; differentiation of intrusives
Seismic	Ground, marine	Coal, uranium, heavy minerals, buried placer deposits, sand and gravel deposits, fractured rocks, lithological changes
Remote sensing	Airborne, satellite	Geological and structural mapping, tectonics, alteration

by more-accurate methods; however, it is still applied in remote areas where on-site results are required quickly.

Data Interpretation

Statistical analysis is the best method to assist the exploration geologist in interpreting large amounts of geochemical data. Numerous commercial software packages exist, ranging from simple statistical calculations (e.g., the mean and standard deviation, and normal and log-normal frequency distributions) to more-advanced geostatistical analysis including Kriging, cluster analysis, discriminant analysis, factor analysis, correlation, and multiple regression analysis.

All data are entered in digital databases and later integrated into GISs and spatially related through the Global Positioning System. This allows for compiling various layers of information to facilitate interpretation. Many software programs, either canned or proprietary, are used for plotting different kind of maps, profiles, and block models.

GEOPHYSICAL METHODS OF EXPLORATION

Geophysical exploration is based on measuring the contrast between natural and induced physical properties of materials. Geophysical surveys begin with airborne reconnaissance methods to outline broad geologic features. Radiometric, magnetic, electromagnetic, and, more recently, gravimetric methods have been applied successfully in airborne surveys. The exploration geologist continues to use more-detailed geophysical methods that provide more-detailed information, including downhole measurements, searching directly for indications of concealed mineralization.

Geophysical exploration methods have gained considerable popularity in the last few years. More precise and easy-to-use instruments have been developed. Modern surveying techniques and software aid in interpreting results faster and with more reliability than previously.

Table 3.2-2 shows the main geophysical methods used in the exploration of various minerals. Survey design and the nature of the data are dependent on the characteristics of the detecting instruments. The geophysical detection of mineral deposits requires that they possess physical properties different than those of their host rocks.

Geophysical Methods

Highly sensitive gravimeters can detect anomalies caused by differences between the local measured gravitational acceleration and the regional expected value for a given point on the earth's surface. Multiple corrections (for geographical and topographical effects) to the field data, however, must be calculated before a final interpretation can be made. Gravity data is analyzed using digital techniques to obtain regional anomalies, first and second derivatives, residual anomalies, and horizontal gradients.

Magnetic Surveys

Magnetic surveys are based on the earth's magnetic field. Similar to gravimetric surveys, magnetic surveys show anomalies caused by differences between values obtained locally from survey readings and that calculated for the normal intensity of the earth's magnetic field at any given point. Aeromagnetic surveys have proven very useful in regional reconnaissance exploration. The application of remote magnetometry has also been used on research ships in oceanographic studies, and, even more remotely, from artificial satellites.

Modern magnetometers are highly sensitive, can produce results in a short amount of time, and are relatively inexpensive to operate. In addition to the detection of ferromagnetic mineral deposits (mainly magnetite and pyrrhotite), remote magnetometry helps in regional mapping of lithological, structural, and tectonic features.

Downhole magnetic logging has been widely used when magnetic minerals are present in detectable amounts in the survey area. Magnetometric methods can also help define concealed formation contacts, unconformities, major faults, and thicknesses of sedimentary cover.

Electrical Methods

Electrical methods operate by measuring natural or induced electrical fields. Although there are many varieties of electrical methods, resistivity, induced polarization, and self-potential are most commonly applied in mineral exploration. Electrical methods may be used to define structural and tectonic features, but they are more useful in the search for metallic conductors associated with ore bodies. Downhole logging is used widely and successfully to define ore bodies. In the proper geological setting, surface data should be employed in conjunction with downhole survey data.

Resistivity is controlled by porosity and fluid content of the rock and the presence of conductive minerals; therefore, different rock types will have different natural resistivities. The depth of penetration and investigation is dependent on the energy injected into the ground by the power source, and by the spacing and array of electrodes. Resistivity has been used broadly in detecting all kinds of mineral conductors and in groundwater investigations.

Induced polarization is often applied in parallel with regular resistivity surveys, making use of the same electrode array or with minor modification. It is based on the detection of electrochemical activity and/or the polarization at a mineral interface brought about by the application of an electric or magnetic field.

Induced polarization is very effective in detecting disseminated mineralization such as the sulfide zone in porphyry coppers or disseminated ore minerals in precious metal deposits.

Self-potential is used when the effect of a "natural battery" is produced, such as occurs by the flow of groundwater and, in the case of minerals exploration, by the interface of a sulfide ore body and its oxide zone. These occurrences are related to outcropping ore bodies or those that have been oxidized corresponding to phreatic zone fluctuations.

Electromagnetic Methods

Several coil arrays are used in electromagnetic methods to measure a combination of induced long-wave electrical fields and the earth's natural electromagnetic fields. An electromagnetic field is generated by a transmitter creating a secondary magnetic field, which is registered together with the primary magnetic field. Subsequent reductions and corrections are calculated so that a final interpretation is done in conjunction with geological and geochemical data.

A superconducting quantum interference device (SQUID) uses an extremely sensitive magnetometer that registers very low magnetic fields. Sometimes it is used in mineral exploration together with coil sensors to detect deeper conductors and to isolate the signal from that coming from overlying conductive cover. Many successful applications have occurred in exploring for deep massive sulfide deposits.

Radioactive Methods

Radioactive methods measure natural or artificial radioactivity. Uranium and thorium are the main sources of radioactive emissions. Regular radioactivity detectors can only reach shallow depths of no more than a few meters; however, emanometry, which detects radon emissions, can "see" deeper into the subsurface and is sometimes used together with regular radioactivity detectors.

Airborne gamma-ray surveys have wide applications exploring over large areas and are useful in geological mapping, taking advantage of minor radioactive emissions from naturally occurring rock-forming minerals.

Decaying radioactive elements produce gamma rays, which are easily detected by modern gamma-ray spectrometers. Potassium, uranium, and thorium are common radioactive trace elements that occur in many rock-forming minerals. As they decay, they provide natural sources of gamma rays that are detected in these surveys.

Seismic Surveys

With limited applications in mineral exploration, seismic methods measure changes in the velocity of shock waves produced by explosive charges or mechanical vibrators (even those from sledgehammer impacts). Reflection and refraction waves are recorded and used to define formational and structural contacts in the subsurface. Seismic surveys are relatively expensive but can be applied directly on the earth's surface or from sea vessels. They are more widely used in petroleum exploration.

Remote Sensing

Infrared sensors register changes of temperature, which have been used successfully in environmental studies and in delineating groundwater zones and exothermic zones of mineral alteration. High-resolution satellite imagery and imaging spectroscopy are gradually displacing traditional aerial photographic methods.

Imaging spectrometry uses multispectral detecting devices. The National Aeronautics and Space Administration's Airborne Visible/Infrared Imaging Spectrometer (AVIRIS) is among the most advanced in use with 224 channels. Other sensors are listed as follows:

- HyperCam infrared hyperspectral imaging camera (ground based or taken by aircraft)
- Moderate Resolution Imaging Spectroradiometer (MODIS; aboard the Terra EOS AM-1 satellite)
- Medium Resolution Imaging Spectrometer (MERIS; aboard the European Space Agency's Envisat satellite)
- Hyperspectral Digital Imagery Collection Experiment (HyDICE) spectrometer
- Hyperion sensor (aboard Earth Observing-1 satellite)

The large multichannel feature gives the sensors the capability to observe atmospheric and planetary phenomena in environmental studies and mineral exploration. Absorption of spectral energy caused by chemical bonds allows the detection of different types of vegetation and many synthetic materials, minerals, and mineral assemblages. Interpretation is not simple and requires advanced training and experience along with the use of sophisticated software.

Costs are relatively high and depend on the size and characteristics of the area to be covered. The popularity of imaging spectrometry is certain to increase in the future.

ORE DEPOSITS

The exploration geologist must have an idea of the type of ore deposit being sought prior to starting a mineral exploration program. In regional studies, the general geologic setting provides the first clues. The proximity to any mineral district or predefined metallogenic province also helps to anticipate the type of mineralization.

Ore Deposit Models and Classification

Most modern authors refer to the initial, and still popular, classification of ore deposits proposed by Lindgren in 1933, or that proposed by Schneiderhohn (and later modified by Niggli in 1929). Many other classifications have been proposed but these still prevail.

Modern studies have established geological and geophysical ore-deposit models whose parameters provide important guides for mineral exploration. Each deposit, however, should be studied or explored on its own merit.

The most widely accepted classification schemes are based on genetic characteristics. Five groups are defined: magmatic, hydrothermal, sedimentary, metamorphic, and surface accumulations. Morphological classifications, such as vein type, disseminated, massive, or strata-bound are normally subdivisions of these. Each group is subdivided into classes, families, or types, and very often there is some overlapping between two or more groups. Some ore-body classifications are based on mineralogical associations. Although there may be some economic or genetic implications, the result has limited relevance in exploration. Greater significance is placed on classifications based directly on geological setting and ore-formation processes.

In all cases, it is very useful to model the deposit under exploration after a similar well-known or previously studied deposit. The geologist should be prepared to confirm or change the model as more data become available.

When evaluating an exploration prospect (or any mining property), the explorationist must adhere to terms and definitions for resources and reserves, which are now required and regulated internationally. The most widely accepted standards are those in National Instrument NI-43-101 (2005), proposed by the Canadian Institute of Mining, which is similar to the Joint Ore Reserves Committee (JORC) Code from Australia (AusIMM 2004). Both are accepted in international mining stock exchanges. The two documents also define a "qualified person" who is required to sign-off on any technical reports.

Zoning

Mineralogical and geochemical zoning constitute valuable guides in mineral exploration. They reveal subtle clues of certain indicator chemical elements or accumulations of ore minerals. Zoning occurs in regional, district, or ore-body scales. Regional zoning is related to metallogenic provinces; district zoning is observed in areas with several mines; ore-body zoning is particular to an identified deposit type and is more relevant in mineral exploration.

Ore-body zonal arrangements are manifest in three dimensions. Their proper identification is especially helpful in lateral or vertical mine planning, in the selection of new exploration targets, and in decisions to terminate drilling and mine-development programs, or both.

Generalized zoning patterns have been recognized for most types of ore deposits; however, each case is different. Most ore-body zonation is characterized by several pulses of mineralization, which often cause patterns of overlapping zones. These must be identified, registered, and properly mapped as soon as the information becomes available. It is not uncommon to follow two or more zoning patterns in mine production. Figure 3.2-1 shows an idealized zoning pattern for a hydrothermal gold–quartz vein deposit.

Zoning is often related to paragenetic sequences and therefore is greatly influenced by changes in temperature and pressure, and by the composition and stability of the mineralizing fluids. The concepts are well known and cited abundantly in the technical literature. Among the subtle zoning guides mentioned previously are geochemical indicators. Geochemical zoning reflects the elements present in mineralizing solutions, and these depend on their relative mobility in and through the rock environment.

Wall-Rock Alteration

Wall-rock alteration is related to the action of hydrothermal fluids in epigenetic and high-temperature remobilization processes, which are observed most often in felsic rocks. It may also be caused by some metamorphic or diagenetic processes. Weathering of outcropping deposits may cause "strong rocks alteration" as well, due to the oxidation of iron-rich minerals and some argillization. All alteration processes are the result of chemical exchange and mineralogical modification.

Alteration halos are typically present in zonal arrays emanating from mineralized circulation channels outward to fresh country rock. They often extend outward several times the perimeter of the ore deposit, making them easier to detect. However, they may also be of very limited dimensions, depending on the reactivity and permeability of the rock, and the amount of circulating fluid.

The most common alteration assemblages are as follows:

- Potassic involves the introduction of K-spar and other potassium minerals.
- Sericitic, also referred to as phyllic, consists of the presence of micaceous potassic silicates derived from primary felsic minerals.
- Argillic is represented by kaolinite and montmorillonite after plagioclases and amphiboles. Advanced argillic is a variation that results from strong acid-leaching.
- Silicic involves the abnormal injection of siliceous solutions (and the almost universal presence of free quartz).
- Propylitic is produced by low temperature–pressure formation of epidote, chlorite, and calcite.
- Fe oxidation, although not properly a result of hydrothermal alteration, gives rise to the common decay or alteration of Fe-containing minerals. These are highly visible and widely used in minerals exploration. This phenomenon has developed into the study of leached and oxidation caps (or gossans), which, in their diverse assemblages, provide excellent clues to the presence of ore deposits and their sources.

All these alteration processes produce color changes in the original rocks. White to bright reddish and yellow are common, passing through shades of green and pink. They all constitute primary guides for the explorationist.

Fluid Inclusion Studies

In idealized deposition models for many ore deposits, temperature and chemical zoning becomes useful in understanding mineralization and its origin. Identifying these features can be accomplished through the study of fluid inclusions, particularly during the drilling of a prospect, or when the upper portions of an ore body have been eroded away, or when structural or tectonic movements have modified the original ore-body's position.

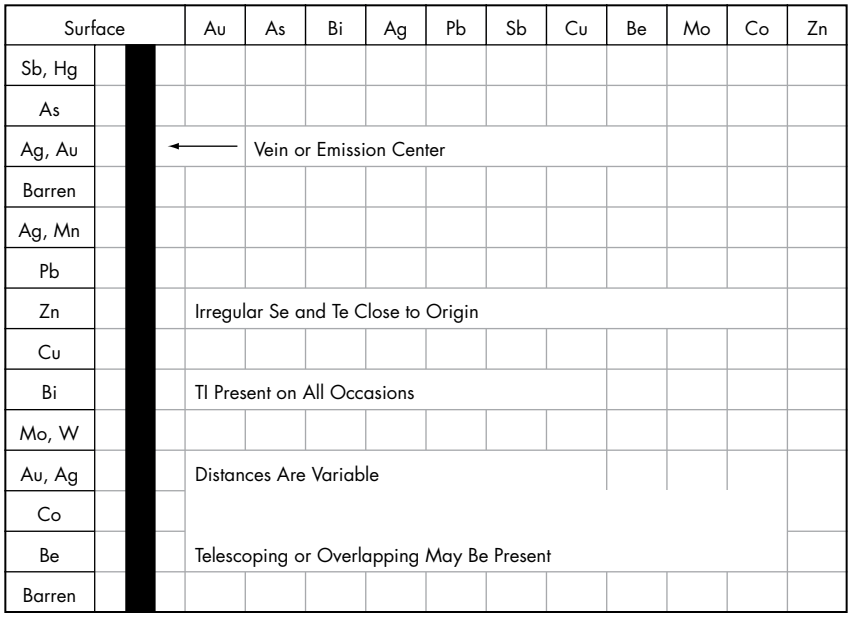


Figure 3.2-1 Idealized vertical and lateral zoning in gold–quartz veins

Fluid inclusion studies are based on the collection of transparent mineral crystals that have trapped gases and/or liquids in vacuoles at the time of deposition. The vacuoles are observed under the microscope and then subjected to temperature manipulation by special heating–cooling stages to determine the temperature of homogenization. Their chemical compositions can be defined by one of many analytical techniques, such as laser spectrometry or electron microprobe.

Isotopic Studies

Some isotopes have been used to study the sources and compositions of mineralizing fluids in many ore deposits and to determine their geochronology. Isotopes of hydrogen, carbon, oxygen, sulfur, strontium, and lead are most commonly studied. Distinctions between barren and productive intrusive and volcanic packages have been made successfully by this method.

Isotopic studies are performed mostly as academic research because of a lack of knowledge and experience of the technique in the mining industry; however, its importance in mineral exploration is becoming more relevant as more experimental data (and its interpretation) are published.

Geometallurgy

Geometallurgy is a relatively recent technique in mineral exploration. It is based on precise quantitative mineralogical and chemical measurements using electron microprobes or similar instruments, followed by statistical analyses to determine variabilities in physical, mineralogical, and geochemical characteristics of a mineral prospect. The information obtained is used in all stages of exploration, as well as in the development of an ore deposit, reserve and resource evaluation, and metallurgical processing.

Geometallurgy may be applied to stream sediment samples, mineralized outcrops, drill cores, or even in productive mine stopes. Ore-deposit models can be proposed, and strategies for further exploration may then be designed. It is also very useful in ore-body modeling, predicting possible extensions and determining the quality of mineral accumulations.

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CHAPTER 3.3

Geophysics Prospecting

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INTRODUCTION

Geophysics prospecting is defined as "making and interpreting measurements of physical properties to determine subsurface conditions, usually with an economic objective, for example, discovery of fuel or mineral deposits." Measurements include seismic travel times and wave shape changes, electrical potential differences, magnetic and gravitational field strengths, and radiometrics (Sheriff 2002).

Geophysicists have developed methods for estimating physical properties from surface measurements. It is a difficult problem to solve because there is often much ambiguity in the solution. This problem is overcome to some extent by constraining the problem with factual information from surface geological mapping, measured physical properties, or drill-hole information. Therefore, it is critical to understand the ranges, or errors, in data acquisition—those introduced through processing and those introduced through the assumptions made while interpreting the data. The "answer" is never exact; nevertheless, some geophysical tools have major technical and financial impacts on mine planning, often with significant influence on the costs of development.

OUTLINE OF METHODS AND TECHNIQUES

Applied geophysical methods are classified according to the transmitting energy source (Table 3.3-1). Detection of the resulting signals is achieved by contact with the ground or remote-sensing platforms. In near-mine or on-mine exploration, this is usually done through physical contact with the ground. Greenfield exploration projects mostly tend to use airborne methods.

Natural sources include the following:

- Potential fields such as magnetic and gravity fields
- Electromagnetic energy due to sun flares and sunspots
- Radiometric methods based on the radioactive emissions from thorium, potassium, and uranium
- Seismic energy from earthquakes and tremors, as well as rockfalls (passive seismic)

Active (human-made) sources include the following:

- Grounded current electrodes and inductive loops powered by batteries and/or generators
- Seismic energy from surface excitation (hammers, vehicles, dynamite, vibrators, core drilling, etc.)

The application of a geophysical technique in the mining environment is usually constrained by the mining infrastructure. Geophysics is usually most effective in mining at the prefeasibility stage. For instance, conducting a three-dimensional (3-D) seismic survey at the prefeasibility stage establishes a base case for future mine planning. To undertake such a survey after the mine has been developed is disruptive to mining and production schedules, and it increases the cost while producing inferior signal quality due to infrastructure "noise."

NATURAL SOURCES

The earth's magnetic, gravitational, and natural electromagnetic fields are included in the potential-fields category. Magnetic and gravitational fields are important contributors to geophysical prospecting. Accuracy is a function of position (vertical and horizontal) and the sensitivity of the instrument. The application of potential fields in prospecting is summarized in Table 3.3-2 by exploration objective.

Table 3.3-2 can assist the engineer in selecting the most appropriate geophysical tools for an exploration objective. The "output and products required" column is intended as a guideline for discussion with the contracting geophysicist when writing the scope of the work. (All costs in this chapter are in U.S. dollars.)

POTENTIAL FIELDS—MAGNETICS

Magnetic surveys should be designed for the target objective (see Table 3.3-2). For example, for targets >100 m deep, airborne systems are the most cost-effective way to collect the data. Shallow target objectives require high spatial resolution surveys (<100 m target objective) on the ground with relatively high sample density per unit area.

Two sets of internationally accepted units can be used to denote the magnetic field (H). The International System of Units is the tesla (T), and the centimeter-gram-second (CGS) unit is the oersted (Oe). One nanotesla (nT) equals 10^{-5} Oe. The earth's magnetic field is about 0.1 Oe.

J.A.J. (Barry) de Wet, Director and Consulting Geophysicist, BDW Geophysics Consulting, Brisbane, Queensland, Australia

Table 3.3-1 Geophysical methods by energy source

Source	Method	Detection	Configuration	Applications		
Natural Sources						
Earth tremors and rockfalls	Seismic	Seismic earthquake detection	Mantle seismic tomography	Earthquake locations and predictions	Diamond and nickel exploration	Crustal studies
	Piezoelectric	Seismoelectric	—	—	—	—
	Passive seismic	Micro-seismic tomography	Surface and underground arrays	Rock mechanics	Rockfall predictions	—
Atmosphere (lightning), 1 to 10 kHz	Audio frequency magnetotelluric (AFMAG) ground and helicopter	Electromagnetic spectrum	Induction coils	Minerals exploration	—	—
Hydromagnetic (magnetosphere), <1 kHz	Plane-wave electromagnetic	Electromagnetic and electrical field	Induction coils and grounded potential electrodes	Deep crustal studies	Diamond and nickel exploration	—
Potential fields	Magnetic	Total field magnetic field, vector, and gradient tensor measurements	Ground, airborne, satellite, and downhole logging	Downhole three-component magnetics	Downhole susceptibility	—
	Gravity	Vertical field and gravity gradiometry	Ground, airborne, and downhole gravity logging	Engineering, foundations, sinkholes, and excess mass	Exploration, groundwater, and sulfide minerals	—
Nuclear radiation	Radiometric	Counts per second from radiation in sodium iodide crystals	Ground, airborne, and downhole logging	Detection of nuclear waste products in mining	Minerals exploration	—
Active Sources						
Electromagnetic induction	Horizontal, vertical, and in-loop	Time domain B and dB/dT; multifrequency amplitude and phase measurements	Ground, fixed wing, and helicopter	Groundwater and agriculture salinity mapping	Exploration for conductive sulfides	—
Electromagnetic grounded wire	Controlled-source audio magneto-tellurics (CSAMT)	Multifrequency magnetic and electrical field	Far-field grounded wire	Minerals exploration for resistive and/or conductive ore	—	—
Direct current grounded wire	Resistivity sounding; induced polarization	Current and electrical potential	Expanding or profiling arrays	Groundwater and agriculture salinity mapping	Sulfide mineral exploration	Conductivity mapping for geological structure
Radio waves	Radio imaging method (RIM), ~520 MHz and ground-penetrating radar, ~16 to 1,600 MHz	Various antenna arrays	Ground surveys and in-mine surveys	Shallow structure (0–30 m); civil engineering problems	Exploration for laterite ores	—
Seismic	Dynamite, vibrator trucks, hammer, and weight drops	P and S waves, either three-component or single vertical component geophones	2- or 3-D surveys; reflection, refraction, and vertical seismic profiles	Oil exploration and production	Coal exploration and mine planning	Witwatersrand gold and Bushveld platinum exploration

The earth's magnetic field originates within the earth and acts like a giant bar magnet located near the center of the earth's core. Buffeting of the earth's magnetosphere by solar wind is the primary cause for the diurnal field. Field reversals (i.e., the North Pole and the South Pole swap positions) have been common throughout geologic history. A magnetic pole is located where the inclination of the magnetic field is vertical. At the North Pole the field enters the earth and at the South Pole the field leaves the earth (by convention). At the magnetic equator the inclination of the field is horizontal.

Susceptibility, k, is the ratio of the magnetic moment per unit volume, M, to the magnetic field strength, H, or $k = M/H$ (the quantity has no units). Values are positive for paramagnetic materials and negative for diamagnetic materials. This is the fundamental property exploited by the magnetic method. This can be rewritten as $M = k \times H$. Normally the earth's magnetic field magnetizes earth materials proportional to the magnetic susceptibility of the material.

Types of Magnetism

All matter reacts to a magnetic field, but only two classes of matter actually exhibit strong interactions: ferromagnetic and ferrimagnetic (Clark et al. 2003).

1. Ferromagnetic refers to materials that one normally thinks of as magnetic, such as Fe, Ni, Co (such as primary ore for iron), nickel sulfides, and volcanogenic sulfides. They exhibit parallel alignment of moments resulting in large net magnetization even in the absence of a magnetic field.

Table 3.3-2 Interpretation of potential fields by objective

Objective	Methods	Output and Products Required	Cost (U.S. dollars)	Improvement If Multiproperty Survey (e.g., magnetics + gravity)
Dykes, intrusives, extrusives (basalt), and faults	Magnetic survey, with gravity optional	• Total field, reduction to the pole, analytical signal, and first and second derivative contour/gridded maps. • Profile modeling across dykes and faults to determine depth, width, and strike	• Airborne: $20/km • Ground: $120/km	• Adding gravity helps to detect nonmagnetic dykes and improves fault detection when faults are not filled with magnetic material.
Sedimentary basin/ thickness mapping	Magnetic survey, with gravity and/or electromagnetic (EM) optional	**Magnetics:** • Total field grid and contour • Depth to magnetic basement grid and contour • Vector magnetic amplitude 3-D inversion, depth to magnetic basement (no remanence)	• Airborne: $20/km • Ground: $120/km	• Remanent magnetic effects affect positional accuracy and body geometry. Depths are usually acceptable as long as the basement is more magnetic than sediments.
		Gravity: • Joint inversion using vector magnetic and gravity 3-D inversion • Euler deconvolution produces depth to basement on both physical properties • Profile interpretations to check depths, width, and strike of basement features	• Airborne gravity and magnetic on single platform: $160/km	• Two physical properties better resolve basement depths. Both suffer from lack of horizontal layer resolution.
		Electromagnetics: • Magnetics products as above. EM products include conductivity depth profiles, conductivity and conductance bullet. • Layer thickness maps • Structural maps (lineaments) • Depth to resistive or conductive layers	• Helicopter magnetic and EM on one platform: $120/km	• Mapping capability improves substantially by resolving layers with high conductivity contrast to depths of <100 m. • Reflection seismic is best for resolving layers but expensive (see later).
Bauxite thickness profile	Ground-penetrating radar (GPR) and ground magnetics	• Profile images of conductivity that look like seismic sections • Depth to unweathered rock • Often saprolite thickness contours if lines are dense enough • Analytical signal of magnetics to map intrusives and dykes	• Ground magnetics as above • GPR: $3,000/d	• Adding seismic refraction helps to verify depth to unweathered rock. • Cost benefit is questionable.
Nickel laterite	GPR and ground magnetics	• Profile images of conductivity that look like seismic sections • Depth to unweathered rock • Often saprolite thickness contours if lines are dense enough • Analytical signal of magnetics to map intrusives and dykes	• Ground magnetics as above • GPR: $3,000/d	• Adding seismic refraction helps to verify depth to unweathered rock. • Cost benefit is questionable.
Nickel sulfide	Combination of magnetic and EM methods is most effective. Gravity must be used when ultramafic is altered by carbonate fluids (no magnetite development).	• Detailed magnetic surveys required • Maps of total field, analytical signal, and first and second derivative maps • Reduction to the pole useful • Profile interpretations to estimate depth to sulfides; but if uncorrected for remanent magnetization, large errors are possible • EM using in-loop and large-loop methods • B-field sensors preferable	• Airborne magnetics: $20/km • Ground magnetics: $150/km • Ground EM: $2,000/d	• Downhole EM is mandatory for all holes drilled. • It is essential for detecting off-hole anomalies.
Iron oxide copper-gold (IOCG)	Magnetic and gravity and/ or induced polarization optional	• IOCG source in hematite alteration produce high-density anomalies without magnetic anomalies. • Magnetic anomalies are often associated with sulfides (e.g., Cannington deposit, Australia). • Joint 3-D inversions are essential for spatial distribution of anomalies.	• Airborne gravity and magnetic on single platform: $800/km^2	• Induced polarization helps to map sulfides but is often misleading.
Broken Hill type silver, lead, and zinc	Magnetic and gravity and/ or induced polarization optional	• BHT source in hematite alteration produce high-density anomalies and magnetic anomalies. • Magnetic anomalies are sometimes associated with sulfides (e.g., Cannington deposit, Australia). • Joint 3-D inversions are essential for spatial distribution of anomalies.	• Airborne gravity and magnetic on single platform: $160/km	• Induced polarization helps to map sulfides but is often misleading.

(continues)

Table 3.3-2 Interpretation of potential fields by objective (continued)

Objective	Methods	Output and Products Required	Cost (U.S. dollars)	Improvement If Multiproperty Survey (e.g., magnetics + gravity)
Iron ore	Magnetic and gravity surveys	• Hematite alterations produce high-density anomalies and magnetic anomalies. • Magnetic anomalies are associated with banded ironstone. • Joint 3-D inversions are essential for spatial distribution of hematite iron bodies.	• Airborne gravity and magnetic on single platform: $160/km	—
Porphyry copper	Magnetic surveys followed by regional induced polarization (IP). IP anomalies are followed up with IP profiles to discriminate between shallow clay anomalies and sulfide alteration in porphyry copper alteration.	• Porphyry copper deposits have characteristic magnetic halos and are often associated with major lineaments. • Maps of analytical signal, total field, and derivatives are required to map structure.	• Airborne: $120/km	• IP costs are $3,000/d.
Kimberlite	Combination of magnetic and EM methods are most effective. Gravity must be used when Group 2 kimberlites are dominant in an area.	• Maps need to show coincident magnetic high and conductive anomalies. • Gravity lows are characteristic of weathered kimberlite.	• Helicopter magnetic and EM on one platform: $120/km	• The addition of gravity, either on an airborne platform or on the ground, confirms kimberlite geometry and reduces risk of failure substantially.

2. Ferrimagnetic materials exhibit a more complex alignment of atomic moments, but they exhibit the same general magnetic character as ferromagnetic material (magnetite, maghemite, pyrrhotite mostly in ore bodies derived from alteration processes such as iron oxide copper-gold [IOCG] deposits and Ni laterites).

The most common magnetic material within the earth's crust is magnetite (Fe_3O_4), which is mostly found in iron ores, banded iron formations, metamorphic and igneous rocks, and in small concentrations in some sedimentary rocks. Magnetite is often associated with metallic sulfide ores such as pyrrhotite ($Fe_{1-x}S$). Ilmenite ($FeTiO_3$) is the next most common magnetic mineral (but to a much lesser extent than magnetite).

The following list describes the types of magnetization:

- **Remanent magnetization:** Remanent magnetization is the magnetization remaining in the absence of an induced magnetic field, such as in a common iron bar magnet. It is important because interpretations of magnetic data are highly complicated by remanent magnetization, and this can result in serious and costly errors if disregarded.
- **Normal remanent magnetization (NRM):** NRM is the residual magnetization retained by rock and other material in situ; unless otherwise qualified, this is the implied meaning.
- **Thermo-remanent magnetization (TRM):** TRM remains after a sample has been cooled to a temperature below the Curie point in a magnetic field.
- **Chemical remanent magnetization (CRM):** CRM is acquired when a magnetic substance is chemically formed or crystallized in a magnetic field at a temperature below the Curie point.
- **Depositional or detritus remanent magnetization (DRM):** DRM is acquired in sediments when magnetic mineral particles are preferentially aligned by the ambient magnetic field during deposition, usually by settling through water.
- **Isothermal remanent magnetization (IRM):** IRM is remanent magnetization in the ordinary sense; that is, it is the magnetization after application and subsequent removal of a magnetic field. IRM is not involved in palaeomagnetism, except for the effect of lightning currents in exposed surface rocks.
- **Pressure or piezo-remanent magnetization (PRM):** PRM is remanent magnetization acquired under the application of stress; the effects generally become more pronounced as the strain proceeds from elastic to plastic deformation.

The magnetic properties in rocks are affected by many geological and physical processes that all contribute to the complexity of the final magnetic vector direction, and this is the reason why the magnetic surveys are so useful. Magnetic surveys are useful for mapping geological structure, lineaments, and intrusive volcanic rocks. From the earlier explanation of remanence in rocks, the practice of assuming a uniform induced magnetic direction (inclination and declination of the induced field) is simplistic and an unreliable approach. Recent developments in 3-D inversion will produce block models of the vector magnetic components that are useful in discriminating different lithologies. Recent work shows that it may discriminate alteration zones in mineralized sequences.

Magnetometers

A magnetometer is an instrument that measures magnetic field strength. The most common magnetometer used today is the proton precession magnetometer, also known as a proton magnetometer. It measures the resonance frequency of protons (hydrogen nuclei) in the magnetic field due to nuclear magnetic resonance. Because the precession frequency depends only on atomic constants and the strength of the ambient magnetic field, the accuracy of this type of magnetometer is very good.

A direct current flowing in an inductor creates a strong magnetic field around a hydrogen-rich fluid, causing some of the protons to align themselves with that field. The current is then interrupted, and as protons realign themselves with ambient magnetic field, they precess at a frequency that is directly

Table 3.3-3 Survey types and intended uses

Survey Type	Intended Use	Flight Line Spacing
Total field magnetics	For regional surveys	Line spacing about 2× flight height
Total field magnetics	For structural interpretation using gradients	Flight height equal or less than line spacing
Gradient and full tensor	For detailed structure and unexploded ordinance	Flight height about half the line spacing

proportional to the magnetic field. This produces a weak alternating magnetic field that is picked up by an inductor, amplified electronically, and fed to a digital frequency counter whose output is typically scaled and displayed directly as field strength or output as digital data.

The relationship between the frequency of the induced current and the strength of the magnetic field is called the proton gyromagnetic ratio and is equal to 0.042576 Hz/nT (hertz per nanotesla).

Cesium vapor magnetometers are commonly used in airborne systems. Optically pumped cesium vapor magnetometers are highly sensitive (0.004 nT/√Hz) and are well suited to higher-resolution surveys.

Superconducting quantum interference devices (SQUIDs) measure extremely small magnetic fields; they are very sensitive vector magnetometers, with noise levels as low as 3 $fT \cdot Hz^{-0.5}$ in commercial instruments and 0.4 $fT \cdot Hz^{-0.5}$ in experimental devices (fT stands for femtotesla, 1 $fT = 10^{-15}$ tesla).

Magnetic Surveys and Survey Design

Magnetic data are obtained using a magnetometer transported over land, water, or in the air—referred to as ground, marine, and airborne surveys, respectively. Normally, a stationary reference magnetometer (base magnetometer) measures the diurnal magnetic field during data acquisition. A Global Positioning System (GPS) is used on most airborne and marine surveys and is standard practice in ground surveys. Data are normally obtained along parallel lines perpendicular to geological strike. Tie-line data are collected at right angles to survey lines but at a much wider spacing (usually 10×). Tie lines help correct for magnetic instrument drift and diurnal variation in the earth's magnetic field. Data collected along a line is referred to as a magnetic profile.

The resolution of a survey is dependent on the sensor height, and line and station spacing. Table 3.3-3 can help prevent oversampling of the field (and therefore unnecessary acquisition costs), depending on the depth of the target.

Table 3.3-3 summarizes the maximum line spacing used in various types of surveys to avoid misleading results. If the target is 100 m below the magnetic sensor (h), it makes little sense to collect data at a line spacing (Δx) of less than 200 m to produce a useful contour map. However, if one needs to resolve lineaments and dykes, then one should fly 100-m flight line spacings so that gradient maps can be calculated. The maximum height of the sensor should be 50 m if single anomalies need to be modeled at a 100-m line spacing.

Line bearings should parallel magnetic north-south when the geological strike varies considerably; otherwise one should choose lines at right angles to geological strike. Common practice is to fly tie lines at right angles with intervals approximately 10× the flight line spacing. With a modern GPS or flight-line photographic tracking is no longer required for positioning and drift leveling of the magnetic map compilation. Survey costs for airborne surveys vary from $10 to $120/km, depending on the access and remoteness; the wide variance is due to a number of factors, which will be discussed later.

Guidelines for Scope of Work for Magnetic Surveys

General specifications for a scope of work should include the following.

- **Project scope:** The area outline and number of line kilometers to be surveyed needs to be clear.
- **Time:** The time period of the contract needs to be specified.
- **Authorities:** In some countries, there is a no-flyover policy over areas licensed to another explorer's properties. Permission is required to traverse across such tenements. One may be required to access private property to map geological structures effectively, so permission must be sought.
- **Deliveries:** It is important to ensure that the contractor has performance criteria for the project, including health and safety, environment, and cultural standards, and criteria for work schedules and the delivery of data and reports.
- **Legalities:** Property access and aboriginal rights are the primary legal concerns.
- **Payment:** The basis of payment needs to be specified.
- **Excluded costs:** The contractor should indicate if accommodation, transport, and fuel access are excluded and should indicate whether goods and services taxes are included or excluded.
- **Missing or substandard data:** Noise specifications should be realistic. The issue of "drop-outs" (i.e., when the magnetometer is not properly orientated and data are lost) needs to be addressed, and an agreement on possible reflights is required.
- **Contract price:** A price quote needs to be submitted.

Survey equipment specifications should include the following:

- Instrument or survey type
- Minimum data accuracy and frequency of measurement; data acquisition system
- Minimum navigational accuracy and navigation systems
- Ground monitoring requirements
- Digital recording medium for delivery

Survey flying specifications need to include the following:

- Traverse line spacing and, when necessary, control line spacing
- Base station for diurnal variations in the magnetic field recorded in close proximity to the survey area and the variation envelope defined, and magnetic storms from sun flares monitored
- Flight altitude tolerances (rugged terrain becomes a major consideration for airborne surveys because the data sensitivity is compromised)
- Instrument calibration requirements
- Navigation and flight path tolerances
- Base-map source and coordinate systems to be selected

Data compilation and interpretation include the following:

- Flight path recovery (method and accuracy)

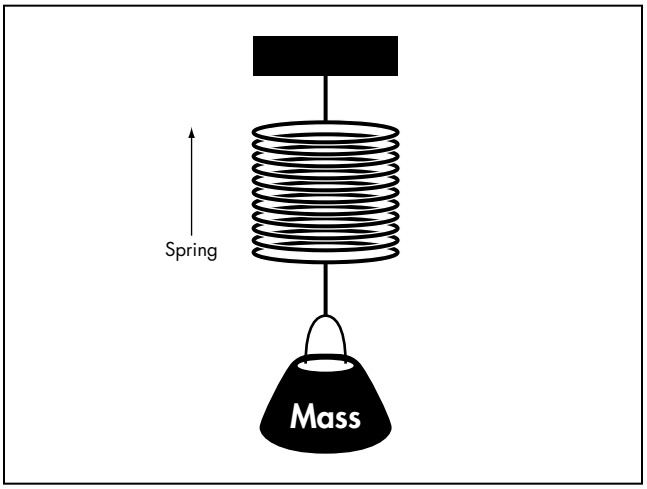


Figure 3.3-1 Simple gravity meter

- Data corrections required (leveling procedure for magnetic data)
- Final compilation products required, including map and chart scales, titles, and legends
- Interpretation (if required) and presentation products and procedures

Survey costs depend on the following:

- Survey type
- Number of parameters measured
- Number of required products delivered
- Type of platform required (ground, helicopter, or fixed wing)
- Acceptable measurement and navigation tolerances
- Time constraints of the survey
- Location of the survey
- Current fuel costs and/or fuel adjustments
- Mobilization time required for contractor to arrive at the survey area (usually quoted at cost)
- Other required products (e.g., interpretation products)
- Survey size

POTENTIAL FIELDS—GRAVITY

Gravity instruments measure the acceleration due to gravity (g). The average value of g at the earth's surface is 9.80 ms^{-2}. Gravitational attraction depends on the density of the underlying rocks, so the value of g varies across the surface of the earth.

A simple gravity meter (Figure 3.3-1) can be thought of as a mass on a spring. The spring exerts an upward and opposite force to exactly oppose the downward component, g_D, of gravity acceleration. The extension of the spring is proportional to the force exerted. The meter requires a stable and level platform for operation.

Airborne gravity and gradiometer systems have been developed for petroleum and minerals exploration. The airborne gravity systems are commonly used for petroleum exploration basin analysis, whereas the gravity gradiometer systems are used in minerals exploration where the detectability requirements are for small, compact ore bodies (Figure 3.3-2).

Costs range from $150/km for gravity gradiometer surveys (~0.1 mGal at 200 m half-wavelength) and range from $75/km to more than $1,000/km for petroleum gravity surveys accuracy (2.0 mGal at 3 km half-wavelength). (Note: Gal is the CGS unit of acceleration. One Gal is an acceleration of 1 cm/s^2. In minerals exploration, the gravity anomalies over structures are very small, so they are measured in thousandths of a Gal, or mGal.)

The detection of ore bodies is highly variable, but in general, iron-rich ore bodies are most readily detected, as seen in Figure 3.3-2. The size of the anomaly is proportional to the total mass or density × volume.

Variations of g due to density differences in rocks near the surface of the earth are very small compared to the g due to earth's overall mass. Many corrections to the data are needed so that density variations can be observed and recorded. There are two types of gravity measurements: absolute gravity and relative gravity.

1. **Absolute gravity:** To determine absolute gravity, an object is dropped inside a vacuum chamber and gravity is measured at reference points.
2. **Relative gravity:** Relative gravity is a change in gravity from one place to another.

In exploration it is necessary to measure small changes in gravity accurately. Gravity surveying is carried out with a portable gravimeter that determines the variation of gravity relative to one or more reference locations. Several corrections are required to transform gravity accelerations into useful exploration information, as follows (Rose and de Wet 2005):

$$g_{obs} = g_{1,967} + g_{free\text{-}air} + g_{Bouguer} + g_{terrain} + g_{tidal} + g_{isostatic} + g_{geo}$$

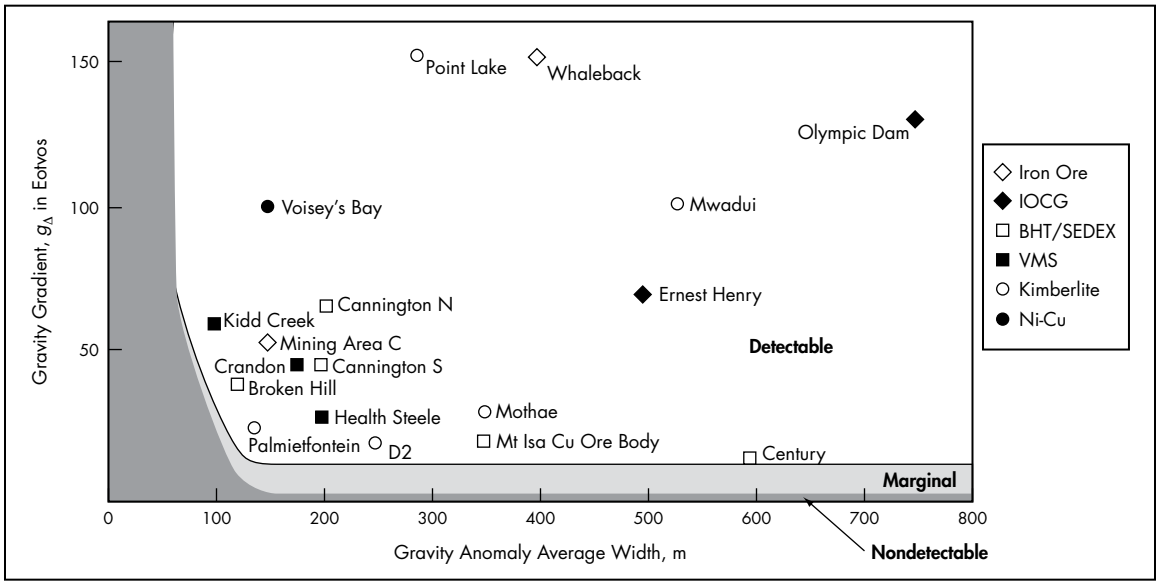
where

g_{obs} = observed gravity (980.000 mGal)
$g_{1,967}$ = reference ellipsoid gravity, rotation, earth bulge (970.000 mGal)
$g_{free\text{-}air}$ = gravity effect of elevation above sea level (0 to 1,000 mGal)
$g_{Bouguer}$ = approximate gravity effect of mass between observation and sea level (0 to 500 mGal)
$g_{terrain}$ = gravity effect of terrain variations (0 to 50 mGal)
g_{tidal} = gravity effect of earth-tides (0 to 0.3 mGal)
$g_{isostatic}$ = only required when doing large-scale regional surveys, calculated as for terrain effects to compensate for root zones in the crust
g_{geo} = gravity effect of geology (residual) (0.1 to 100 mGal)

Gravity Instruments

Ground gravity meters have accuracies between 5 and 15 mGal. The primary application of ground gravity is in engineering geophysics. In minerals prospecting, gravity often is used to verify airborne gravity survey anomalies prior to drilling. The cost of ground gravity surveys range from $20 to $50 per point, depending on terrain and size of surveys.

Gravity survey design is similar to magnetic survey design for both airborne and ground surveys. Gravity meters are usually calibrated at a national gravity-grid base-station point. Information on the locality of national gravity grid points is available at all government geological departments;



Courtesy of Fugro Airborne Surveys, Perth, Australia.

Figure 3.3-2 Gravity gradiometer response of some ore bodies

for example, for Australia one can obtain information from Geosciences Australia (Wellman et al. 1985).

Gravity measurements are the vertical component of the gravity field. The potential field due to the density of rocks does not have the vector complications that the magnetic field has, such as the remanent magnetic field superimposed on the earth's magnetic field. Interpretation of the gravity potential field is similar to magnetic interpretation, and 3-D inversions are common practice in interpretation.

ELECTROMAGNETIC FIELDS FROM NATURAL/ PASSIVE SOURCES

Natural electromagnetic (EM) sources come from a variety of processes—from within the earth's core and from distant galaxies. Frequencies for minerals and mining exploration are in the range 0.001 to 10,000 Hz. The only two sources that qualify are from the atmosphere and the magnetosphere. Electric storms in the lower atmosphere generate EM frequencies ranging from 1 Hz to 10 kHz. Frequencies lower than 1 Hz originate mainly from hydromagnetic waves in the magnetosphere. The concept of "skin depth" (δ) estimates the depth of exploration as given by Zonge and Hughes (1991) as approximately $\delta \cong 503\sqrt{(\rho/f)}$ meters, where ρ is resistivity of a half-space uniform earth and f is the frequency in hertz. Figure 3.3-3 illustrates the skin depth graphically, showing the increase in depth with increased resistivity.

Passive EM methods include magnetotellurics and natural source audio-frequency magnetotellurics (NSAMT). As seen in Figure 3.3-3, a broadband frequency system can explore from near surface to great depths using natural EM energy making the use of artificial sources superfluous when using the appropriate measuring equipment and software. The EM fields generated by lightning activity are seasonal, and one needs to make sure the surveys are conducted during the times of maximum EM activity (Goodman 1995).

The trend in brownfields exploration is toward deeper depths of exploration, which makes high-powered artificial current sources unacceptable in busy mine-site environments.

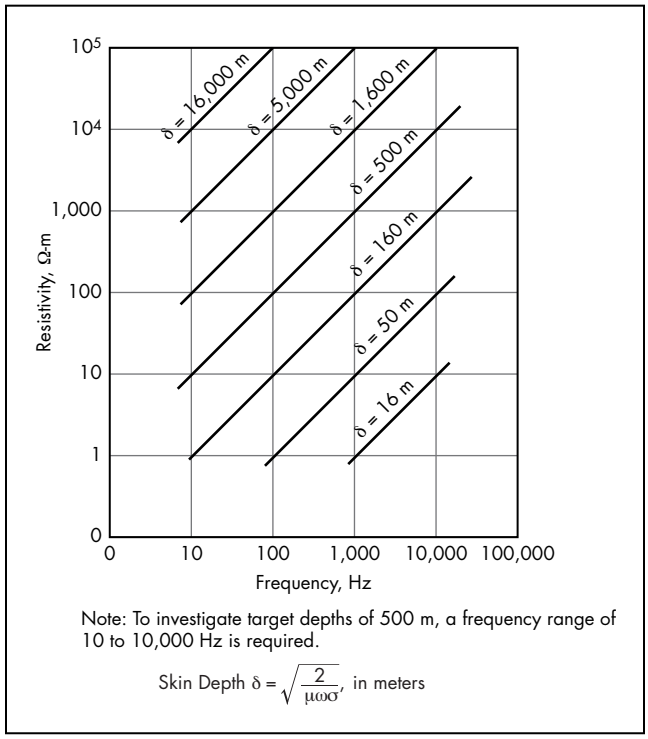


Figure 3.3-3 Example of skin depth

NSAMT is a viable alternative that minimizes the risk of electrical accidents.

Active Sources in Electromagnetic Methods—Principle of Induction

Faraday's law states that "if the magnetic flux perpendicular to the plane of the loop changes with time, then an electromotive force (emf) of the same magnitude as the rate of change of flux ($\delta B/t$) will be induced in the loop." The induced emf causes

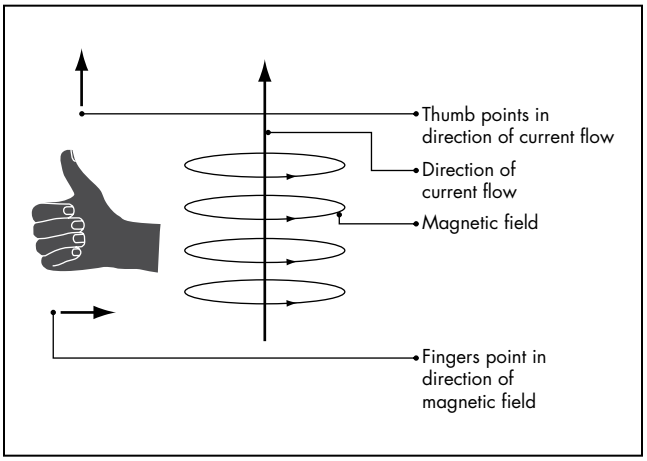


Figure 3.3-4 Right-hand rule in electromagnetic induction

current to flow in the loop so that the magnetic field of the current opposes the change of flux (Lenz's law)—so if the applied B-field is increasing, then current will flow in the loop in a direction so as to reduce the rate of increase (Figure 3.3-4).

Active-Source Methods

Active-source methods employ an artificially generated inducing field. Configurations used in minerals exploration using inductive loop sources include the following:

- Fixed-transmitter methods
- Moving-transmitter methods
- Downhole electromagnetics (DHEM)
- Airborne electromagnetics (AEM)
- Controlled-source audio-frequency magnetotellurics (CSAMT)
- Very low frequency (VLF) methods

Quantities and Units

The EM field may be described by the following vector functions:

B = magnetic induction, Wb/m^2 (tesla)
E = electric field intensity, V/m
H = magnetic field intensity, A/m
D = dielectric displacement, C/m^2

When measuring EM fields, the following units are used:

σ = conductivity, S/m (Seimens per meter)
ρ = resistivity, Ω-m (ohm-meters)
μ = magnetic permeability, H/m
μ_0 = magnetic permeability of vacuum, $4\pi 10^{-7}$ H/m
f = frequency, Hz
t = time, s

Usually one assumes that $B = \mu_0 H$ (linear, isotropic media).

Because the current in the transmitter varies with time, the primary magnetic field also varies with time and will induce secondary currents in nearby conductors according to Faraday's law. The secondary currents have their own secondary magnetic field, which can be detected at a receiver (Rx) coil.

EM fields can be generated by diverse human-made sources. Electrical currents are generated by specialized lightweight generators (traditionally sourced from aviation generators) and batteries.

Time domain electromanetic (TDEM) systems (Figure 3.3-5) are popular because the data are recorded during the off time, which eliminates the need for the high precision geometry required by the frequency domain system. Frequency domain systems are very sensitive to geometry of the receiver and transmitter orientation. However, TDEM systems are sensitive to telluric noise, which seriously affects the late time information. Frequency domain noise can readily be filtered using Fourier analysis.

Galvanic resistivity and induced polarization (IP) can be thought of as special cases of EM. The lower the frequency of the signal, the greater will be the depth of EM penetration within a conductive earth.

Airborne Electromagnetic Methods

The use of AEM methods has become mandatory in many exploration strategies because of their ability to map conductivity contrasts to relatively large depths. The broadband nature allows for depth sounding that, when stitched together, can be converted to conductivity depth images (CDIs). These data, integrated with magnetic and gravity inversion images, effectively reduce ambiguity in the interpretation.

AEM methods produce good-quality CDIs because of the excellent electronic technologies available. One can expect improvements in both functionality and miniaturization in the future. AEM technologies are fast approaching the resolution achieved by ground EM methods, so that the cost benefit of ground EM surveys is rapidly diminishing. AEM surveys are orders of magnitude more cost-effective than groundwork, except when small areas and/or mine infrastructure surveys are required. Mine infrastructure is always an issue when working with EM methods because of the electrical interference (i.e., noise) generated by mining activity; the high electrical currents deployed are seen as a safety risk.

Figure 3.3-6 shows a typical helicopter TDEM system in flight. The transmitter loop is at an altitude of about 30 m, the receiver is at the center of the loop, and a magnetometer is halfway between the helicopter and the transmitter loop.

Many publications describe the theory of EM prospecting; Figure 3.3-7 reviews and recommends EM methods by commodity, and although not exhaustive, it is a convenient and quick guide to most common exploration situations.

Galvanic Methods

Galvanic EM methods involve grounded electrode current sources, which cycle below a frequency of 1 Hz. The convention is to refer to conductivity in relation to EM methods and to resistivity in galvanic resistivity/induced polarization methods. The units of resistivity are ohm-meters, and the units of conductivity (the inverse of the resistivity) are Siemens per meter.

Electrical conduction in rocks occurs in the following ways:

- Electronic (motion of electrons)
- Electrolytic (movement of ions)
- Dielectric (displacement of electrons and positively charged atomic nuclei)

Most rocks are electronic conductors (semiconductors). The conductivity of rock matrices is generally poor because there are usually only a few unbound electrons. The bulk conductivity of rocks in the upper few kilometers of the crust is due mainly to electrolytic conduction (i.e., through the movement of ions contained in pore fluids).

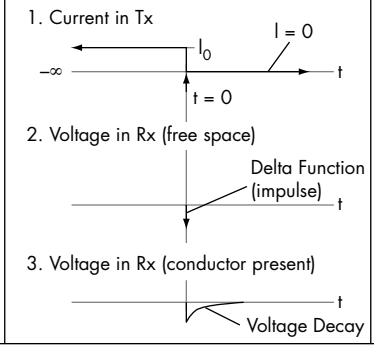
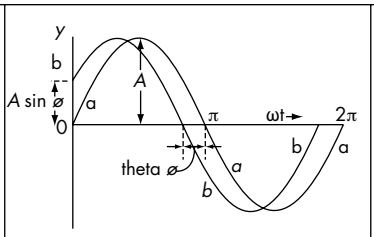
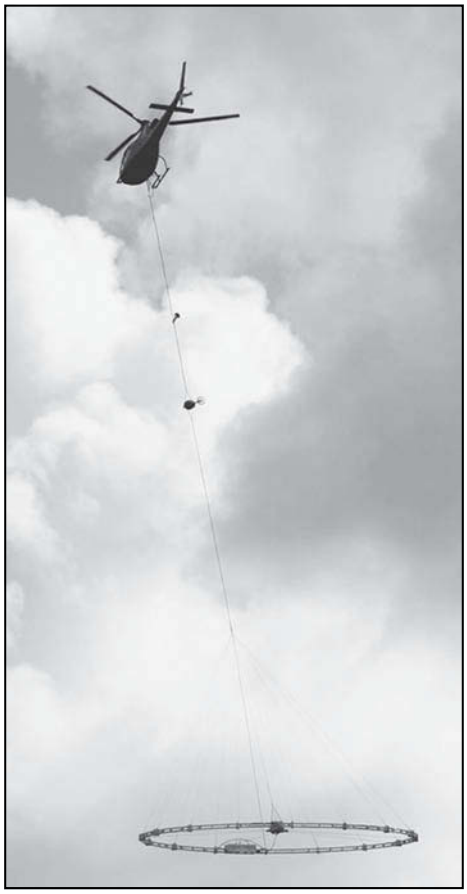
Time Domain Method				
Concept	A pulse of current energizes the earth and then is turned off. The decay of the electric and magnetic field tells us about the conductivity of the earth.		When transmitter (Tx) is on, a static magnetic field exists in the earth. When Tx current is stopped, currents start to flow in the earth that resist the collapse of the magnetic field. These currents can be approximated by a filament of current that grows in diameter with time and increases in depth in the earth. The magnetic field generated by these currents is called the primary field.	Because the current in the transmitter is time-varying, the primary magnetic field is also time-varying and will induce secondary currents in nearby conductors according to Faraday's law. The secondary currents have their own associated secondary magnetic field, which can be detected at a receiver (Rx) coil.
Frequency Domain Method				
Concept	A continuously varying current energizes earth and is altered in phase and amplitude as it passes through rocks of different conductivity. This variation in phase and amplitude tells us about the conductivity of the earth.		The cosine of the phase angle is called the in-phase component and the sine of the phase angle is called the out-of-phase component (or imaginary or quadrature component).	In frequency electromagnetic methods, the Tx current is typically sinusoidal with a given frequency. The Tx current is always on, and measurements are therefore always made in the presence of the B-field due to the current in the Tx wire (primary field).

Figure 3.3-5 Active electromagnetic sources



Courtesy of Fugro Airborne Surveys, Perth, Australia.

Figure 3.3-6 Typical helicopter time domain EM system. Note the magnetometer halfway down the tow cable. The receiver is at the center of the transmitter loop.

Major factors affecting electrolytic conductivity of rocks include the following:

- **Porosity:** Conductivity increases as porosity increases. This includes macro- and microscale porosity (e.g., fracturing, jointing, intergranular porosity, and vugs/vesicles).
- **Water saturation:** Conductivity increases with increased saturation.
- **Water salinity:** Conductivity increases with increased salinity.
- **Connectivity of pore spaces (permeability/pore tortuosity):** Conductivity is higher in permeable rocks.
- **Clay content:** Conductivity increases with increasing amounts of clay.

Other factors affecting electrolytic conductivity of rocks include the following:

- Fresh crystalline rocks typically have low conductivity because of their low porosity; however, they can be relatively conductive if extensively fractured/jointed.
- Unconsolidated materials (weathered layers) are generally conductive due to high porosity and clay content (and the presence of saline groundwater).
- Alteration of rocks into clay increases the conductivity of unmineralized rocks.
- Geological processes that decrease porosity (e.g., silicification) will decrease conductivity. Processes that increase porosity (e.g., fracturing, weathering) also increase conductivity.

Electrode Arrays Used for Galvanic Resistivity Surveys

The exploration objective usually defines which electrode array is most suitable and cost-effective to image the target; the selection also depends on whether the survey is required to map a large two-dimensional (2-D) area (e.g., for detecting

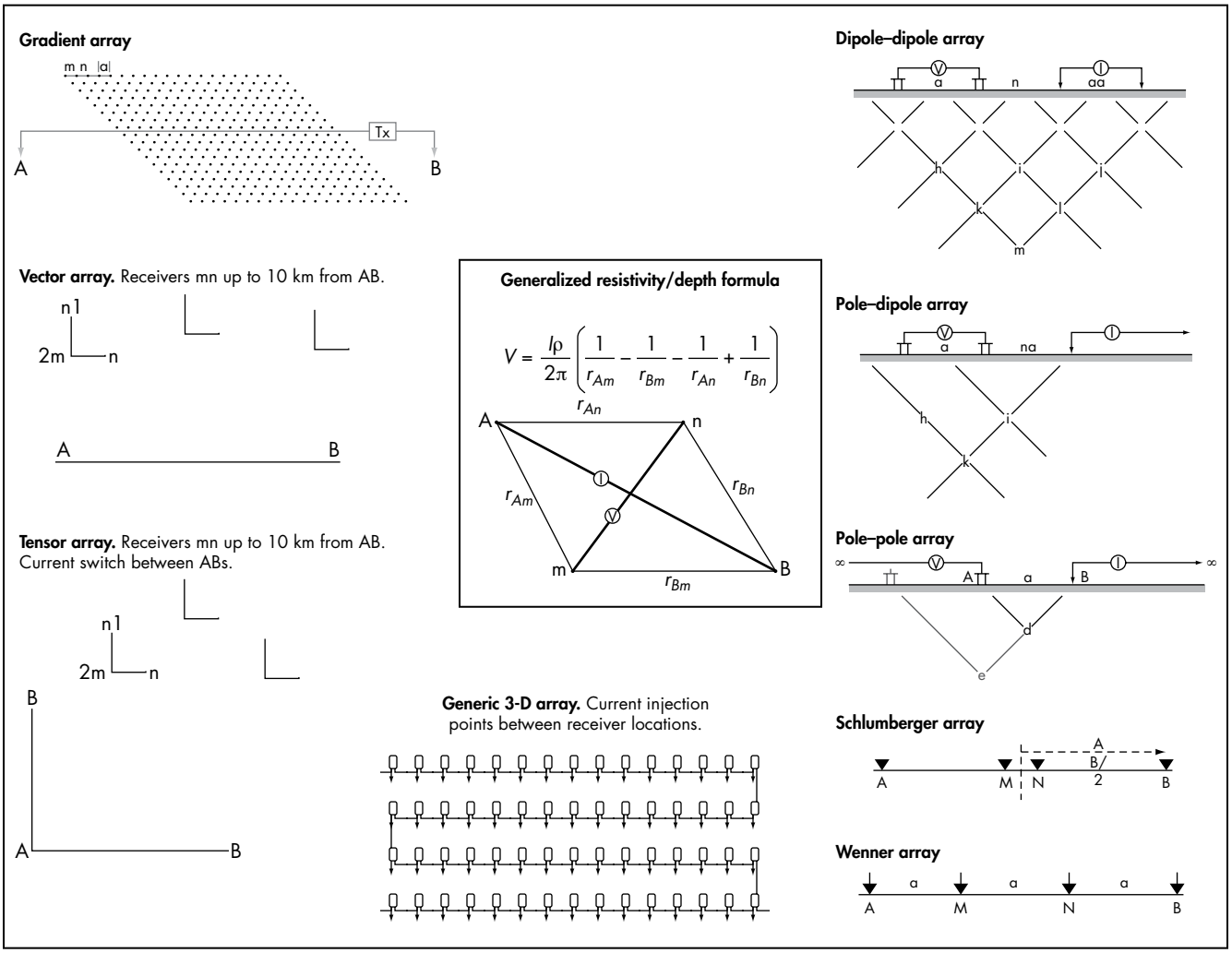


Figure 3.3-7 Galvanic resistivity arrays commonly used in exploration

gravel channels, for groundwater, or for porphyry copper targets) or whether the survey is for preparing a drilling campaign where finer resolution is required using a 2- or 3-D approach. Several arrays, illustrated in Figure 3.3-7, were developed and include the following.

- **Regional mapping arrays**
  - *Gradient:* Grounded current electrodes are fixed throughout the survey. Data are collected on a grid within an area approximately one-third of the current electrode separation. This method produces a resistivity map of the area.
  - *Vector:* This array is designed for rapid mapping of resistivity and induced polarization of large 2-D areas. Current electrodes separations are 2 to 5 km, and potential fields are measured parallel and at right angles to the azimuth of the current electrodes.
  - *Tensor:* This array has the same setup as for vector array except current electrodes are deployed at right angles, requiring measurements from each of the current orientations for calculating tensor fields. This array is useful when there is no reliable geological map for an area where geological strike directions (which could control current channeling) are uncertain.
- **Targeting for drill-hole positioning arrays**
  - *Dipole–dipole:* The electrode array is symmetrical, which simplifies the interpretation. The array is sensitive to steeply dipping structures.
  - *Pole–dipole:* This array is more labor efficient than the dipole–dipole array because only one current electrode needs to be moved. The asymmetrical array is difficult to interpret unless 2-D inversion software is used to create a CDI.
  - *Pole–pole:* This array has only two moving electrodes, making it even more labor efficient. Interpretation is effective when using 2-D inversion software.
  - *Schlumberger expansion and profiling:* This is the most popular method for groundwater and sedimentary basin mapping. It is sensitive to flat-lying sedimentary layers for coal and oil exploration. A symmetrical array makes interpretation relatively easy.
  - *Wenner expansion and profiling:* This is much the same as Schlumberger soundings, but it is more sensitive to lateral effects such as dykes and faults.

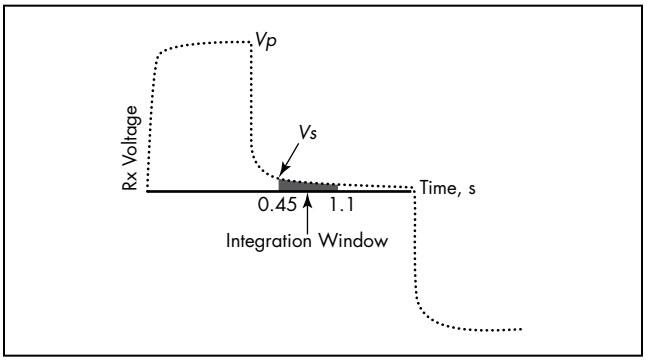


Figure 3.3-8 Time domain wave shape

  - *Three-dimensional:* Similar to seismic 3-D surveys, the current field is active in three dimensions so that all 3-D geological effects are mapped. This is becoming an important prefeasibility mapping tool for resource estimation.
  - *Marine:* Recently developed for oil exploration, hydrocarbons are imaged as resistive layers in the sedimentary sequence. Interpretation is relatively simple using inversion software.
- **Drill-hole resistivity arrays**
  - *In-hole:* Current and potential field electrodes are built into a flexible tube allowing similar survey methods as for surface mapping. This method is essential for reviewing IP targets.
  - *Mise-a-la-masse:* This method maps the surface image of an ore body at depth. Current electrodes are positioned in two drill holes at the sulfide intersection, and potential fields are measured on a grid at the surface. The method is based on the premise that current is channeled along the conductor.
  - *Downhole radial:* This is similar to the mise-a-la-masse method but with only one current electrode in the drill hole. Potentials are measured along radial lines starting from the drill collar. Used for detecting extensions when only one drill hole has intersected ore.

Induced Polarization

The arrays described previously are all used to measure IP on an exploration play. The method is used for detecting the presence of sulfide at depth. Unfortunately, IP effects are also produced by other minerals such graphite, clay, and some shales.

IP is a measure of the earth's storage capacity for electrical energy. When current ceases to flow in the presence of an IP (but no EM coupling), the voltage will drop instantaneously (Figure 3.3-8) to a secondary voltage value and then decay toward zero. The ratio of the secondary voltage (Vs) to the steady-state or primary voltage (Vp) is the (true) chargeability (M), which is a dimensionless parameter where $M = Vs/Vp$. By transmitting current into the ground through two grounded electrodes and measuring the electrical potential between two other electrodes, IP and resistivity are measured.

Transmitted currents will range from 0.1 to ≥15 A; larger currents improve the signal-to-noise ratio. An IP transmitter will put out up to 1,000 V. If the transmitting circuit has a resistance of 100 Ω, the transmitted current will be 10 A (Ohm's law: $I = V/R$). Except for the contact resistance between the electrodes and the ground, all parts of the circuit, including the earth itself, have negligible resistance. So contact resistance is the only important parameter. Therefore, special attention is given to the position of electrodes to lower the contact resistance during a survey.

Table 3.3-4 Relative cost of IP surveys based on a dipole–dipole coverage equal to 1.0 (typically $2,000 per day)

Survey Method	Relative Cost
Vector IP	0.1
Tensor IP	0.2
Gradient IP	0.5
Conventional pole–dipole/dipole–dipole	1.0
3-D array	3.0

IP costs are represented in Table 3.3-4, which shows the relative costs (based on a dipole–dipole coverage equal to one) of arrays previously discussed. Vector IP would be about one-tenth the cost of dipole–dipole surveys. Similarly, a 3-D array would cost about three times that of a dipole–dipole survey.

Electromagnetic Methods

EM methods are an important geophysical technique in the exploration for most ore bodies. Table 3.3-5 is a guide for selecting the appropriate EM method by exploration objective.

SEISMIC METHODS

Seismic methods are discussed to assist the engineer in understanding the scope and benefits to mining.

Table 3.3-6 shows the seismic methods deployed using natural/passive sources as well as those more generally used in exploration geophysics. Natural sources are primarily used for crustal and mantle studies. Earthquake prediction and mine seismicity monitoring are used for safety management and are mandatory applications in all underground mining ventures. These are outside the scope of this chapter but nevertheless are very important applications of geophysics.

Active source seismic methods are employed as either refraction or reflection surveys. The seismic method ranks as a high-resolution technique because of its ability to map relatively thin beds and local discontinuities. It is the only effective method to map geological unconformities.

Snell's law (Figure 3.3-9) is the guiding principle of seismic exploration. The law of reflection states that the angle of incidence of a ray is equal to the angle of reflection. The law of refraction states that a ray will refract at an interface where its velocity changes.

A wave is refracted when the velocity or refractive index of a medium changes. As shown in Figure 3.3-9, a reflection occurs when an amount of energy is reflected from a subsurface back into the same medium; the interface is known as an "acoustic impedance contrast," and the reflection coefficient, Rc, is the ratio of the amplitude of the reflected ray to that of the incident ray. The acoustic impedance, ρV, is density times velocity. The amount of energy reflected and transmitted depends on the ρV on either side of the interface, where the amount of energy reflected is the reflection coefficient, which is calculated as follows (Waters 1981):

$$Rc = (\rho_2 V_2 - \rho_1 V_1)/(\rho_2 V_2 + \rho_1 V_1)$$

The most direct way of determining seismic resolution is to measure it in a drill-hole geophysical log. The minimum

Table 3.3-5 Interpretation of geoelectric data by objective

Objective	Methods	Output and Products Required	Improvement If Multiproperty Survey (e.g., magnetics + gravity)
Deep crustal soundings	• Magnetotellurics (natural field source)	• Conductivity depth sections to depths of up to 50 km • Reveals deep structures acting as conduits for magma to the surface (e.g., Olympic Dam mine, nickel source rocks, and kimberlite)	• Adding gravity and magnetics helps to discriminate rock types. • Seismic tomography combined with magnetotellurics is ideal for such studies.
Petroleum exploration	• Controlled-source electromagnetics (CSEM) • Multichannel transient electromagnetics (MTEM)	• Conductivity depth sections to depths of up to 5 km	
Bauxite thickness profile	• Ground-penetrating radar • High-resolution time domain electromagnetics (TDEM)	• Profile images of conductivity depth images (CDIs) • Look like seismic sections • Depth to unweathered rock • TDEM output in CDIs • TDEM required to discriminate layers	• Adding seismic refraction helps to verify depth to unweathered rock. • Cost benefit is questionable.
Nickel laterite	• Ground penetrating radar • High resolution TDEM soundings	• Profile images of CDIs • Look like seismic sections • Depth to unweathered rock • TDEM output in CDIs • TDEM required to discriminate layers	• Adding seismic refraction helps to verify depth to unweathered rock. • Cost benefit is questionable.
Nickel sulfide	• Airborne EM methods to locate targets • Ground B-field TDEM methods most effective	• TDEM using in-loop and large-loop methods • B-field sensors preferable • dB/dT coils do not discriminate sulfide from sedimentary sulfides in banded ironstone	• Downhole EM is mandatory for all holes drilled. • This is essential for detecting off-hole anomalies. • See Table 3.3-3 for potential field contribution.
Iron oxide copper-gold	• Induced polarization (IP) • TDEM optional	• Copper sulfides usually associated with high-density hematite anomalies and low magnetic susceptibility anomalies • IP detects sulfides, but some hematite species also produce IP anomalies • Supergene copper usually conductive (e.g., Ernest Henry mine, Australia)	• IP helps to map sulfides but is often misleading. • Adding potential fields (see Table 3.3-3) improves the ability to discriminate.
Broken Hill type (BHT) silver, lead, and zinc	• IP • TDEM optional	• BHT mineralization usually associated with sulfides, thus very conductive (e.g., Cannington mine, Australia) • Airborne EM and ground TDEM useful for targeting drill sites	• Adding potential fields improves the ability to discriminate. • IP and TDEM helps to map sulfides but is often misleading.
Iron ore	• Resistivity section mapping (electrical resistivity imaging) • Output as resistivity depth image	• Hematite alteration produces resistivity anomalies in channel iron deposits • Clay degrades quality and can be mapped with resistivity	• Integrates with potential field data, especially density. • Discriminates iron ore from host rock (see Table 3.3-3).
Porphyry copper	• IP • TDEM optional	• Porphyry copper deposits have characteristic pyrite halos (donuts) and are often associated with major lineaments. • Because of the large size of alteration halo, they can be mapped using vector IP and/or gradient IP.	• See Table 3.3-3 for integration with potential fields.
Kimberlite	• Combination of helicopter magnetic and TDEM methods most effective	• Maps need to show coincident magnetic high and conductive anomalies. • Gravity lows are characteristic of weathered kimberlite.	• Addition of gravity, either on an airborne platform or on the ground, confirms kimberlite geometry and substantially reduces risk of failure.
Gold	• Airborne magnetic and EM with IP follow-up	• Gold deposits are found in diverse geological settings. • Basic mapping for structure, magnetite veins, quartz veins, etc. • Best accomplished through airborne EM and potential field methods • Ground IP detects sulfides in host rocks such as banded ironstone, magnetite veins, and quartz veins.	• TDEM methods not often applied but are only used when geologically justified.

Table 3.3-6 Seismic surveys by target objective

Objective	Output and Products	Improvement Adding Other Geophysics
Coal	• 2- and 3-D seismic high resolution surveys • Outputs are fault, thrust, intrusive, and coal seam geometry • Refraction for weathering profile	• Potential fields help to focus on dykes and sills that are not well mapped by seismic methods • Ground-penetrating radar helps on open-pit benches for roof of coal depth mapping • Wire-line logging is used for accurate measurement of coal thickness, quality, and borehole-to-borehole lithology correlation.
Nickel/platinum	• In layered intrusive, platinum reef and hazards such as dunite "pothole" • Faults and post-mineralization intrusives • Depth to reef and reef geometry	• Magnetic surveys help to discriminate intrusives such as dykes and porphyries.
Kimberlite	• Geometry of the walls of a kimberlite pipe with host rock	• Potential fields help to define geometry
Potash	• Primary purpose is mapping hazards in the roof of the deposit (water seal) • Faults and stratigraphy • Lateral changes in potash depositional environment	• Wire-line logging is used for determining the roof strength by means of sonic logs and for lithology correlation.

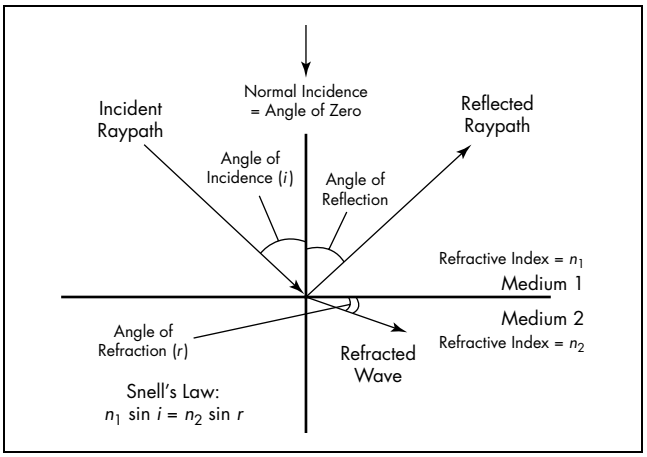


Figure 3.3-9 Snell's law showing reflection and refraction

quantities would be the P-wave velocity and the density logs. Unfortunately, a drill hole only gives the information for a short distance from the actual hole, so that a number of holes need to be logged to estimate the average for an area. These controls are essential when converting from time to depth during the interpretation.

Planning for reflection surveys is an involved process, and its complexity is often underestimated. Reflection seismic crews can number as many as 150 people with a variety of equipment and machinery—more so when drilling of shot holes are required. Typical seismic reflection costs are about \$15,000/d for two dimensions and about \$300,000/km^2 for a high-resolution, 3-D seismic reflection survey. Table 3.3-7 is a checklist that provides an outline of the planning and organizing required. The objective is to have the right people in the right jobs communicating effectively.

Table 3.3-7 Checklist for planning a seismic reflection survey

Planning Step	Checklist
Presurvey	☐ Familiarization introduction: Presurvey project team scoping ☐ Scheduling regular meetings ☐ Production of meeting action minutes and addressing of each item at meetings until closed out ☐ Determination of the survey schedule project/license time requirements ☐ Survey design: Project team to provide technical information required ☐ Survey data ☐ Regulatory approvals ☐ Budget
Seismic acquisition contract*	☐ Pre-tender contractor meetings ☐ Contract preparation
Other contracts	☐ Review of seismic acquisition contract ☐ Determining list of other contractors
Health and safety	☐ Contractor prequalification
Environment and community	☐ Environment management plan
Pre-mobilization	☐ Project execution plan
Mobilization	☐ Crew audits, which include follow-up and close-out of prior audit items
Field operations	☐ Survey progress reporting—includes joint venture partners and government ☐ Data harvesting
Postsurvey	☐ Accountable contractors ☐ Financial reconciliation ☐ Delegation of responsibility for invoice settlement

*This contract often causes the most objections from the contractor's legal representatives, and it can be onerous to negotiate a final agreement.

WIRE-LINE AND DOWNHOLE GEOPHYSICAL LOGGING

The ability to constrain surface geophysical interpretations with additional information such as downhole logging increases the value of the surface data. The most common methods include the following:

- Petrophysical logging in coal exploration is used to calibrate surface seismic processing and interpretation. In coal exploration, the information is critical for coal quality, thickness, ash content, and stratigraphic mapping. Tools are usually density, resistivity, gamma, acoustic impedance, susceptibility, and self-potential.
- Downhole electromagnetic (DHEM) sensors are high-resolution, three-component sensors used primarily for nickel and other sulfide exploration. DHEM can detect

conductors up to 150 m from the drill hole, defining a target vector for drill orientation. The receivers measure mostly in the time domain and use the popular University of Toronto electomagnetic (UTEM) on-time transmitted signal for better sensitivity (e.g., as is used in nickel exploration in Sudbury, Canada).

- Downhole gravity has been common in petroleum exploration for decades. The advantage over the conversional neutron logging sensors is that it can measure the relative gravity field that can be used to detect off-hole density targets. The gradient information can also be transformed into real density measurements. A limiting factor with neutron tools is the maximum density that can be reliably measured is about 2.8 kg/t, whereas the gravity-logging tool will measure higher gravity values, making it ideal for resource estimation for commodities such as iron ore.
- Three-component, downhole magnetic surveys are uncommon and are sometimes employed for mapping the off-hole presence of magnetic minerals, such as magnetite and pyrrhotite, and for mapping magnetic intrusive plugs. The difficulty with magnetic logging is in accurately tracking the orientation of the three-component sensor while traveling downhole.
- Downhole IP is useful for calibrating surface IP interpretations for drilling. Good practice dictates that all drill holes targeted on IP anomalies should be logged.

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CHAPTER 3.4

Geochemical Prospecting

Jeffrey A. Jaacks, L. Graham Closs, and J. Alan Coope

Exploration geochemistry, or geochemical prospecting, includes any method of mineral exploration based on the systematic measurement of one or more chemical or chemically influenced properties of a naturally occurring material. The property measured is most commonly the trace concentration of some chemical element or group of elements. The naturally occurring material may be rock, soil, stream sediment, glacial sediment, surface water, groundwater, vegetation, microorganisms, animal tissues, particulates, or gases.

HISTORY OF GEOCHEMICAL PROSPECTING

It is clear from historical records that the principles of geochemical exploration have been applied in prospecting over several thousand years. The prospector who panned for gold and traced the colors upstream to a bedrock source used mineralogical observations in a similar way to the modern geochemist who utilizes sensitive chemical analyses to outline patterns of dispersion in the surficial environment. Geobotanical indicators were recognized as early as the eighth and ninth centuries. The mid-16th century work by Agricola (1556) describes the analysis of natural waters, springs, and their residues.

Modern methods of exploration geochemistry were first used in the early 1930s in the Soviet Union. Shortly thereafter, the methods were applied in the Scandinavian countries, particularly Sweden. In North America, the earliest geochemical surveys were carried out between 1938 and 1940 by H. Lundberg in Newfoundland and in 1944 by H. Warren in British Columbia. The first comprehensive geochemical exploration studies commenced at the U.S. Geological Survey under the leadership of H.E. Hawkes in 1947 and at the Geological Survey of Canada with R.W. Boyle in the early 1950s. The Applied Geochemical Research Group was established at the Imperial College of Science and Technology in London in 1954 under the direction of J.S. Webb, and in France, research related to exploration geochemistry began in 1955.

The successful application and adaptation of geochemical exploration techniques in all parts of the world, the rapid development of analytical and computer technology, and improvements in field transportation have made geochemistry one of the more effective and widely applied exploration disciplines. Analytical capability is such that relatively rapid, sensitive analysis can be achieved for virtually all metals of economic interest. New technological advances are expanding our established capability to cost-effectively detect and interpret dispersion patterns related to mineral deposits in a wide variety of environments. Recent geochemical research has focused on detecting ore bodies hidden beneath transported overburden and development of geochemical dispersion models to aid in the interpretation of anomalies developed in these environments (Cameron et al. 2004).

Through multielement analysis, geochemical data can reveal signatures related to distinct geological units and geochemical processes. This capability, when applied to rock samples, permits geological correlation as well as the more precise delineation of otherwise invisible alteration features related to mineralization. When applied to soil and other types of samples, multielement data can help outline major geological units and the extent of alteration systems, and identify the presence of mineralization buried under extensive cover.

GEOCHEMICAL PROSPECTING METHODS

The various components of a geochemical exploration program include sampling design, choice of sample media, sample collection, sample analysis, quality control of analyses, data visualization and evaluation, and interpretation. These components of a geochemical survey are interdependent and should be focused on the geologic objective to be achieved. Good design and planning (including orientation) and well-conducted sampling programs constitute the foundation of all sound geochemical exploration work (Closs and Nichol 1989). Inappropriate sample preparation prior to analysis can destroy the integrity of the well-chosen sample. It follows that no matter how accurate and precise the techniques used for analysis or how advanced the statistical treatments

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and computer programs used for data handling and interpretation, they cannot resurrect the lost quality and representativity and restore the decreased probability of exploration success caused by poor planning, improper field sampling, faulty sample preparation, or mismatch between sample media and digestion method.

GEOCHEMICAL PROGRAM PLANNING

Experience has confirmed that the success of a geochemical survey depends largely on the correct assessment of all the natural factors that influence the mobility and dispersion of the metallic elements of interest. These natural factors, which are described more fully in textbooks by Rose et al. (1979) and Levinson (1974, 1980), are

- Physical and chemical properties of the elements or parameters of interest,
- Nature of the geology and mineralization,
- Geomorphological history of the field area,
- Vegetation, and
- Topographic and climatic conditions.

These factors are completely interrelated, and any change in one condition—such as a geologic change from an acidic to a basic environment, a local climatic change caused by elevation, or a topographic change from rolling to mountainous terrain—can significantly influence element mobilities and the lateral extent of the dispersion from any mineralization type. Consequently, procedures for sampling, sample preparation, and analysis that prove to be satisfactory in one field area may be inadequate and unreliable in an adjacent region.

The processes of oxidation, weathering, erosion, transportation, sedimentation, and diagenesis that characterize the surficial environment are too complex to describe in detail in this summary. The physiochemical conditions determine whether the weathering products of mineralization disperse either in solution in a hydromorphic form or in a solid or clastic form. The low pH, high Eh conditions of the acid-weathering environment (e.g., oxidizing massive sulfides) promotes the solubilization of many base metal and other elements and their more widespread hydromorphic dispersion in surface and groundwaters. Alkaline conditions developed over a weathering limestone or in a semiarid climatic condition prohibit or arrest this hydromorphic dispersion. Under these conditions, the insoluble metal particles disperse mechanically to form clastic patterns.

A series of publications sponsored by the Association of Exploration Geochemists develops conceptual models based on conditions in Canada (Bradshaw 1975), Scandinavia (Kauranne 1976), the Basin and Range province of the western United States (Lovering and McCarthy 1978), Australia (Butt and Smith 1980), tropical and subtropical terrains (Butt and Zeegers 1992), and arctic and temperate terrains (Kauranne et al. 1992). These conceptual models describe the principles and mechanisms of formation and configuration of anomalies and dispersion patterns revealed through geochemical exploration surveys in these regions. This format has been expanded in the *Handbook of Exploration and Environmental Geochemistry* published by Elsevier, currently with 11 volumes covering many aspects of geochemical exploration.

GEOCHEMICAL ORIENTATION SURVEYS

When contemplating a geochemical survey in a new region, the most reliable method of determining the extent and nature of dispersion patterns is to conduct an orientation survey. The objective of orientation sampling is to determine and outline the existence and characteristics of dispersion patterns or anomalies associated with mineralization and also background levels in similar environmental conditions. The specific sample media used are dependent on a knowledge of the field area, the prospecting problem, and, if available, previous experience, but may include any of the following: rocks, soils, stream sediments, surface waters, groundwater, glacial sediment, lake sediment, vegetation, soil gases, and microorganisms.

The orientation survey commonly involves the collection of a number of relatively closely spaced samples over and in the vicinity of known but preferably undisturbed mineralization with the express purpose of outlining the dispersion patterns in the available sampling media. This information can then be used to select the most reliable sampling method, sample preparation, and analytical techniques capable of detecting similar anomalies under similar environmental conditions. The detailed examination of the nature and shape of the dispersion patterns invariably yields information on the principal natural factors responsible for the observed distribution of anomalies. This is a significant aid in the development of interpretation procedures.

Table 3.4-1 is a synthesis of the important parameters that can be derived from a properly planned and executed orientation survey. Based on this information, the optimum physical parameters (sample depth, sample interval, analytical technique, etc.) for routine surveying can be chosen that necessarily take into account the defined dispersion characteristics as well as the physical, logistical, and economic conditions pertaining to the project. Large samples should be taken to provide sufficient material for the full evaluation of the parameters listed in Table 3.4-1, and representative samples should be collected from nonmineralized areas to adequately define background conditions and contrast of anomalies, where contrast is defined as the ratio between background and threshold value of anomalous samples.

A survey conducted over gold-silver vein mineralization at Mount Nansen in the Yukon Territory of Canada (Coope 1966) illustrates the application of the orientation approach. Detailed profile sampling of soil and overburden exposed in a trench across virgin mineralization produced patterns of lead, antimony, and zinc, as illustrated in Figure 3.4-1. It is apparent from these patterns that the dispersion behavior of lead and antimony is similar but quite contrasting with the patterns for zinc. All metals have been influenced by downslope movement in the overburden, but the zinc pattern is compatible with dispersion in solution along the bedrock surface. Examination of the patterns also indicates that near-surface sampling (0 to 300 mm, or 0 to 12 in.) would not reliably indicate the mineralized vein. It was concluded that a sample depth of 460 to 610 mm (18 to 24 in.) and a sample interval of one-third the anomaly width at this depth (5 m, or 15 ft) were the optimum parameters necessary for routine surveying (Figure 3.4-2). Observation of these sampling criteria led to the discovery of several previously unknown veins.

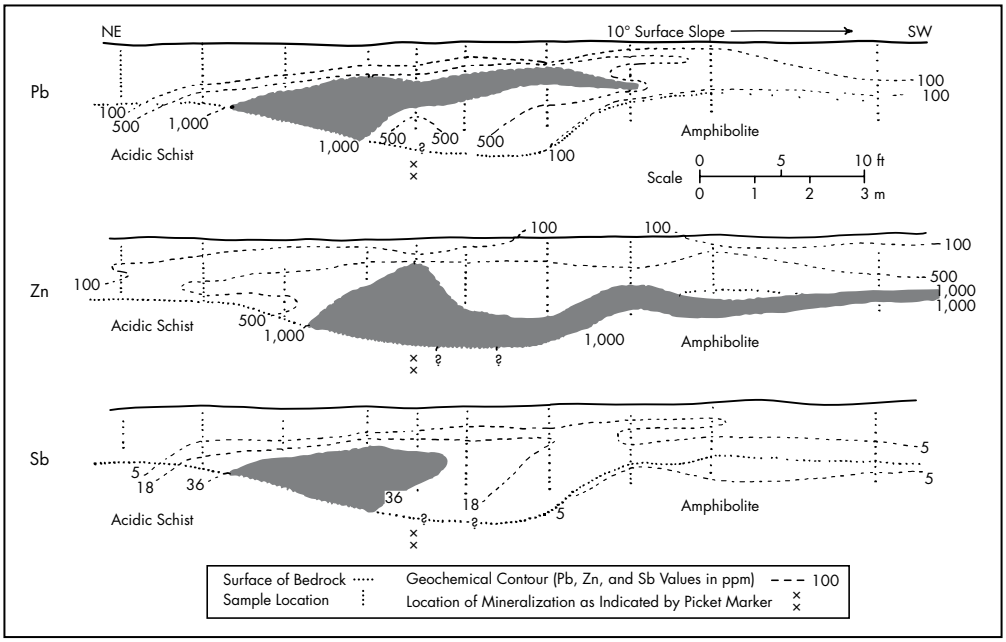
The soil profile interval in an orientation survey over more extensive types of mineralization will be proportionately larger than the intervals chosen for narrow-vein-type mineralization.

The sample intervals employed in orientation surveys will vary according to the anticipated size of the target

Table 3.4-1 Parameters to be derived from a properly planned orientation survey

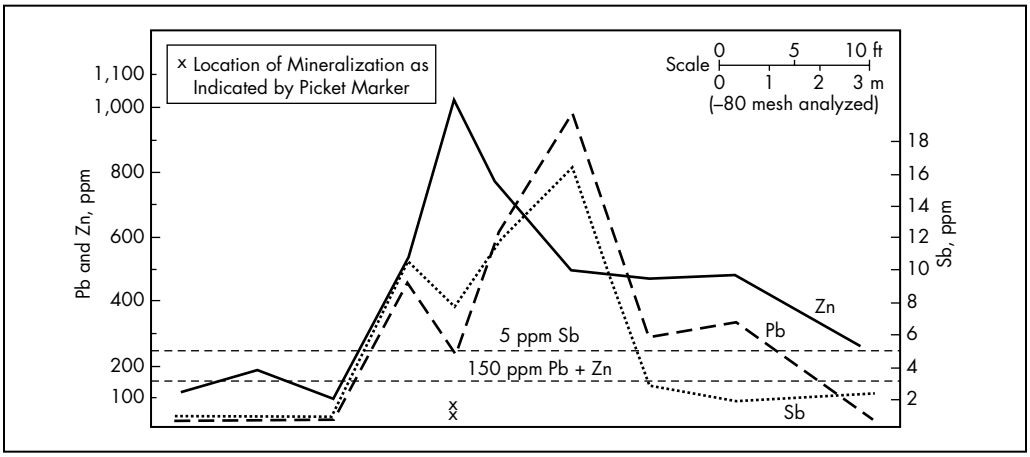
Rock	Soil	Sediment	Water	Biogeochemical	Gas
Shape, extent, reproducibility, homogeneity, and controls of geochemical patterns					
Most suitable indicator elements and best analytical methods					
Reproducibility of sampling and analysis					
Sampling pattern and density					
Amount of sample required					
Required field observations					
Possible contamination sources					
—	Effects of topography, drainage, vegetation, geology				
—	Seasonal or temporal variations				
—	Optimum sampling depth or horizon			—	Sampling depth
—	Optimum sampling fraction (size, heavy minerals, organic fraction)		—	—	Sampling volume
—			—	—	—
Sample material (rock, vein, etc.)	Nature of overburden (residual vs. transported)	Correlation with Fe-Mn oxides	Optimum material (ground or surface)	Optimum species (distribution, ease of anomaly recognition, ease of sampling, and depth of root system)	Effect of soil type
	Mechanism and direction				Effect of soil depth
Type of sample (grab, channel, etc.)	Soil profile development	Correlation with organic matter	Control of water flow	—	Effect of temperature
	Depth variation of indicator elements	—	Difference in aquifers	—	Effect of soil moisture
Effects of weathering and alteration	—	—	Relation to recharge areas	Part of plant to be sampled	—
Applicability of mineral separations	—	—	Variation with depth	Effect of aspect (sunlight, shading)	—
Mineralogy (sulfides, biotite, limonite, calcite, etc.)	—	—	Variation with type of lake	—	—

Source: Adapted from W.B. Coker, personal communication.



Source: Coope 1966.

Figure 3.4-1 Profile distribution of lead, zinc, and antimony over mineralized vein, Mount Nansen area, Yukon Territory



Source: Coope 1966.

Figure 3.4-2 Profiles of lead, zinc, and antimony concentrations in soil over a mineralized vein at 45- to 60-cm depth, Mount Nansen, Yukon Territory

mineralization and the dispersion characteristics of the metals of interest. In areas where no previous experience exists, a short interval is recommended so that three or more samples are collected within the distance spanning the zone of mineralization. This interval should then be progressively expanded with distance from the metal source to the limits of the known or anticipated dispersion pattern. Samples must also be collected from nonmineralized areas to establish the background range, and sufficient material should be collected at each site to allow for the determination of optimum size fractions, analytical techniques, and other factors listed in Table 3.4-1.

SAMPLE COLLECTION AND HANDLING

Effective sampling of all surficial media requires well-trained personnel capable of recognizing and describing the correct sample material and the sample site characteristics. Samplers should be able to recognize and, if possible, avoid situations where contamination from human activity or changes in the natural physiochemical conditions can produce spurious or unusual results. In most situations, these sampling duties can be undertaken by trained technical personnel with geochemical exploration experience. For some specialized types of surveys, where identification of the correct sample material is critical (as in biogeochemical or glacial till sampling programs), it is prudent to employ qualified specialists (e.g., botanists and quaternary geologists) to conduct orientation surveys and instruct and supervise the sampling teams.

Sampling tools vary according to the medium and the field situation. Non-contaminating equipment is essential, and care should be exercised in not only choosing noncontaminating steels for shovels, trowels, augers, and so forth, but in ensuring that lubricants and adhesives, weldings, and solders are metal free too. This awareness of geochemical cleanliness extends to the dress of the sampler, who should avoid wearing metal buckles, gold rings, and so forth, and avoid handling coins, which might lead to contamination by chipping or transfer of metal on fingers.

The same caution is necessary in the choice of sample containers. Kraft paper (with noncontaminating waterproof glue and closures), olefin, and plastic bag containers of appropriate size are frequently used. Kraft and olefin allow samples to be dried without transfer, while plastic bags are commonly used for larger samples. More rigid polypropylene bottles can be utilized in water sampling, and a variety of sampling devices are available for the sampling of gases and particulates.

Collecting Different Sample Media

Soil

Soils vary considerably in composition and appearance according to their genetic, climatic, and geographic environment. Classified into residual and transported types according to their relationship to their substrate, soils are mixtures of mineral and biologic matter and may be distinctively differentiated into a series of soil horizons. Residual soils characteristically contain detectable dispersion patterns developed during the weathering of mineralization in the underlying bedrock, and these patterns are revealed by careful sampling of appropriate soil horizons. Transported soils present more difficult sampling problems, but meaningful sampling is possible in many areas after the genetic origins of the transported cover are understood.

With all but a few exceptions, soils are sampled along traverses or grids in the follow-up or detailed prospecting stage of a geochemical program. Orientation programs define criteria such as sample depth or soil horizon to be sampled, sample interval, and the size fraction for analysis. It is essential that these criteria be applied consistently through the survey (Hoffman and Thomson 1987).

Stream Sediments and Surface Water

Stream sediment is one of the more commonly used media for geochemical reconnaissance surveys. The sediment at any point in a stream is a natural composite sample of erosional materials from upstream in the drainage basin and will include clastically, hydromorphically, and biogenically derived products from weathering mineralization. The length of anomalous dispersion trains will vary with the nature of the mineralization, source, and physiochemical environment of the field area or drainage basin. In humid, actively oxidizing environments, dispersion trains from sulfide-rich base metal deposits may extend downstream for several kilometers.

Active stream sediment—material constantly or most frequently washed by stream waters—is collected from the

center of a drainage, avoiding sites that may be contaminated or influenced by bank collapse. In most survey programs, approximately 500 to 1,000 g (1.1 to 2.2 lb) of fine-grained material are collected from the upper few tens of millimeters of the sediment. If heavy minerals are to be examined, larger samples from deeper in the streambed are collected from carefully selected sites. In all surveys in new areas, the critical parameters of sample interval, sediment size fraction, appropriate analytical procedures, significant anomaly contrasts, and background levels are determined through orientation surveys (Hale and Plant 1994).

In the regional reconnaissance prospecting mode, stream sediment surveys can be designed to systematically cover areas up to several hundred square kilometers. To pinpoint source areas in more detail, anomalous indications of mineralization can be followed up with more detailed sediment sampling. Follow-up sampling of seepage areas is particularly effective in delimiting anomalous groundwater sources containing metal derived from oxidizing mineralization. If appropriate, soil sampling can be used to define suboutcropping mineralization in the anomalous source areas defined by the sediment survey.

Regional reconnaissance can also be achieved by sampling the waters of actively flowing streams where metal is dispersing in solution. A prospecting approach similar to the sampling of stream sediments is necessary, collecting waters in clean 250-mL (0.07-gal) polypropylene or glass bottles. Sampling of groundwater seepage sites is an integral part of stream water surveys. To keep the dissolved metals from adhering to the bottle walls, a few milliliters of acid is routinely added to the water samples after collection. Temperature, pH, total dissolved solids, conductivity, and certain other measurements are commonly made at the sample site.

Lake Sediments and Lake Water

Lake sediments and lake water sampling have been developed into effective geochemical reconnaissance techniques in areas of Canada and Scandinavia where lakes are common, conditions are swampy, and/or where stream drainages are inaccessible or poorly developed. In low-relief regions, the lake sediment medium is dependent on the hydromorphic dispersion of metals into the lake environment through groundwaters and the adsorption of this metal onto hydrous oxides and the organic-rich muds deposited on the lake bottoms. The sampling focuses on the collection of these organic muds using specially designed sampling devices. In more mountainous areas, fine-grained clastic dispersion into the lake sediment becomes a more important factor. In all areas, satisfactory sample locations are found well away from lake shores and are reached using boats, float planes, or helicopters. Lake waters are commonly collected at the same sites as the lake sediments (Coker et al. 1979).

Glacial Deposits

Extensive quaternary glacial deposits occurring over most of Canada and the northern United States, northern Europe, and northern Asia have presented major challenges to exploration. As a better understanding of the origin and formation of these glacial sediments has grown, their blanketing presence has become progressively less formidable, and effective exploration techniques have been developed.

Mineralized boulder tracing in glaciated regions is an established technique of the traditional prospector in Scandinavia and parts of Canada. Methods were developed for sampling tills in the 1950s, and this technique is now the preferred sampling method in modern geochemical exploration programs. Approximately 70% of lodgment till is locally derived, and most of the early success with till sampling was in areas of shallow till cover (<10 m [30 ft]) where the sample medium is reasonably accessible. In the 1960s, lightweight percussion drills, such as the Pionjar and Cobra models, were adapted to collect small samples of till from immediately above the suboutcropping bedrock to geochemically categorize anomalous geophysical features at depths of up to 21–24 m (70–80 ft).

Overburden drilling technology, particularly reverse circulation and sonic drilling, advanced rapidly with the utilization of larger drills in programs for uranium and gold deposits in glaciated areas. Because most types of gold deposits are not detectable by conventional geophysical methods, lodgment till sampling using overburden drills to depths of 100 m (330 ft) has been used routinely in prospecting for gold in the Canadian Shield through the 1980s. Large samples of till (~10 kg [20 lb]) are recovered in these programs from which the heavy mineral fraction is separated and examined both visually and chemically for gold and other metals (Averill 2001).

The correct interpretation of these data is dependent on an understanding of the till stratigraphy and provenance of the transported material. The technique is expensive, with combined drilling, sample treatment, and analytical costs ranging from $20 to $50/m ($60 to $100/ft), but it is cost-effective in this deep overburden-covered environment where other methods in gold exploration have not been as successful. In contrast with the heavy mineral sampling of tills practiced in Canadian exploration, Scandinavian explorers place a greater reliance on the 63 μm (–240 mesh) fraction of till. This fraction has successfully indicated the presence of several types of mineralization, including gold.

Rocks

Rock sampling, or lithogeochemical surveys, comprise systematic sampling of outcrops, trenches, drill cores, or drill cuttings. As with other types of geochemical surveys, the sampling procedures and the sample material collected in lithogeochemical work should be standardized as much as possible.

Lithogeochemical sampling necessarily must take into account the geological environment and the type of mineral deposit of interest to the explorer. Very briefly, syngenetic lithogeochemical patterns (which developed at the same time as the rocks that enclose them) can develop, for example, in (1) intrusives genetically associated with specific mineral deposit types, (2) volcanic stratigraphy where exhalative activity has dispersed detectable quantities of metals during the formation of volcanogenic massive sulfide deposits, or (3) the vicinity of sedimentary deposits. The scale of sampling necessary for detection of these and other types of patterns will be determined by orientation surveys but may include the regional sampling of individual plutons or more detailed sampling of specific parts of an exposed stratigraphic section.

Epigenetic lithogeochemical patterns (those developed long after the rocks that enclose them) can develop as diffusion aureoles in the rocks hosting epigenetic mineral deposits or as leakage aureoles along fractures and other structures that mark the pathways followed by hydrothermal or groundwater fluids prior to and subsequent to the deposition of the significant mineral deposits. Both these processes introduce the concept of mineral zoning of geochemical aureoles. The

ratios of the different elements introduced during the mineralization period varies with distance from the principal deposit according to the properties of these elements and the host rocks and the physiochemical conditions at the time of deposition. Leakage anomalies can extend for hundreds of meters or feet from a deposit, whereas diffusion anomalies rarely exceed 30 m (100 ft). The scale of dispersion, the presence of zoning patterns, and the sampling parameters for epigenetic geochemical patterns are determined by appropriate lithogeochemical orientation surveys.

It is clear that surveys designed to detect leakage anomalies will focus on a standardized collection of fault or fracture zones and, possibly, bedding structures. In contrast, the preferred lithogeochemical sample material for the detection of syngenetic haloes is likely to be unfractured and the scale of sampling much more detailed. In all instances, geochemical analysis of lithogeochemical material has the potential of delimiting dispersion patterns associated with mineralization beyond visible alteration.

Vegetation and Humus

Early scientific observers dating from the eighth and ninth centuries recorded that the morphology and distribution of certain plants were affected by the presence of metals in the soils. Such visible variations in a plant species are referred to as geobotanical indicators. Many other plants, while not showing any visible variations, are capable of concentrating metals in their tissues, and the presence of anomalous metals in the soils or groundwater is often reflected in the metal content of leaves, twigs, or other plant organs. These invisible metal concentrations are known as biogeochemical indicators (Brooks 1983).

The seasonal fall of leaves and needles transfers some of the accumulated metals to the surface soil where they are incorporated in the humus layer. Sampling of this humus (alternatively known as mull, A_o, or A_h material) by Scandinavians in the 1930s revealed its potential for geochemical prospecting. This is especially true in areas of transported material where the root penetration of the plant exceeds the thickness of this cover and obtains nutrients from the underlying mineralized bedrock and groundwater.

In addition to the direct recognition of geobotanical indicators, the most attractive feature of vegetation sampling is the ability of biogeochemical and mull prospecting to see through thicknesses of barren transported overburden.

Plants are complex organisms and so is their metabolism. Different species respond differently to the same conditions, and, consequently, some species are more effective biogeochemical indicators than others. Deep-rooted plants (e.g., the mesquite) are much more effective prospectors of the deeper groundwater than the shallow-rooted flora of the southwestern U.S. deserts and are, therefore, preferred species in biogeochemical work. Evapotranspiration has been suggested as a mechanism for movement of metals into the nutrient depth of these plants.

Some species preferentially concentrate metals in specific tissues such as leaves, twigs, bark, or wood. It is therefore important to establish the most favorable tissues for sampling after a useful species has been identified (Dunn 2007). This complexity is accentuated by the fact that metal uptake may vary with aspect, and degree of uptake commonly varies with the seasons. In temperate forest regions, accelerated uptake and higher concentration commonly occur during the spring growth following a dormant winter season. In hot desert regions, following the exhaustion of available near-surface water during the dry season, deep-rooted plants will tap the deeper, more metal-rich groundwater.

These variables make biogeochemical sampling a specialized exercise. The expertise of an experienced geochemist or botanist is essential during orientation studies and the supervision of vegetation surveys. Because of the seasonal variations, biogeochemical surveys must be completed quickly, and sampling in the spring and fall is generally avoided.

The same complexities do not affect the humus or mull medium. Dead tissues are not subject to seasonal variations, and annual accumulation has an integrating effect. Weathering, leaching, and bacterial decomposition will work to diminish the metal contents, but signatures in mull are generally preserved.

Soil Gas

Under certain conditions, weathering mineral deposits produce gaseous emanations that can be detected by specialized measurements. Radon, for example, is produced during the radioactive decay of uranium and radium. Survey techniques measuring the alpha particle emissions during radon decay have been used extensively in the search for uranium. Helium is another gas produced during radiogenic decay and is considered by many to be also of deep-seated origin. Mercury-bearing minerals, which can include sphalerite and other sulfides, often release mercury vapor during oxidation.

The oxidation of sulfides leads to the generation of sulfur dioxide, dihydrogen sulfide (H_2S), and carbon dioxide (CO_2). Because of the consumption of oxygen in the oxidation process, the atmospheric proportions of carbon dioxide and oxygen change in the vicinity of oxidizing sulfides, and these imbalances can be measured in the soil gas to detect buried mineralization. More recently, organic gases have been used to define mineralized and alteration zones located beneath transported cover. Organic gas species are zoned relative to mineralization, with heavier reduced compounds occurring over ore and lighter, more oxidized gas species occurring at the boundaries of mineralization (Klusman 1994).

SAMPLE PREPARATION

Inappropriate sample preparation can completely nullify the careful work of the sampler who has invested time and expertise in the selection and collection of representative material. Furthermore, it is impossible to restore the integrity of the poorly prepared sample by enterprising analytical treatments and interpretational procedures.

Drying

Prior to mechanical treatment, surficial and rock geochemical samples have to be dried. In some climates, this can be achieved by exposure to the sun, but most samples are dried in drying ovens. Heating these ovens to temperatures in excess of 70°C (160°F) can lead to the loss of volatile elements (Hg, As, Sb) that may be of value to the exploration program. Organic samples, including humus, may be dried in a conventional drying oven or in a microwave oven at temperatures not to exceed 70°C (160°F).

Sieving and Crushing

After drying, surficial samples should be agitated and disaggregated to achieve complete separation of component

particles without crushing. This can be done with a pestle and mortar or other suitable non-contaminating mechanical device that can be thoroughly cleaned between samples. In earlier geochemical prospecting work, orientation studies on soils, stream sediments, and other surficial materials revealed that the separation of –80 mesh material for analysis was appropriate in many surveys. With some elements, such as those commonly concentrated in residual minerals (W, Sn) and other elements dispersed in weathering products in arid/semi-arid environments, fractions coarser than 80 mesh give superior geochemical patterns with better contrasts. Surveys in glacial till environments benefit from the analysis of the –240 mesh (63 μm) or even finer fractions. Careful size fraction analysis during orientation will not only indicate the most appropriate fraction for routine work, but the metal distribution throughout the sample reveals invaluable information on metal behavior for interpretation purposes. Sieves must have a non-contaminating composition. Those most frequently used are made from stainless steel.

Heavy Mineral Separations

Heavy mineral separations from stream sediment, glacial till, and rock samples commonly utilize the liquids tetrabromoethane (specific gravity [sp gr] 2.9) and methylene iodide (sp gr 3.3). More recently, solutions of sodium polytungstate, with specific gravity ranging from 1.0 to 3.1 according to dilution, can also be used to avoid the disposal issues associated with tetrabromoethane and methlyene iodide. Usually these liquid separations are carried out on coarser sample size fractions (e.g., –30 + 80 mesh), but separations can be made on material down to 200 mesh if the metal of interest is present in the finer size fractions. If necessary, heavy mineral concentrates can be further subdivided by electromagnetic separation into magnetic, paramagnetic, and nonmagnetic fractions and analyzed separately.

Vegetation Samples

Ideally, vegetation and humus samples should contain no clastic material. Dust on leaves and twigs can be removed by rinsing with demineralized water, but humus material is rarely 100% organic. The presence of excessive clastic material may dilute or contaminate the metal content of the organic material and adversely interfere with lower detection limits possible on organic material using the neutron activation analytical technique.

After drying at temperatures of less than 70°C (160°F) in a standard drying or microwave oven, vegetation samples are macerated in a Wiley mill to a –2 mm mesh size. This material is compressed into either 8.0- or 30.0-g pellets for direct analysis by neutron activation or can be ashed in a muffle furnace at 450° to 470°C (842° to 878°F) for approximately 15 hours. The plant ash is then analyzed in a similar manner to clastic sample material.

Rock Samples

Some surveys call for the analysis of specific minerals in rock samples. Such minerals may include magnetite, biotites, feldspars, and sulfides that may be concentrated by magnetic separation or with heavy liquids from the most suitable size fraction, after appropriate sample treatment (Govett 1983).

Most rock samples collected in geochemical programs are analyzed for their whole rock or trace element contents. This requires crushing and pulverizing and, if the contained metals of interest are heterogeneously distributed (e.g., as with coarse gold), the principles of sampling theory must be observed (Pitard 1993). The pulverized whole rock product that is commonly analyzed has a grain size of less than 150 to 200 mesh. The sample is rarely reduced to this fine grain size in its entirety, but, as a general rule, the finer the sample can be crushed or ground prior to sample splitting, the more representative the split is likely to be.

The emphasis on gold exploration in recent years has focused attention on sample heterogeneity. This has stimulated some significant developments in the design and adaptation of crushing and grinding equipment for improved sample preparation. In addition to sample representativity, the geochemist must be concerned with sources of contamination in the sample preparation equipment. Grinding plates and blades of different composition are available so that the obvious contamination from chrome steel, tungsten carbide, and other materials can be avoided. Similarly, loss of sample representivity through *smearing* of native copper, native gold, molybdenite, or other soft material on pulverizer plates can be avoided in many instances by the selection of alternative equipment with a different comminuting action.

Because sample weights analyzed in geochemical work are small (0.1 to 50 g [0.0002 to 0.1 lb]) compared with the original sample size, sample homogeneity is important. Improved mixing can be achieved, when necessary, by careful blending and the pulverization of coarser fractions to the –200 mesh size.

ANALYTICAL TECHNIQUES

Analytical Methods

A prime requirement for cost-effective geochemical exploration surveys is the availability of analytical procedures capable of high productivity, low detection limits, high precision, and acceptable accuracy. These criteria were met in the 1950s by a series of colorimetric analytical techniques with productivities ranging from 20 to 100 samples per day. Technological developments have subsequently led to the introduction of atomic absorption spectrophotometry (AAS), inductively coupled plasma (ICP) spectrometry , X-ray fluorescence (XRF), and instrumental neutron activation analysis (INAA) with far greater analytical sensitivity and vastly increased productivity.

These analytical techniques provide accurate, precise determinations with detection limits of less than 5 ppb to 1 ppm for many of the elements commonly measured in exploration geochemical surveys. The ICP-mass spectrometer instrument is capable of even greater sensitivity. Comparison of some of these analytical methods is shown in Table 3.4-2.

In the minds of the inexperienced, the spectacular analytical capabilities of these instruments often overshadow the critical importance of the sample preparation and sample decomposition stages of sample treatment.

It was emphasized in the preceding section on sample preparation that the most informative fraction of a geochemical sample may be the heavy mineral fraction, a relatively coarse fraction (–30 + 80 mesh), a fine fraction (–80 mesh), or a very fine fraction (–250 mesh), depending on the material being analyzed or the particular property of interest in a sample. If an improperly chosen solid sample material is correctly analyzed, the results can be no better than mediocre in the context of the exploration program and may even be misleading.

Table 3.4-2 Comparison of characteristics of analytic methods

Method	Capital, $	Multielement	Determinations per Day	Comments
Colorimetry	1×10^3	No	20–100	Very simple; adaptable to field
AAS	2×10^4	No	500	Easy to set up; several elements can be determined on same solution but not simultaneously
ICP	$1–3 \times 10^5$	Yes	>2,000	Requires skilled analyst to supervise plus computer; sample in solution
XRF	$1–5 \times 10^5$	Yes	>1,000	Requires skilled analyst to supervise plus computer; solid sample analyzed
INAA	$0.8–20 \times 10^6$	Yes	<3,500	Requires skilled analyst to supervise plus computer; solid or liquid sample analyzed

Source: Adapted from Fletcher 1987.

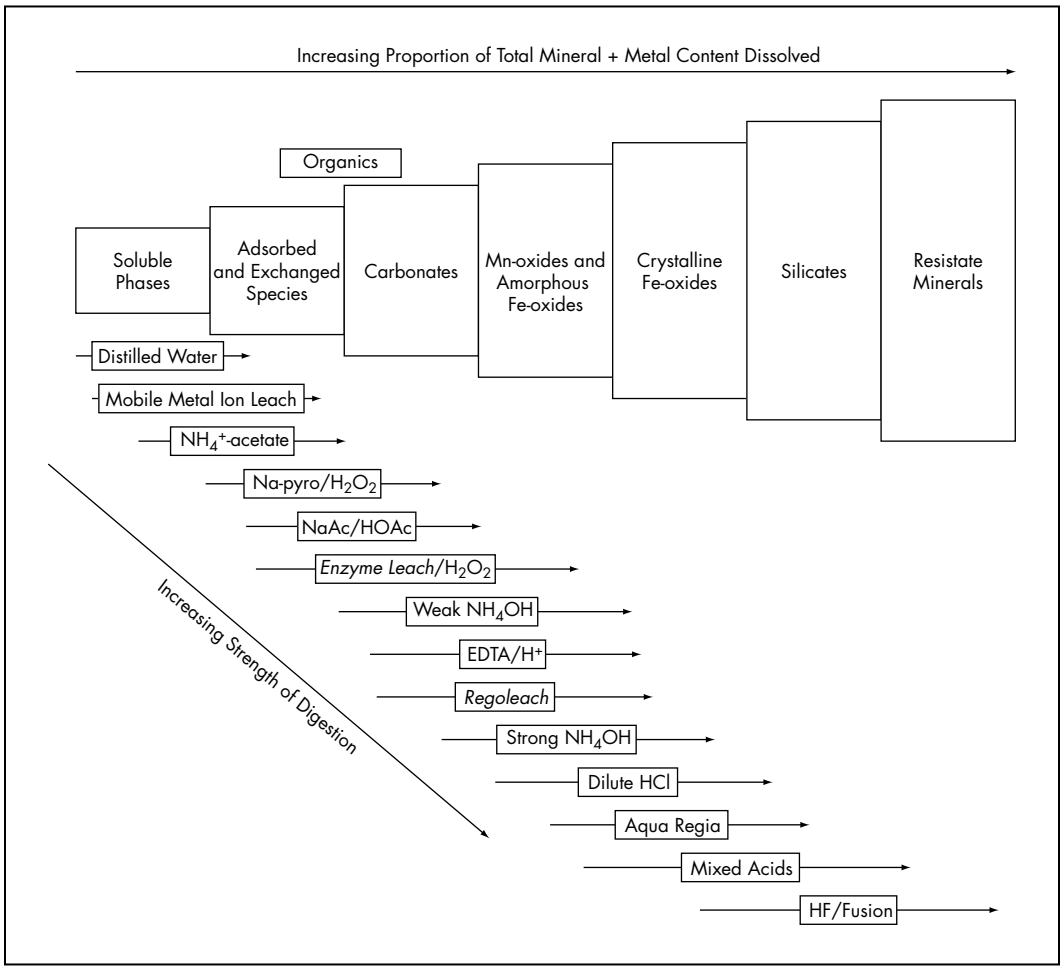


Figure 3.4-3 Relationship between various selective geochemical extractions and the extent of regolith mineral components attacked

Digestion Methods

Sample decomposition presents even greater complexities. The newer analytical methods require that elements of interest be introduced to the instruments in solution. The choice of method to take elements into solution requires careful consideration of the mineralogy of the material (Figure 3.4-3). Strong decompositions of a geochemical sample can be achieved through treatment with hot concentrated acids or by fusion. Some resistant minerals (e.g., chromite) may not be soluble in hot concentrated acids but are broken down by fusion.

The amount of metal extracted by the strong decompositions achieved with nitric, hydrochloric, or perchloric acids or their mixtures is not total and will vary with the mineralogy of the sample. Hydrofluoric acid is the only acid decomposition medium that will dissolve silicate minerals in the typical clastic or lithogeochemical sample. Even with acid mixtures, extraction efficiency will vary with the acid, sample-to-acid ratio, and the duration and temperature of extraction (Fletcher 1981). Of the fusion techniques, lithium metaborate can be effective in attacking resistant minerals and extracting specific elements.

Comparison of the quantities of metals extracted by techniques capable of different degrees of decomposition can be informative. Metals in silicate lattices of rock-forming

minerals commonly constitute the background or threshold level of the geochemical sample material. The mineralization component of an anomalous sample is generally contained in sulfides, iron or manganese oxides, or in adsorbed positions on clay minerals. The metal on the clays, and that contained in the sulfides and hydroxides, is more easily extractable than the background/threshold component contained in the rock-forming minerals. Stronger decompositions, which break down the rock-forming minerals, will effectively dilute the anomalous metal components by releasing the lower concentrations of metal in the background/threshold component. Partial decomposition techniques utilizing cold or weaker acids and other reagents that do not break down the rock-forming silicates enhance the more readily extractable mineralization component, resulting in a much greater contrast between the anomalous and background values in the survey. In numerous geochemical surveys, therefore, the data from partial decompositions, when ratioed against total decompositions, can be much more definitive in target delineation than data from strong or total decompositions. Examples of different decomposition techniques and the corresponding mineral phases digested into solution are shown in Figure 3.4-3.

Sequential extractions are series of digestions of progressively increasing strength, which can be used to selectively remove mineral phases in succession within a sample. This protocol has been used to successfully delineate copper mineral phases in copper deposits and delineate ore types for mineral processing (Parkison and Bhappu 1995).

Gold Analyses

Gold can be analyzed using geochemical or assay methods. For reconnaissance and follow-up programs, gold can be analyzed using *geochemical methods* including aqua regia digestion with an inductively coupled plasma–optical emission spectrometry (ICP-OES) finish. For resource evaluations, gold should be analyzed using traditional *assay methods,* including 30-g fire assay digestion followed by an ICP-OES or AAS finish if the grades are below 2 ppm and by gravimetric finish if grades exceed 2 ppm. This would be true for any element of ore grade concentrations. If the samples are to be analyzed and incorporated into a resource evaluation, the assay methods are the preferred method to determine concentrations for the element(s) of choice in the resource determination. If a nugget effect exists, *metallics* assays should be used to establish grades.

QUALITY CONTROL

In addition to observing the basic principles of orientation, sampling, sample preparation and analysis, adequate assurance of analytical quality, representativity, accuracy, and precision are essential when all data are evaluated. The term *QA/QC* is used in discussing quality control programs. QA indicates quality assurance, which is the plan for implementing a quality control program. Quality control is the analysis and evaluation of the results of the analytical control program.

Quality controls should be introduced at all stages in the geochemical program. Specially prepared certified reference samples (standards) with known metal contents provide essential material for checking the accuracy of an analytical laboratory on a batch basis and also for monitoring analytical drift with time when numerous batches are forwarded to the same laboratory over the life of a project. It is important to know that analytical data from all stages of a project are comparable without any significant bias between analytical batches. The reference materials should be prepared from a similar matrix as the project samples using a vendor accustomed to preparing certified reference materials. Failing that, personnel responsible for monitoring QC programs should acquire certified reference materials (standards) from a reputable source.

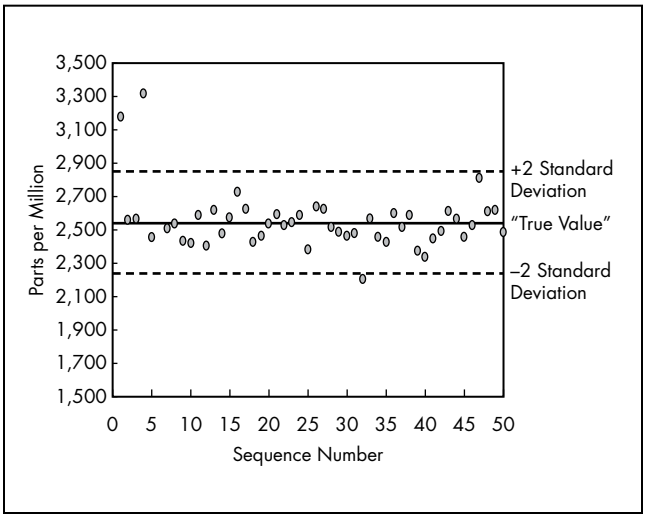


Figure 3.4-4 Standard quality control chart

Duplicate sampling can be used at every level where the sample volume is reduced and then subsampled to measure sample, sample preparation, and analytical reproducibility. Duplicate sampling of the designated material at the same sample site produces two samples (sample duplicates) that will give a measure of the sample reproducibility. Splitting of a subsample after the appropriate sample preparation (preparation or crush duplicates and pulp or analytical duplicates) will provide information on sample preparation reproducibility and analytical variance.

The accepted practice is to include a suite of standard reference samples, as well as sample, preparation, and analytical duplicates, to provide a 10% QC volume within every analytical batch. For each batch, the client should include a higher-grade standard, a lower-grade standard, a coarse blank that needs to undergo sample preparation, a sample duplicate, and an analytical duplicate as a minimum. Where samples are being submitted to determine ore reserves, the frequency of standard insertion should be greater.

Routine scanning of results from these standards and duplicates will give an immediate indication of unsatisfactory accuracy and precision and sample inhomogeneity. A set of pass/fail criteria should be established for the project. Acceptable industry practice recommends that standards should report within ±2 standard deviations of the mean using the mean and standard deviations determined from the reference material (standard) certification process. Figure 3.4-4 illustrates a QC chart for presenting standard analyses. Samples can be arranged by analytical sequence number or date—in this illustration by sequence number—to display accuracy. Lines indicating the certificate value (solid line) and ±2 standard deviations (dashed lines) are also plotted. Those analyses that exceed the +2 standard deviation pass/fail criteria are considered failures, which need to be investigated.

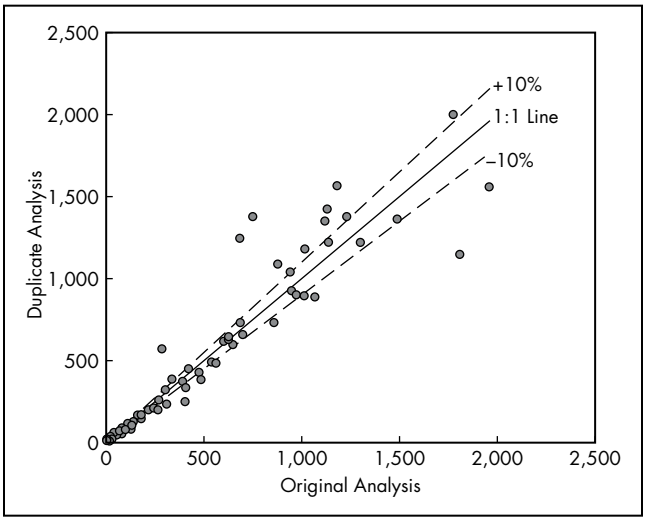


Figure 3.4-5 Duplicates quality control chart

Generally, sample duplicates should be reproducible to within ±20%, preparation duplicates to within ±15%, and analytical duplicates to within ±10%. Figure 3.4-5 illustrates a QC chart used for presenting results of duplicate analyses—in this case, for analytical duplicates with a precision envelope of ±10%. Any control samples exceeding the pass/fail criteria should be reanalyzed and reevaluated.

Because it is the objective of the geochemical laboratory to produce good and reliable data and maintain an ongoing, profitable customer relationship, quality control is of concern to both the laboratory and the exploration client. If irregularities are noted, the laboratory should be contacted immediately and the discrepancies discussed. Every reputable laboratory will have its own standard reference samples and other controls and should rerun samples if the quality of its output is questioned. If erratic analytical results are experienced with the duplicate and batch samples but not with the standard reference materials, sample inhomogeneity (*nugget* effect) is indicated. Experienced commercial laboratory personnel can assist with these problems by utilizing special preparation techniques designed to overcome sample inhomogeneity.

INTERPRETATION PRINCIPLES

Geochemical interpretation does not begin only after all samples have been collected, prepared, and analyzed. Interpretive methodologies develop progressively throughout a geochemical project. The patterns observed during the orientation program are directly related to the dispersion characteristics of the metallic elements, the nature of the overburden, and the overall general geochemical environment. Such recognition contributes to the formulation of interpretation procedures, and this understanding directly affects the sampling, sample preparation, and analytical procedures that are selected for the geochemical program.

The cumulative experience of geochemical behavior gained from orientation surveys and case histories in a broad spectrum of environments has enabled geochemists to compile models that represent dispersion behavior in a wide variety of landscape configurations. The simplified example in Figure 3.4-6 illustrates the formation of geochemical anomaly as patterns related to natural factors within the landscape. All dispersion models have certain common features (Hoffman and Thomson 1987):

- A body of mineralization or another source that may mimic mineralization
- The relative distribution of bedrock, overburden, soil, groundwater, surface water, vegetation, and other factors
- Highlighted dispersion pathways related to mineralization and anomaly formation
- Preferential sites or geochemical barriers, where metals concentrate to form anomalies
- Portrayal by each model of dispersion as a series of patterns related to and controlled by a variety of identifiable natural factors

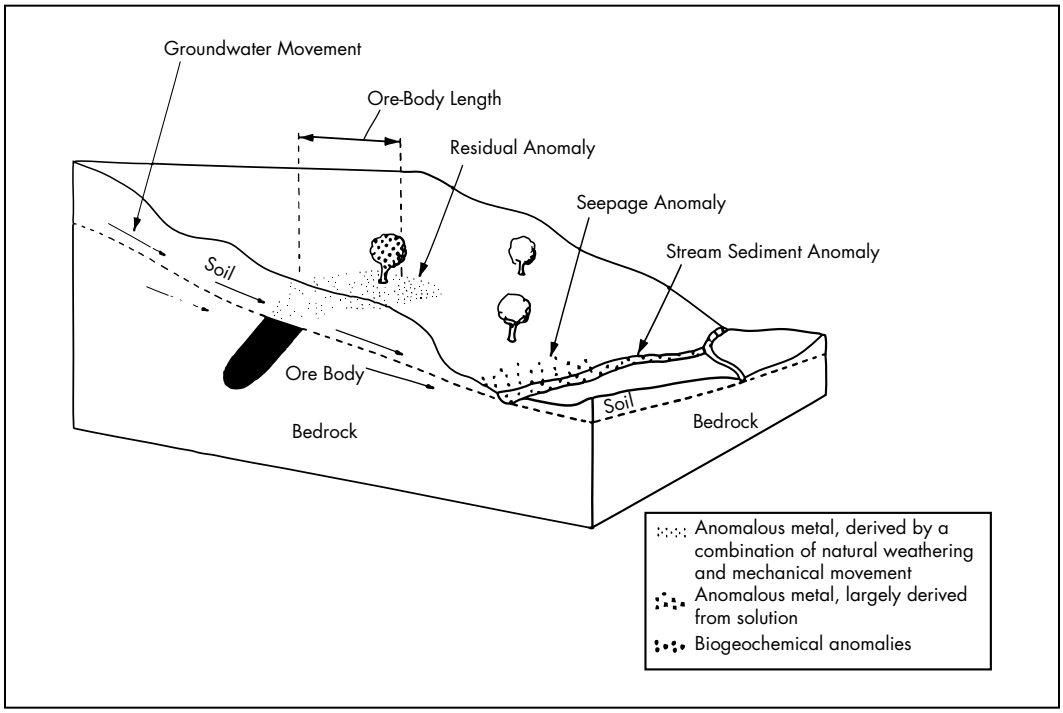
Fundamentally, geochemical interpretation involves the recognition of these patterns, the identification of the factors causing them, and the extrapolation of the patterns back to a mineralized or other source.

The importance of the preferred emphasis on patterns of dispersion rather than the magnitude of the geochemical values in units of parts per million or parts per billion can be illustrated by reference to Figure 3.4-7. Precipitation and accumulation of hydromorphically transported metal in the seepage anomaly area adjacent to the stream channel can invariably result in concentrations markedly higher than in the surface horizons of a residual soil anomaly on a well-drained slope. In such a situation, reference only to the magnitude of values would result in first priority for follow-up being assigned to the seepage anomaly, whereas recognition of the location and shape of the seepage anomaly would immediately indicate its origin and direct the interpretation to the source areas upslope.

In parallel with pattern recognition, geochemical interpretation requires a knowledge of the anomalous, threshold, and background values of the elements of interest in the survey. The fundamental observations leading to the identification of these values come from the orientation survey. In the simplest orientation scenario, background values are not influenced by the presence of mineralization and in Figure 3.4-7 are represented by the relatively homogeneous areas of low values at the extremities of the hypothetical soil traverse. Anomalous values are, in this example, the higher values peaking a short distance downslope of the suboutcropping vein zone. Weaker mineralization, disseminated in the rocks on either side of the vein zone, also gives rise to anomalous but less spectacular values, which are also influenced by the topographic slope.

The upper limit of the background population is referred to as the *threshold*, and it is clear that contouring this data at the threshold level will outline an anomalous pattern related to the mineralization in the bedrock. It is common for background levels to shift with changes in lithology or alteration systems, which also need to be determined during interpretation of the data, such as the lower and the elevated background of the volcanic rocks illustrated in Figure 3.4-7. Introduction of geological and geomorphological complexities such as those shown in Figure 3.4-7 can also produce spurious anomalies, which need to be considered during interpretation.

Conversely, during mineralization processes, some elements may be leached from the host rocks by the mineralizing fluids. In such a situation, a depletion in values below the regional background for the element will cause an *anomalous low* that is geochemically and economically significant.



Source: Hoffman and Thomson 1987.

Figure 3.4-6 Simplified model showing formation of geochemical anomalies

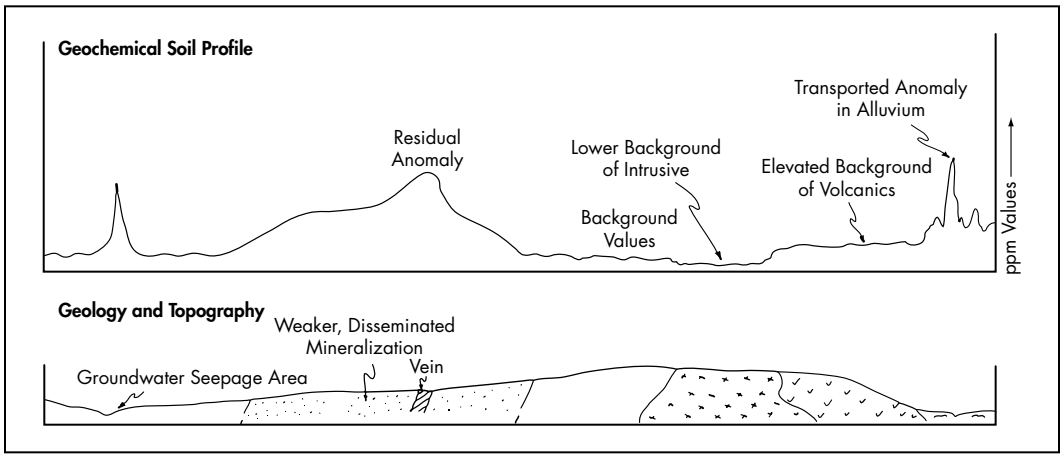
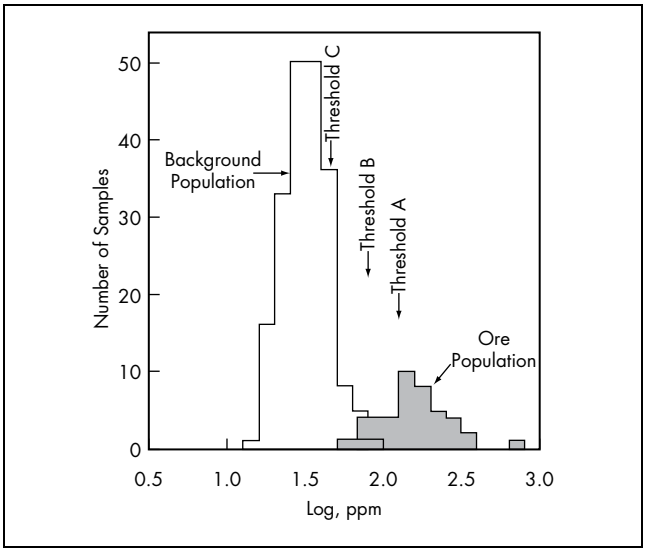


Figure 3.4-7 Hypothetical soil profile illustrating anomalies related to geological and geomorphological complexities

The interpretation of geochemical data and the correct identification of significant anomalies therefore require a fundamental awareness of the geochemical environment as presented in the geochemical model (Figure 3.4-6); a knowledge of the geology, structure, and other characteristics of the type of deposit sought; and an underlying understanding of the geochemical behavior of the elements of interest. This procedure is clearly described and illustrated with numerous actual examples in a workbook format by Levinson et al. (1987).

During the past several decades, geochemists have adapted statistical methods of evaluation to assist in geochemical interpretation. Effective application of statistical methods, whether *univariate* or *multivariate*, requires not only the same full appreciation of the geochemical environment, the geology, and the chemistry of the elements as described in the preceding paragraphs but also an understanding of the statistical technique employed. In addition, this effectiveness is also dependent on correct design, representativity, and quality of the sampling and analytical phases of the program. Given these essential understandings, statistical techniques provide useful and often powerful tools for geochemical data analysis. They can assist in explaining previously unrecognized characteristics in a data set and making significant anomalous patterns more easily recognizable.

A basic assumption applied in the statistical treatment of geochemical data is that the data are unbiased and continuous. Most geochemical data are discrete, but fortunately, in practice these discrete values commonly are sufficiently abundant



Source: Rose et al. 1979.

Figure 3.4-8 Histogram with overlap of values in background and ore populations

and close that they can be assumed to be continuous. A review of basic statistics is beyond the scope of this presentation, but a comprehensive treatment is covered by Howarth (1983).

Univariate statistics allow the geochemist to quickly become familiar with voluminous sets of data. Histograms are used to provide visual information on the total range of values, the existence of one or more populations, and possible thresholds separating background and anomalous values, as illustrated in Figure 3.4-8. Probability plots are used to determine partitioning of populations and the estimation of threshold values, as well as to detect outliers that can bias the initial statistical analyses (Sinclair 1976).

The significance of identifying populations is that individual populations represent geologic or geochemical processes. One can often separate lithology, alteration, and ore signatures by isolating individual populations and plotting these samples spatially on maps with an appropriate scale to identify a given geologic or geochemical process within the survey area.

Multielement chemical analysis of samples is extremely cost-effective, and the resulting data sets provide the opportunity to more specifically characterize features in the data based on their multielement geochemical signature. Correlation analysis is used to determine intraelement relationships, which are examined visually using scatter diagrams showing the relationships between individual element pairs.

Multivariate data analysis is used to investigate relationships between elements and can include other types of exploration parameters based on geological, geophysical, or other data. Based on this information, it is possible to investigate intraelement relationships using a variety of techniques (principal component, factor analysis, cluster analysis, multiple regression, discriminate and characterization analysis) (Howarth 1983). Elements that are highly correlated with one another represent geochemical associations, which indicate geologic or geochemical processes. Plotting coincident anomalies of these highly correlated elements spatially on maps shows where the process is operative in the survey area.

Figure 3.4-9 illustrates the principles discussed previously for a property prospective for sediment-hosted gold mineralization. Figure 3.4-9A displays the anomalous population of gold from a soil grid. Several gold anomalies occur throughout the survey area. Figure 3.4-9B displays an imaged gold geochemical signature (of Au+As+Sb+Hg) as determined by principal component analysis of multielement analyses on soil samples. The geochemical anomaly presented in the images is controlled by fault intersections, which acted as conduits for mineralizing fluids and favorable stratigraphy where mineralization was deposited. The gold geochemical signature (Figure 3.4-9B) indicates the extent and controls of the mineralizing system. The anomalous gold image (Figure 3.4-9A) indicates that the highest population represents gold deposited at structural intersections of northwest- and northeast-trending faults.

OTHER GEOCHEMICAL PROSPECTING TECHNIQUES

The remarkable advances in new technology have provided numerous avenues of research and the development of new methods that can be applied in mineral prospecting. Usually, each new method experiences a breaking-in period during which the technology is applied in a variety of situations, leading to a better understanding of the properties being investigated and refinements in interpretation procedures.

Before these methods are described briefly, it is appropriate to note that certain well-established techniques of prospecting are still as effective today as they were in the earlier days of mining. Such techniques include the skillful use of heavy mineral panning, particularly in the search for gold, platinum, cassiterite, tungsten, barite, and other resistant metals and minerals of high specific gravity. Although undocumented, panning may be responsible for the discovery of more ore and mineral occurrences than any other technique.

The ultraviolet light, or blacklight, method is another established technique with specific application in the search for naturally fluorescent minerals such as scheelite, willemite, and fluorite.

Portable XRF analyzers, initially developed in the 1960s, today use both radioisotope sources and excitation tubes to irradiate the sample material with gamma rays (Glanzman and Closs 2007). The resultant X-ray spectrum is selectively filtered, and specific elements can be measured directly at the outcrop, the drill site, or the underground face in concentrations as little as in the low parts-per-million range for selected elements.

The penetration of the sample surface by X-rays emitted by the radioactive source is limited and generally less than 3 mm (0.1 in.). The accuracy of any determination is dependent, therefore, on the sample surface exposed to the detector. Measurements on unprepared rock surfaces, whether the sample is a rock face, a hand specimen, or piece of drill core, display greater variation and represent, at best, a semiquantitative measurement. To obtain quantitative results, it is necessary to homogenize and mix the sample through grinding and pulverization. The sensitivity of the portable XRF analyzers is generally less than more conventional geochemical or assay techniques, but nevertheless, the positive identification of metals of interest in the field contributes to an increased prospecting effectiveness. The metals that can be identified and quantitatively measured by the portable analyzers include barium, copper, iron, lead, molybdenum, nickel, silver, titanium, tungsten, tin, zinc, and zirconium.

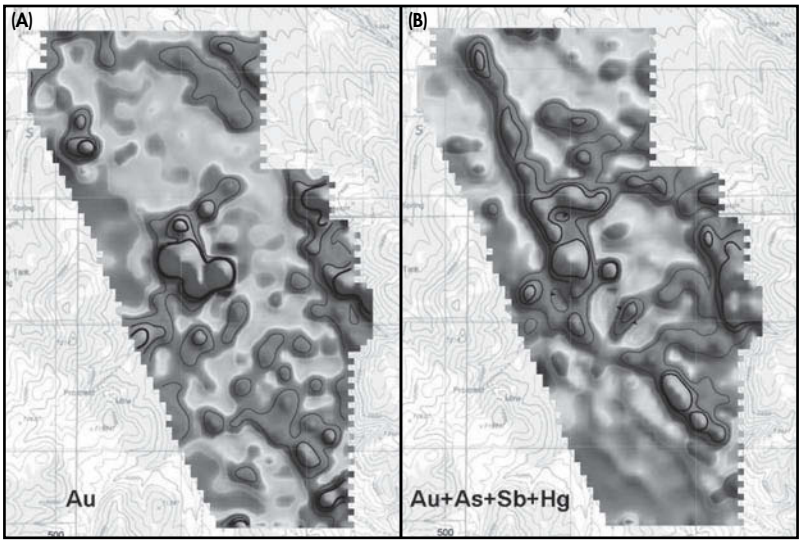


Figure 3.4-9 Contoured images of (A) anomalous gold population and (B) multielement Au+As+Sb+Hg geochemistry association, as determined by principal component analysis

Portable infrared mineral analyzers are field spectrometers that measure infrared reflectance spectra of core and crushed rock samples, which allows the identification of minerals. These instruments have been used extensively by those exploring alteration systems associated with porphyry copper deposits and epithermal gold deposits.

MINE SITE APPLICATIONS OF GEOCHEMISTRY

This section provides an overview of applied geochemistry and highlights specific aspects of geochemistry that impact resource development and mine site responsibilities of geologists and mining engineers.

Sampling

Representative sampling of mine materials is one of the most important and difficult tasks. When done poorly, it generally leads to erratic and noisy data, and poor decisions. The purpose of sampling is to provide a representative estimate from which valid decisions follow. Appropriate sampling starts with geology and encompasses both what and how. It also includes mining considerations—for example, the bench height and block size used to estimate the deposit characteristics for mine planning purposes.

Once the initial sample is obtained, subsampling aspects need to be considered to maintain the representativity of the sample prior to the measurements of properties under consideration (Springett 1984; Pitard 1993). Alternatively, sample separation of the original material collected can be undertaken for specific informational purposes. For example, when a new section of the mine is opened, samples from the new section should be tested to determine if any changes in ore mineralogy or metal content might affect recovery. This process must also follow protocols that ensure the representative aspect of the subsample.

Chemical Analysis

Chemical analysis of samples can provide information about the anticipated commodities to be recovered, elements that may potentially provide by-products (thereby enhancing value), elements that may be penalty metals during processing or be environmental liabilities, and input to overall mine planning and scheduling. Initial investigations of mineralization types should be assessed via high-quality multielement data to identify those elements that may be either value enhancers (depending on deposit type—e.g., silver or cadmium) or potentially penalty constituents (e.g., arsenic, mercury, or cadmium). Analytical methods can be selected to aid in assessment of mine design—for example, fire assay data to obtain estimates of total metal versus cyanide extraction of gold—to address the options of either milling or heap leaching plant design.

Mineralogy, particularly clay mineralogy, is also important to assess. It can affect rock competency and stability, milling and heap leach plant design, and degree of weathering and oxidation in mine and tailings facilities.

Quality Assurance and Quality Control

As noted earlier, QA deals with selecting the appropriate analytical method for the issue at hand. An example is the choice of selecting total analysis of gold via fire assay versus cyanide extractable gold as input to plant type selection. QC of analytical assessment of data is essential in making initial production and development decisions, in construction of the database on which project economics and financing are based, and in reconciliation and ongoing project management. Typically, production analyses are carried out on-site. QC responsibilities often fall on mine staff geologists and mining engineers who use these data on a daily basis. External independent checks should be a routine part of this process. Properly designed QA and QC protocols quantify, monitor, and document how well important variables are known and where corrective action is required.

Deposit Zoning and Mine Sequencing

The distribution of mineralization types in an ore deposit is significant in mine planning. This knowledge allows for consideration of ore blending during mine extraction sequencing to potentially mitigate the negative impact of included penalty metals.

Baseline Studies

Initial exploration and development studies, ongoing assessment during production and reclamation, and monitoring during postproduction to satisfy environmental regulatory requirements are all based on correct assessment of the site before, during, and after mining. A departure point for these studies is the geological and geochemical nature of the material being mined. Geoenvironmental models developed initially by the U.S. Geological Survey (du Bray 1995; Filipek and Plumlee 1999) provide a starting point for identifying potential problems and developing a plan to address these issues at the beginning of a project rather than later when a more expensive retrofit is often required.

Communication

Applied geochemistry has a great deal to offer to the efficient, effective, and profitable execution and ongoing operation of a mine. Many different professional skills are required. Each professional has a responsibility to appreciate the contribution of fellow professionals. The overall management and responsibilities commonly fall to mining engineers. Integrating knowledge and effective communications with fellow professionals will allow each to more efficiently and successfully carry out individual responsibilities.

ACKNOWLEDGMENTS

This chapter in the previous edition of the handbook was originally written by J. Alan Coope, who is now deceased. The current chapter has been restructured and updated by J.A. Jaacks and L.G. Closs. During the preparation of this summary, the authors benefitted considerably from wise and pertinent comments from a great number of people. Their contributions are gratefully acknowledged.

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PART 4

Deposit Assessment

CHAPTER 4.1

Geological Data Collection

A.J. (Joe) Erickson Jr. and Jeffrey T. Padgett

PROJECT AND MINING GEOLOGY

Exploration geology focuses on activities that lead to the discovery of a potentially valuable mineral deposit, recognizing that the deposit may develop into a mine. *Project geology* following discovery focuses on the more detailed evaluation of the mineral deposit, up to and including feasibility studies, and evolves into *mining geology*, which is directed toward the planning and operation of the mine. Methods employed for project and mining geology require an engineering discipline to ensure that the data and information provided are appropriate for use in project evaluation and mine production. The most important aspect of the geologist's work is to discover and delineate the mineral deposit and to prepare a detailed definition that describes the deposit's location, size, shape, variability, and grade continuity. This detailed definition ensures accurate and reliable mineral resource and reserve estimates.

Every mineral project or mine is based on a geologic entity—an ore (mineral) deposit. A well-defined mineral deposit and its geologic characteristics are the only aspects of a project that cannot be altered. Mine plans can be modified to exploit the deposit using alternative approaches that often yield similar results. Varied processing methods also are typically available for producing comparable results. Major modifications can be made to other aspects of a project without substantially changing project economics. Regardless of the approach taken, however, all engineering and metallurgical aspects of a mine must be accommodated to the specific location and unique geologic characteristics of the deposit being evaluated.

Geologic data and interpretations form the foundation for both mine evaluation and mine production, providing essential information for estimating resources and reserves and for mine planning and process design or control. Proper geologic work requires a keen awareness of and an ability to anticipate the technical requirements of mining engineers, metallurgists, geotechnical engineers, hydrologists, and other technical specialists who all rely on the geologic data. Geologists are integral members of the project evaluation or production team. Presentation of pertinent data in a usable format and frequent communication of new geologic knowledge to the other technical specialists are integral components of the geological program. Thus, excellence in written and oral communications of geologic information is essential to guide the evaluation process and to achieve production goals.

The ultimate objective of the exploration geologist is to find ore; that of the project geologist is to define the ore; that of the mine geologist is to keep the mine in ore. Project geology sequentially follows exploration, after a discovery of potentially economic mineralization has taken place and evaluation and development proceed. Mining geology begins on commencement of production, though sometimes it has been defined to include project geology. By comparison, project geology combines many elements of both exploration and mining geology through the delineation of mineralization and the estimation of resources and reserves. With the large number of mineral deposits that have been developed in recent decades, project geology has become recognized as a separate discipline requiring special knowledge distinct from either exploration or mining geology. Feasibility studies and development decisions involving large capital outlays require great accuracy in mineral deposit definition and resource and reserve estimation. The greatest single cause of mine failure worldwide is unreliable reserve estimates. In addition to involvement with these aspects of the work, the project or mine geologist is expected to identify various metallurgical ore types as well as potential ground stability and hydrological problems.

Ore reserves are the basic wealth of mining and minerals companies and the principal source of future earnings. A mining company's existence, growth, and survival depend on its ore reserves. It is necessary to periodically review and update resource and reserve estimates to account for changes related to additional ore discovery, mine ore depletion, upgrading or downgrading of resource categories, or fluctuations in economic conditions. During the 1990s, the international mining industry established standardized definitions for the terms *resources* and *reserves* that are generally accepted throughout the world. In 2005, the Society for Mining, Metallurgy, and Exploration

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(SME) established guidelines in *The SME Guide for Reporting Exploration Results, Mineral Resources, and Mineral Reserves*, which it revised to accommodate requirements of the U.S. Securities and Exchange Commission (SME 2007). The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) also adopted its rules in the *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*, which uses the standard international definitions (CIM 2003). The CIM guidelines were subsequently codified and are generally accepted throughout North America, and in many countries elsewhere in the world where North American mining companies are established, to ensure best-practices project management and to comply with National Instrument NI-43-101, *Standards of Disclosure for Mineral Projects* (2005).

Additional support provided by geologists to other specialists in mine evaluation and production includes

- Gathering and assessing geologic data and samples for geotechnical analysis;
- Collecting groundwater data for hydrological investigations;
- Defining the ore body and distinguishing ore grades and types for mine planning and production;
- Exploring for additional ore bodies and other materials in the district;
- Collecting samples for metallurgical testing;
- Evaluating geology and ore potential of sites designated for waste dumps, a mill, leach pads, shops, offices, and associated facilities; and
- Assisting with land, legal, environmental, and permitting studies.

GEOLOGIC DATA COLLECTION AND RECORDING

Mining is a physical endeavor that extracts some valuable resource from the earth. For a mining company to do this effectively, it is essential that company management has as accurate as possible a characterization of resource-zone geometry. A good understanding and clear representation of the shape, size, quality, variability, and limits—the geologic characteristics of the resource zone—are needed at the evaluation, development, and productive stages of a project. This characterization requires a high-quality geologic database so the geologist can provide management with the information needed for critical project decisions.

The financial success of the mining venture is directly related to the accuracy and completeness of the geologic database and the quality and understanding of the characterization that describes a resource zone. The geologist who fails to provide the best geologic characterization based on data available at the time "is delinquent in his or her responsibility for providing management with the best possible information for intelligent decision making" (Barnes 1980).

This chapter deals with basic geologic data collection principles and the need to improve the accuracy of this deposit characterization and modeling by collecting more and better-quality geologic data. General comments are provided first, followed by a review of the type of data needed. Also presented are common symbols and abbreviations typically used in geologic data collection, mapping, and core or cutting logging.

General Comments

Geologic data vary greatly within a deposit and from deposit to deposit. This applies if we are concerned with metallic, coal, or nonmetallic commodities. Specific geologic features differ considerably, and likewise the importance of any specific feature varies from deposit to deposit. Geologists are faced with the task of collecting many types of geologic data and seldom know in advance which features are critical; therefore, they must collect detailed data on all features of potential importance. Data may be collected via surface or underground mapping; drilling; geophysical or geochemical surveys; or specific studies examining such features as structure, rock mechanics properties, specific gravity, alteration effect and distribution, or mineralogy.

The data may be collected using the long-established pencil-and-paper-based method of mapping, posting, and compiling. Data collected via this conventional method can be converted to digital electronic form through the use of computer-aided design (CAD) systems, computer spreadsheets, a geographic information system, and other software in computer-facilitated systems. Alternatively, the development and evolution of computer-based systems and software has enabled collection of data by direct digital means through the use of portable or pen-and-tablet computers hosting the appropriate software. In the latter method, data are recorded electronically on computer screens that host digital base maps (probably topographic maps) or, alternatively, property-specific simple grid sheets that are stored in the computer and commonly linked to Global Positioning System (GPS) methodologies. This allows the user to create electronic geologic maps or drill logs for direct use on computer screens or to provide hard-copy printouts as desired.

Collecting adequate geologic data requires a great deal of time, effort, and expense; and with either data collection methodology, similar geologic skills are required. It is essential that data collection systems be planned in advance so that all data and descriptions are systematically gathered to ensure high quality and completeness. Data collection should employ some standardized system or format to ensure consistency, accuracy, neatness, legibility, objectivity, quantification, and timely completeness. Fact must be discernible from inference. Personnel should be trained in the requirements imposed by either the conventional or computed-assisted or computer-based recording system. It does little good to have two people, biased by personal experience, making data recordings on the same outcrop or sample, the ultimate records suggesting two entirely different geologic units. Quantification of the geologic variable should be done wherever possible. This requires estimation, which—although imprecise—is far superior to vague generalizations such as "much" or "strong amounts" of some particular mineral species.

Accuracy is a recurring theme in the foregoing discussions of any of the numerous types of geologic data normally collected. This accuracy requirement implies that an appropriate QA/QC (quality assurance/quality control) system or algorithms of some type are in place to ensure that the data collected are checked for accuracy and correctly entered into databases, and that automatic backup is available. Not uncommonly, these algorithms are present in data entry programs that check to ensure that drill-hole data are entered sequentially from top to bottom and that transposition errors are minimized. Some data entry programs contain algorithms that identify and flag for review or reduction assay values that appear high or out of normal for the type of deposit under study. Numerous other checks and balances of this nature exist.

Adequate security and restrictions against database access should be in place to prevent corruption of data by inappropriate access or improper data handling or updating. Geologic data—the facts—must remain available and in an unaltered and periodically updated and secure form to ensure that as interpretations are developed, they do not inadvertently become part of "the facts."

Geologic data are extremely important and costly to obtain, and they are essential for proper interpretation, evaluation, and ultimately mining and processing ores from a deposit. The data collection may be a one-chance occurrence due to constraints imposed by mining or distance. A second observation of a critical area may be impossible as the drift or bench may be mined out or the core crushed for assay. If data are collected in the manner just described, they will provide a useful record that is timeless in character.

Ultimately, it is essential that geologic data be converted to some digital format. This conversion will enable their incorporation into the database to support interpretations, control resource and reserve estimation, characterize mineralogical and metallurgical conditions, or clarify other mining-related issues.

Further observations on these topics are made by McKinstry (1948) and Malone (1995). Malone discusses the roles of the geologist and the mine geologist; the importance of comprehensive, standardized mapping and core logging; areas where geologic input is essential in operational support; and positive and less desirable aspects of computer-based logging and mapping systems. He points out that while "computer systems do not reduce the geologic effort and skill required for accurate mapping and logging," they do, however, "provide much greater flexibility and speed in manipulating and using the data." Malone also suggests that

> it is never good for management to try to save money by reducing the quality of geologic mapping. Attempts to constrain mapping to fit an artificially simplified geologic model are always counterproductive in the long run. Accurate geologic mapping, faithfully recording the exposed geology (the geology that is revealed not what is expected), is the best insurance a mine can have against unexpected disasters.

The practice of geology is not easy and requires a great deal of patience, diligence, discipline, and simply hard work. Adequate training and a high level of professionalism are required.

Required Data

Every effort must be made to observe, objectively record, and describe all geologic features that may be of importance in characterizing the size, shape, and variability of the resource and its associated environment under study. Broad categories of data to be collected routinely are location information and data on lithology, mineralogy, assay samples, alteration, and structural and rock competency. As experience is gained in specific areas, deposits, or suites of rocks, the capability to subdivide various units into key subunits typically will be developed. This will enable the geologist to improve description, correlation, projection, and understanding of the genesis of the deposit. More importantly, it will allow for superior resource estimation and will improve recommendations needed for management decision making.

Geologic Data Collection: Key Features

For the following categories of data collection, the keys are careful observation and clear description.

Location Data

Sample, map, mine, or drill location should be recorded on each sheet. This may include geographic data such as state, county, section, township, range, latitude, longitude, coordinates, elevation, mining district, mine, pit, bench, level, working, claim, claim corner, or any and all information that will clearly identify the unique location of the geologic data points. Data cannot be used if the geologist does not know where they came from.

Lithologic Data

Typical data to describe rock, sample, or unit should include color, texture, mineralogical characteristics, lithology, and rock type. Appropriate descriptive modifiers, stratigraphic information if known, top and bottom data, age relationships, and general gross features such as hardness, competency, and bedding characteristics should be included. Subjective generic terms should be avoided unless well established or qualified to distinguish inference from observable facts. Primary sedimentary structure and sedimentologic features (e.g., bedding, laminations, casts, soft-sediment deformation, graded bedding, burrows, bioturbation, and fossil content, as well as banding, foliation, and lineation with appropriate attitudes) should be noted where possible.

Structural Data

Secondary structural features that postdate rock formation should be described. Data should include a clear description and attitudes of joints, fractures, and faults; breccias with quantitative description of selvages, gouge zones, and fragment size; and healed or re-cemented character of breccias. Information related to rock competency such as rock quality designation (RQD) and natural fracture frequency is important. These data are best if collected at the drill site prior to boxing of core, although useful data are frequently accumulated after the core has been boxed. Folds, drag folds, crenulations, lineations, and foliation should be noted. Age relationships, mineralization association, and overall effect on rock mass are important. Weathering and oxidation intensity data are usually critical and commonly structure related but may be included with lithologic data. Quantification of structural data where possible is extremely important as it may play a key role in determining mineability of a deposit.

Alteration Data

Alteration data include nature, mineralogy, intensity, and distribution of features. This should include color, texture, mineralogy, intensity, fracture or vein/veinlet relationship, control, stages, mineralization association, and pervasiveness with respect to the overall effect on rock mass. Weathering and oxidation intensity are important but may be included with lithologic data. Quantification where possible is extremely useful, as is description of age relationships of various alteration features.

Mineralization Data

This category includes nature, intensity, mineralogy, and distribution of the desired resource. It should include primary and secondary classification; estimates of specific and total

quantity of various minerals; intensity; character of veinlets, vein, or disseminations; supergene features; weathering and oxidation intensity; and associated gangue mineralogy. As an example, estimates of total sulfide content, mineral and metal ratios, and gangue mineralogy are of use in deposit description, in support of metallurgical studies and testing, and in waste characterization. Vein age relationships tied to mineralogy, alteration, or lithology provide important data in understanding both zoning and grade estimates and overall deposit genesis. Assay work should include desired ore elements, deleterious elements (arsenic, etc.), and iron or sulfur or both to calculate total sulfide content for the previously mentioned waste characterization. Geologists must clearly understand the methods and the significance of sampling, sample preparation, sampling procedure, and sampling protocol. Individual ore deposits typically host multiple metallurgical ore types based on mineralogy, alteration, or oxidation, most of which are based in geology and therefore require careful geologic description and metallurgical testing to determine distributions, possible process modification, or varying mining sequences to ensure optimum recoveries.

Coal Data

In addition to standard lithologic and structural data, it is important to map or log any and all features that aid in correlation, understanding the distribution of sedimentary facies, and constructing a depositional model of the coal bed(s) and coal-bearing sequence. Detailed descriptions of horizons immediately above and below the roof and floor are critical, as are accurate measurements of depths and thickness of all units associated with the coal. Some key features include abundance and type of marine or freshwater fossils, slickenside in roof or floor rocks, the presence of roots representing old soil horizons, pyrite bands, nodules or streaks, siderite or ironstone nodules, and plant debris. Description of individual coal beds, either the banded or nonbanded groups, requires careful measurement or estimates of the banded lithotypes, vitrain, clarain, durain, and fusain content (Ward 1984). A more practical system (Schopf 1960) describes the thickness and amount or concentration of vitrain and fusain bands in a matrix of atrital coal. The latter is described by five luster levels that range from bright to dull. Description of nonbanded sapropelic coals and boghead and cannel end members relies on identification of these massive, faintly banded, fine-grained accumulations of algae or spores and usually requires a microscope for adequate description. The nature of cleats, partings, bone, and shale layers needs description and careful thickness measurements to separate net from gross coal-bed thicknesses. Coal-bed description, while straightforward, requires some supervised training to ensure adequate data recording.

Other Features

Other features that may supply extremely important information with direct bearings on mining and/or metallurgy should be recorded. This may be reasonably objective (fracture frequency, rock quality determination measurements, longest and shortest unbroken core recovered in a run) or more subjective (an overall estimate of rock strength, friability, or competency). Total sulfide content or assay is extremely important for waste characterization as well as metallurgical process development. Metallurgically significant features such as hardness, which affects grindability; grain size, which controls grinding for particle liberation; or oxidation intensity should be noted—as well as mineral species and alteration mineralogy, which affects flotation recoveries. As an example, supergene copper mineralization coatings on sphalerite provides a challenging metallurgical problem, as does activated pyrite due to similar chalcocite coatings on pyrite beneath more thorough supergene copper enrichment zones. Metallurgical personnel need to be made aware of the presence of these features. Numerous other examples could be cited. Added testing is almost always needed here; however, geologic data collections should indicate these and other potential problem areas requiring specialized study.

SYMBOLS

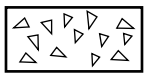
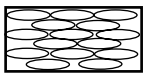
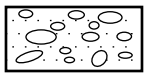
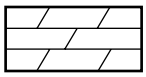
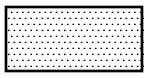
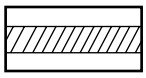
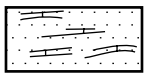
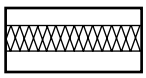
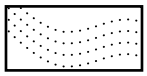
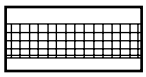
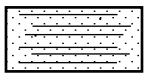
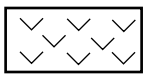
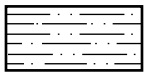
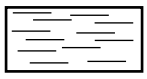
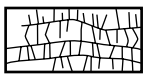
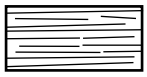
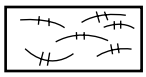
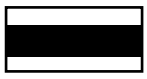
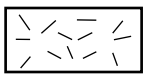
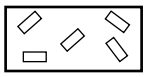
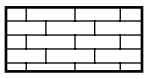
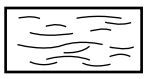
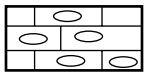
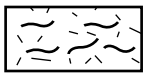
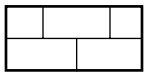
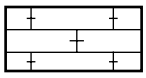
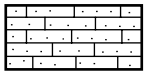
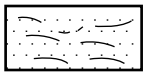
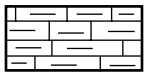
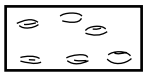
Geologic representation, geologic mapping, core logging, data compilation, and interpretation are heavily dependent on visual presentation as "geologists think and communicate best in pictures. The basis of almost all geologic work is the making of maps, plans, sections and the like. The graphical form is the most efficient way of displaying and explaining geological observations and ideas" (Dixon 1979). Standardization in the way geologic data are collected, recorded, or displayed is desirable as the concepts being portrayed are frequently difficult, and varied symbolism may cause confusion. A simple line or symbol on a map is a powerful decision-influencing tool and may have innumerable connotations, and it therefore should be presented in a clear, self-explanatory way. Symbol standardization, a very difficult objective, helps in obtaining the desired clarity.

There is no single standardized set of geologic and mine-related symbols or abbreviations in use, but the symbols compiled in Table 4.1-1 are commonly used in the United States and elsewhere. They were compiled from works by Peters (1987), Compton (1985), Dietrich et al. (1982), Berkman and Ryall (1982), Reedman (1979), Lahee (1952), and McKinstry (1948). Additionally, the first of the excellent five-volume Geological Society of London Handbook Series by Barnes (1981) is very useful and practical. Blackadar et al. (1968) illustrate symbols used in Canada, and several chapters in Finkl (1988) cover symbology and other topics on field methods.

Table 4.1-1 is not all-inclusive, but it includes many commonly used symbols. Simplicity, clarity during reproduction, and practicality should be the guides.

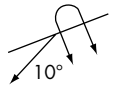
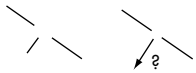
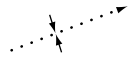
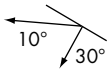
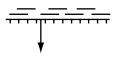
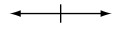
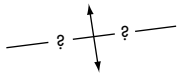
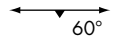
Some mapping systems, such as the Anaconda system developed in the early 1900s in Butte, Montana, United States (Brimhall et al. 2006), use black for culture, lithology, and rock data; blue for structure; and red for mineralization. This practice is convenient, useful, and readily understood. These authors also discuss the expansion of this three-color system to 10 colors to record vein types, ore, and alterations minerals to better record geologic data during the evolving study and mapping of additional ore deposits. Still other systems collect all data in black, which is quite useful for modern copying technology, although color copying equipment is readily available in most areas. Modern computers, printers, plotters, and software typically offer an endless array of colors. Any system should be well planned in advance, simple to learn and use, consistent, well documented, and should aid, not hinder, data collection and understanding.

Table 4.1-1 Common mapping symbols

Lithological Symbols			
Breccias		Chert	
Conglomerate		Dolomite	
Sandstone, bedded		Gypsum	
Sandstone, calcareous		Ahydrite	
Sandstone, crossbedded		Salt	
Sandstone, shaly		Volcanics	
Siltstone		Tuffs, breccias	
Mudstone-claystone		Flows, basic	
Shale		Flows, other	
Coal		Granite	
Calcareous shale		Porphyritic	
Limestone, bedded		Schists	
Limestone, cherty		Gneiss	
Limestone, massive		Marble	
Limestone, sandy		Quartzite	
Limestone, shaly		Serpentine	

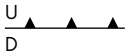
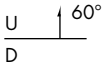
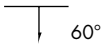
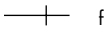
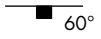
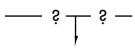
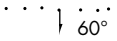
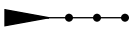
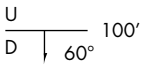
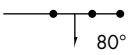
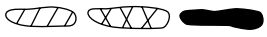
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Table 4.1-1 Common mapping symbols (continued)

Geological Symbols*			
Strike and dip of bedding plane		Anticline overturned, limb dips, axial plunge	
Uncertain		Syncline axis and plunge	
Generalized		Syncline inferred	
Horizontal bedding		Syncline uncertain	
Vertical bedding		Syncline concealed	
Overturned bedding		Syncline overturned, limb dips	
Strike, dip, lineation plunge		Dome	
Contact		Lineation plunge	
Contact with dip and lithology		Lineation horizontal	
Contact inferred		Lineation plunge and bedding, magnitude unspecified	
Contact uncertain		Vertical beds	
Contact concealed		Vertical beds, horizontal lineation	
Contact vertical		Foliation showing dip	
Anticline axis and plunge		Foliation vertical	
Anticline uncertain		Foliation horizontal	
Anticline concealed		Foliation, horizontal lineation	

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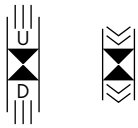
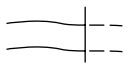
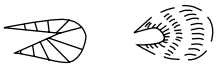
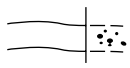
Table 4.1-1 Common mapping symbols (continued)

Geological Symbols (continued)*			
Cleavage		Thrust fault, teeth on upthrown rock	
Cleavage vertical		Fault, reverse	
Cleavage horizontal		Fault, relative displacement	
Fault and dip		Fault zone	
Fault vertical		Joint with dip	
Fault inferred		Joint vertical	
Fault uncertain		Joint horizontal	
Fault concealed		Vein	
Fault, sense, amount of displacement, dip known		Vein attitude	
Fault, ball or hachure on downthrown block		Mineralization, ore bodies showing increasing intensity	
Sedimentological Symbols			
Laminae, plane parallel		Underclay slickenside	
Laminae, curved parallel		Clasts, nodules	
Laminae, plane nonparallel		Organic material	
Laminae, curved nonparallel		Plant fossils	
Laminae, wavy parallel		Marine fossils	

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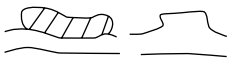
Table 4.1-1 Common mapping symbols (continued)

Sedimentological Symbols (continued)			
Laminae, wavy nonparallel		Brackish fossils	
Large cross-stratification		Pyrite	
Ripples		Load cast	
Flaser bedding		Flute cast	
Bioturbated		Scoured surface	
Rooted		Lag	
Claystone (seat rock) underclay			

Culture Symbols, Surface or Underground			
Vertical shaft		Shaft, through level	
Shaft, flooded, caved		Shaft bottom	
Inclined shaft		Inclined shaft (chevrons down)	
Portal (blocked)		Raise, winze (head)	
Portal and cut		Raise, winze (through level)	
Prospect, open cut		Raise, winze (foot)	
Trench		Chute, manway	
Pit, quarry		Working blocked	
Dump		Working filled	

(continues)

Table 4.1-1 Common mapping symbols (continued)

Culture Symbols, Surface or Underground (continued)			
Mine (abandon inverted)		Lagging, cribbing	
Sand, gravel (abandon inverted)		Stope	
Drill hole, no., (inclination)	MH31 –90°	Drill hole, no., inclination	MH31 –17°
Drill hole, no., elevation, depth	2116′ MH31 TD397′	Survey point, no., elevation at back, distance to sill	1571 –9.3

*Bedding dips show options with and without double arrowheads. The arrowhead prevents map confusion if the blocky limestone symbol is used. It is advisable to use half arrow on veins and faults as it allows clear symbol identification and shows relationships where considerable detail is involved.

ABBREVIATIONS

The extensive observations and descriptions essential in geologic work are commonly facilitated by the use of abbreviations. Standardization of these abbreviations is desirable because logs and maps may be used by numerous individuals, and they therefore require some basic commonality. Additionally, a long lag time may be involved between data recording and actual use. Abbreviations as shown in Table 4.1-2, largely abridged from Chace (1956), are fairly standard and commonly used in the United States and other English-speaking nations. Other lists of abbreviations appear in Finkl (1988), Compton (1985), Berkman and Ryall (1982), and Blackadar et al. (1968). Quantification of these abbreviations is achieved by the addition of superscripts or subscripts, as py^2 or py 2-10, which indicate estimates of 2% or 2% to 10% pyrite content in a rock. The abbreviations cp_1 and py_2 indicate 1% and 2% chalcopyrite and pyrite, respectively, whereas cp:py = 1:2 or cp1:py2 indicates a ratio of chalcopyrite to pyrite of 1 to 2. Combinations are endless. The only requirements are that the estimate be reasonable and that an explanation or legend describing the notation accompany each log, map, or project.

GEOLOGIC MAPPING

Surface or underground mapping requires uniformity and standardization as well as systematic, unbiased, and objective data collection and recording. Note-taking, abbreviation, and symbology are best if they employ a company-wide established methodology that reflects common initial training of personnel involved. In addition to being consistent with logging, to be discussed in the following section, mapping data must be accurately located and tied to known locations that preferably have been surveyed conventionally or using survey-quality GPS.

In general, regardless of the resource type, geologic mapping is for the purpose of providing data on lithology, alteration, mineralization, structure, and ground conditions, as well as analytical data (i.e., assays or coal quality) for resource evaluation. The following eight-item list summarizes the typical requirements, considerations, and steps in the mapping process. The list illustrates how one might proceed in what is essentially an exercise in detailed mineral and rock identification, characterization, and record keeping, which will be discussed in more detail later in this chapter. These steps assume a basic knowledge and understanding of the symbols and abbreviations generally used in geologic data collection.

1. Conduct a pre-mapping review based on general geologic knowledge of the area and deposit type under investigation. This is to determine the purpose of the mapping, to identify geologic parameters of probable importance identified according to guidance in the "Geologic Data Collection: Key Features" section in this chapter to consider the scale(s) to be used, and to determine the local physical geography of the area under investigation.
2. Secure base maps, air photos, grid sheets, survey information, claim locations and ownership, survey points, and geographic information system (GIS) data. Secure approval for property access as appropriate.
3. Secure typical items useful during the mapping process, such as mapping vest, compass, tapes, hand lens, and so on (the "Core/Cutting Logging Process" section lists other items), for conventional work or in support of observation entry into portable laptop computers.
4. Select recording bases as available, such as air photos or topographic maps; data may be recorded directly on the base map or on acetate overlays attached to the base. If no base maps are available, notes on observations of geologic data can be taken on simple grid sheets and locations can be determined through the use of compass and tape or GIS data. Cut selected recording base maps, air photos, or grid sheets to the size that is appropriate for the aluminum sheet holder to be used.
5. Make one or more visit(s) to the field location (i.e., deposit, exploration/evaluation area, or mine site) to make observations and collect and record geologic data with appropriate symbols and abbreviations on selected base maps/air photos or grid sheet. Alternatively, use laptop computers to record geologic data directly and construct maps using mapping software.

Table 4.1-2 Geological abbreviations

Term	Abbreviation
Igneous Rocks	
agglomerate	agg
amygdaloid	amyg
andesite	and
anorthosite	ano
aplite	ap
basalt	bas, bt
breccia	bx, bc, br
dacite	da, dct
diabase	db
diorite	di, dio
dolerite	dole
dunite	dun
gabbro	gb, ga
granite	gr
granodiorite	grd, gd
hornblendite	hbt
igneous	ig
lamprophyre	lamp
latite	la
lava	lava
monzonite	monz, moz
norite	nor
obsidian	obs
pegmatite	peg
peridotite	pd
phonolite	phon
pillow lava	pl
porphyry	po, por, p
pumice	pu
pyroxenite	pyxt
quartz diorite	qd
rhyolite	rhy, ry
sperulitic pillow lava	spl
syenite	sy
trachyte	tryt, tyt
trap	trp
tuff	tuff
Sedimentary Rocks	
argillite	argl
arkose	ark, ak
asphalt	asph
banded iron formation	bif
bitumen	bit
chert	ch, ct, cht
claystone	cs
conglomerate	cg, cong, cgl
diatomite	dtm
dolomite	dol
edgewise conglomerate	ewcg
graywacke	gw
iron formation	if
ironstone	is, ir
limestone	ls
marl	ml
mudstone	ms
phosphate	phos
quartzose iron formation	qif
sandstone	ss
sedimentary	sed
shale	sh
shaly iron formation	shyif
siltstone	slt, st
travertine	trv
Metamorphic Rocks	
amphibolite	amp, amph
anthracite	anth
ellipsoidal greenstone	el gs
gneiss	gns
greenstone	gs
injection gneiss	inj gns
magnetic graywacke	mag gw
magnetic slate	mag sl
marble	mb
metamorphic	meta
orthogneiss	orgns
paragneiss	pagns
phyllite	phy
quartzite	qte,qtzt
schist	sch
serpentine	serp,sert
slate	sl
Minerals and Metals	
acanthite	ac
actinolite	act
adularia	adu
aguilarite	agu
aikinite	ai
alabandite	abn
albite	ab
algodonite	alg
allanite	all
altaite	at
alunite	al
amalgam	amal
amblygonite	amb
amphibole	amp
analcite	anal
andorite	ad
anglesite	ang
anhydrite	anh
animikite	anm
ankerite	ank
annabergite	anna
anorthite	an
antimony	Sb
antlerite	ant
apatite	ap
aragonite	ara
argentite	arg
argyrodite	ay
arsenic	As
arsenopyrite	asp
atacamite	ata
augite	aug
autunite	aut
azurite	az
baddeleyite	bad
barite	ba
bauxite	baux
becquerelite	bec
bementite	bem
biotite	bio
bismuth	Bi
bismuthinite	bm
bornite	bn
boulangerite	bl
bournonite	bo
brannerite	bran
braunite	br
bravoite	bv
breithauptite	btp
calamine	cala
calaverite	ca, cl
calcite	calc
canfieldite	can
carbon	C, cbn
carbonate	carb
carnotite	carn
carrolite	car
cassiterite	cs, ct, cx
celesite	cel
cerargyrite	cer
cerussite	ce
chalcanthite	chln
chalcocite	cc
chalcopyrite	ccp, cp
chalcostibite	cb
chalmersite	cm
chert	ch, cht
chloanthite	cln, cl
chlorite	chl
chromite	cr
chrysocolla	chrys
cinnabar	ci
clausthalite	ct
clay	clay
cobalt	Co
cobaltite	cob
coffinite	cof
coloradoite	colo
columbite	colu
coolgardite	cool
copper	Cu
corundum	cor
cosalite	cos
covellite	cv
crookesite	ck
cubanite	cn
cummingtonite	cum
cuprite	cup
cylindrite	cy
datolite	dat
descloizite	des
diaphorite	diap
dickite	dick
diopside	diop
dolomite	dol, dolo
domeykite	dom
dyscrasite	dy
emplectite	emp
enargite	en
enstatite	ens
epiboulangerite	epib
epidote	ep
erythrite	ery
erythrozincite	eryz
euxenite	eux
famatinite	fm
feldspar	fs, felds
ferberite	ferb
fergusonite	ferg
fluorite	fl
franckeite	fc, fr
franklinite	fk
freieslebenite	freis
galena	gn
gangue	G
garnet	gar
garnierite	garn
geocronite	gc
gersdorffite	gf
gibbsite	gibb
glaucodot	gld
goethite	goe, gt
gold	Au

(continues)

Table 4.1-2 Geological abbreviations (continued)

Term	Abbreviation
goldfieldite	glf
goldschmidtite	glm
gossan	gos
graphite	graph
greenockite	gk
grunerite	grun
guanajuatite	gt
guitermanite	gm
gummite	gum
gypsum	gyps
halloysite	hl
hausmannite	hs
hematite	hem
hessite	hs
hewettite	htt
histrixite	his
hornblende	hb
huebnerite	hub
ilmenite	il
iron	Fe
itabirite	ita
jalpaite	jl
jamesonite	jm
jarosite	jar
jasper	jas
jasperoid	jasp
jordanite	jd
kalgoorlite	klg
kaolinite	kaol
kermesite	km
keweenawite	keew
krennerite	kr
kyanite	ky
labradorite	lab
laterite	lt
laumontite	laum
lead	Pb
lepidolite	lep
limonite	lim, lm
linnaeite	ln
livingstonite	lv
loellingite	lo
luzonite	lz
magnetite	mag, mg
malachite	mala, mc
manganite	man, mng
marcasite	mar, ms
matildite	mt
melonite	melon
meneghinite	mene
metacinnabarite	mc
miargyrite	my
millerite	ml
mineral	min, mnl
mohawkite	mk
molybdenite	mo
muscovite	mus, mv
nagyagite	ng
naumannite	nm
nephelite	neph
niccolite	nc
oligoclase	ol
olivine	ov
orpiment	orp
orthoclase	or
oxide	ox
patronite	pat
pearcite	pc
pentlandite	pn, pent
petzite	pet
phosphate	phos
pitchblende	pthb
plagionite	pg
platinum	Pt
plumbojarosite	pbj
polyargyrite	plgy
polybasite	plb
polydymite	pm
polyhalite	pyh
prehenite	prh
proustite	pu, pru
psilomelane	psi
pyrargyrite	pr
pyrite	py
pyrochlore	pyl
pyrolusite	pyrl
pyrrhotite	po, pyrr
pyromorphite	pymp
pyroxene	px, pyx
quartz	qz, qtz
quicksilver	Hg
rammelsbergite	rm
realgar	rl
rhodochrosite	rhod
rhodonite	rho
rutile	rt
safflorite	sf
scheelite	shee
semseyite	sems
sericite	ser, sr
serpentine	serp, sert
siderite	sid, sd
siegenite	sg
silicate	sil
sillimanite	sill
silver	Ag
skutterudite	sk
smaltite	sm
smithite	smtt
smithsonite	smith
specularite	specul
sperrylite	sperry
sphalerite	sp
sphene	sph
stannite	stan
stephanite	stp
stibnite	stib
stilpnomelane	stilp
stromeyerite	strom
stutzite	stut
stylotypite	sty
sulfide	sulf
sulfur	S
sylvanite	sv
talc	tc
tantalite	tan
teallite	teal
telluride	tell
tellurium	Te
temiskamite	tk
tennantite	tn
tenorite	to
tetradymite	ty
tetrahedrite	td
thomsonite	thom
thorite	thor
thucholite	thuc
tiemannite	tie
tin	Sn
titanite	tit
topaz	tz
torbernite	torb
tourmaline	tl
tremolite	tm
troilite	tr
tungsten	W
turgite	turg
turquoise	turq
tyuyamunite	tyuy
ullmannite	ul
umangite	um
uraninite	uran
uranophane	urp
vanadinite	van
vein quartz	vqz
vermiculite	verm
violarite	vl
voltzite	vo
warrenite	wa
whitneyite	wh
willemite	wy
wolframite	wf
wollastonite	woll
wulfenite	wulf
wurtzite	wurt
zeolite	zeo
zinc	Zn
zincite	znc
zinkenite	zk
zippeite	zip
zircon	zr
zunyite	zun
Colors	
black	blk, bk, bl
blue	blu, bl, bu
bright	brt
brown	brn, br, bn
buff	buf, bf
chocolate	choc, cho
dark	drk, dk
drab	drb
gray	gry, gy
green	grn
light	lgt, lt
orange	oran
purple	purp, pp
red	red, rd
steel	stl
violet	vio
white	wht
yellow	yel, yw, yl
General Terms	
batholith	bath
dike	dk
lopolith	lopo
mineral	min, mnl
pluton	plut
rock	rk
specimen	spec
stock	stock
stone	stn
volcano	vol

(continues)

Table 4.1-2 Geological abbreviations (continued)

Term	Abbreviation	Term	Abbreviation	Term	Abbreviation
Descriptive and Structural Terms		extrusive	extr	oolitic	ool
abundant	abun	fault	flt, f	opaque	opq
alteration	altn	favorable	fav	outcrop	otp,oc,otc
altered	alt, alt'd	ferruginous	ferr	oxidized	ox
angular	ang	fine	fn	pebble	peb
arenaceous	aren	fissle	fss, fsl	phenocrysts	phen
argillaceous	argill	fissure	fiss, fisr	porphyry	por
asphaltic	asph	flakes	fks	predominantly	pred
band	bnd	flow cleavage	flclvg	proportion	prop
banded	bndd	fold	fld	rare	rare
banding	bndg	foliation	foli	refractory	rfty
bed	bd	foot wall	F.W., FW	residual	resd
bedded	bdd	formation	frm	residue	res
bedded, thick	tkbd	fossil	foss	rock	rk
bedded, thin	tnbd	fracture cleavage	frac cleav	rocks	rx
bedding	bdg	fracture	frac	round	rnd
below	blw	fragmental	fragl	sand	sd
bentonitic	bent	fragments	frag	sandy	sdy
between	btwn	friable	fri	scarce	scar
bitumen	bitum	glass	gls	schistosity	schis
bleb	blb	gneissosity	gny	sediment	sed
border	bor	gouge	go	sheared	shrd
bottom	bot	gradational	gradl	sheeting	shtg
break	brk	grade	grad	siliceous	silic
broken	bkn	grain	grn or grain	slickensides	slick
calcareous	calc	grained	grn, grnd	slightly	sly
carbonaceous	crb,carbon	granular	gran	soft	sft
cement	cmt	ground	gro	specimen	spec
cherty	chty	hanging wall	H.W., HW	sticky	stky
cindery	cndy	hard	hd	stock	stock
clear	clr	heterogeneous	hete	stone	stn
cleavage	clvg	high grade	h.g., hg	strained	strained
coarse	cse	igneous	ig	streak	strk
compact	cpt	impregnated	impreg	striae	striae
composition	comp	inclined	incl	strike	str
concentration	conc	inclusions	inclus	strong	strg
concretion	concr	indurated	indur	structure	struc
condense	condense	interbedded	intbd	tarnished	tnshd
condition	cond	intrusive	intr	thick bedded	tkbd
conductivity	conduct	irregular	irreg	thick	tk
considerable	consid	joint	jo	thin bedded	tnbd
cross-bedded	x-bdd,xb	jointing	jotg	unaltered	unalt'd
crumpled	crump	lamination	lam	undulating	und
crushed	crsd	lean	ln	unfavorable	unfav
crystalline	xln	lineation	lin	unoxidized	unox
crystalline, coarsely	xln-c, cxln	location	loc	variable	var
crystalline, finely	xln-f, fxln	low grade	l.g., lg	vein	vn
crystallized	xld	magnetic	mag	veined	vnd
dark	drk, dk	massive	msv	veinlet	vnlt
dense	dns	material	mat	very	vy, v
dike	dk	medium	med	volcanic	vol
discard	dscd	metamorphic	meta	wash	wash
disseminated	dissm	mineral	min, mnl	weak	wk
distributed	distr	mineralization	mzn	weathered	wth
dragfold	dfld	mineralized	minz, mzd	with	+, w
ellipsoidal	elp	mixed	mxd	without	w/o
enriched	enr	mottled	mot		

Source: Adapted from Chace 1956.

6. As mapping proceeds, it is important to carefully note and record the location of important cultural features such as roads, buildings, key topographic features, or survey points for later data compilation and development of the geologic map. If surface or underground mine workings are involved, secure ground outlines as available, annotated with known survey point locations if possible.
7. *Unless the geologist is extremely experienced, it is inadvisable to enter old mine workings alone.*
8. Compile note sheet data on larger sheets to build the geologic map of the project.

Historically, compass and tape(s) commonly have been used to construct ground outlines for data recording, to locate outcrops, and to tie in culture or survey control; and if used

carefully, they will provide sufficient accuracy. This is a fairly straightforward procedure and consists of stretching a cloth measuring tape or tapes from or between known points and determining the bearing of the tape with a compass. This tape line is then plotted to scale in its proper orientation on the field note sheet, and tick marks are posted and identified each 3 m (10 ft) along the bearing of the tape line. Following this, offset measurements are taken at right angles to the cloth tape from the tape to the edge of the drift or pit bench. A small pocket tape is used to take these offset measurements at 3-m (10-ft) intervals along the cloth tape. Points corresponding to these offsets are then plotted on the field note sheet and connected, and an outline of the drift or edge of the pit bench is thus created for geologic note-taking. Figure 4.1-1 illustrates the methodology of developing a ground outline and shows some simple geologic notes. Location, scale, date, geologist, and orientation are clearly indicated on each example.

In typical mine-related work, geologic field note-taking is commonly done on a relatively large scale, such as 1:240 or 1:600 (1 in. = 20 ft or 1 in. = 50 ft). Smaller scales—1:1,200 and 1:2,400 (1 in. = 100 ft and 1 in. = 200 ft)—are also used, generally to collect data on overall resource setting or to simplify more detailed work in the mine itself. For regional work, scales of 1:12,000 or 1:24,000 (1 in. = 1,000 ft or 1 in. = 2,000 ft) may be appropriate. Some variability is necessary, and the actual mapping scale used depends on needs of specific projects. Advance planning here is useful. Although detailed 1:240 (1 in. = 20 ft) mapping in a vein or massive sulfide deposit is desirable, attempting to get the same detail in a 13.5-Mt/a (15-million-tons/yr) open-cut coal mine or a 1.8-Mt/a (2-million-tons/yr) underground coal operation would be inappropriate.

Mapping techniques are described in a number of good references, which vary somewhat in perspective. Proper supervised training is desirable. Peters (1987) provides considerable detail in a good description of general surface, surface open-pit, and underground work. He describes an outcrop mapping system in which three acetate overlays (shown in modified form in Figure 4.1-2A) are superimposed over a base map or air photo and used to take notes describing geology, mineralization, and alteration for each outcrop. The field manual by Compton (1985) provides good coverage of geologic mapping techniques, as do earlier textbooks by McKinstry (1948) or Forrester (1946), both of which remain excellent sources on techniques in mine mapping.

The geologic data may be collected conventionally using the long-established pencil-and-paper-based mapping and compilation system or methodology. This system can be augmented with the use of digitizers or other computer-facilitated electronic databases to prepare maps, sections, or other displays for use.

Alternatively, because of the rapid development of computers, enlarged electronic storage capacity, and software, the data may be collected by direct digital means using portable or pen-and-tablet computers. With these devices, it is possible to record data directly on computer screens in the form of spreadsheets, commercially available digital base maps visible on the screen, or perhaps property- or company-specific custom grid sheets. The geologic data, contacts, attitudes, mineralogy, structure, and so on, are posted directly on the screen and captured digitally (XYZ coordinates) through the software and saved in the storage medium of the computer. The stored data are available for later use as desired in plan map or cross-section construction, three-dimensional (3-D) manipulation and study, geologic or resource model development, resource estimation, or other needs.

Walker and Black (2000) review the development of a computer-based field mapping program at a midwestern university. They note that computer use is increasing in most aspects of geologic work, that the location-dependent nature of geologic data corresponds to both the importance and the limitations of digital information, that direct digital recording may save time, and that digital topographic data are generally readily available. Additionally they supply some key references, identify hardware and software issues and some products, and conclude that in addition to teaching conventional field skills they will continue to use computers in field courses at the university.

Another technological development is the portable X-ray analyzer, which can provide useful analytical data to incorporate into geologic mapping.

Brimhall et al. (2006) provide a thoughtful review of the fundamental importance of geologic mapping and some history of the development and modification of the Anaconda system of geologic mapping. The system was initially developed in the early 1900s at Butte, Montana, and underwent subsequent modifications and expansion (to 10 colors and the use of specific symbols and plotting positions on the field sheets) in Chile in South America, Nevada in the United States, and elsewhere. Einaudi (1997) discusses this mapping technique in an excellent, well-illustrated, unpublished Stanford University paper. Geologists interested in ore deposit mapping may wish to secure a copy of this paper. Additionally, an excellent paper by Proffett (2003) demonstrates and documents the results that can be expected through the use of careful detailed field mapping and core logging or relogging (discussed and referenced in Chapter 4.2 of this handbook) techniques that serve as the basis for the geologic evaluation of the Bajo de la Alumbrera copper–gold deposit in Argentina.

In addition to fully describing and illustrating the use of this paper-based mapping system, Brimhall et al. (2006) also discuss digital mapping based on pen-and-tablet portable computers as well as the GIS revolution in mapping and data manipulation with CAD systems. Importantly, both conventional and computer-based digital methodologies require the same basic geologic skills, and both have their advantages and disadvantages (Malone 1995). Adequate training is needed for either methodology.

Geologic mapping in underground coal mines can greatly increase the geologic understanding, productivity, and ultimately the profitability of a mine. Krausse et al. (1979a, 1979b) provide excellent examples of underground mine mapping methodology and the effect of lithology and structural features on mining. Their work and that of Ledvina (1986) further describe the increasing use of geologic mapping in underground coal mines and stress the importance of roof rock characterization. Moebs and Stateham (1984) summarize contract studies investigating the relationship between geologic factors and roof stability in coal mines. The work is well referenced and clearly identifies mine and core mappable geologic features that control roof stability.

The Coal Mine Roof Rating (CMRR) system developed by the National Institute for Occupational Safety and Health (NIOSH; Anon. 2008) incorporates rock composition, structural defects, and thickness into a rating index that ranges from 1 to 100. The system is increasingly being used as a basis

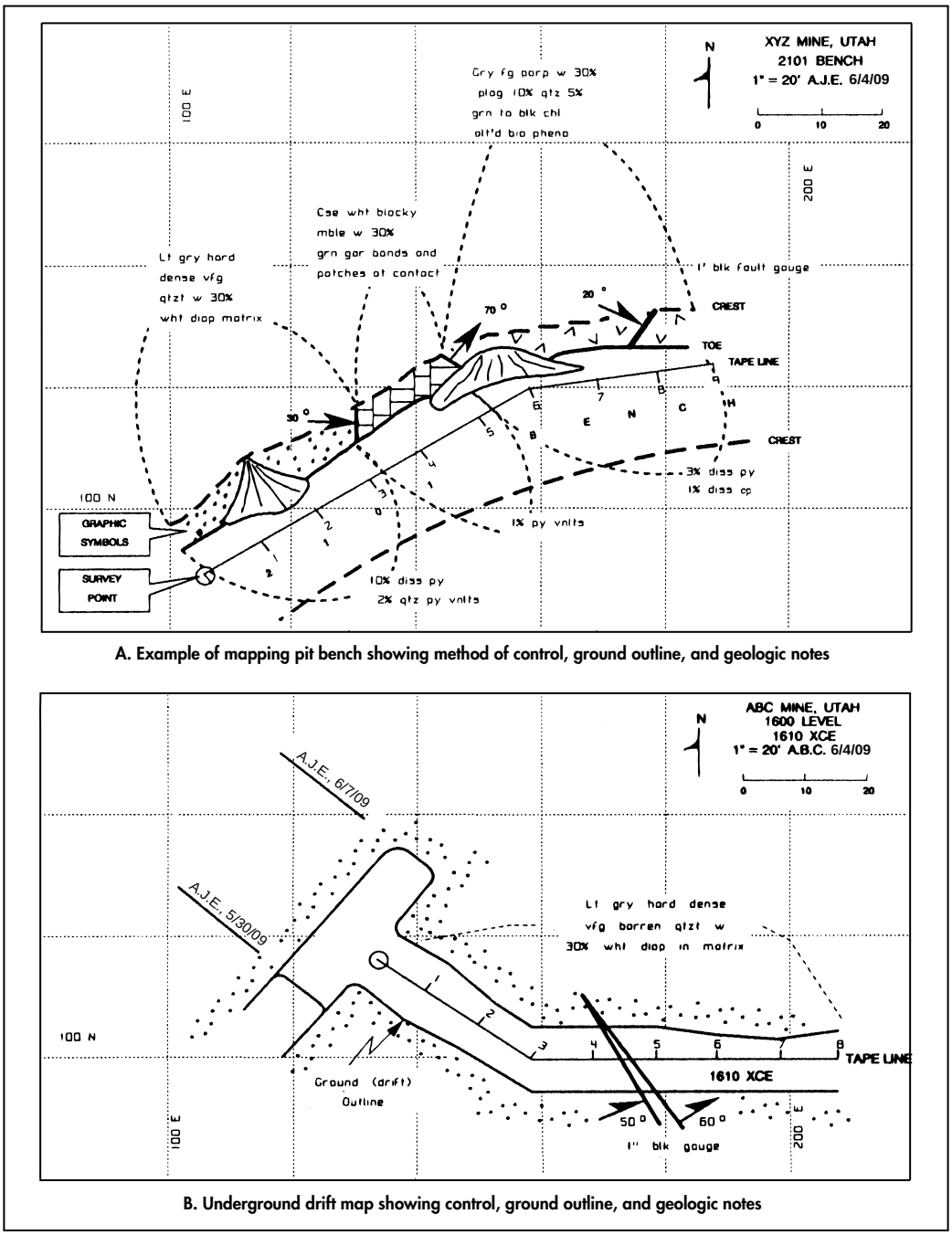


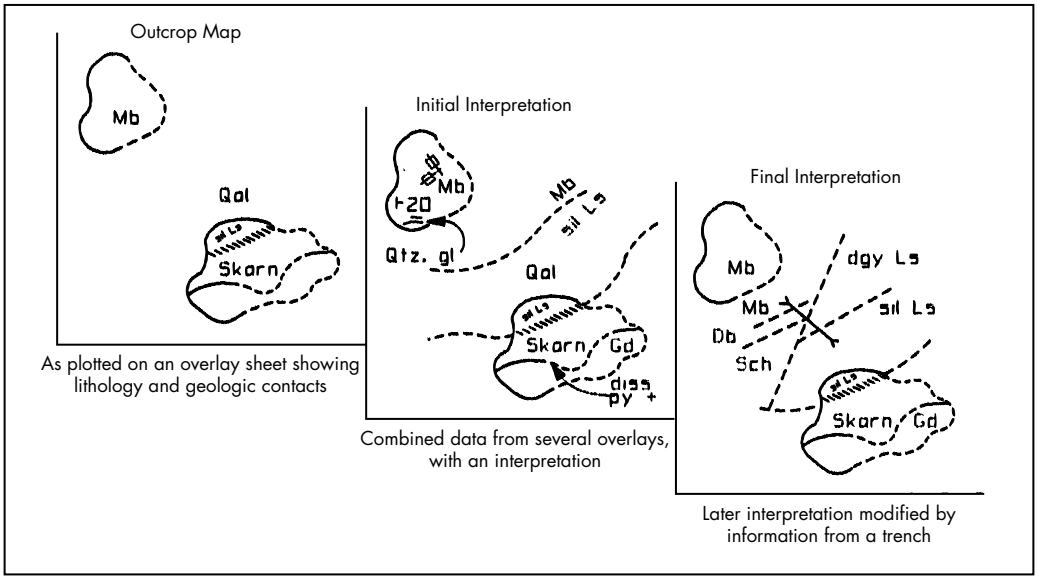
Figure 4.1-1 Simple geologic field note sheet

for mine design applications. The current version is available as a download file from the NIOSH Web site.

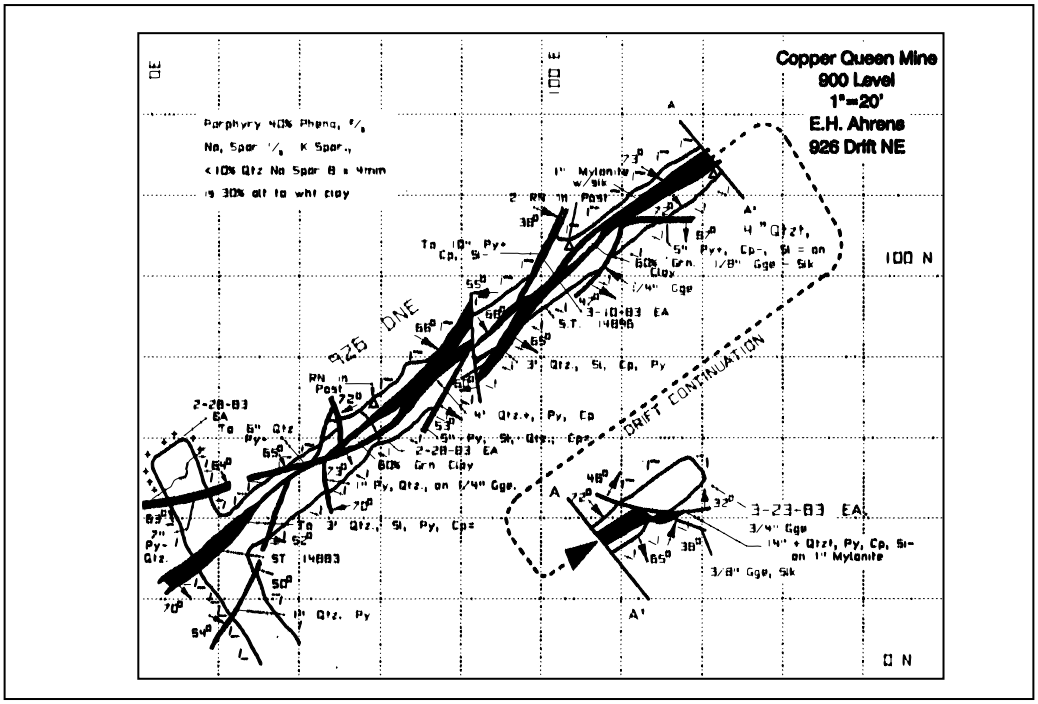
Coal geologic work tends to depend heavily on computerized geologic data handling with its extensive reliance on structure contour, isopachs, numerous coal quality maps, and other geology-based elements that influence operations.

Early papers by Linforth (1914, 1933), McLaughlin (1933), McLaughlin and Sales (1933), and Sales (1941) clearly show techniques and discuss the importance of careful geologic mapping.

Information and comments on specific software packages useful for geologic work have been outlined by several authors in the *AAPG Computer Applications in Geology, No. 4* (Gibbs 2000) and in *Mining Magazine* (Anon. 2006). The packages vary from simple one-purpose programs to fully integrated production programs, commonly with powerful 3-D capability, for geologic data gathering, analysis, resource estimation, feasibility studies, and mine planning. Software programs for computer assistance in data gathering and analysis are readily available in geochemistry, geologic modeling, geology, geophysics, geotechnical areas, hydrology, resource modeling, and resource estimation. Properly used, they can provide important, timely support and allow geologists to consider alternate options or respond to the effects of changes



A. Part of an outcrop map using overlays



B. Detailed mineralization and structural notes taken during underground mine mapping of a vein in intrusive rocks

Source: (A) Adapted from Peters 1987; (B) adapted from Ahrens 1984.

Figure 4.1-2 Field note sheets

in mine operational parameters. Examples are changes in cut-off grade, reserve estimation, production planning, and others. Additionally, the Internet site www.miningsoftware.com (Gibbs Associates 2008) provides excellent coverage and descriptions of various geologic software programs.

The following sections on surface mapping, pit mapping, and underground mapping review additional important points to consider.

Surface Mapping: General Points

The following comments and recommendations are offered for surface mapping:

- Use aluminum sheet holders for 216 × 279 mm (8½ × 11 in.) gridded sheets, topographic bases, or air photo base maps, which are superior to bound notebooks, as they allow sheet removal and filing, and reduce the chance of data loss.

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CHAPTER 4.2

Geologic Interpretation, Modeling, and Representation

A.J. (Joe) Erickson Jr.

INTRODUCTION

In Chapter 4.1, discussion centered on the importance of clear, complete, accurate, detailed, and systematic collection of varied and descriptive geologic data, and methodologies to accomplish this goal were suggested. Narrative guidelines and narrative-graphical formats were provided to ensure the desired standardization in the acquisition of geologic information. The area of activity in that chapter is referred to as the data collection step in the three-step geologic information processing effort required in resource evaluation.

This chapter builds on Chapter 4.1 in discussing the purpose and recommending methodology for carrying out the remaining two steps: data analysis and interpretation. All three of these steps are required to discharge the critical responsibility of resource characterization for quantification, evaluation, mine planning, and extraction. *Resource characterization* is defined as the determination of the shape, size, quality, quantity, and variability of the geologic entity. Equally importantly, resource characterization determines the limits of various parameters or features of the geologic entity. Correct characterization has both operational and exploration implications, as it provides the information for synthesis of often subtle features into an accurate, predictive description of the resource environment. This description—the geologic model—can then be used either in support of mining operations or in the ongoing search for additional reserves.

GEOLOGIC INTERPRETATION

Thompson (1992) has stressed the importance of the development of computers on geologic data integration and modeling, noting the great increase in and accessibility to computer power and software for modeling varied data and the speed with which two-dimensional and three-dimensional (3-D) data analysis may now take place. He notes that "complex data sets can be integrated directly with ore deposit models based on empirical features…and supported by genetic or process concepts." There is improved ability to visualize multiple data sets graphically and statistically. He also notes a trend toward computer use in all areas of economic geology and warns of possible abuse if computers are seen as magic black boxes requiring no understanding of fundamentals on the part of the researcher. Inappropriate analysis, prediction, and estimation may be the result. To be successful, computer use and analysis must be based on careful geologic data gathering, mapping, and logging—all of which are discussed in Chapter 4.1 of this handbook.

Mason (1993) points out that technical failures in mine development have been frequent and attributes this to the "pronounced decline of economic geology as a field-mine based discipline" and to the "significant advance in computing technology" that has made reserve calculation "painless [and] frequently inaccurate." After recognizing the style and type of mineralization, and after drilling and sampling to allow the mineralized zone to be evaluated and defined, the most important parameters for a geologic database to address include "boundaries to mineralization and ore, continuity of mineralization and ore, classification of mineralization and ore types, and ore quality (grade)" (Mason 1993). To this can be added mineral variability with respect to metallurgical processes and mining conditions.

Continuity relates directly to the type of deposit (Mason 1993) and hence to the understanding of the appropriate geologic model. Knowledge of structural complications is critical. Classification of mineralization as to deposit type and ore type is important for resource/reserve estimation, and both conventional and geostatistical estimates require careful geologic evaluation and awareness of ore types.

Like Journel (1988), Mason (1993) stresses that the need has never been greater for more and improved "geologic control" to eliminate "computer cover-up [caused by] inadequate data bases and interpretation of such data bases." In both estimation and production of ore reserves, says Mason, 3-D modeling has put "greater demands on the appropriateness and accuracy of geological data bases."

GEOLOGIC MODELING

Current terminology refers to resource characterization as the geologic model, or perhaps the 3-D geologic model, whereas previously the model concept was conveyed using the approximately synonymous term *zoning patterns*. The model consists

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of a compilation of all geologic data, observations, and studies available at the time, assembled to display and explain the observations and limits from an empirical and/or genetic point of view. The model may be extremely simple or highly complex depending on the nature of the resource, the data available, or the degree of sophistication in studies of the resource.

One may compare a volcanogenic massive sulfide (VMS) deposit model in simpler form (Franklin et al. 1981) with a more complex genetic model (Franklin et al. 2005). Both are useful and essentially correct as they are based on information, data, and understanding available at the time. The empirical model represents the compilation and integration of numerous types of chemical, mineralogical, structural, and—not uncommonly—numerically quantifiable zoning studies. The conceptual or genetic model attempts to explain the distribution and origin of these features in a useful and meaningful way. Development of a geologic model forces the geologist to think about and explain the rock, mineral, alteration, and other relationships described in the mapping and logging process; to develop an understanding of their relationships; and to provide a predictive tool in support of ongoing exploration and production. The geologic model is frequently incomplete and, as discussed by Peters (1984), probably will change or be revised as additional geologic data are acquired.

Eckstrand (1984), Cox and Singer (1986), Roberts and Sheahan (1988), and references in Erickson (1992) synthesize geologic data from a broad spectrum of mineral deposits into a number of both empirical and genetic models. Stoeser and Heran (2000) provide a CD version of more recent extensive work by the U.S. Geological Survey on mineral deposit models.

Hutchinson and Grauch (1991) provide historical perspective on the development of genetic concepts for VMS and other deposits in their publication on varied deposit types, whereas Keays et al. (1989) review gold in VMS and other deposit types. Large (1992) provides perspective on approximately 30 Australian VMS deposits, including illustrations that show styles of VMS deposits and, importantly, the effect of deformation on interpreted seafloor morphology on several deposits. LaBerge (1996) provides data about several Wisconsin (United States) VMS deposits—Ladysmith, Lynn, Crandon, and others. Two additional important references are Barrie and Hannington (1999) on modern and ancient settings and Hannington and Barrie (1999), the definitive volume on the Kidd Creek VMS deposit in Canada.

Key references on porphyry-related deposits and models include Lowell and Guilbert (1970), one of the best-known works; Gustafson and Hunt (1975), describing detailed mapping, vein relationships, geology, and a genetic model for El Salvador, Chile; Titley (1982) on southwestern United States porphyry deposit studies and descriptions; Nielsen (1984), discussing the evolution of porphyry models, identifying variation, and discussing some associated problems; Pierce and Bolm (1995) on the American cordillera deposits; and Camus et al. (1996) on new Andean discoveries and studies. Proffett (2003) provides an excellent study regarding the mapping, description, analysis, and emplacement genesis of the Argentinean Bajo de la Alumbrera deposit, including the first in the large-format *Economic Geology* Map Series. Hedenquist et al. (2005) reviews porphyry and other deposit types and includes papers on environments, processes, and metallogeny. Sillitoe (2010), in an invited general overview–summary paper on giant porphyry copper systems, provides an excellent field-oriented review paper on these deposits. Key geologic features are discussed from a regional, district, and deposit-scale perspective. Sillitoe discusses regional and district characteristics; stresses the multiple-occurring nature of these deposits, which are commonly found in "clusters" or in "alignments"; the variability in deposit "footprint"; deposit-scale features; and zoning. He presents excellent illustrations of empirical and genetic models of these intrusion-centered systems.

Useful resources on gold deposit models include Macdonald (1986) and Hausen et al. (1990). The latter provides a review of deposit geology and contains several papers that discuss metallurgical process problems associated with these deposits. Skinner (1981) contains good papers on gold deposits, and Corbett and Leach (1998) describe the different styles of gold–copper deposits in the southwest Pacific Rim countries.

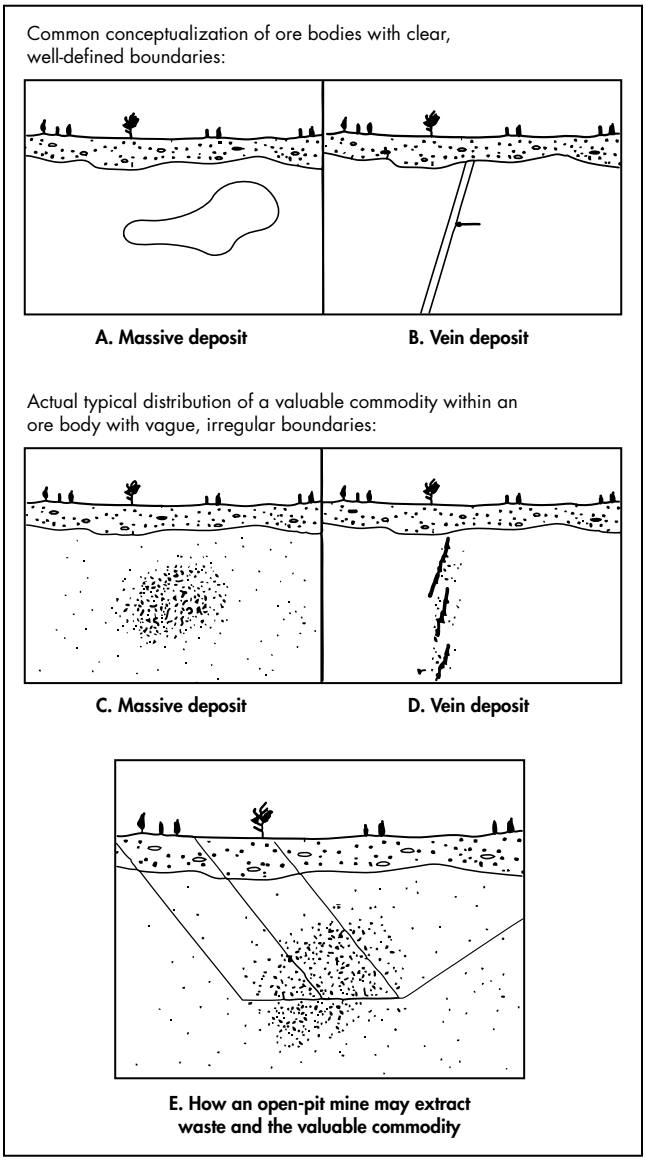
Two textbooks, *Geology of Ore Deposits* (Guilbert and Park 1986) and *Exploration and Mining Geology* (Peters 1987) remain extremely useful references.

Work by Dapples and Hopkins (1969), Kaiser et al. (1980), Ayers (1986), and Thomas (2002), as well as references cited in Erickson (1992), provide excellent descriptions of coal deposits and the associated geologic environments and processes. Models described in all of these references provide a state-of-the-art picture of the products and of geologic processes and events. They are extremely useful for broadscale exploration activity or more tightly controlled resource evaluation and operational support work. These useful volumes are compilations from and syntheses of numerous varied studies of long-established mining districts or regions. Typically these are areas where extensive exploration, development, and operational geologic programs have provided large factual geologic databases and extensive material for laboratory studies. Although each district or region may exhibit specific variations, the volumes summarize observations and studies from many deposits of a similar nature. Hence they provide a guide to the commonly observed typical features to be expected in a particular deposit type—the geologic model. They are extremely important and useful guides for interpreting geologic observations while work is being conducted on projects, when new district-specific models are being constructed, and as aids during ongoing exploration or development programs.

Models should attempt to explain all observable facts. Geologic data should not be ignored or discarded if they do not fit the perceived model. The lack of agreement between facts and theory may indicate incorrect data, incorrect models, or insufficient data, or all three, and this lack identifies the need for further investigation.

The fundamental problem associated with resource characterization for evaluation and estimation is in taking a very limited amount of geologic data, correctly analyzing and interpreting the data, extending these interpretations into unknown areas, and then making summary quality, quantity, and limit statements about the area. In simplest terms, resource characterization answers these questions: Where is the resource (ore)? What are its limits (shape)? What is the quality (grade)? What is the nature of the associated environment?

It would be easier to answer these questions if the boundaries were sharp, but that is the case in only a few specific types of deposits. More often one is dealing with an irregular, poorly constrained distribution of values of a commodity (Gentry and O'Neil 1984). Figure 4.2-1 is a lucid and simple



Source: Adapted from Gentry and O'Neil 1984.

Figure 4.2-1 Ore-body representation

explanation of problems related to resource estimation, external and internal dilution, and limit or edge problems that plague many operations.

Sound, careful geologic data analysis, review, and verification during model construction go a long way toward solving the previously described fundamental geologic problems in resource characterization. The geologic data and interpretation (the model) comprise the foundation of the evaluation process and are essential for resource estimation and extraction (Ranta et al. 1984). As Call (1979) indicates in his discussion of statistical data handling in development drilling programs, "the validity of any analytical model must ultimately be determined by geologic interpretation." Sound geologic judgment, discipline, and hard work are needed to deduce the detailed geologic setting from drill holes, mapping, and sampling programs and lead to the construction and understanding of the geologic model and its variations. The model is based on numerous varied geologic studies; assuming that all other factors (sampling, hole or data locations, analysis, assays, or others) are correct, the model ensures the best possible reliable resource estimate and an understanding of ground conditions. Data about drill holes (or any other data) can be misleading if interpretation is incorrect (Ranta et al. 1984), and manual, computer-assisted, or the more sophisticated geostatistical estimates may also be incorrect. The simple line on a map or cross section depicting the limit of a geologic feature (e.g., alteration type, lithology, ore) is an extremely powerful decision-influencing item, and only proper geologic interpretation ensures its correctness. Sound interpretation is the only way a geologist can fulfill the challenge issued by Barnes (1980) to "accept the responsibility of recognizing and recording boundaries" that characterize the deposit or distributions within the deposit. It is imperative that the geologist draws the line establishing these boundaries.

In summary, "good geology based on well-understood data is still the only recipe for good reservoir/site/deposit characterization" (Journel 1988). The geologist must thoroughly understand the total geologic environment and distributions to reliably estimate resources, delineate areas for mine planning, and support day-to-day operations.

Methodology

Previous discussion in this chapter has introduced the concept of geologic characterization—the interpretative model—and has emphasized the importance of developing a good model for accurate resource evaluation. The discussion now turns to guidelines on how best to construct a geologic model of a resource. Emphasis is on an empirical model, one that accurately records the factual geologic observations. The same principles apply whether developing a conventional or computer-based geologic model.

The typical steps in constructing and interpreting a geologic model are summarized in the following list. These steps are generally time-consuming because of the dependence on data collection through mapping and logging. They vary somewhat depending on the resource being evaluated.

1. Begin with fundamental knowledge and understanding of various deposit types and their models.
2. Accurately and precisely locate surface and drill-hole data points; accurately collect the appropriate geologic data. Accuracy is essential.
3. Carefully plot drill-hole locations and the data on drill-hole traces on a set of cross sections.
4. Identify correlatable contacts, geologic units, mineralization, or other features on the drill-hole traces.
5. Correlate the interpreted limits of geologic features on a drill-hole-to-drill-hole basis on individual cross sections.
6. Carefully plot data, transfer information, and correlate interpreted limits of the geologic features on a drill-hole-to-drill-hole basis on a second set of cross sections at right angles to the first.
7. Construct a set of plan maps at desired elevations using drill-hole intersections on the plan maps and interpreted limits of units as developed on sets of cross sections.
8. Construct simple illustrations (the geologic model) for discussion and comparison.

The geologist usually starts with surface outcrop data and drill-hole logs that, as reviewed in Chapter 4.1, are of high quality and have been collected using rigorous standardized methodology and appropriate quality assurance/quality

control (QA/QC) measures. This will provide unbiased factual data for compilation and analysis. The geologist builds a geologic outcrop map from surface work that contains all geologic observations. This map (hard copy or on a computer screen) must clearly differentiate factual outcrop data from geologic interpretation and inference between outcrops. A common graphic methodology uses bold solid lines and patterns or dark colors for outcrops and dashed or dotted lines of similar colors, but applied in a paler or lighter fashion, for interpretations. Herness (1977) emphasizes that "there must be no screening of data to eliminate 'unimportant' facts" because frequently "the importance of insignificant data is realized 20 or 50 years later" when mine workings or core may not be available for remapping.

With respect to core availability, the deliberate disposal of core from known ore deposits or established mining districts is unconscionable. Stored core is simply one of many items in the geologist's files that will be constantly examined and reviewed as new ideas or data become available. E. Klohn, manager of the Geology Department of Compañía Minera Disputada, comments, "Core is our record; we must always go back to it" (personal communication). Numerous ore discoveries have been made because core was available for reexamination, re-sampling, and reinterpretation. Braun (1991 and personal communication) emphasizes the importance of the availability of old drill core for relogging and re-sampling that may lead to the discovery of a new gold ore body.

More recent presentations by Leonardson et al. document a significant reserve expansion in a mature mining district in the northwest portion of the open pit at the Barrick Goldstrike deposit, Nevada, United States (Leonardson 2004; Leonardson et al. 2005). A world-class geologic map and model existed in the pit area, and ore controls were understood. However, a deficiency in the model existed in the area of drill-hole logging due to the transitional nature of, and resultant inconsistencies in, the picking of unit and formation contacts and because stratigraphic information from early drill holes was not in the database. As a result, projection of ore-controlling structures outside the existing pit wall was difficult. An 82,300-m (270,000-ft) core and cutting relogging campaign that was primarily stratigraphic in nature but included lithology, structure, and alteration was carried out with team members jointly reviewing their work to ensure consistency. Current bench mapping was included. The greatly improved data were incorporated into the computer-based geologic model of the property, which allowed for the definition of numerous structural/stratigraphic drilling targets. Approximately 300 holes were drilled over a 2-year period, and the property reserve expanded by more than 68 million g (2 million oz) of gold. Significant mining plan changes were made that required pushbacks of the originally planned "final pit wall" that ranged from 154 to 610 m (500 to 2,000 ft) and averaged 305 m (1,000 ft). The work documents the importance of saving core and cuttings, consistency in the geologic work, and the importance of a good geologic model.

Rivera et al. (2009) further document the importance of maintaining drill core from earlier drilled holes in reporting on a brownfield exploration program in the Chuquicamata District in Chile. Here the "exploration potential was determined by the review of old drill holes" that led to a program that discovered the so-called Toki Cluster of five completely concealed, individual, closely adjacent porphyry centers containing an "oxide and supergene sulfide resource on the order of 1.6 Bt [billion metric tons] of mineralized rock averaging 0.4% Cu."

Gustafson et al. (2001) summarize ongoing geological work on the multiple intrusion/mineralization centers at the El Salvador copper deposit, Chile. These authors reemphasize that porphyry copper deposits tend to cluster, may be closely spaced and overlap (which is difficult to identify and not a widely recognized concept), and require careful mapping (field and core) and mineralization study. They believe that studies of this type are frequently neglected at operating properties and that the failure to recognize multiple, truncated, or incomplete zoning patterns may result in the lack of appropriate additional drilling. The work they describe led to the drilling, discovery, and development of new ore in an existing mine in which ore had first been identified 40 years earlier. Discoveries of this type "within the mining complex" tend to be far more valuable than new discoveries farther removed from an operation.

Future discoveries will be made because old core or cuttings or data were available for examination in the light of new data, new ideas, or other commodities, or, unfortunately, simply because a better or more conscientious geologist was involved. Geology is a continually evolving discipline, and new knowledge and understanding are accumulating at an astonishing rate. Discarding already logged core or cuttings is the equivalent of destroying survey or production records or a computer database after a set of maps or financial algorithms has been completed. It should not be done.

After or in conjunction with the development of a surface geologic map, it is essential that a detailed set of geologic cross sections (and subsequently plan maps) be constructed, preferably at right angles to each other. This is easier if the critical aspect of a resource can be satisfactorily approximated by a set of north-south and east-west sections tied to a coordinate system. Other orientations are possible, although this can introduce measuring and posting problems in relationship to grid coordinates. Coordinates may also be rotated to provide local grids, but this practice may cause serious survey correction problems and difficulties in tying locations to the Public Land Survey System or various state plane-coordinates systems. A reference line to aid in registration and data posting should be drawn on sets of sections, particularly those that do not parallel the coordinate grid system. All data from surface mapping and careful systematic core logging should be posted on appropriate sections and plotted, as indicated by Call (1979), "without interpretation on reproducible sheets. From these, copies can be made [or they can be developed on computer screen displays] for use in interpretation. This process will maintain the distinction between observed facts and interpreted geology." These sections can be constructed manually as the project or drilling proceeds. If an appropriate computer-assisted package is available, data can be entered into an expandable database for rapid factual data posting to provide base maps for interpretation.

As new data are acquired (subject to the appropriate QA/QC measures), they are posted on the factual data sheets. Prints or computer screen displays are produced and new, revised interpretations are developed. Additionally, completed interpretations can be transferred to a second set of reproducible bases (copies of the factual data) so multiple sets of interpretative maps can be reproduced for several users. Drill-hole traces and associated data should be plotted on the sections at a scale sufficient to allow for posting of multiple

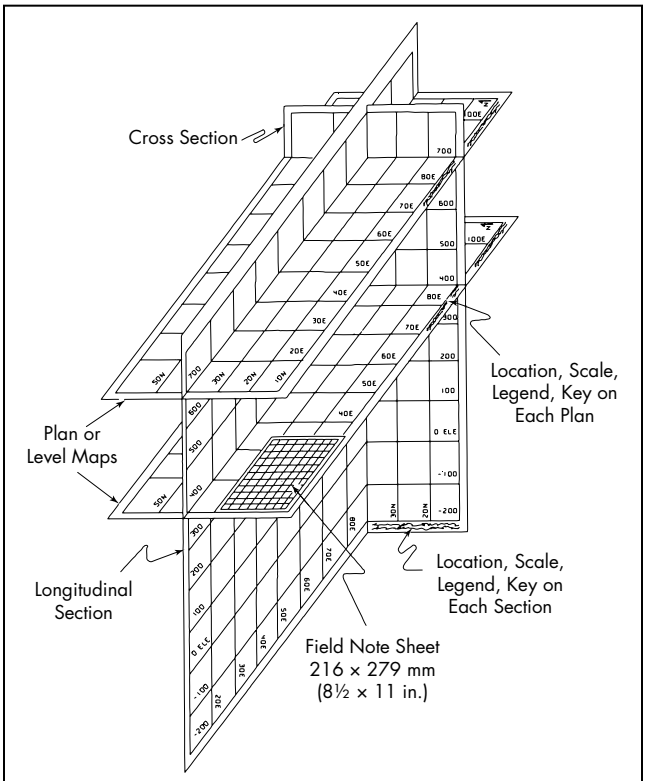
variables along the trace of the hole. The important parameters are lithology, structure, alteration (if present), grade (as a pattern, a color bar, histograms, or numerical values), and perhaps total sulfide or sulfur content. The particular parameters posted depend on the resource being evaluated. This posting of several parameters allows for the determination of critical relationships that normally are of use in guiding interpretation and projections. An extremely risky interpretation is the simple correlation of assays (grade) from hole to hole with no consideration of associated geologic features. This process, which should be avoided, commonly leads to incorrect interpretation, overestimation of resources, and incorrect, usually overly optimistic, evaluation.

Clear cross-sectional construction allows one to determine hole-to-hole relationships and continuity of numerous key geologic features such as lithology, alteration, mineralogy, grade, and structure and, perhaps, features that may influence mining such as rock quality designation or the distribution of varied metallurgical ore types (hardness, alteration, mineral species, or others) requiring different metallurgical processes. The construction of a second set of cross sections at right angles to the first allows for determination of hole-to-hole relationships in two directions and hence section-to-section relationships in three dimensions.

The final step is usually development of a third set of illustrations, plan or level maps (sometimes referred to as slice maps) from the drill-hole pierce points in the plane of the level maps, and projections from the two sets of cross sections. This allows one to conform the sections and plans so all common points have similar XYZ coordinates in the now completed 3-D interpretation and resource characterization. As indicated, multiple sets showing the important relationships of variables—the geologic model—is the product.

In areas of little or no outcrop, such as areas covered by glacial debris or thick overburden, a bedrock geologic map at the subcrop (i.e., at the base of the glacial debris or overburden), rather than a surface geologic map, is commonly constructed. This is carried out in conjunction with construction of the plan maps of the deposit at varied elevations. Data for the subcrop map are derived from the drill holes and consist of the core or cutting information obtained when the drill encounters bedrock. Likewise, interpretations developed during cross-section construction are typically projected to this bedrock surface.

Construction of this set of plans, sections, and illustrations described previously allows one to develop the fundamental geologic understanding of the resource necessary for mine planning and financial evaluation. In the case of coal or uranium, the sets of plans and sections are commonly supplemented or replaced by isopach, isograde, grade-thickness, or structure-contour maps of important features. Of particular significance in this understanding is the determination of ore/resource controls such as lithology, primary structural features, secondary structures, structural intersections, or combinations of features that are directly responsible for the specific localization of the resource. Sufficient understanding, particularly of limiting features, should be developed in the process to allow the input of proper geologic controls in either conventional or computer-assisted resource-estimation methodologies, such as block modeling or the gridded seam technique. This limiting aspect is critical in resource/reserve work to prevent the building of computer-generated reserves in unmineralized areas.



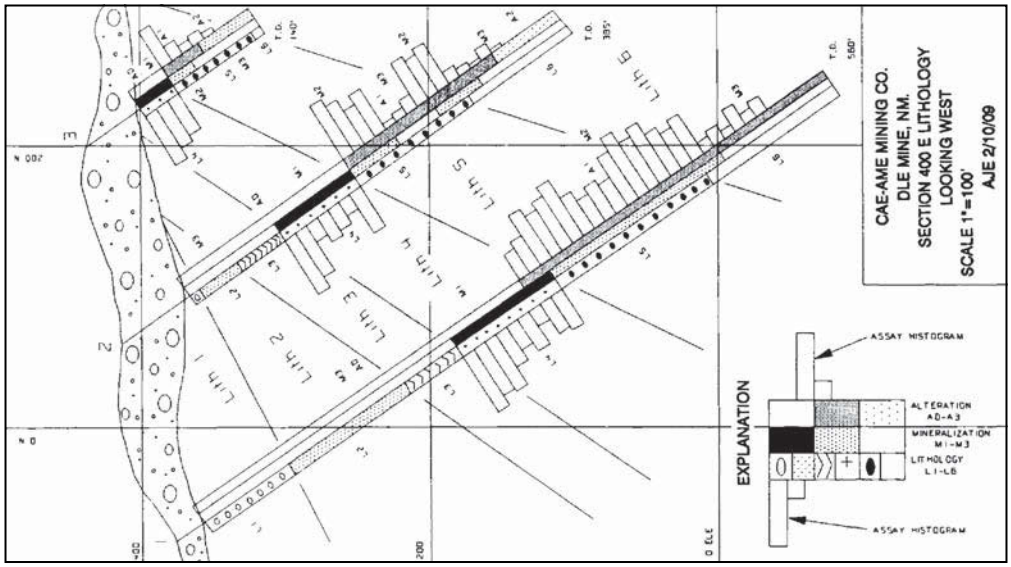
Source: Adapted from Herness 1951.

Figure 4.2-2 Relationship of 3-D plan, section, and field note sheet

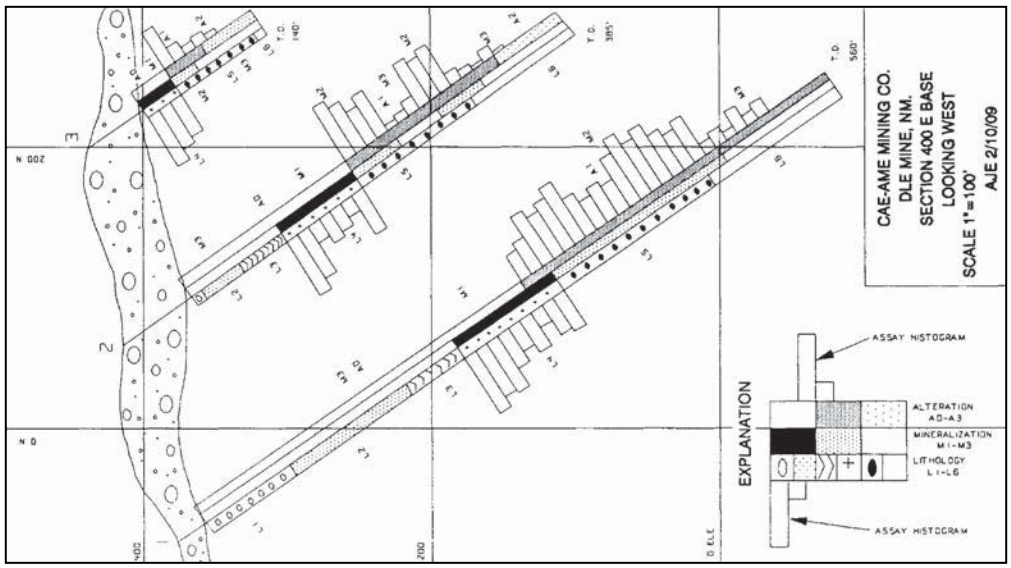
Figure 4.2-2 illustrates the relationships of an orthogonal set of plan maps and cross sections, including the smaller field note sheet subdivisions, used to collect and compile data and support interpretations in the development of a geologic model. Herness (1951) provides a very detailed discussion, illustrations, and recommendations (some dated and others still current) on the essentials of setting up a unified geologic data recording and representation system.

Figures 4.2-3 through 4.2-6 depict the development of factual data and interpretative plans and sections needed to build the geologic model characterizing a particular resource, in this case a VMS deposit. Combination of features such as mineralization and lithology, not shown here, is common. These types of illustrations ultimately form the basis for VMS models (Figure 4.2-7). Figure 4.2-7, which shows mineral and metal distributions and features to be expected in the deposits, portrays one of several idealized models of VMS characteristics developed from studies of numerous deposits.

Figure 4.2-8A is a well-known model that portrays the distribution of alteration, mineralization zones, and sulfide occurrences in a porphyry copper system (Guilbert and Park 1986). This illustration portrays only primary, or hypogene, mineralization and does not show the effects of the secondary, or supergene, processes of weathering (oxidation and enrichment), which are extremely important in these deposits. The effects and changes in the mineral distribution resulting from the action of these secondary processes on the porphyry copper deposits are illustrated (modeled) and discussed elsewhere in



Lithologic interpretation developed from data posted on the cross-sectional base (bottom) of this figure

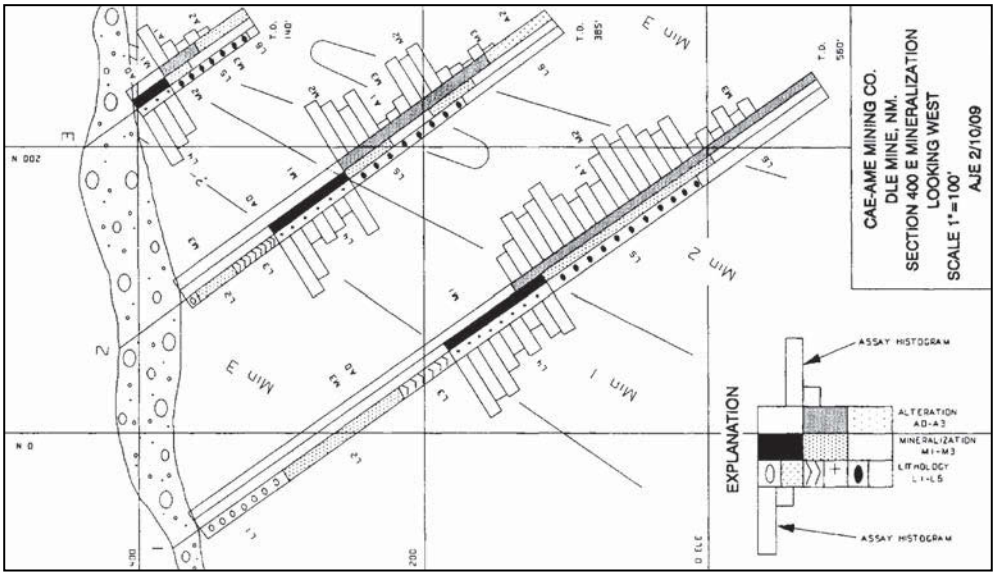


All factual data manually posted or computer generated from a database on a cross-sectional base showing coordinates, elevations, drill-trace annotated with multiple geologic variables, and histograms representing metal grades

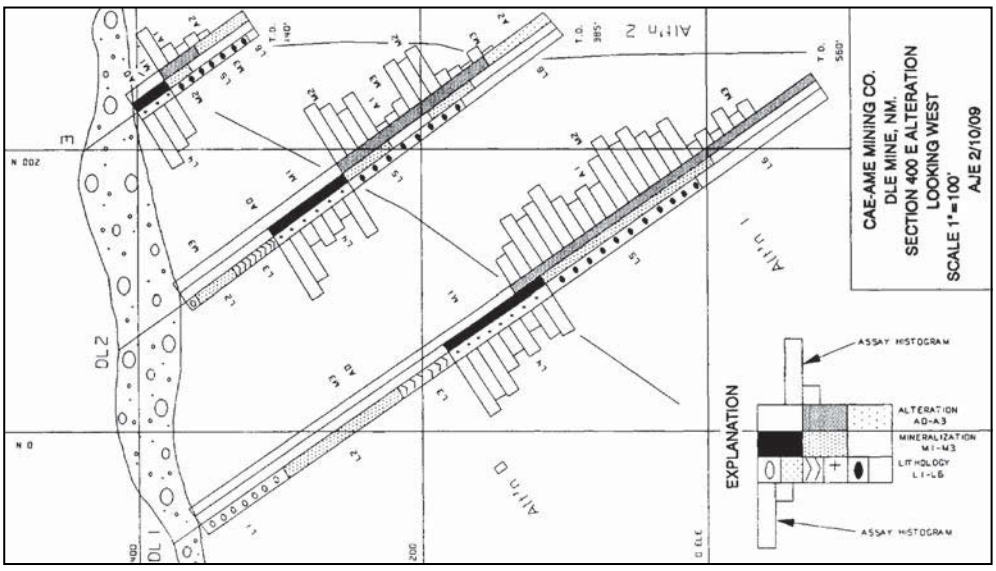
Figure 4.2-3 Idealized cross section—factual data and lithologic interpretation

the above-referenced text as well. In addition, Sillitoe (2005) and Titley (2009) provide important reviews, discussions, and illustrations of oxidation and supergene enrichment environments, processes, and resultant products. Figure 4.2-8B illustrates an idealized model of an intrusion-centered porphyry copper–molybdenum–gold deposit and possible other related deposit types that *may* be related to and/or associated with the porphyry deposit (Sillitoe 1995).

Figures 4.2-7 and 4.2-8 portray idealized models of a VMS and porphyry deposit and potentially occurring deposits in an intrusion-centered mineralization system. It is essential for the geologist to know of, recognize, and understand the mineralogical variations and variability of the deposits represented by these and other deposit-type models—to know where (in what geological zone or subzone) he or she is working, that is, collecting data in a mineralization system.



Mineralization and ore-body interpretation developed from data posted on the cross-sectional base (bottom) in Figure 4.2-3



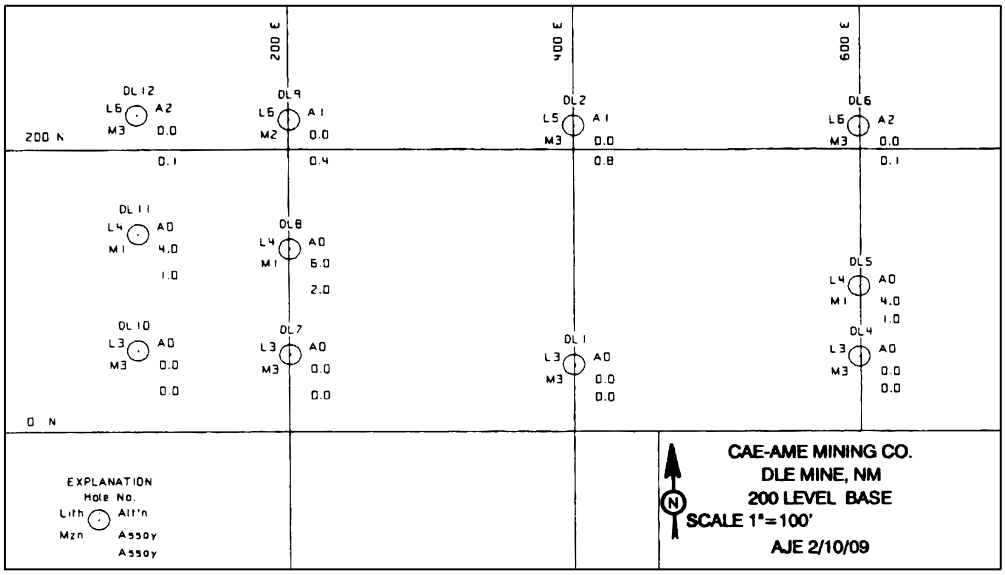
Alteration interpretation developed from data posted on the cross-sectional base (bottom) in Figure 4.2-3

Figure 4.2-4 Idealized cross section—alteration and mineralization interpretations

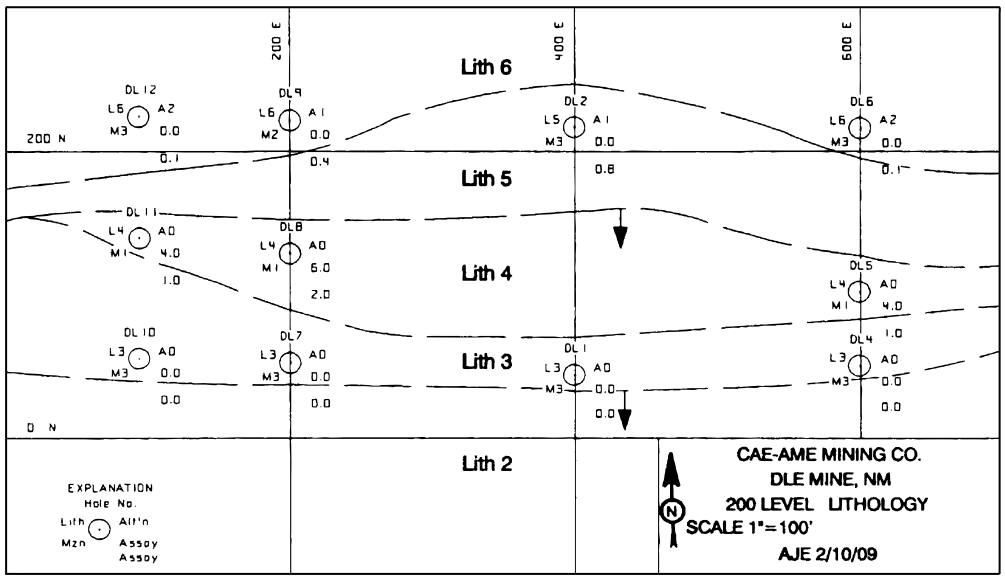
In reality, because of erosion or tectonic events such as folding and/or faulting, the geologist is commonly working on and evaluating only a portion of a total mineralization system, and any model he or she develops from the data should reflect this fact. The geologist in turn must recognize the nature and complications of this situation and the consequences and opportunities it presents.

Purpose

The geologic model is constructed to provide a clear picture of the 3-D geologic relationship of numerous features that limit varied distribution in the geologic resource. The plans and sections used to develop the model form the basis for resource estimation in conventional schemes for calculating reserves. They provide the input for limiting controls, generally as digitized polygons, in computer-assisted methodologies and are the primary standard of comparison between conventional and computer-generated resource estimates. The model provides the limits needed to prevent algorithms from estimating resources in unmineralized areas. A well-constructed model ensures that the algorithms are being used on correlatable units or features.



A. Plan map at the 200 elevation showing factual data in drill holes



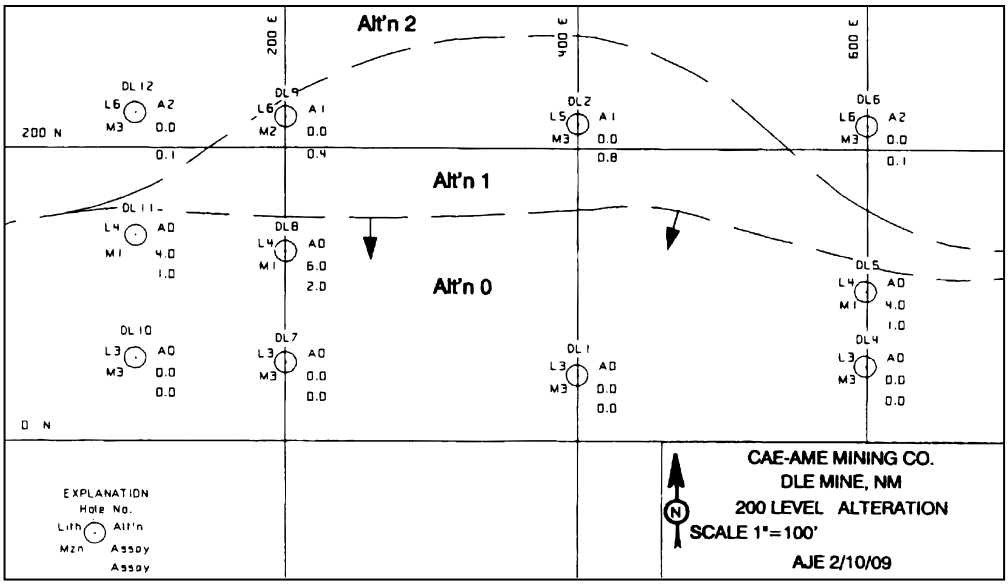
B. Lithology interpretation at the elevation of the plan map developed from factual data in the drill holes and interpreted contacts transferred from cross section 400E in Figure 4.2-3 (top)

Note: Only contacts transferred from one cross section are shown, whereas in actual practice, data from many sections would be used.

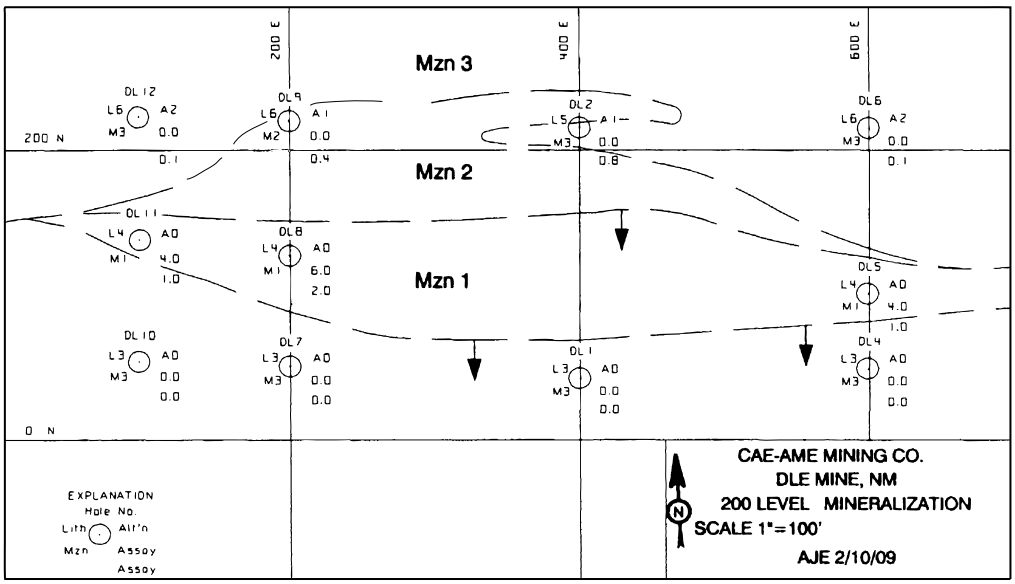
Figure 4.2-5 Level maps—factual data and lithologic interpretation

Frequently, simplified illustrations that depict the model and explain the observed geologic relationships are developed. These illustrations show the gross features of the deposit and perhaps the surrounding environment at time of formation. They are used for discussion, illustration, and perhaps comparison with generally accepted models of the type of deposit in question to ensure the appropriateness of the model. Although these simple illustrations are unfortunately sometimes referred to as cartoons, they are extremely useful for conveying concepts in discussions with technical experts involved in deposit evaluation and in management presentations. Figures 4.2-7 and 4.2-8 illustrate this type of display.

It is becoming increasingly apparent as more sophisticated computer-modeling techniques are developed that, assuming correct analytical data and posting are available, the fundamental control of sound geologic interpretation is the most important factor in resource evaluation. Superior models require superior data collection and interpretative work to provide superior resource calculations.



A. Alteration interpretation at the elevation of the plan map developed from factual data in the drill holes and interpreted contacts transferred from cross section 400E in Figure 4.2-4 (bottom)



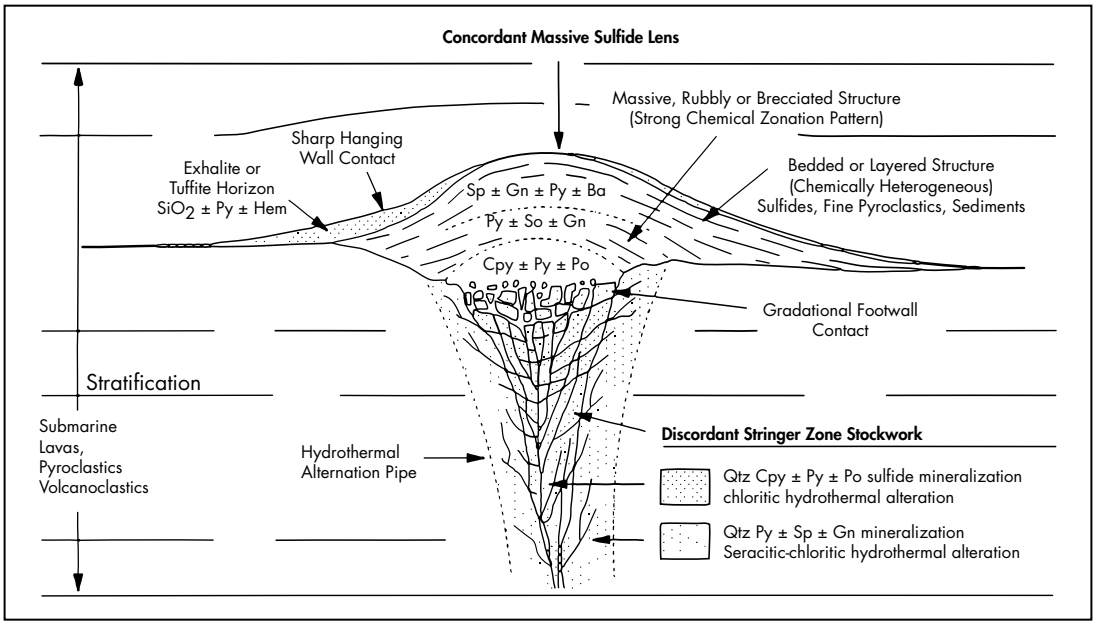
B. Mineralization and ore-body interpretation developed from data in Figure 4.2-4 (top) in a similar manner to lithologic and alteration interpretations shown in Figures 4.2-5B and 4.2-6A

Figure 4.2-6 Level maps—alteration and mineralization interpretations

Considering the risk associated with financial commitments in the resource industries, unquestionably, in the evaluation of all but the smallest resource, the best way to handle the massive amounts of data and to test sensitivities as parameters are varied is through computer modeling of the resource. These models, whether block, gridded seam, or others, take considerable work to construct and control and require sound geologic judgment and experience. When the computer model is constructed, the parameters can be rapidly varied and analyzed to determine sensitivities. It is imperative to have a correct resource determination to support evaluation and risk assessment and ensure that the heavy financial commitment required for a new mine will lead to a profitable outcome. The only way to effectively discharge this important geologic responsibility is through construction of an accurate geologic model.

GEOLOGIC REPRESENTATION

Geologic features of mineral resources are highly variable. A nearly constant change in 3-D shapes and relationships is the rule rather than the exception. Many parameters of varying



Source: Adapted from Roberts and Sheahan 1988.

Figure 4.2-7 Empirical geologic model of VMS deposit

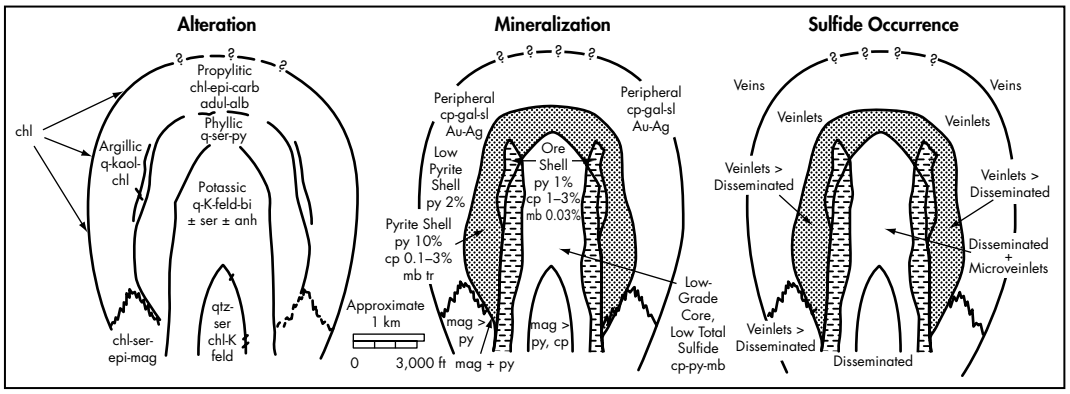
significance such as lithology, alteration, mineralization intensity, structural complexity, or mineralogical variability perhaps affecting resource recovery must be considered in order to understand the resource and its normally complex geometrical shapes. It is essential that presentations or illustrations (hard copy, computer screen plans and sections or 3-D visualizations) used in describing these complex relationships present sufficient, but not excessive, detail to provide accurate, clear, qualitative, and where possible quantitative, or at least semiquantitative, representations of the important features. The illustrations must be systematic, consistent, preferably simple, and standardized, because professionals from various disciplines (frequently nontechnical) will be using them. Relationships depicted or documented by the illustration should be instantly obvious with a minimal chance for confusion or misinterpretation. The user must be able to quickly grasp the significance of relationships. Additionally, illustrations should be easy to update to ensure that new data can be rapidly assimilated and available to support the decision-making process.

The basic illustrations commonly used to depict important geologic relationships should consist of an integrated set of plan maps and sections showing distributions of lithologies, alteration, and the resource under study. Structural features are commonly shown on all sets. Combinations of various parameters such as mineralization with lithology or perhaps isograde reflecting sulfur distribution within mineable areas of a specific coal seam are common. Almost any feature (parameter) showing variability can be mapped and, if it is important to resource characterization, illustrated to depict the importance of the relationship. Sets of illustrations should have common coordinates and points of origin so they include identical areas. Sheet size should be standardized—neither so small that it prevents adequate coverage of important areas nor so large as to be unwieldy if frequently used. Sheets 760 × 1,070 mm (30 × 42 in.), a standard size readily available, are a good choice for most geologic work. Sheets should have coordinate grid lines with full numerical values of the coordinates identified, not coordinates with some digits removed for ease in posting, as this leads to questions concerning the actual XYZ location of the data point(s) under discussion (e.g., 2,550,000 N should be stated as such and not abridged to 50,000 N). Similarly, elevations on cross sections should not be abridged or modified to reflect anything other than true elevation as related to sea level. Although schemes are used that add or subtract constants to true elevations, they tend to confuse and are best avoided.

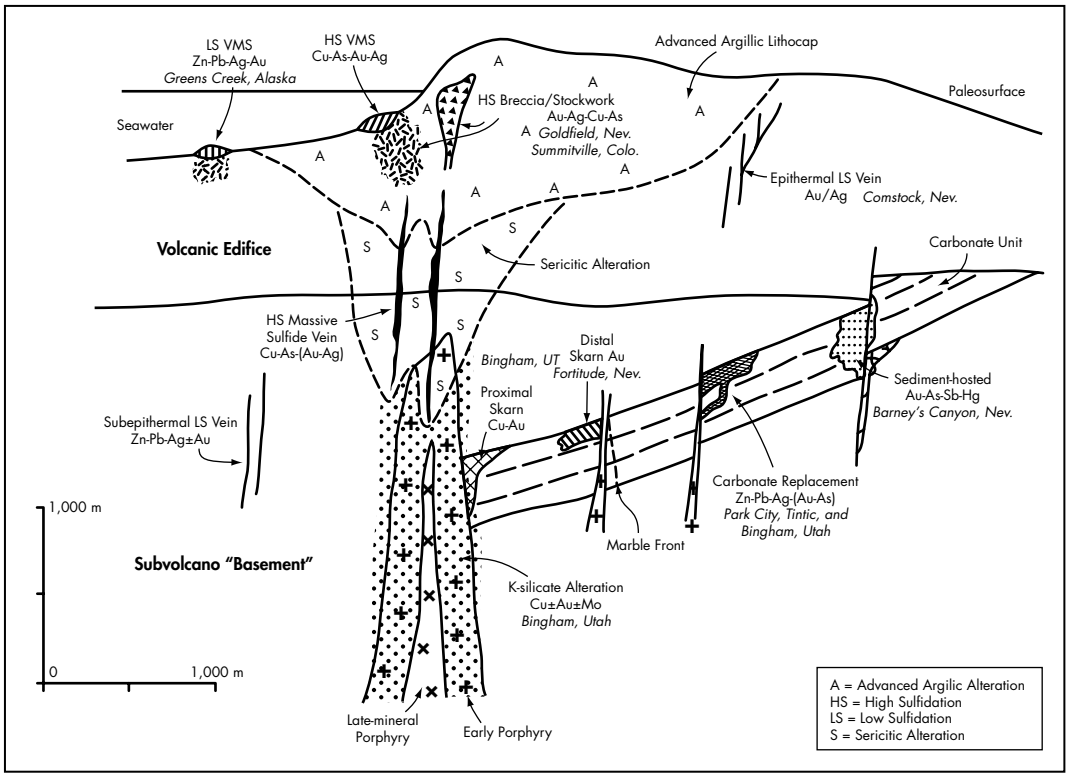
Legends or explanations should be complete and should appear on each illustration. The legend should include all symbology and should show the scale as both a bar and ratio or representative fraction. Location name, date, topic and/or subtopic, and a north arrow(s) as appropriate showing true, magnetic, or grid north should be included. Magnetic declination on plan maps may or may not be needed.

As indicated previously and shown in Figures 4.2-3 through 4.2-6, illustrations should be clear and reproducible with sufficient detail included to provide appropriate understanding and with fact clearly discernible from inference. A base reproducible set showing all factual data should be maintained in order to provide easy copies for interpretative work. The interpretative sets, constructed from the base set, are the result of the data analysis process and form the basis for geologic predictions and projections into unsampled areas. Copies, either or both hard paper or as electronic files, of these interpretations and resource estimates should be archived, as they will be needed for review, both as new data becomes available and during the operating life of the mine. These reviews, typically plan versus actual production reconciliations studies—commonly an ongoing process in producing operations—must be based on original data, interpretations, and estimation that should be available in the archived material.

Key simplified sets such as mineralization outlines, grade distribution, perhaps isograde, alteration, or coal thickness with sulfur content contours will typically be made available



A. Empirical model of mineral zones in a porphyry copper deposit based on work on the San Manuel–Kalamazoo (Arizona, United States) ore bodies



B. Model showing lateral and vertical metal and deposit type zoning related to an intrusion-centered system with selected U.S. deposits identified. Lateral: Porphyry Cu to sediment-hosted Au deposit.

Source: (A) Adapted from Guilbert and Park 1986; (B) adapted from Sillitoe 1995.

Figure 4.2-8 Geologic models

to operating personnel for mine planning and operational support.

Typical scales of illustration may vary as follows.

- For initial data compilation in a base metal deposit: the extremely detailed and seldom used 1:120 (1 in. = 10 ft) or the more common 1:240 (1 in. = 20 ft) or 1:600 (1 in. = 50 ft)
- For interpretive compilations and mine planning: 1:1200 (1 in. = 100 ft) or 1:2400 (1 in. = 200 ft)
- For summary presentations or district or regional overviews: 1:4800 (1 in. = 400 ft), 1:6000 (1 in. = 500 ft), 1:12,000 (1 in. = 1,000 ft), 1:24,000 (1 in. = 2,000 ft), or a scale in which an inch equals a mile (1:62,880 [1 in. = 5,280 ft])

Numerous other types of maps and illustrations can be constructed on an as-needed basis to depict or clarify important geologic relationships. Most useful among these are the following: structure contours or isopachs of geologic units or

parameters, isograde of metal values, total sulfide content, or perhaps dilutents as sulfur or ash content of coals, and ratio maps or combinations of parameters such as grade and thickness, common in uranium work. Any geologic parameter that can be quantified can be mapped or illustrated and may provide extremely useful data on trends, distributions, or variability; hence it contributes to interpretation and a better understanding of the geologic conditions associated with a resource. The key points on the use of any of these are accuracy, clarity, simplicity, and reproducibility.

Herness (1977) discusses subsurface geologic representation and provides a list of essentials for effective field and office representation of geologic data, abridged and summarized as follows:

- Techniques should be easy to master and rapid to use.
- All megascopically recognizable features should be capable of being represented.
- Representation legends should be logical, systematic, chromatic, or geometric sequences clearly depicting trends.
- Systems should be planned to prevent duplication, unnecessary maps, and map overlap.
- Ongoing work maps should be easy to revise.
- Sections and plan maps should be integrated to allow for efficient utilization and prevent data loss in files.
- Note sheets should be uniform in size, kept in a logical available file, and never discarded.
- Office maps should not be excessively large to ensure ease of handling and prevent damage. (However, a good set of large, highly generalized, attractive display maps should be available for presentations.)
- Coordinate grids should be parallel to the edges of note and map sheets, and maps should conform to a district-wide grid and not overlap.
- Maps and note sheets should be clear, neat, and pleasing in appearance. They must have sales appeal because difficult concepts are being portrayed and a project's funding is commonly dependent on clarity of presentation. Impressive, well-executed note sheets and maps generally attest to the quality of the geologic work and of the geologist doing the work.

The variety, types, styles, and purposes of geologic illustration are too varied to allow an example of each. Any of the professional technical geologic journals can be reviewed for style and content. In addition, the following references are particularly good in their content of illustrations: LeRoy et al. (1977), Titley (1982, particularly the included papers by M.T. Einaudi); Dixon (1979), Roberts and Sheahan (1988, very clear and simple, sometimes small), Barnes (1981), as well as the references cited previously in the "Geologic Modeling" section of this chapter.

In summary, there is no substitute for high-quality resource characterization and evaluation based on sound geologic understanding and judgment and presented in an accurate, clear, and lucid manner.

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CHAPTER 4.3

Sample Preparation and Assaying

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The sample preparation and assaying procedures discussed in this chapter apply equally to in-house facilities as well as commercial laboratories. They are applicable to samples derived from reconnaissance exploration through development drilling to mine operations and mineral processing. These procedures assume that the samples submitted were collected properly and a representative portion of the original samples is submitted for preparation and analysis. The character of the material being sampled influences the size of the sample that should be collected and the manner in which it should be prepared for analysis. Ingamells and Pitard (1986) present a good review of the necessity for the collection of appropriate samples as well as techniques for establishing proper sample sizes. Abbott (2007) emphasizes the importance of quality assurance and quality control procedures throughout the life of a project.

The focus herein is on operations in metal mines; indeed, much worldwide mining activity at this writing is concerned with exploration for and development of gold deposits. The requirements for the preparation of samples of many gold-bearing materials are more stringent than those for many other metallic deposits. Thus, the practices described here generally fulfill the requirements for most metallic mineral deposits as well as many nonmetallics.

The preparation and testing of coal samples is a specialized field, and these procedures are summarized later in this chapter. As is true for coal, the preparation and testing of samples of industrial minerals raw materials is highly specialized. For many products, the preparation and testing procedures are end-product sensitive; that is, the preparation and testing procedures to be used are controlled by the properties desired in the final product. One must attempt to select sample preparation and testing procedures that will produce a product comparable to one produced by the actual process procedures to be used. *Industrial Minerals and Rocks* (Kogel et al. 2006) presents perhaps the best overall background on a wide variety of industrial minerals products, although little specifically on sample preparation and testing. Few additional general publications exist, and no specific guidelines can be given here, other than to suggest that sample preparation and testing procedures be established for individual deposits through frequent and close communication among the exploration and mining staff, laboratory personnel, mill managers, marketing staff, and end users.

SAMPLE PREPARATION

Sample preparation is the process of converting samples of geologic materials from the larger sample collected in the field or mine into finely divided homogeneous powders suitable for chemical analysis or other testing. This is accomplished by the screening of soil or sediment samples or the mechanical reduction of pieces of rock to a smaller particle size in a stepwise sequence, alternating with the reduction of sample volume or mass by an unbiased splitting process.

Error can be introduced in many ways during sample preparation. As a consequence, attention to detail and thorough cleaning of equipment between samples is necessary. The desired end result of sample preparation is a powder, or pulp, that contains the elements to be analyzed in the same concentrations and proportions as in the original sample received. The reduction in particle size will be affected by many factors, including particle shape, hardness, specific gravity, lubricity, malleability, residual moisture, and the quantity of clay minerals or organic matter present.

Selecting a Sample Preparation Procedure

Virtually every mineral deposit has its own characteristics, and an individual sample preparation procedure should be developed for each, unless the deposit is known to be fine-grained and relatively homogeneous. Soil and sediment samples typically are dried, sieved through 10-mesh and 80-mesh screens, and in some cases pulverized before analysis. Rock samples routinely are dried if necessary, crushed (in stages if necessary) to –10 mesh, riffle split, and a 250-g to 1-kg portion pulverized to a nominal 150 or 200 mesh.

A "safe" sample preparation procedure is given by Royle (1988) based on a method originally developed by Gy (1977). Similarly, Pitard (1993) presents sampling nomographs that enable one graphically to analyze an existing

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sample preparation process and to develop an optimum protocol. Pitard markets programs for personal computers that describe tests to perform on gold ores of unknown characteristics and enable one to plot sampling nomographs from which an appropriate sampling protocol can be developed. In their Chapter 1, titled "Sampling," Ingamells and Pitard (1986) present a good review of the necessity for intelligent sampling, both before and during the sample preparation and analytical stages.

Sample Preparation Equipment

The equipment required to adequately prepare mineral samples for analysis depends to some extent on the nature and quantity of the samples, and even on the climatic environment. In hot desert environments, samples can be adequately air dried under the sun, and a laboratory with only a handful of small rock samples to crush and pulverize per day could do an adequate job with a manual bucking board and muller.

Dryers

Electric or gas-fired ovens are used to remove moisture from samples before crushing and pulverizing. An airflow is maintained through the oven to remove water vapor released from the samples. For routine assay purposes, oven temperatures are usually maintained from 104° to 140°C (220° to 285°F), the higher temperatures being used on clays, although the temperature should not exceed 37°C (100°F) if mercury is to be determined. The submission of larger and excessively wet drill-cutting samples to high-volume minerals laboratories has started a trend toward drying rooms or even buildings equipped to dry large quantities of samples.

Screens

Soil and sediment samples should be screened through screens with both frame and screen as well as a pan made of stainless steel, the screen pressed or welded in, not soldered. Screen sizes typically are 10 mesh to remove coarse fragments and 80 mesh for the final product.

Crushers

Jaw crusher. The first stage of crushing normally is accomplished in a laboratory-sized jaw crusher. Some of these can be choke-fed through a hopper with feed up to 100 mm (4 in.) across, and they will produce a 2.4-mm (–8 mesh) product in one pass while the operator is attending to other duties. A newer double-acting jaw crusher with one jaw top driven and one bottom driven has a 25:1 reduction ratio and will produce a –10 mesh product. Jaw crushers have a relatively high productivity and reduction ratio and are generally easy to clean between samples.

Cone crusher. Normally used in a second stage of crushing, cone crushers produce a uniform-sized product with a smaller percentage of fines, considered better for metallurgical testing, and most can crush to 2 mm (10 mesh). Cone crushers are not effective on clays, but work best on hard, siliceous materials. They have a relatively low productivity and are difficult to clean, especially after clayey samples.

Roll crusher. As an alternative to cone crushers for second-stage crushing, roll crushers have a higher productivity and can produce a 2-mm (10-mesh) product. However, they produce a poorly sorted product with a higher percentage of fines and are noisy, dusty, and difficult to clean. The feed to roll crushers must be sized to 10 or 12 mm (⅜ or ¼ in.), and the feed rate must be controlled to prevent choking.

Hammer mill. Hammer mills have a high productivity rate and the potential to produce a product suitable for splitting and pulverizing in one pass from feed as large as 100 mm (4 in.). However, they are extremely noisy, dusty, hard to clean, and subject to excessive wear when processing tough, siliceous materials. Their product is not well sorted, typically consisting of a large percentage of fines with a small percentage of very coarse fragments. Hammer mills are more often used to crush clays, limestone, coal, and similar softer materials.

Splitters

At some point in the sample size-reduction process, it becomes impractical and unnecessary to further reduce the particle size of the entire sample. The sample volume is then reduced by half or more, depending on the procedure selected for the material at hand, by using a sample splitter.

Riffle splitter. The riffle splitter, or Jones splitter, is most commonly used for sample size reduction. The technique of splitting must be carefully monitored to ensure that statistically valid splits are taken. Splitter chutes should be at least three times as wide as the diameter of the largest particles in the sample, and the delivery pan should be no longer than the distance across all of the chutes. The sample should be evenly distributed along the length of the delivery pan and should be poured along the center line of the splitter—not against one side or the other. The rate of pouring must be slow enough to avoid choking the chutes. The splitter must be cleaned between samples.

Rotating sectorial splitter. The most effective splitter, in terms of sampling error, is the rotating sectorial splitter (Allen and Khan 1970). Shop-made laboratory-sized versions of the rotating sectorial splitter are in use in some sample preparation facilities. One design consists of a rotating circular table driven by a variable-speed motor at about 10 rpm. Four plastic cartons of 1- or 2-L (1-pt or 1-qt) capacity rest on the rotating table. The gaps between the cartons are covered by pieces of angle iron. The sample is fed from a feed hopper by a vibratory feeder. Professionally manufactured rotating sectorial splitters are available that will produce a split of any proportion from 2% to 50% of the original sample.

Pulverizers

After the sample is reduced to an appropriate weight (typically 100 to 500 g, or ¼ to 1 lb), it is then pulverized to a nominal 150 to 75 μm (100 to 200 mesh). Two basic types of pulverizers have been in common use for many years—plate pulverizers and vibratory ring mills or swing mills—although plate pulverizers are being phased out.

Plate pulverizers. Plate pulverizers reduce the particle size of samples through a shearing action as the samples pass between a fixed and a rotating plate. Plates are made of steel, or ceramic if metallic contamination is to be avoided. Controlling the spacing between the plates (and thus the product size) has been a problem and has required almost constant attention by the operator, but newer models of plate pulverizers have better mechanical or even pneumatic controls that maintain a constant plate separation. They can pulverize a large sample and are easily cleaned, but they are dusty and their product is inhomogeneous, requiring

thorough blending before assaying. Mikli (1986) does not recommend plate pulverizers for the final pulverizing of nuggety gold ores, as the plate pulverizer does not significantly reduce the particle size of gold nuggets. Plate wear is high, requiring frequent changing and refinishing or replacing of plates.

Vibratory ring or swing mills. Two basic versions of vibratory mills are made. The ring mill consists of a steel bowl with lid, the bowl containing a cylindrical steel puck plus one or two steel rings that surround the puck. The crushed sample is placed in the open space between the wall of the bowl, the rings, and the puck. The bowl is clamped in a housing which is made to oscillate around a vertical axis by an electric motor carrying an eccentric weight on its shaft. The oscillatory motion causes the puck and rings to revolve in a planetary motion inside the bowl, thus grinding the sample.

The second version of this mill also consists of a bowl with lid but only a single grinding element, a discus or "flying saucer"shaped oblate spheroid of steel with a flat rim. The center of gravity of this steel element is off center so that when the bowl oscillates, the grinding element revolves in a planetary pattern within the bowl. Bowls of this style have a capacity of 800 g to 5 kg (1.8 to 11 lb) of sample, whereas the puck and ring bowls range from 50 to 450 g (0.1 to 1 lb) in capacity. An alternative to the single grinding element is a split discus consisting of two elements with matching concave or convex upper and lower surfaces that oscillate independently in a planetary pattern.

Vibratory mills exhibit the following advantages: they require no operator adjustment, produce a relatively homogeneous product that requires no further blending, create little dust, are available in a variety of materials to avoid sample contamination, have a low noise level (because of a supplied noise-suppressant cabinet), and have a moderate productivity of some 10 to 20 samples per hour. The productivity depends on the character of the samples, with the lower productivity being achieved on high-clay samples, which require just a few minutes to pulverize but several minutes' cleanup time between samples. Vibratory mills have been described by Mikli (1986) as the only type of pulverizer that can actually reduce the particle size of gold nuggets. However, to reduce the particle size of coarse nuggety gold or to pulverize a 2-kg (4.5-lb) or larger sample thoroughly to 150 μm (–100 mesh) or finer can require 10 or more minutes and result in excessive bowl wear. Coarse nuggety gold is best screened out, weighed, and assayed separately from the fine fraction of the sample.

Blenders and Pulp Splitters

Pulps prepared on plate pulverizers, or pulps that have been extensively vibrated during transport, require blending. The most common approach (although not the most effective) is to roll the pulp on a rubberized cloth. Taggart (1945) gives instructions for proper rolling: "Rolling is accomplished by drawing the corners of the cloth horizontally toward diagonally opposite corners, causing the sample to roll over and over on itself. If the corner is lifted instead of drawn horizontally, the sample merely slides along the surface of the cloth and no mixing occurs." Plastic sheeting should not be used for blending because of the static charges that build up, causing retention of some particles and difficulty in cleaning.

A superior alternative to rolling is the use of a mechanical blender; however, small sizes suitable for blending assay pulps are not common. Individual sample blenders are slow. A multisample mechanical wheel blender that meets the productivity requirements of a high-volume minerals laboratory has been described by Gilbert (1987).

A simple approach to splitting a pulp is to roll it on a rolling cloth into a "sausage," flatten the sausage to the height of the scoop being used, and then cut several increments from the sausage using a flat-bottomed vertical-sided scoop until the required weight has been withdrawn. Mechanical rotating sectorial splitters appropriate for splitting pulps are available and are used in some automated sample preparation systems.

Automated Sample Preparation Systems

Mine laboratories that process large numbers of samples of similar materials on a routine basis have invested in automated sample preparation systems. Designed individually for each operation, the systems range from simply crushing and splitting to complex systems involving two stages of crushing, rotary splitting, two stages of pulverizing in continuous flow ring mills, rotary splitting of the resulting pulp, and even weighing into test tubes or beakers for analysis. Automated sample preparation systems cost from several tens of thousands to a few millions of dollars, but their cost is offset by high productivity and dramatically lower labor costs.

ASSAY METHODS

Two basic classes of assay methods historically have been available: geochemical and quantitative, but current instrumental methods of measurement and standardized dissolution techniques have tended to merge the two classes of analysis. Geochemical procedures typically are used in prospecting and the early stages of exploration when results of high accuracy and precision are not as necessary but low levels of detection are required. Quantitative procedures are used during exploration drilling, sampling and analysis for ore reserve estimation, and subsequent stages of mine development and operation. Geochemical methods of analysis in the past have been considered semiquantitative, but most of these now approach quantitative methods in accuracy and precision, and they offer the advantage of considerably lower levels of detection.

The following procedures are not applicable to all materials encountered in the mineral industry. For example, placer samples should be processed by gravity methods (panning, sluicing, jigging) to produce results comparable to the recoveries to be expected from gravity production techniques. Nuggety gold-bearing rock samples are best analyzed either by screen fire assays or by bottle-roll cyanide leach tests of large samples in preference to routine fire assays or wet chemical analyses of smaller samples.

Whether one is establishing an in-house laboratory or selecting a commercial laboratory, there is no substitute for a chemist with experience in the analysis of earth materials. The wide variety of materials to be analyzed, constituents to be determined, and matrix compositions to be encountered pose a myriad of challenges to the minerals analyst. Constructive comments on the art of geochemical analysis are given by Ingamells and Pitard (1986). Many methods of analysis of earth materials are given in publications of the U.S. Geological Survey, the Geological Survey of Canada, and several Australian organizations.

Wet Chemistry

Most analytical procedures today involve wet chemical digestion of the sample, followed by instrumental measurement of the elements of interest.

Digestion Techniques

The most common digestion techniques in current use are aqua regia, four acids (hydrofluoric, hydrochloric, nitric, and perchloric), and fusions using either lithium borate, lithium metaborate, or sodium peroxide followed by acid digestion of the fusion product. Aqua regia digestions may not liberate all elements quantitatively, especially from more resistive minerals, but the acid-leachable results are considered by some to be valuable in geochemical exploration. Fusions followed by acid dissolution produce the most quantitative liberation of a wide suite of elements, but are somewhat slower and more expensive than simple acid digestions. Purposely weak selective leaches are used in some geochemical exploration programs. They include demineralized water, ammonium acetate, hydroxylamine hydrochloride, and sodium pyrophosphate, as well as the proprietary procedures known as Enzyme Leach, Mobile Metal Ion Leach, and Terrasol Leach.

Instrumental Measurement

Instrumental methods of measurement follow the wet chemical dissolution of the sample and have dominated minerals laboratories since the development of the atomic absorption spectrophotometer (AAS) in Australia in the 1950s and the inductively coupled plasma emission spectrometer (ICP-ES) in the 1970s. AASs are used when results for only a few specific elements are desired, whereas ICP-ESs measure a large number of elements either sequentially or simultaneously. The simultaneous instrument is faster for routine work on similar materials, whereas the sequential instrument is more versatile, as it can be tuned for specific elements.

The AAS enables the analyst to measure the concentration of some cations down to a fraction of a part per million and with specialized attachments down to a part per billion. The AAS instrument generally is specific for the element selected, although there are various interferences and operational nuances that the analyst must be aware of and either avoid or compensate for.

ICP-ES instruments utilize the high energy of an argon plasma to excite atoms of various elements in a sample solution that is aspirated into the plasma. The excited atoms emit light that is passed through a spectrometer wherein the energy of light at various wavelengths is measured electronically and converted into the concentration of each element in the sample solution. A more sensitive version of the ICP-ES couples plasma excitation of the sample with a mass spectrometer to measure isotopes of many elements, enabling the reporting of more than 60 elements, in some cases at detection limits of a few parts per billion. ICP-ESs have a linear response over some five to seven orders of magnitude of concentration of the element in question, a much broader operating range than the AAS.

Several other instruments are used for specific determinations in minerals laboratories. Specific ion electrodes, for example, are used to measure the concentration of cyanide in dilute solutions as well as the content of fluorine, chlorine, and several other constituents in earth materials.

Fire Assay

A fire assay is a chemical fusion method for separating, concentrating, and measuring the content of gold and silver in exploration samples, ores, and concentrates. The pulverized sample is weighed, mixed with a lead oxide–alkali carbonate flux and a small amount of a reducing agent such as flour, and fused in a fire-clay crucible. The reduced lead collects the precious metals as it settles down through the melt. The molten charge is then poured into a mold to cool. The lead sinks to the bottom of the mold and is broken from the glassy slag when cool. The precious metals are separated from the lead by cupellation. The lead button containing the precious metals is placed in a cupel of bone ash or magnesia, which is heated in the furnace under oxidizing conditions. The cupel acts as a semipermeable membrane, allowing the lead oxide formed from the button to be absorbed into the cupel, leaving the precious metals in a tiny bead on the surface of the cupel. The bead is weighed, the silver is chemically separated from the gold, and the resulting gold bead is either weighed or dissolved and measured instrumentally. Fire assay is the standard method of the industry. Details, theory, and variations of the method are described in references such as Bugbee (1940), Haffty et al. (1977), and Heady and Broadhead (1976).

Nondestructive Analysis

X-Ray

The X-ray diffractometer is used in some well-equipped minerals laboratories to determine the mineral species present in a sample by recording their characteristic crystallographic patterns. The instruments are not extensively used in mine and project assay work.

X-ray fluorescence spectrometry (XRF) is used for the rapid analysis of silicate rocks and the routine determination of the concentration of selected elements in exploration samples, ores, concentrates, and mill products. XRF analysis is most satisfactory when applied to a continuing series of samples of similar matrix. The method is subject to matrix effects and interelement interferences, most of which can be corrected for in the comprehensive computer software that accompanies all modern instruments. XRF instruments require for calibration previously analyzed standards similar in bulk composition to the unknowns. The instruments are capable of excellent precision, but without proper calibration and intelligent operation, they can be rather inaccurate.

Neutron Activation Analysis

Neutron activation analysis (NAA) is available, principally through commercial laboratories with access to nuclear reactors. Most of the laboratories offering commercial neutron activation analyses of earth materials at a reasonable cost are Canadian based. NAA is indicated when (1) a conventional technique does not have acceptable limits of detection for the element of interest, (2) the sample is unique and cannot be consumed in analysis, (3) only a small quantity of sample material is available, or (4) conventional methods of analysis are unacceptable because of interferences or inherent instrumental errors. The sample typically is pulverized, loaded into a capsule (rabbit), and introduced to a reactor in which it is bombarded by neutrons. After recovery of the capsule from the reactor, the radioactivity induced in the sample is measured and analyzed, thus giving a measure of the concentration of

each element present in the sample. NAA currently has much application in the analysis of vegetation or mull for trace amounts of gold, and in the analysis of the platinum-group elements, the rare earth elements, and uranium.

COAL PREPARATION AND ANALYSIS

The procedures used in the preparation of coal and coke samples for analysis are similar to those of rock samples. The ASTM International standard method of preparing coal samples is ASTM D 2013-07 ("Standard Practice for Preparing Coal Samples for Analysis"; ASTM International 2009). The principal difference in coal sample preparation is the use of lower temperatures and, indeed, even air drying to preclude oxidation as much as possible. Crushing and pulverizing is accomplished with much the same equipment as used on rocks. However, samples are pulverized only to 250 μm (–60 mesh), and only 50 g (0.1 lb) is retained for analysis.

A variety of tests are available for coal samples. Some of these include sieve analyses, washability, Hardgrove grindability, moisture, sulfur, ash content, carbon and hydrogen content, and calorific value. Two of the most common determinations are proximate and ultimate analyses. A proximate analysis, as described in ASTM Standard Method D 3172-07 ("Standard Practice for Proximate Analysis of Coal and Coke") covers the determination of moisture, volatile matter, and ash, as well as the calculation of fixed carbon (ASTM 2009). According to ASTM, the results of a proximate analysis are used to establish the rank of coals, to show the ratio of combustible to incombustible constituents, to evaluate the coal for beneficiation and other purposes, and to provide a basis for buying and selling.

An ultimate analysis, according to ASTM Standard Method D 3176-89 (reapproved 2002), "Standard Practice for Ultimate Analysis of Coal and Coke," when tabulated along with a proximate analysis, provides the data for a cursory valuation of coal for use as fuel and coke for metallurgical purposes (ASTM 2009). An ultimate analysis includes the determination of carbon and hydrogen as well as sulfur, nitrogen, ash, and the calculation of oxygen by difference. Typically moisture is reported as well. The procedures for an ultimate analysis are also specified by ASTM International.

In addition to the analytical procedures described previously, data on the major, minor, and trace elements in coal and coke ash often are of use in the evaluation of coal quality. Accordingly, methods for these determinations are presented in ASTM D 3682-01 (reapproved 2006), "Standard Test Method for Major and Minor Elements in Combustion Residues from Coal Utilization Processes"; ASTM D 4326-04, "Standard Test Method for Major and Minor Elements in Coal and Coke Ash by X-Ray Fluorescence"; and ASTM D 6357-04, "Standard Test Method for Determination of Trace Elements in Coal, Coke, and Combustion Residues from Coal Utilization Processes by Inductively Coupled Plasma Atomic Emission, Inductively Coupled Plasma Mass, and Graphite Furnace Atomic Absorption Spectrometry" (ASTM 2009).

QUALITY CONTROL

The necessity of establishing a quality control program for every project is presented by Abbott (2007). Most laboratories have an adequate quality control program covering their analytical work, and many commercial laboratories currently are certified under the International Organization for Standardization quality control standard ISO 9001:2000, currently being supplanted by ISO 9001:2008. How intensively the quality of sample preparation is controlled is unknown, yet it is just as important as the chemistry, for, as A.G. Royle (1989, personal communication) has commented, "Wait until you see the numbers that show all the horrible things that happen in sample preparation of gold samples!"

During sample preparation, maximum particle size can be readily monitored by screening at various stages as the material is crushed and pulverized. Other parameters are more easily checked by actually analyzing the material. Analysis of several pulps prepared from different splits of the crushed bulk sample will provide data on reproducibility of splitting. Analogously, the homogeneity of a pulp can be checked by taking a number of replicate portions for analysis from the same pulp and calculating the precision of replication. A simple procedure to evaluate the effectiveness of sampling drill cuttings is presented by Schwarz (1989).

Analytical precision and accuracy are best established and maintained through the use of reference standard samples and analytical control samples (Hill 1975). Certified standard samples of mineral materials are available from mineral industry suppliers and some governmental agencies and professional or trade groups in several countries, but such standard materials are expensive and limited in quantity. The elements present and concentration ranges in certified standards may not adequately cover the character of materials that the laboratory is engaged with. Nevertheless, such standard materials may help to provide initial calibration. Matrix-matched control samples to provide batch-to-batch and day-to-day calibration and quality control in the laboratory can be prepared and blended from the reject samples from the project. The analytical development of these in-house control samples can be calibrated to the certified standards. Ideally, every set of analyses produced by the laboratory should have one or more control samples and duplicates included for quality control and assurance. Some governmental agencies that contract out a large quantity of sample preparation and analytical work arrange for 3 of each 20 samples to be controls or replicates. ASTM gives guidelines for evaluating laboratories performing analysis of coal and coke (Method D 4182-97, reapproved 2004, "Standard Practice for Evaluation of Laboratories Using ASTM Procedures in the Sampling and Analysis of Coal and Coke"; ASTM 2009).

When the results of analysis of the controls do not agree with the established values within acceptable limits, the entire group of assays should be rejected, the problem identified and resolved, and the group of samples rerun. The timely realization and resolving of sampling, sample preparation, and analytical problems is the essence of quality control.

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CHAPTER 4.4

Ore-Body Sampling and Metallurgical Testing

Philip King

The evaluation and development of an ore body requires the input of many disciplines, including geology and related subjects, mining engineering, process engineering, and environmental sciences, as well as marketing and financial modeling. Work commences during the exploration phase with extensive sampling programs designed primarily to evaluate the grades and tonnages of mineralization present.

In a feasibility study, a deposit's ore reserve grades and tonnages are quoted according to strict accounting procedures such as those of the Australasian Joint Ore Reserves Committee (JORC), its variants in the west, and the State Commission on Mineral Reserves of Russia (GKZ) in the former Commonwealth of Independent States countries. However, methods for evaluating exactly what is recoverable from an ore body are not well defined and it is usually up to the process engineer, with assistance from geologists and mining engineers, to develop a metallurgical test program.

QUALITY MONITORING IN CHEMICAL ANALYSIS

The general quality standards (not specific to laboratories) of the ISO 9000 series developed by the International Organization for Standardization (ISO) are guidelines to ensure that a quality system exists and is followed, but they do not assess actual quality or competence. ISO 17025:2005 specifies general requirements for the competence of laboratories to carry out tests and calibrations, including sampling. It covers testing and calibrations performed using standard, nonstandard, and laboratory-developed methods. It is applicable to all laboratories regardless of the number of personnel or scope of testing activities. It requires a laboratory to state its quality policies and procedures, provide building facilities and equipment appropriate to the tests performed, use suitable methods, recruit and train competent staff, maintain good document control, and keep thorough records.

Laboratories seeking accreditation must prepare a list of the tests they perform and describe the quality control (QC) procedures associated with these tests. An independent national or private organization then assesses the laboratory for accreditation. If the laboratory is accepted, it receives a registration document showing the tests for which it is accredited. Follow-up visits, usually annually, are made to maintain accreditation. Under ISO 17025, prospective clients must confirm that any laboratory that they are considering using is registered to perform their required tests in addition to having overall ISO accreditation. Although ISO 17025 is achieving international acceptance, some countries still maintain their own national systems.

With the drive toward stricter reporting conditions for mineral exploration and the requirement for so-called competent-person reports, the need for laboratories to be accredited is of increasing importance. This applies not only to assay laboratories but also to laboratories providing services such as mineral-processing test work and mineralogy.

Assay laboratories should have written procedures under the following main headings:

- Sample handling
- Sample preparation
- Sample analysis
- Quality control
- In-house QC procedures for both sample preparation and assay
- External control by assay of samples with other accepted laboratories and by participation in round-robin analytical programs such as those organized by Geostats and Canmet
- Sample documentation

Accreditation is not the same as certification. The latter requires an independent third party to give written assurance that a service conforms to specified requirements. In the context of mineral exploration and mining, where companies may have varying requirements, it is probably better that certification of a laboratory (whether external or in-house) be carried out as part of a company's own QC program.

Overall, a comprehensive QC program should increase the number of individual analyses by about 40% compared with the original number of samples being analyzed. The laboratory should maintain tables of its QC performance on all of the above factors, with charts as appropriate, and make these available to current and prospective clients. For in-house mine and

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process-plant laboratories, QC data should form part of monthly reports and be open for discussion at management meetings.

Laboratory clients should not rely on only laboratory-produced data. As part of their own QC programs, they may wish to insert duplicates, blanks, and standards as part of their sample batches.

The following terms are used in the analysis of samples:

- **Accuracy:** Closeness of agreement between a measured value and the true value. Lack of accuracy can be measured and quantified in terms of bias or systematic error.
- **Bias:** Statistically significant difference between a measured value and the true value.
- **Precision:** Magnitude of randomly distributed variations in the measurement procedure.

GEOSTATISTICS

Geostatistics is a branch of statistics concerned with analysis of not only data values but also the positions of data samples and time-related data variations. Although originally developed in the mining industry, it is now widely applied in a range of disciplines.

Basic Principles

The basic principles of geostatistics involve summary statistics, variance, variograms, and kriging.

Summary Statistics

Normal analysis of any geological data usually starts with the production of summary statistics to provide an initial view of the data ranges and distributions. Summary statistics include not only the mean, standard deviation, variance, median, and mode, but also the coefficient of variation (standard deviation/mean) and log estimates of the mean. Graphical analysis usually includes histograms, log histograms, and log probability plots that enable analysis of whether different populations are present and whether some sort of domaining is necessary. A decile analysis can also be helpful in assessing outlier grades; the data set is ordered by grade, then the metal content contained in each 1/10th of the data set is calculated and 1/100th parts for the top-most decile are analyzed.

Variance

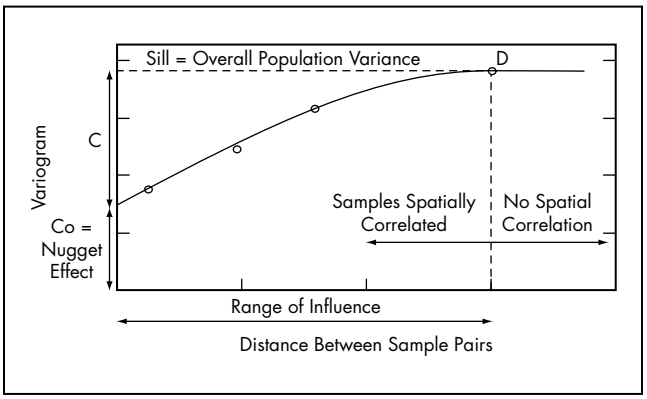
For the purposes of estimation, the variance of grade distributions is as useful, if not more, than the mean. Variance is a fundamental measure of variability and precision.

Variance between pairs of samples a specific distance apart is a measure of the interdependence of grades for points separated by this distance. Points closer together are normally more closely related (have lower variance) than are points farther apart. For pairs of samples formed by, for example, check-sampling from different laboratories, specific techniques exist for the analysis of variance (ANOVA).

Variance also depends on the size of a sample, so composites usually have a lower variance than do raw samples. As many geostatistical processes are involved in ANOVA, irregularly sized samples must usually be composited to a regular length (greater than the majority of raw sample lengths) so that the composites have the same approximate size (support) for subsequent analysis.

Variograms

In comparisons of a large number of sample pairs, some differences are positive and some are negative. Squaring these differences gives a new set of all-positive values. Differences measured between samples separated by similar distances can then be averaged, giving a variance. A graph called a variogram can be developed of variance versus distance of separation (lag) between points. Figure 4.4-1 shows a sample variogram (strictly it should be called semivariogram, as it is the variance/2), fitted with a spherical model variogram. The distance at which the variogram levels off is called the range of influence; samples separated by more than this distance are uncorrelated.



Courtesy of Adam Wheeler.

Figure 4.4-1 Sample variogram

Different variograms can be created and modeled in different directions, enabling measurement of the anisotropy of mineralization. The position where the variogram cuts the y-axis gives a measure of the so-called nugget effect (designated Co), which describes how well the sampling results can be reproduced by repeated sampling at the same location. This measure incorporates the natural inherent variability of the deposit plus the variability due to sample size, preparation, and analysis. A very high nugget effect relative to sill height may therefore indicate poor data quality, as might result when sample data stems from a mix of different laboratories over many years. A low (or near-zero) nugget effect indicates very homogeneous mineralization.

Kriging

An important use of variogram parameters in grade estimation is the so-called kriging method of estimation. Kriging involves the following steps:

1. Perform a structural study of the sample data to determine the variogram.
2. Select samples to be used for evaluation of a particular block.
3. Calculate variogram values between all selected samples as well as between each sample and the block itself to set up a kriging system of equations.
4. Solve the kriging system of equations to obtain weighting coefficients for each sample.
5. Estimate the block grade from the weighted average.

In mining and geostatistical software systems, model variogram parameters and search parameters are user defined. The search parameters themselves, particularly the distances involved, also largely stem from the variography analysis. The model software uses the parameters to complete the previous kriging steps and produce a separate kriged grade for each block of a supplied block model.

Drilling and Sampling

During a geostatistical study, variogram ranges and other derived model parameters can be helpful in determining the minimum spacings required for future drilling or sampling. If different ranges exist in different directions (anisotropy), this information can be used to guide different drilling spacings in the corresponding directions.

Ore-Body Modeling

Computer-based resource estimations usually involve the generation of block models. Each cuboid block represents a volume of ground and has a number of numeric (typically grades, densities, or metallurgical parameters) or alphanumeric (rock-type) attributes assigned to it. A parent block size is usually assigned; the size is established when the model is initially generated, although some mining software systems also allow sub-blocks to be generated within the same framework so that more complicated geological or topographical features can be accurately represented.

Volumetric Modeling

The first stage in resource modeling is generally to build a volumetric block model that uses topographical and geological surfaces and envelopes to split and code blocks. This process often involves modeling mineralized zones into which grade values are subsequently interpolated. Mineralized-zone boundaries can stem from drill-hole or other sample data and can take into account cutoff grades, lithological differences, and extrapolation of other geological features. Some mining software offers advanced facilities for extrapolating geological and ore-body features.

Grade Estimation

The next stage is generally to use grades stemming from the sample data in order to estimate grades in the mineralized-zone blocks. Often the boundaries used to define mineralized zones have also been used to set corresponding codes onto the sample data. These data can then also be composited, either into regular lengths (for large ore bodies) or cross-intersection composites (for veins or seams). Search parameters need to be defined to control which composites or samples are used for each block estimate. Generally, geostatistical analysis helps set these search parameters. An estimation method is then applied to assign weights to each selected composite/sample during a block's estimate. The block value is derived from a weighted average.

The following are commonly applied estimation methods:

- **Indicator methods.** Instead of grade values being weighted directly, composites are first set to either 0 or 1, depending on whether they are above or below a specified cutoff grade. A series of different indicator 0/1 values (for a range of cutoffs) is estimated for the block model, possibly by means of kriging. These interpolated indicator values (decimal numbers between 0 and 1) are reconstituted into a block-model grade value. A number of indicator methods and variants are available, including multiple-indicator kriging.
- **Inverse-distance weighting.** Weights are assigned that are inversely proportional to the distance between the sample and block. The distance can optionally be raised to a power (typically 2 or 3) before weights are assigned. The method is easy to apply, but does consider clustering of sample data and generally performs poorly beyond the edge of the available sample data.
- **Kriging.** Parameters defined by the model variogram are used, and the resultant weights are dependent on the sample pattern and spacing. Clustered samples automatically have their individual weightings reduced. Many variants of kriging are available, the commonest of which is probably ordinary kriging.
- **Nearest-neighbor (polygonal) weighting.** The block grade is set to the grade of the nearest available sample/composite. Results are similar to those produced by sectional evaluation, where polygons are assigned sample/composite grades.
- **Simulation methods.** In a derivative of kriging called conditional simulation, a number of alternative block models are generated, which are smaller than those used for conventional estimation methods. Grades differ for each block but can be thought of as equally likely outcomes for the same input sample/composite data. The types of grade pattern produced by this method are generally more realistic, on a smaller scale, than are those produced by conventional methods. Having a series of block-model outcomes enables good probabilistic assessment of results that can be used for resource classification, mine design, and prediction of production tonnages and grades.

Model Validation

Regardless of the estimation method used, it is recommended practice to complete a number of validation steps before performing final resource calculations, as a check on the model-generation procedures. Validation steps can include the following:

- **Visual examination.** Superimpose the sample or composite data onto sectional views of the block model and coloring and/or annotating grade data.
- **Global-average comparisons.** Determine the average grade of samples and composites within principal divisions of the mineralized zones, and compare these with corresponding average grade values from the block model.
- **Local-average comparisons.** Depending on the geometry of the ore body, capture sample and model data onto regular parallel slices (e.g., vertical sections or benches), then produce graphs depicting the variation in grade by slice position (e.g., easting). Variations in average grade from samples, composites, and models can be compared in the form of model grade profiles (swath plots) that can include principal grade fields (estimated, for example, by kriging) and also alternatively derived grades (estimated perhaps by inverse-distance or nearest-neighbor weighting).
- **Historical comparisons.** When older estimates are available, compare these with newer revised estimates to help understand differences in the updated procedures and/or data.
- **Test-block analysis.** Isolate particular blocks in the block model and study the associated grade estimates in more detail. Retrieve the sample/composite data used for estimation and plot it along with the weightings that the estimation method applied. If kriging was used, determine other parameters associated with the estimation process

along with the resultant block grade, and from these assess how "well informed" that particular block may be. Analysis parameters can include kriging variance, which is the expected value of the squared error between the actual and estimated grades and is a useful indication of the geostatistical confidence in a given block with respect to the sample data configuration. Analysis can also include comparison with reconciliation data (planned and produced tonnages and grades) when available.

Resource Evaluation

The following terms are used in ore-body assessment:

- **Average grade:** Average quantity of an ore's valuable constituents; the average relative quantity or percentage of ore, mineral, or metal content in an ore body
- **Bulk density (dry or wet):** Dry or wet weight of an object or material divided by its volume including pore spaces, often expressed in metric tons per cubic meter, or t/m^3 (for dry bulk density, the sample must have been oven-dried to a constant weight at 105°C)
- **Contained metal:** Weight of a given metal contained within a unit amount of ore or mineral-bearing material
- **Density:** Ratio of the mass of a substance to its volume
- **Specific gravity:** Ratio of the weight of a unit volume of a substance to that of water, based on the volume of solid material, excluding pore spaces
- **Stripping ratio:** Unit amount of spoil, overburden, or waste that must be removed to gain access to a unit amount of ore or mineral-bearing material, generally expressed in cubic meters of overburden to raw metric tons of mineral material (m^3/t)

Resource Classification

Along with estimated grades, some assignment of resource categories is necessary so that results can be reported according to normal resource-estimation practice. In most international systems, resource categories are measured, indicated, and inferred, reflecting decreasing levels of confidence. Only measured and indicated resources should be used as the basis for mine design and resultant ore reserve calculation.

These class categories can be set as attributes directly in the block model. Alternatively, separate physical limits can be constructed, again so that block-model evaluation can be subdivided into classes. Criteria involved in assigning classes should include the following:

- Drilling, sampling, and assay integrity
- Sample spacing and patterns
- Mineralization continuity, often described by means of model variogram parameters
- Estimation method and block size
- Samples/composites encountered during estimation; for each block, the number found and proximity to the nearest and general coverage are important
- Sample types; in some instances, older less-reliable data sets might have been used for inferred resources only
- Other modeling parameters (such as maximum distance of extrapolation)
- Reliability of other supporting measurements (such as density)
- Likely mining dimensions, as resources must be reported to a realistic degree of selectivity

Other more sophisticated procedures relevant to resource classification might include use of results from kriging operations, such as maps of kriging variance, or other forms of risk assessment (such as use of conditional simulation). For producing mines, the ore production rate may also be used such that the confidence associated with measured, indicated, and inferred resource categories corresponds to the confidence associated with tonnage/grade predictions in short-term, medium-term, and long-term mine planning.

Evaluation Methods

For ongoing or future mine development, the resource block model usually becomes the main tonnage and grade reference source for mine planning. All embedded attributes, grades, resource classes, and resource properties are useful. For open-pit mines, the model is useful for pit optimization and determination of future exploration-drilling requirements. For underground mines, the model is useful for stope optimization. Mining software provides a variety of planning tools for interactive design work using the block model as a three-dimensional backdrop.

After a resource block model has been set up with some means of resource classification, it can be evaluated in various ways. Resource evaluation can be subdivided by rock or ore type, resource classification, cutoff grade interval, or model increment such as benches. The model can be evaluated as a whole or by retrieval within defined bounds: physical bounds in the case of a mine design or a particular zone or range defined by coordinates.

Evaluation results are commonly displayed by means of grade–tonnage curves that show the available tonnage and average grade of that tonnage for a range of different cutoff grades. Figure 4.4-2 shows a sample curve. According to the figure, for a Zn cutoff grade of 4%, 11,500 t are available with an average grade of 7.6% Zn.

In these curves, the x-axis represents the cutoff grades applied. Generally the left-hand y-axis represents the tonnage above cutoff and the right-hand y-axis represents the average grade above cutoff.

It is important to be aware of the selectivity associated with a grade–tonnage curve. If the curve is produced directly from a block model, selectivity is a function of the block size and the grade-estimation parameters used. Curves produced from mining blocks defined by polygons or three-dimensional shapes may differ in shape and selectivity. Thus different curves can exist for the same deposit at different levels of selectivity.

When an economic mineral has a specific gravity that is significantly higher than those of the waste or gangue minerals, the relationship between ore density and grade should be evaluated. This can be readily achieved by plotting ore grade versus density as a scattergram and then performing regression analysis. The density of an ore containing two minerals of different specific gravities is

$$100/\Delta = x/\delta_A + (100 - x)/\delta_B$$

where

Δ = density of the ore
x = percentage of mineral A
δ_A = density of mineral A
δ_B = density of mineral B

This equation can be expanded for more than two mineral phases.

OBJECTIVES OF A METALLURGICAL TEST PROGRAM

The main objectives of a metallurgical test program are to

- Define the recoveries of the metals or minerals to a salable product;
- Determine the grade or quality of the products;
- Conduct a liberation analysis process;
- Develop one or more flow sheets for ore processing;
- If reserves are to be declared, demonstrate the economic viability of the process;
- Specify and size process equipment;
- Estimate capital and operating costs; and
- Evaluate the characteristics of tailings or waste products.

The stage of the study determines the level of detail and degree of accuracy required for a metallurgical test program. For example, a conceptual study typically has relatively little measured ore resources and only conceptual mining and process plant flow sheets, and therefore capital and operating cost estimates might be accurate to within only ±30%. In contrast, a detailed feasibility study typically has detailed design criteria including balanced process flow sheets and completed equipment designs based on firm supply quotations, and estimates might be accurate to within ±10%. The detailed study then provides the formal basis for the final contractual design of the process and the selection and sizing of plant equipment and other associated facilities. It also includes all operating criteria, including throughputs, operating schedules, design capacities, feed characteristics, and mass balances.

TYPES OF SAMPLES USED FOR METALLURGICAL TESTING

Samples for metallurgical studies can be obtained from a variety of sources, including drill cores (of diameters up to ~200 mm), trenches, small pits, and channel samples. Residual samples from initial exploration programs or from dedicated metallurgical sampling exercises can also be used.

Field-Sampling Methodologies

The prime objective of field sampling is to create a sample that is suitable for laboratory processing and representative of its source. This is the most critical step in the development of a sampling flow sheet. Errors incurred at this point are usually large (have great variance) and cannot be rectified by subsequent treatment. Major types of error include splitting in the field, loss of sample, and contamination.

Assay Pulps

The ore body at this point has been subjected to a sampling program to determine the grades of metals or minerals present, and a considerable amount of sample material often remains from this exercise. Such material can be in the form of a dry solid called an assay pulp, either moderately crushed (~12 mm), finely crushed (~1 mm), or pulverized (~75 μm). The latter is of little use in a metallurgical test program, but the crushed materials have some value. An advantage of using assay pulp is that it is usually plentiful and reasonably representative of the deposit. A disadvantage is that minerals may have deteriorated during storage, and testing based on whole-rock response (e.g., crushability) is precluded.

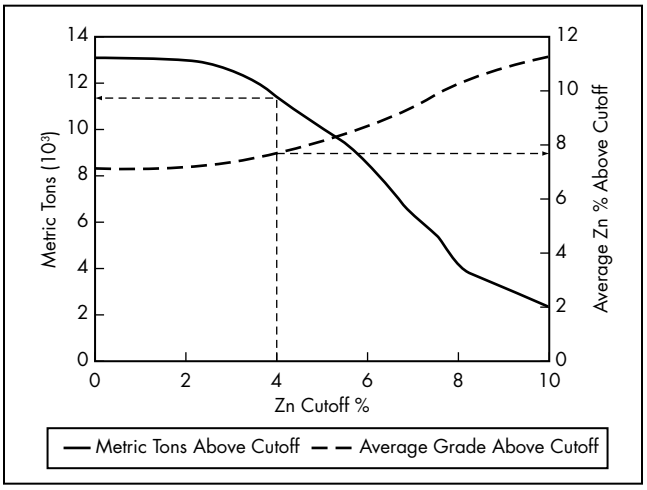


Figure 4.4-2 Sample grade–tonnage curve

Split Core

In an exploration program, diamond-drill core samples are generally cut in half. One half is sent for chemical analysis; the other half is then generally quartered, and one quarter subjected to metallurgical testing and the other quarter kept for reference.

Trenching and Pitting

Near-surface ore bodies can be sampled by means of pitting or trenching. Both are relatively inexpensive means for obtaining bulk samples for testing. A disadvantage is that the samples taken may not represent the mineralization below the level of pitting, particularly where the ore body has been subjected to weathering near the surface.

Channel Sampling

Where there is access to the ore body, the mineralized face can be channel sampled. This involves cutting a channel over the mining width without preferential recovery of highly mineralized or softer minerals. The width of the sample can be extended into the footwall or hanging wall so as to include mining dilution.

GY'S SAMPLING FORMULA

Gy's sampling formula (Gy 1979) is widely used when subsampling particulate material for a metallurgical test program. The formula predicts sampling errors incurred when a subsample is taken from a known weight of crushed or ground material. It also determines to what size a sample must be crushed and the size of subsample that must be taken to achieve satisfactory sampling accuracy. However, the formula does not determine the size of sample that should be taken when sampling an ore body, drill-core sample, or mine face.

Gy's sampling formula can be simplified to

$$M = Cd^3/s^2$$

where

M = minimum sample weight
C = sampling constant
d = top size of the material being sampled
s = a measure of the statistical error incurred during sampling

The formula considers particle size, mineral content, degree of liberation of the minerals, and particle shape. The sampling constant C depends on the material being sampled and takes into account the mineral content and its degree of liberation. It may vary with d, and often must be experimentally calibrated for the material being sampled. The term s can be used to measure the confidence in the results of the sampling exercise.

Applications of the formula involve the introduction of sampling constants, shape factors, size-range factors, liberation factors, and mineralogical factors, and the concept of fundamental error. The formula is also widely used in the sampling of alluvial material. Alluvial deposits generally contain very low levels of economic minerals and the formula can determine the size of sample that must be taken to achieve a given degree of sampling accuracy. The required size is often such that a small-scale or pilot gravity processing plant must be constructed on-site.

SAMPLE SELECTION FOR METALLURGICAL TESTING

If the style of mineralization is the same throughout an ore body, with the same minerals present in the same proportions with no significant degree of alteration, then a single composite sample can be taken for testing.

If the style of mineralization is not the same or if some doubt exists regarding ore consistency, then several samples should be taken and subjected to a range of basic mineralogy and scoping beneficiation tests. If all samples yield the same test results, the samples can be combined into a single composite sample for testing. However, it is more likely that mineralization is variable, with different styles and processing characteristics. The most common examples of this are gold and base-metal ores that have both primary and oxide ore zones, often separated by a transition zone. Mineralization within these zones is clearly different, and each zone must be tested separately. Other examples of ore-body variability include mineralization hosted within different lithologies; spatial variation in grade, either with depth or laterally; mineral alteration; and variation in the ratios of economic minerals where more than one is present.

The distribution of deleterious elements can also affect the selection of test samples. Classical related pitfalls of sample selection include the following:

- The sample is not representative of the mineralization being sampled.
- The mineralization being sampled is not representative of the rest of the deposit.
- The sample represents the average characteristics of the deposit but does not consider factors such as range of head grades, mineralogy, and physical characteristics.

LABORATORY TEST PROGRAMS

Laboratory test programs are undertaken in a number of phases, each of increasing detail, complexity, and cost.

Scoping Test Program

The first level of testing, scoping testing, identifies the processing characteristics of the ore and any potential factors that may hinder realization of its perceived economic value. The aim at this point is to identify minerals of economic interest and broadly define the processing characteristics of the ore while minimizing testing costs.

If test results are favorable and the ore body is deemed potentially viable, more detailed test programs are then undertaken. The scale of testing increases as knowledge of the deposit and confidence in the ore reserves increase.

Scoping tests can be undertaken on samples weighing as little as a few kilograms. In particular, basic optical mineralogy can reveal the limits of recoverability of certain mineral phases or the grade of mineral concentrate that can be produced. Mineral phases or elements that are deleterious to final product quality can also be identified and quantified at an early stage.

Preliminary Test Program

The second level of testing, preliminary testing, defines more clearly the processing characteristics of the ore and provides basic test data needed to size crushers, primary grinding mills, beneficiation plants, and thickeners. Preliminary testing involves more detailed analytical, mineralogical, and beneficiation tests, often on a range of samples taken from different areas of the deposit.

Preliminary tests investigate the ore's response to processing at a range of grind sizes determined from the mineralogical analysis. Beneficiation tests are performed using a range of techniques appropriate to the minerals to be recovered.

For gravity and flotation testing, initial batch tests can take the form of so-called rougher tests where the sample is processed in a single stage to produce a number of products of varying grades. Depending on the beneficiation method used, concentrates from a rougher test can then be subjected to cleaning tests to determine final product quality. A batch test does not give a final measurement of either quality or recovery, as some mineral values are lost with material rejected during cleaning. In a continual process, these cleaner tailings are normally recirculated to some degree, which increases product recovery but may well reduce the final product grade.

Physical tests performed at this stage are normally limited to the Bond ball mill work index test.

Detailed Test Program

The third level of testing, detailed testing, is normally undertaken as part of a full feasibility study. At this stage, the grade of ore to be processed should be known to a high degree of confidence. A mining plan should also have been established, although it is in the nature of feasibility studies that mine-plan optimization is normally run concurrently with the final stages of metallurgical testing and plant design.

It is important that dialogue be maintained among geologists, mining engineers, and process engineers, as mine plans can often change rapidly when computer mine-modeling systems are used. Mining engineers determine the optimum ore extraction rate and convert the geological resources into mining reserves by applying mining-recovery and dilution factors as well as conceptual mine-design and commodity-price considerations.

Samples of the products from the test program may be sent for vendor testing, particularly if performance guarantees are to be sought, and such tests are often undertaken for dewatering of tailings or concentrates. Laboratory settling tests, using measuring cylinders, can readily be undertaken on relatively small samples. Similarly, vacuum filtration tests can be undertaken using standard laboratory filter leaf tests, although tests for pressure filtration require considerably larger samples than are normally generated in a laboratory bench-scale test program. Bulk samples of concentrates or tailings for such

tests may be produced from pilot-scale testing, but if no such samples are available, vendors can size equipment based on their experience with other operations.

Bulk samples of concentrates or tailings can be produced from pilot-scale tests. If no such samples are available, vendors can size equipment based on their experience with other operations.

Variability testing should be undertaken on a range of samples to determine how the processing characteristics of the ore body vary. It may be advisable to test composite samples that represent the ore that will be treated during the crucial first 2 years of production.

TEST-PROGRAM STAGES

Test-program stages include sample preparation, chemical analysis, and mineralogical analysis.

Sample Preparation

Sample preparation generally involves the preparation of one or more composite samples, often from many individual samples. Samples may require drying and crushing to improve representivity and to initiate the comminution process.

Drying should be undertaken at a temperature that does not alter the physical or chemical nature of the minerals and can be undertaken under ambient conditions (air drying) or in an oven. Some minerals may need to be dried in an inert atmosphere.

Crushing of a dried sample is generally done with jaw, cone, or roll crushers. The crushed material is then thoroughly mixed, either manually or by rotating drums. The mixed material can be divided into subsamples by means of a Jones riffle splitter or rotary splitter or by cone and quartering techniques.

Chemical Analysis

The following are techniques for determining the major chemical elements of interest in a sample:

- Atomic absorption spectroscopy is widely used to analyze base metals and is also suitable for analyzing precious metals.
- Fire assay is the most widely accepted technique for analyzing gold and platinum group metals.
- Inductively coupled plasma is used as the main analytical method and can also produce reasonably accurate semiquantitative scans of minor and trace elements.
- X-ray fluorescence can detect major rock-forming elements and is a preferred method for analyzing niobium, tantalum, tin, tungsten, and other elements that are difficult to dissolve in acid.

Mineralogical Analysis

Optical microscopy is used initially to determine the mineralogical composition of a sample in order to identify the economic minerals present, identify the gangue or other rock-forming minerals, estimate the likely liberation size of the minerals of interest, and evaluate the degree of alteration of the minerals.

X-ray diffraction (XRD) can also be used to identify minerals, although the method is, at best, only semiquantitative. Scanning electron microscopy (SEM) can be used to identify minerals that cannot be identified optically. Electron probe microanalysis can be used for chemical analysis of mineral grains to a low level of detection.

In recent years, automated SEM image analysis has become possible due to the advent of QEMSCAN (Quantitative Evaluation of Minerals by Scanning Electron Microscopy) and Mineral Liberation Analyzer systems. These systems rapidly scan a sample with an electron beam and analyze the resulting X-rays to identify mineral composition. Data output includes mineralogical maps of the sample and information on grain size, shape, and mineralogical associations. The output can be used to produce limiting grade–recovery curves that can assist in determining the maximum degree of separation efficiency for a particular sample. Analysis is usually performed on discrete size ranges, and large amounts of data can be generated from even a single sample. Careful interpretation of the data is required to gain maximum benefit from this developing technology.

BENEFICIATION TESTS

After chemical and mineralogical testing is completed, a beneficiation test is usually performed. Beneficiation tests exploit chemical and physical differences between the one or more minerals to be recovered and the host rock or gangue minerals.

Beneficiation testing can include froth flotation, gravity, hydrometallurgical processing, magnetic, electrostatic, pyrometallurgical, and screening tests.

PILOT-PLANT TESTS

After laboratory testing is complete, it may be necessary to perform continuous pilot-plant tests. Pilot-plant tests become practical in any of the following situations: (1) the test process is technically complex and involves the recycling of product streams, (2) the test process is technically straightforward and pilot test facilities can readily be sourced, or (3) plant capital expenditures are high.

Contract pilot-plant testing is expensive, and relatively few facilities offer the service. There is clearly little point in performing pilot-plant tests unless a truly representative sample can be obtained. Access to the ore body is normally required unless sufficient material is available from drilling programs. Furthermore, contract facilities may not have the correct sizes of equipment for the flow sheet being tested and compromises are often made. A high degree of operator knowledge is required or else the exercise becomes one of testing the pilot plant itself rather than the process.

Pilot testing can also be undertaken by treating bulk samples through an existing facility that uses a flow sheet similar to that being proposed for the new operation. This might be the case when an ore body is being developed near an existing operation that is treating a similar ore type.

For larger-scale projects, it may be possible to justify purchasing a pilot plant and performing tests on-site. Doing so enables evaluation of site-specific conditions such as water quality, temperature, and atmospheric conditions, and allows local operators to be trained. Although it may be possible to purchase secondhand equipment, on-site testing is almost always expensive. It is important to ensure that equipment used in pilot-plant testing is well matched and capable of the required throughput.

Pilot-plant testing enables the testing of specific equipment such as that required for pressure filtration, high-rate thickening, column flotation, and developing technologies that cannot be readily evaluated on a laboratory scale. It also enables production of bulk samples of concentrate for marketing purposes or for further specialized testing. Bulk samples

of tailings can also be produced for backfill, paste, or liner tests.

Pilot-plant test data requires careful evaluation and should, whenever possible, be audited by independent observers. A thorough review of metallurgical balances is required to ensure that data was obtained at steady-state operating conditions. Chemical analyses should also be audited and check samples should be submitted to reputable assay laboratories.

Whether or not to perform pilot-plant tests is a complex decision but ultimately comes down to the question of risk. Constructing and operating an on-site pilot plant is expensive but, if testing is performed to a high standard, minimizes technical risk. A concentrator designed on the basis of small-scale batch laboratory tests undertaken on a few kilograms of material probably has a relatively high chance of failure, depending on the commodity being investigated.

Notably, gold ores are rarely pilot-tested, as the tests involved are well understood and can easily be performed on relatively small samples. In recent years, even complex sulfide ore flotation flow sheets are less likely to be pilot-tested because locked-cycle tests (explained in the following section) can be used to simulate continuous flotation performance accurately.

SPECIFIC TEST METHODOLOGIES

The following tests are related to minerals-processing test work.

Physical Testing

Physical tests evaluate the resistance of ore to breakage and determine the crushing and grinding power required during comminution. Abrasion tests determine the consumption of grinding media and liners during grinding. The following tests are available:

- **Bond abrasion index (Ai) test.** Each test requires at least 1,600 g of 19 × 13 mm (0.75 × 0.50 in.) material. The test provides an index from which the wear rates of grinding media and liner wear rates can be determined.
- **Bond ball mill work index test.** Each test requires ~10 kg of sample, stage-crushed to pass 3.36 mm. The test uses a closing screen size of 100 μm and a range of factors to determine the grinding power required to achieve a given product d_{80} (80% passing size).
- **Bond low-energy crushing work index (Wi_c) test.** Each test requires at least 20 pieces sized between 51 and 76 mm.
- **Bond rod mill work index test.** Each test requires ~15 kg of sample, stage-crushed to pass 12 mm.

Autogenous grinding (AG) and semiautogenous grinding (SAG) milling tests can also be performed. Several types of tests can be undertaken to evaluate how an ore will respond to milling:

- **Advanced media competency test.** A suite of tests covers impact crushing, rod and ball mill work indices, abrasion index, and uniaxial compressive strength tests to determine whether a material is suitable for autogenous or semiautogenous milling. An advantage of this test suite is that it can be used on PQ core material (85 mm).
- **Autogenous media competency test.** Each test requires 200 kg of material in discrete size ranges from 152 × 140 mm down to 114 × 102 mm. The sample is loaded into a drum (1.83 × 0.3 m) and rotated for 500 revolutions while power draw is measured. A range of Bond crushing, rod, and ball mill tests are performed on the mill product and on fresh ore.
- **JK Tech drop-weight test.** Each test requires 100 kg of crushed ore in the size range 75 to 12 mm. If diamond drill core is used, the core diameter should be at least 50 mm. The test measures impact-breakage and abrasion-breakage parameters. The former is determined by a tumbling test, the latter by a high-energy impact device called a JK drop-weight tester. These parameters are then used in a computer model (JKSimMet) to predict AG and SAG mill performance.
- **MacPherson autogenous work index test.** Each test requires 250 kg of material, stage-crushed to pass 32 mm. The test is performed in a mill (of diameter ~46 cm or 18 in.) as a dry grinding process with cyclone classification. It does not evaluate the competence of the ore at the coarse sizes used in autogenous or semiautogenous milling but gives a preliminary indication of whether the ore is better suited to autogenous or semiautogenous milling.

The number and type of samples submitted for testing depend on the complexity of the ore body and the associated host rocks. It is important that the samples include mining dilution. For an underground mine, it is also important to consider the expected proportions of hanging-wall and footwall dilution.

When designing comminution circuits for a concentrator, it is important to know not only the average value of the ore's hardness (resistance to breakage) but also the range of values that will be experienced throughout the life of the mine. It is therefore important to test a range of samples to determine how energy requirements and plant throughput will vary. The choice of these samples should generally be made based on the lithology of the rock types present rather than the ore mineralogy, particularly if the economic mineral is a relatively minor component of the ore.

The final value of the work index used for plant-design purposes depends on the nature of the ore body and the mine plan. For example, in an open-pit mine, if the ore body has a work index that is lower at the top and higher at the bottom of the pit, then it is clearly necessary to design the comminution circuits to treat the harder ore at the bottom.

In an underground mine, ore may vary in hardness, but in practice it is likely to be produced from several stopes in different parts of the ore body, in which case it may be adequate to design not for the hardest ore type but rather for a calculated blend based on the predictions of the mine plan. Comminution circuits are normally designed based on a stated average value of work index as well as the range of values with which the plant is expected to cope.

Gravity Testing

High-density minerals can be recovered by means of gravity processing. The following test methods may be appropriate.

Heavy Media Separation Test

Some ores are amenable to preconcentration using heavy media separation (HMS). HMS can be used to reject a significant portion of ore at a relatively coarse crush size with low loss of mineral value. Tests are performed with heavy liquids, typically with densities in the range 2.6–3.3 g/cc. Crushed ore

is screened into size ranges, and each size fraction is subjected to sink–float separations within closely sized density ranges.

Heavy-liquid test results are usually sufficient to evaluate the efficiency of most HMS processes. It is not necessary to perform pilot-plant testing unless bulk samples of HMS product are required for subsequent processing.

Spiral Test

Gravity separations can be performed using spirals on material in the size range 1.5–0.05 mm. A full-size spiral requires a minimum of 30 kg of material although smaller units can be used. Tests are normally performed in a closed circuit with products periodically removed and fresh material added.

Table Test

Shaking table tests are usually performed in an open circuit. The feed material should be reasonably closely sized. Products can be subjected to further cleaning, and computer models can determine the effect of recycling these products.

Gravity-Recoverable Gold Test

The use of centrifugal concentrators such as those made by Falcon and Knelson is a well-established method for treating gold ore. Concentrators are often installed within a grinding circuit for treating a portion of the circulating load (cyclone underflow).

A standard gravity-recoverable gold test involves stage-grinding a sample (10 kg) and performing a gravity separation after each grinding stage. The gravity concentrate is generally cleaned using a Mozley table or other laboratory panning device, and the gravity tailings are combined and ground to liberate further values. This method simulates gold recovery in a grinding circuit and recovers gold as soon as it is liberated so as to prevent overgrinding.

Gravity concentrate should be cleaned to the point that it is either salable or readily processed by smelting or further processing methods such as intense cyanidation.

Flotation Testing

Froth flotation is a widely used beneficiation method. The response of an ore to froth flotation can be readily tested in the laboratory using bench-scale flotation machines. Samples (typically 1 kg) are ground in a laboratory rod or ball mill and subjected to batch tests in which samples of concentrate are collected for timed periods.

A flotation test program generally involves the following stages:

1. Determining the relationship between mineral recovery and grind size
2. Collector screening
3. Testing the effect of pH
4. Testing modifiers to reduce or increase the floatability of minerals
5. Regrinding rougher or scavenger concentrates
6. Performing cleaning tests

A test program often culminates in a series of locked cycle tests. This laborious procedure involves a series of identical tests, or cycles, in which intermediate cleaner tailings products are added at the appropriate point in the subsequent cycle to simulate closed-circuit cleaning. The tests predict how mineral values that report to the cleaner tailings during the cleaning stages will be redistributed between the final products and tailings during continual processing. The locked cycle test generates a final metallurgical balance for the process and determines the final product concentrate grades and recoveries. Undertaking such tests requires experience, and it is essential that the test be kept in balance or equilibrium such that the weight of metal in the final products of each cycle matches the input of new material.

Gold Ore Testing

The testing of gold ores is often complicated by the low levels of gold present. Sampling errors increase as the grade of gold decreases and the gold particle size increases.

Cyanidation remains by far the most common method of processing gold and is used increasingly in conjunction with gravity processing. The following cyanidation tests are available.

Diagnostic Leach Test

The diagnostic leach test determines the gold mineralogy or gold associations within an ore. The test is actually a series of tests in which the gold in the sample is progressively recovered. Test stages might include the following:

1. Gravity testing and amalgamation to determine free gold
2. An initial cyanidation test to determine cyanide-recoverable gold
3. Treatment of the cyanide tailings with warm nitric acid to break down sulfide minerals, followed by filtering, thoroughly washing of the residue, and a further cyanidation test to determine gold associated with sulfides
4. Fire-assay analysis of the residue to determine gold encapsulated within silicate minerals

There are no standard test methods. Rather, tests are usually tailored for a particular ore type based on the basic mineralogy of the sample.

Bottle Rolls Test

Preliminary cyanidation tests often involve bottle-rolling. A sample is ground wet, often without measuring particle size, and placed in a bottle with lime and cyanide. The bottle is then rolled, typically for 24 or 48 hours, and the amount of soluble gold is determined. Analysis of the solid residue determines the gold recovery. The test is often undertaken by assay laboratories to give an initial indication of the feasibility of using cyanidation for gold recovery. However, the test is generally not sophisticated, and the levels of cyanide and oxygen present, both of which are essential for leaching to proceed, are sometimes not measured.

Agitated Leach Test

Laboratory cyanidation tests can also be performed in a stirred vessel where the pulp is agitated by a mechanical stirrer. The test is akin to industrial methods of processing; it enables air or oxygen to be sparged through the pulp and it is easy to monitor pulp conditions throughout the test.

The main process variables are grind size, pulp density, cyanide concentration, pH, and air requirement. The test determines whether it is advantageous to add carbon during the leaching process (carbon-in-leach) to minimize the effect of preg-robbing, in which soluble gold is adsorbed onto organic carbon that may be present in the ore.

The test can be performed with impellers and reaction vessels that give known scale-up factors. The dimensions and

speed of the impellers should be carefully controlled and the reaction vessel fitted with baffles to improve mixing.

Carbon Adsorption Test

Carbon adsorption tests determine the gold and silver loadings that will be achieved during continuous processing using carbon-in-pulp technology. The test involves adding varying amounts of carbon to samples of leached pulp and determining how the levels of precious metal decrease with time. Test results can predict the number of carbon adsorption stages required.

Column Leach Test

Column least tests determine the feasibility of using heap leach technology to recover gold. The static tests are performed in vertical columns typically ranging from 0.15 to 0.30 m in diameter, although pilot-plant tests may be performed in vessels several meters in diameter.

The ore sample is crushed and placed in the column, often on top of a filter medium consisting of crushed rock and hessian. If the sample contains clay, it may be necessary to agglomerate the ore using cement. In the laboratory, agglomeration is normally performed by rolling the sample in a cement mixer, adding measured amounts of cement and water, and curing for 24 to 48 hours before testing. Cyanide solutions are pumped into the top of the column by peristaltic pumps. The solutions percolate down through the bed of material, leaching gold and silver minerals. They are collected and measured daily and assayed for metal content, pH, and cyanide concentration, then passed through carbon columns to recover the precious metals. The barren solutions are pumped back through the column.

Environmental Testing

Tightening requirements for obtaining environmental permits for mining projects are resulting in similarly increased requirements for testing of tailings, low-grade ore, and other waste products. Test objectives are generally to determine the nature of the products arising from long-term degradation of mineral species.

Tailings Characterization Tests

Tailings characterization tests can involve the following analyses:

- Chemical analysis (major and trace element analysis)
- Mineralogical analysis using optical methods and XRD
- Particle size analysis

Chemical and mineralogical tests and particle size analysis were discussed previously; the goal of these tests is to gain a basic understanding of the nature of the tailings. Chemical analysis should identify the presence of elements that are potentially toxic and have the potential to enter the environment.

Acid–Base Accounting Analysis

Acid–base accounting (ABA) seeks to determine the acid-producing potential of tailings material. The method involves determining the total sulfur and sulfate sulfur contents of a sample and calculating the sulfide sulfur content by difference. The levels of sulfide sulfur indicate the acid-generating potential of the material, it being assumed that all sulfide sulfur present converts to sulfuric acid. Some methods involve only total sulfur analysis and assume that all sulfur present converts to sulfuric acid over time.

The acid-neutralizing potential of a material is determined in a separate test that involves reacting a known weight of sample with hydrochloric acid and determining the amount of acid consumed by titration.

The net generating potential is the difference between acid-generating potential and acid-neutralizing potential, and is normally quoted in metric tons of calcium carbonate per 1,000 t of material.

An alternative ABA method is the net acid generating test, which uses hydrogen peroxide to oxidize sulfide minerals. The pH at the end of the test is regarded as a measure of the sample's ability to generate acid.

There are several variations of ABA procedures, and the methodologies are continually being refined.

Leachate Analysis

Two types of leachate analysis are appropriate:

1. Synthetic precipitation leaching procedure (SPLP) test. The test determines the mobility of toxic organic and inorganic materials into groundwater. It involves shaking a sample (100 g) with a very dilute acid that represents rainwater, and analyzing the filtered leachate for a range of determinands. The strength of the acid used in the test should match the perceived levels of pollution.
2. Toxicity characteristic leaching procedure (TCLP) test. The test determines the mobility of the organic and inorganic phases in a material. It involves shaking a sample (100 g) with an acid buffer for a prescribed period of time; the pH of the buffer depends on the pH of the sample. The pulp is filtered and the leachate analyzed for a specified suite of 40 determinands, although this can be modified depending on the material being tested.

Humidity Cell Test

The humidity cell test is an accelerated weathering test in which a sample (1 kg) is placed in a cell and subjected to the following cycle: 3 days of dry-air permeation followed by 3 days of humid-air permeation and 1 day of washing with a fixed volume of water. The water samples are collected, carefully stored, and analyzed for a range of determinands. The test period can range from 20 weeks to 2 years.

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CHAPTER 4.5

Mineral Resource Estimation

Alan C. Noble

Mineral resource estimation is the process of estimating the tonnage, grade, size, shape, and location of mineral deposits. The *ore reserve estimate* follows the resource estimate, and generally requires at least a prefeasibility study to establish quantity and grade of a mineral that may be profitably and legally extracted from the deposit. Estimation of ore reserves involves not only evaluation of the tonnage and grade of a deposit but also consideration of the technical and legal aspects of mining the deposit, beneficiating the ores, and selling the product.

Estimation of the mineral resource generally involves only the geologist and a resource estimator who may be a geologist, geostatistician, or mining engineer that specializes in resource estimation. This team works together to define a resource model that defines the in-situ characteristics of the mineral deposit. The mineral resource model does not generally require consideration of mining costs or mining method, but it may be convenient to incorporate some mining features such as bench height in a deposit that will be mined by open pit, or minimum mining width for an underground vein mine. Reporting of the mineral resource is a different matter, however, and at least minimal consideration of project economics will be required to determine a cutoff grade above which resources are reported. The standards for reporting resources and reserves are different for the various countries, and the resource estimator must be careful to follow the appropriate regulations. Some examples of reporting requirements and standards include the Securities and Exchange Commission *Industry Guide 7* for the United States, NI 43-101 for Canada, the JORC Code for Australia and New Zealand, and the SAMREC Code for South Africa (SEC 2007; National Instrument 43-101 2005; AusIMM 2004; SAMREC 2007).

RESOURCE ESTIMATION METHODOLOGY

A resource estimate is based on prediction of the physical characteristics of a mineral deposit through collection of data, analysis of the data, and modeling the size, shape, and grade of the deposit. Important physical characteristics of the ore body that must be predicted include the size, shape, and continuity of ore zones; the frequency distribution of mineral grade; and the spatial variability of mineral grade. These physical characteristics of the mineral deposit are never completely known but are projected from sample data. The sample data consist of one or more of the following:

- Physical samples taken by drilling, trenching, test pitting, and channel sampling
- Measurement of the mineral quantity in the samples through assaying or other procedures
- Surveys to determine the location of the samples in three-dimensional (3-D) space
- Measurement of in-situ rock density
- Direct observations such as geologic mapping and drill core logging
- Metallurgical testing to define the amenability of the minerals for upgrading and extraction

Estimation of the resource requires analysis and synthesis of these data to develop a resource model. Methods used to develop the resource model may include

- Compilation of the geologic and assay data into maps, reports, and computer databases;
- Delineation of the physical limits of the deposit based on geologic interpretation of the mineralization controls;
- Compositing of samples into larger units such as mining bench height, seam thickness, or mineable vein width;
- Modeling of the grade distribution based on histograms and cumulative frequency plots of grades;
- Evaluation of the spatial variability of grade using experimental variograms; and
- Selection of a resource estimation method and estimation of quantity and grade of the mineral resource.

The estimation procedure must be made with at least minimal knowledge of the proposed mining method because different mining methods may affect the size, shape, and/or grade of the potentially mineable ore reserve. The most important mining factors for consideration in generating an ore reserve estimate from a mineral resource estimate are

- The range of likely cutoff grades,
- The degree of selectivity and the size of the selective mining unit for likely mining methods, and

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- Variations in the deposit that affect the ability to mine and/or process the ore.

These mining factors often determine the degree of detail that is required for the resource model. For example, a disseminated gold deposit may be continuous and regular in shape, if mined by bulk open-pit methods. The same deposit may be discontinuous and difficult to estimate, however, if mined by more selective underground methods at a higher cutoff grade. Such large differences in deposit shape due to variations in cutoff grade and mining method may require different ore reserve estimation methods for different mining methods.

DATA COLLECTION AND VERIFICATION

Data that must be collected and compiled for the resource estimate are as follows:

- Reliable assays from an adequate number of representative samples
- Coordinate locations for the sample data
- Consistently recorded geologic data that describe the mineralization controls
- Cross sections or plan maps with the geologic interpretation of the mineralization controls. The geologic interpretation may be developed interactively on the computer using 3-D modeling methods, in which case paper plans and sections may not be required.
- Tonnage factors or specific gravities for the various ore and waste rock categories
- A surface topographic map, especially for deposits to be surface mined
- Metallurgical testing for samples representative of the various types of mineralization in the deposit

Although small deposits may be evaluated manually using data on maps and in reports, manual methods of resource estimation are nearly obsolete. Resource estimations are primarily done using computer methods with the resource data entered into a computer database. Computer programs can then be used to retrieve the data for printing reports, plotting on digital plotters, statistical analysis, and resource estimation. The minimum information that should be included in a drill-hole database are

- Drill-hole name, number, or other unique identification;
- Hole length, collar coordinates, and down-hole surveys;
- Sample intervals and assay data;
- Geologic data such as lithology, alteration, oxidation, etc.; and
- Geotechnical data such as rock quality designation.

Entry of data into a computer database is a process that is subject to a high degree of error if not carefully controlled and checked. Some procedures that may be used to ensure that the data have been entered correctly are as follows:

- Verification of the data using independent entry by two persons. This may also include importing of the data by two persons/methods and electronically comparing the results.
- Manual comparison of a random sample of hard-copy data sheets with data in the database.
- Scanning the data for "outlier values"; for example, drill locations outside the project limits, high and low assays, and sample intervals that overlap or are not continuous.
- Comparison of computer-plotted data with manually plotted maps of the same data. Collar location maps and cross sections are especially useful to rapidly locate inconsistent collar locations and downhole surveys.
- An independent audit of the data as part of statutory requirements, which may be required by some regulatory bodies.

Assay data are generally transferred from the analytical laboratory to the client using electronic means (such as e-mail) to transfer the data in electronic format. Although this dramatically reduces data entry errors compared to manual keyboard entry, it creates a new set of problems. A particular issue is that the laboratory may change the reporting units (for instance, from parts per million Au to ounces Au/t, or parts per million to parts per billion) from one report to the next, or even within the same report.

The recent practice of surveying collar coordinates using Global Positioning System (GPS) methods is also a frequent source of errors. Particular problems include reporting using different datum, improper units, incorrect conversion from GPS units to a local survey, and use of a low-resolution, inaccurate consumer handheld GPS units. The geologist or resource estimator should always ensure that the survey method, survey datum, data postprocessing methods, datum conversions, and the type of instrument are reported with the survey data.

Additional care and attention to the detail and accuracy of the resource database are essential. A database with a large number of errors may result in a resource estimate that is inaccurate and requires a complete revision to provide defensible results.

GEOLOGIC INTERPRETATION

The sample database represents a large 3-D array of point locations in a deposit. The sample data are quantitative and have been subjected to minimal reinterpretation after the original measurements. There is another body of geologic knowledge, however, that does not fit this description. This is the interpretation resulting from the geologist's assimilation of the geologic data. These interpretative data are often represented on plan maps or cross sections that show outlines of the geologic features or iso-grade contours that define mineral zones.

The current industry practice is to create a 3-D model of a geologic interpretation known as a "wireframe" model. The wireframe model is created by displaying a slice through the deposit on the computer screen and interactively digitizing the outlines of the geologic feature. The digitized line is frequently attached, or "snapped," to points on drill holes, such as contacts and grade-zone boundaries, to provide a more precise location of the line relative to the locations established by the drilling.

The interpretation process is continued on an adjacent slice through the deposit to extend the interpretation. Adjacent lines are then connected using a mesh of triangles to form a continuous 3-D ribbon of triangles that links the two lines. After this process has been completed across the extent of the deposit, the interpretation is a series of adjacent, connected ribbons that are built from a mesh of triangles. The visual appearance of the resulting object, the wireframe model, is that of a triangular network of wires connecting an irregular set of points. A significant body of mathematical tools has been developed by computer software developers

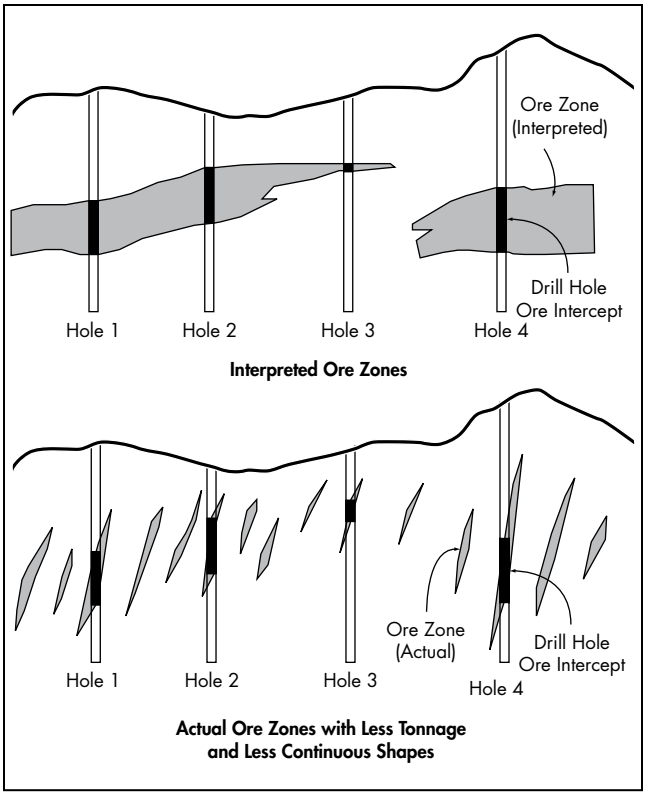


Figure 4.5-1 Overestimation of the size of a mineral deposit based on a geologic model that is more continuous than the actual ore zones

to provide for manipulation and analysis of wireframe models as solid objects for use in resource estimation.

These interpretations provide an interpretative geologic model that is one of the most critical factors in the resource estimation. Failure to develop an appropriate geologic orebody model is the most common reason for large errors in the resource estimates. As shown in Figure 4.5-1, an inappropriate geologic model may lead to errors greater than an order of magnitude.

The geologist's interpretation of the ore body should be used as much as possible in developing the resource estimate. There are, however, practical limits to the amount of complexity that can be included in the resource model, and the geologic interpretation will be limited to critical inputs that define the shape and trends of the mineral zones at different cutoff grades and the character of the mineral zone contacts.

Examples of geologic features that are often modeled include

- Receptive versus nonreceptive host rocks;
- Alteration types that accompany mineralization or create problems in beneficiation;
- Faulting, folding, and other structural modifications;
- Multiple phases of mineralization; and
- Post-mineral features such as oxidation and leaching.

Changes in lithology are often important variables in resource estimation as mineralization can vary because of physical or chemical attributes of the host rocks. The differences may be distinct, such as the sharp contact between a skarn ore body and an unmineralized hornfels country rock. They also may be gradational, such as the gradual decrease in grade that is often observed between a favorable and slightly less favorable host in a porphyry copper deposit. Other important lithologic controls include barren post-mineral intrusive rocks, nonreceptive shale beds, and other unmineralized materials that are contained within the mineralized zone.

The effects of faulting will vary according to whether the faulting occurred before or after the mineralization and what processes accompanied the faulting. A simple post-ore displacement may create a discontinuity in the mineralization trends, preventing simple interpolation across the fault. The same type of fault occurring prior to mineralization may have little or no effect on the mineralization on either side of the fault, or may localize high-grade, vein-type mineralization that must be modeled independently of a more uniformly disseminated ore body. It is also important to determine whether the fault is a thin, well-defined structure or many smaller structures in a complex, wide shear zone. In the first case, the fault is modeled as a simple surface with zero thickness; in the second, the fault zone must be defined and modeled as a volumetric unit distinct from the adjoining rock units.

Folding is particularly significant in sedimentary and stratabound deposits. Modeling of folding depends on whether folding happened before or after ore deposition, on the tendency of the mineral zoning to follow the stratigraphy, on any remobilization that occurred with the folding, and on the creation of mineral traps or other favorable structures. In addition to defining the shape of the folds, it is important to determine whether the mineralization follows the contours of the folds or is independent of the fold geometry.

Multiple phases of mineralization must be defined individually, particularly where they complicate the mineral zoning pattern through overlapping, discordant trends and through post-mineral oxidation or leaching. Secondary enrichment and oxidation will almost always require delineation of these enrichment features as modified ore zones.

The character of the ore zone contact must be determined and input into the resource model. A sharp contact will be handled as a discontinuity (a hard boundary) and the data used independently on either side of the contact. A transitional contact, however, is a broad, gradational boundary (a soft boundary) that may require data selection from zones of 5 m to more than 30 m to achieve true differentiation between the different grade zones. As a transitional zone becomes thinner, it will eventually approach a sharp contact. For practical purposes, any transitional boundary thinner than the smallest selective mining unit will be modeled as a discontinuity.

In addition to definition of these physical ore controls and post-mineral modifications, a clear understanding of ore genesis will always be beneficial in creating a resource model. In the simplest case, the ore genesis will give clues to the behavior of the grade distributions and variograms; in other cases, the genetic structure is so dominant that it can be used as a direct control in the estimation of mineral resources.

Grade zoning is another common method for adding geologic control to the resource model. Grade zones are usually created by manually drawing grade contours on plans or sections through the mineral deposit. Correctly drawn grade zones will synthesize all of the known geologic controls and the assay grade distribution to define a shape for the deposit that is much more informative than just the grades themselves. The grade zone contours may then be wireframed to form 3-D grade shells for use in coding the block model and selecting

assay data for estimation. Some general rules for grade zoning are as follows:

- Grade zones should be fairly smooth, continuous lines. If it is necessary to draw very irregular lines to contour the sample data, grade zoning is generally not appropriate.
- Unless the grade-zone boundary is coincident with a natural break in a grade distribution, the grade-zone boundaries should not be treated as strict hard boundaries for grade estimation. Mineralization is usually gradational across a grade-zone boundary, and it is generally appropriate to treat adjacent grade zones as soft boundaries and nonadjacent grade zones as hard boundaries.
- Defining a grade zone based on the anticipated mining cutoff grade and using that grade zone as a hard boundary is incorrect unless the mining cutoff grade is coincident with a natural break in the grade distribution. The use of the grade zone as a hard boundary in this fashion will create an estimate that is a polygonal estimate along the boundary, and additional dilution is usually required to create an unbiased estimate of the resource grade.

COMPOSITING

Compositing is a procedure used in developing resource estimates in which sample assay data are combined by computing a weighted average over longer intervals to provide a smaller number of data with greater length. Compositing is usually a length-weighted average. If density is extremely variable (e.g., massive sulfides), however, compositing must be weighted by length times density (or specific gravity).

Some of the reasons for and benefits of compositing include the following:

- Irregular length assay samples must be composited to provide equal-sized data for geostatistical analysis.
- Compositing incorporates dilution such as that from mining constant-height benches in an open-pit mine or from mining a minimum height/width in an underground mine.
- Compositing reduces wild variations caused by erratic high-grade values (the nugget effect).

Several methods for compositing may be used depending on the nature of the mineralization and the type of mining. Common compositing methods are (1) bench compositing, (2) constant length, downhole compositing, and (3) ore-zone compositing.

Bench compositing is a method often used for resource modeling for open-pit mining and is most useful for large, uniform deposits. Composite intervals for bench compositing are usually chosen at the crest and toe of the mining benches. Bench compositing has the advantage of providing constant elevation data that are simple to plot and interpret on plan maps. In addition, the dilution from mining a constant-height, constant-elevation bench is approximated by the bench composite.

Downhole composites are computed using constant length intervals that generally start from the collar of the drill hole or the top of the first assayed interval. Downhole composites are used when the holes are drilled at oblique angles (45° or less) to the mining benches and bench composites would be excessively long.

Downhole composites should also be used where the sample interval is long compared to the composite interval. For example, if the composite interval is 10 m and the sample interval is also 10 m, it is possible for a bench composite to straddle two sample intervals. In the worst case, the composite will be composed of two 5-m sections of the 10-m sample intervals and the resulting composite will have exactly the same grade as a 20-m composite and will contain significant excess dilution.

Where the drill holes are drilled in many directions with respect to the ore zone, the composite length may need to be varied based on the orientation of the drill relative to the ore zone. For example, when mineralization in a tabular structure has much better continuity along the structure than across the structure, drill holes oriented perpendicular to the structure should be composited to a short interval, while drill holes oriented parallel to the mineralization should be composited to a longer interval. The ratio of the length of the composites should respect the relative continuity of mineralization in each direction.

Ore-zone compositing is a method of compositing that is used to prevent dilution of the composite when the width of the contact between waste and ore (or low-grade and high-grade mineralization) is less than the length of a composite. Use of bench compositing or downhole compositing in this case may distort the grade distributions by adding low-grade mineralization to the ore population and high-grade mineralization to the waste population, resulting in underestimation of ore grade and overestimation of waste grades.

Ore-zone composites are computed by first identifying the interval containing each ore zone in the drill hole. Each ore zone is then composited individually as follows:

1. The length of the ore zone is divided by the desired length of the composite to estimate the number of composites that will be created;
2. The number of composites is rounded up or down to provide the composite interval that is closest to the desired composite length; and
3. The ore zone is composited using length composites starting at the beginning of the ore zone and length as determined in the previous step.

Geologic codes are usually assigned to composites according to the rock type, ore zone, or other geologic feature. This is often a simple procedure, since most composites will be computed from samples taken from a single geologic unit. Assignment of geologic codes to composites that cross geologic contacts is more complex, since the composite will be computed using data from multiple geologic units. In most cases, the geologic code for the composite is assigned according to the dominant code within the composite.

A special case of ore-zone compositing is encountered in a vein or bedded deposit in that the width of the ore zone is determined by a combination of minimum mining thickness (height) and assay limits. In these situations, composites must be computed for each combination of assay cutoff grade and minimum mining thickness that is used for the resource model.

If the geologic contact is transitional and does not separate contrasting grade distributions, it is appropriate to assign the geologic codes according to the majority rule. If the composite crosses a sharp boundary between contrasting grade distributions, it is best to use geologic unit compositing or to assign the composite to the geologic unit with the most similar grade.

If some sample intervals in the data are missing assays, it is important to determine the reason for the missing data and account for it appropriately. Typical examples follow:

- The missing zone was not assayed because it was low grade or barren by visual inspection, or the sample was missing because of poor core recovery in a barren zone. *Action:* Composite using the average of the barren unit or zero grade for the grade of the missing assay.
- The sample was missing because of poor core recovery in a narrow post-mineral fault. *Action:* Ignore the missing interval when computing composites. The volume of the fault zone is small and the grade will be similar to the grades in the country rock.
- The sample was missing because of poor core recovery in a vein that is higher grade and less competent than the surrounding country rock. *Action:* Ignore the missing interval when computing composites, but retain the length of the interval for use in estimating the width of the vein.

BASIC STATISTICS AND GRADE DISTRIBUTION

Computation of basic statistics and evaluation of grade distributions are the first quantitative analyses of the grade data and are basic tools to provide both feedback to the geologic analysis and input to the resource modeling. Important factors in these basic studies include

- Detection of high-grade or low-grade outlier values,
- Evaluation of different lithologies to determine which are favorable and which are unfavorable host rocks for mineralization,
- Differentiation of complex grade distributions into simple populations for resource modeling, and
- Identification of highly skewed and/or highly variable grade distributions that will be difficult to estimate.

Basic statistics should be computed for sample and/or composite grades in each geologic domain that is suspected to have different characteristics. This may include different lithologies, alteration types, structural domains, grade zones, or other grouping of data that has been recognized (or suspected) to have different grade distributions. Statistics that should be compiled include

- Number of data (samples or composites);
- Average grade, thickness, etc. (mean);
- Standard deviation (std. dev.) and/or variance;
- Coefficient of variation (COV), the standard deviation divided by average grade;
- Histogram of grades; and
- Cumulative frequency distribution (probability plot).

The first item reviewed is the number of data; generally, at least 25 data points are required to make comparisons between different geologic domains. If sufficient data are available, average grades and COVs will be compared among the various geologic domains. General rules for evaluating differences in average grade and guidelines for analyzing COVs are shown in Tables 4.5-1 and 4.5-2, respectively.

Distributions with COVs greater than 25% often have a lognormal grade distribution, and the basic statistics will also be compiled for the natural logarithms of grades. For a perfectly lognormal distribution, the lognormal statistics are related to the normal statistics as shown in Equations 4.5-1 through 4.5-4:

Table 4.5-1 General rules for evaluating differences in average grade

Grade Difference, %	Interpretation
0 to 25	Grade populations that do not usually require differentiation for resource modeling.
25 to 100	Grade populations that require differentiation for resource modeling if divided by a discontinuity such as a fault or if variograms or grade trends are dissimilar.
>100	Grade distributions must be separated for modeling. Differences of 1,000% or more may be observed when barren, mineralized, and/or high-grade populations are present.

Table 4.5-2 Guidelines for analyzing coefficient of variation

COV, %	Interpretation
0 to 25	Simple, symmetrical grade distribution—Resource estimation is easy; many methods will work.
25 to 100	Skewed distributions with moderate difficulty in resource estimation—Distributions are typically lognormal.
100 to 200	Highly skewed distributions with a large grade range—Difficulty in estimating local resources is indicated.
>200	Highly erratic, skewed data, or multiple populations—Local grades are difficult or impossible to estimate.

$$\text{mean} = e^{\alpha + \frac{\beta^2}{2}} \tag{4.5-1}$$

where

α = average of the natural logarithms of grades
β^2 = variance of the natural logarithms of grades

$$\text{median} = e^{\alpha} \tag{4.5-2}$$

$$\text{COV} = \sqrt{e^{\frac{\beta^2}{2}} - 1} \tag{4.5-3}$$

$$\text{std. dev.} = \text{mean} \times \text{COV} = \text{mean} \times \sqrt{e^{\frac{\beta^2}{2}} - 1} \tag{4.5-4}$$

Close agreements between the mean, standard deviation, and coefficient of variation when estimated using both normal and lognormal statistics are indicative of a lognormal population and are required to use lognormal statistics.

GRADE DISTRIBUTION AND GEOLOGIC PARAMETERS

The grade histogram and cumulative frequency distribution are also used to study the relationship between the statistical grade distribution and geologic parameters. The analysis is usually begun with a histogram of sample or composite grades. If the histogram is bell shaped and symmetrical, a normal distribution is indicated and the cumulative frequency will be plotted as a normal probability graph. Normal distributions are not usually found in mineral deposits except for those that are very continuous and have low variability.

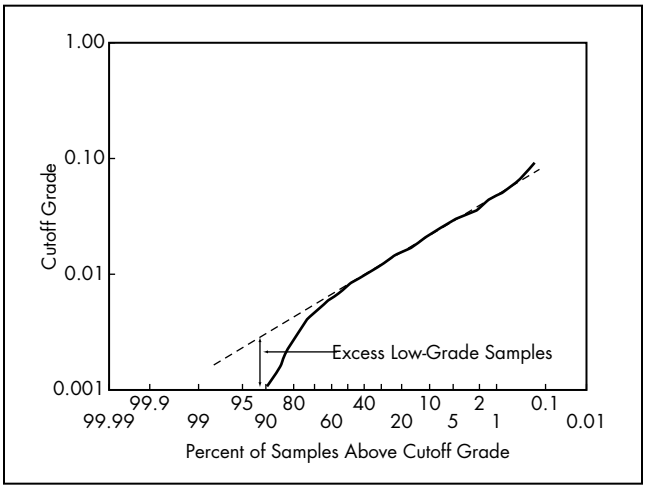


Figure 4.5-2 Cumulative frequency plot showing excess low-grade samples

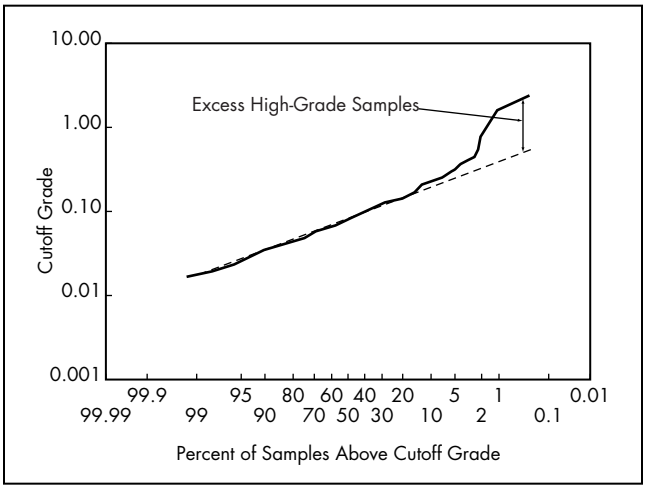


Figure 4.5-3 Cumulative frequency plot showing excess high-grade samples

If the histogram is skewed to the right so that the high-grade side of the histogram is larger than the low-grade side, a lognormal distribution is indicated and the cumulative frequency distribution will be plotted as a lognormal probability graph. Lognormal distributions, or combinations of lognormal distributions, are common in mineral deposits.

A normal probability plot is a specialized plot where one axis (usually the y-axis) is the cutoff grade and the other axis (usually the x-axis) is the cumulative frequency distribution (the percentage of samples above, or below, the cutoff grade). The cumulative frequency axis is scaled so that a normal distribution will plot as a straight line, the slope of the line is proportional to the standard deviation of the distribution, and the 50th percentile is the average of the samples.

A lognormal probability plot is similar to a normal probability plot except that the axis for the cutoff grade is scaled according to the logarithm of cutoff grade. The slope of the line is proportional to the standard deviation of logarithms of grade β, and the 50th percentile is the average of the logs of grades α. A probability axis is generally available in most technical graphing software.

The probability graph may be used to estimate the log–standard deviation based on probabilities from the normal probability distribution, as shown in Equation 4.5-5:

$$\beta = \frac{[\ln(g_{98\%}) - \ln(g_{2\%})]}{4.11} \tag{4.5-5}$$

Often the probability graph will not be a straight line but will be composed of multiple straight lines or curves. A typical deviation from a straight line is a downward curve at the low end of the graph, as shown in Figure 4.5-2. This curve represents excess low-grade samples and in porphyry systems is often attributed to weakly mineralized late intrusions or to post-mineral, barren dikes. On low COV deposits, this type of graph may also represent a normal distribution that has been plotted on lognormal probability paper. The data should be examined to determine the source of the low-grade material and to determine whether that population has been or can be mapped geologically and estimated separately.

Another common deviation from a straight line on the probability plot is a steeper slope at the upper end of the curve, as shown in Figure 4.5-3. This represents excess material in the high-grade population and may be caused by two superimposed populations, such as high-grade veins within lower-grade disseminated or stockwork mineralization. Other causes of excess high-grade assays include small zones that are highly favorable to mineralization because of higher permeability, favorable chemical properties, secondary enrichment, or metamorphic remobilization. Since the high-grade mineralization usually has less continuity than the lower-grade mineralization, the high-grade samples are usually capped to prevent overestimation of the grade of the resource. If the high-grade samples show sufficient continuity to define a continuous volume, a separate high-grade domain may be defined and estimated separately from the lower mineralization.

VARIOGRAM MODELING

The variogram is the fundamental tool used by the geostatistician and geologist to measure spatial continuity of grade data. The variogram (also called the semi-variogram) is usually presented as a graph showing the average squared difference between samples versus the distance between samples. It is computed by averaging the squared differences between pairs of samples that are a given distance apart as follows:

$$\gamma(h) = \text{variogram for distance } h$$
$$= \frac{1}{2N}\sum(g_i - g_{i+h})^2,\ i = 1, 2, 3, \ldots N \tag{4.5-6}$$

where

h = distance between samples
N = number of pairs at distance h

The variogram function, $\gamma(h)$, is computed for a number of different sample distances to provide an experimental variogram that typically looks like the graph in Figure 4.5-4. The most important features of the variogram are the nugget, range, and sill. The nugget value is identified as the y-intercept of the variogram curve and represents random and short-distance variability factors such as sampling error, assaying error, and erratic mineralization. High nugget values are commonly found in mineral deposits where short-distance variability is extremely high, where accurate sampling and assaying of ore is difficult, or where

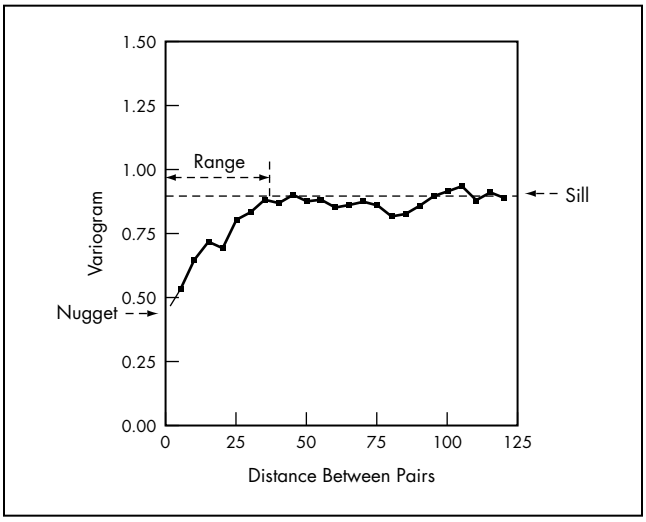


Figure 4.5-4 Typical variogram plot

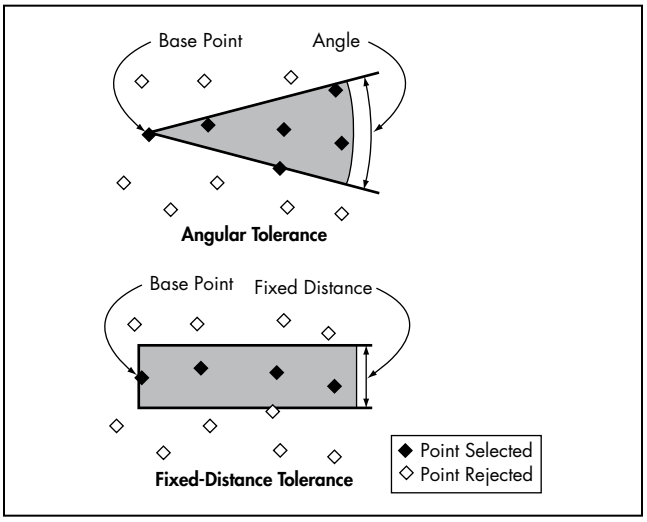


Figure 4.5-5 Simple angular and fixed-distance tolerance geometries for computing variograms

poor sampling and assaying techniques are employed. High nugget effects are found in many gold deposits because of random gold particles that cause large grade changes over small distances. (Note: A "large" gold particle may be as small as 0.25 mm). Similar high nugget values are often found in molybdenum deposits; these are caused by small pockets of pure molybdenite in a disseminated or stockwork mineralization.

Small nugget values suggest good sampling techniques and locally continuous mineralization, and imply that the assays can be used reliably for geologic interpretation and resource estimation. Low nugget values are typically found in many types of deposits, including hypogene porphyry copper, iron ore, and coal. High nugget values have also been found for each of these types of deposits, so each deposit must be analyzed individually. In addition, salting of samples to fraudulently upgrade a deposit may cause a false low-nugget value if a constant amount of finely dispersed mineral is added to samples before assaying. A similar effect has been observed in samples salted by downhole contamination in reverse-circulation drilling.

Most variograms increase in value from the nugget for some distance and then level off to a constant value. This distance is called the range of the variogram, and the variogram value is called the sill. The range is equivalent to the geologist's concept of range of influence; that is, the distance beyond which samples are not correlated with other samples and beyond which grade trends should not be projected. The sill value is usually equal to the sample variance. If the sill is higher or lower than the variance, zonal effects or multiple grade distributions are usually indicated.

The slope and shape of the variogram often vary in different directions, with the range increasing in the direction of greatest continuity of the mineralization. This behavior is referred to as a geometric anisotropy.

Computing an Experimental Variogram

Computing an experimental variogram from a set of randomly spaced data involves finding pairs of data that are oriented in the specified directions, determining the distance between the samples, and then summing the squared differences of the

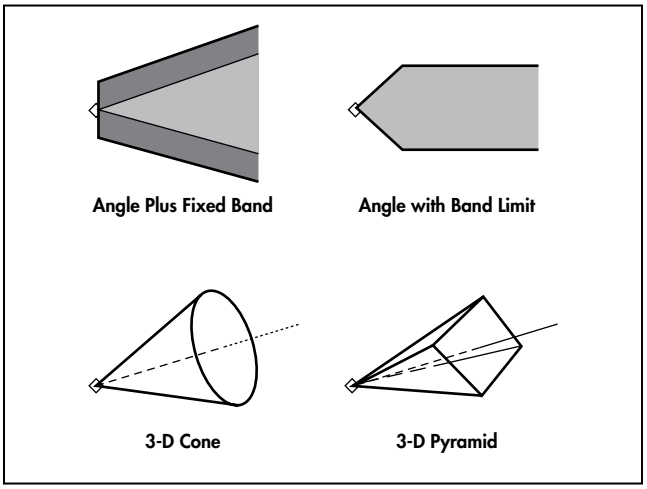


Figure 4.5-6 Composite and 3-D geometries for computing variograms

grades. Since the data are usually sparse, it is necessary to use tolerances for both the directional and distance increments. The directional tolerance is usually achieved with a window angle or a fixed distance, as shown in Figure 4.5-5. These methods may be combined and/or generalized into three dimensions, as shown in Figure 4.5-6. The distance tolerance is a fixed-distance increment (cell size) selected so a reasonable number of samples fall in each cell.

The following list offers some guidelines to aid in computing experimental variograms.

1. Variograms must be computed within continuous zones of mineralization. Contacts should not be crossed between different geologic domains.
2. The maximum distance used should be less than one-half the length of the mineralized zone in the direction of the variogram.
3. The maximum search distance perpendicular to the direction of the variogram must be less than one-fourth the range of the variogram in the perpendicular direction.

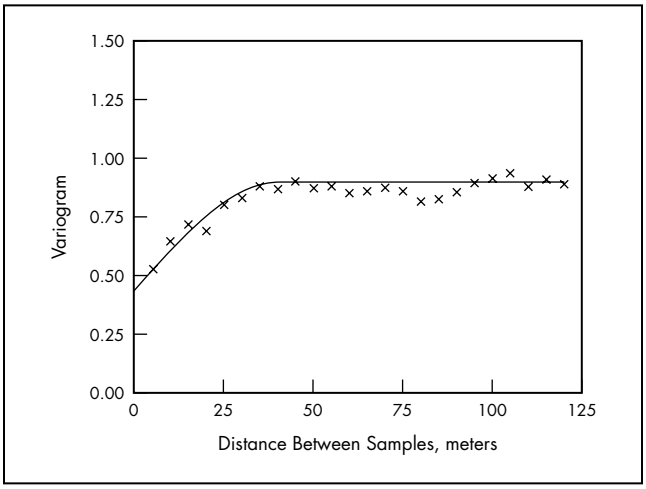


Figure 4.5-7 Spherical variogram model

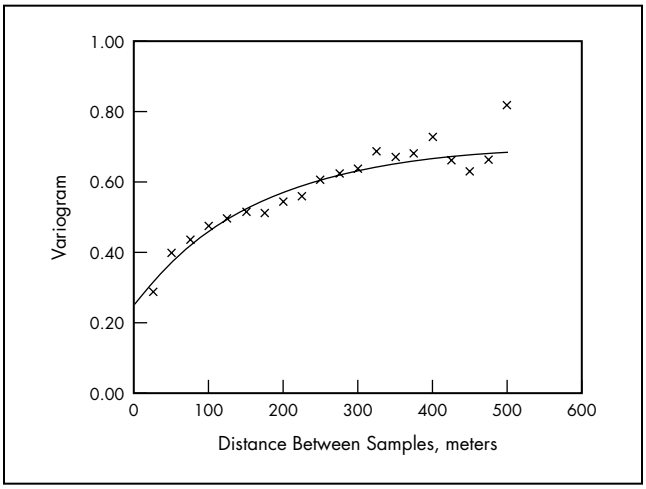


Figure 4.5-8 Exponential variogram model

This is particularly important for planar or bedded deposits that often have much greater continuity parallel to the dominant plane compared to the perpendicular/across-bedding direction.

4. The distance increment should be approximately equal to the average spacing between samples in the direction of the variogram.
5. At least 30 pairs of samples are required to compute a valid variogram. More pairs produce a more stable variogram.
6. All samples must be the same size and should be obtained by the same or similar sampling methods.
7. Where the samples are not evenly spaced, particularly where more samples have been taken in higher-grade areas, data should be declustered before computing the variogram. In addition, a few twin holes in a higher-grade area may give a misleading impression of the nugget effect. The simplest method of declustering is to randomly remove some of the clustered samples. Other methods compute areas of influence for the clustered samples that are used to create weighted averages for statistical analysis and variogram modeling.

A model, or equation, is fitted to the experimental variogram for further geostatistical evaluations such as kriging. The most common variogram models found in mineral deposits are the spherical model, shown in Figure 4.5-7, and the exponential model, shown in Figure 4.5-8. Another model that is rarely seen in mineral deposits but may be useful for classifying resources is the linear model, shown in Figure 4.5-9. Most geostatistical software packages provide functionality for variogram modeling, varying from interactive graphics fitting of the variogram curve to least-squares fitting of the variogram in multiple directions.

Because many mineral deposits have lognormal assay distributions, a proportional effect will be present and grade variability will be higher in high-grade areas than in low-grade areas. This may be dealt with by dividing the raw variogram in each variogram cell by the square of the mean of the samples that were used in the variogram for that cell. The resulting variogram is known as a relative variogram. If the data distribution has been shown to be lognormal during statistical

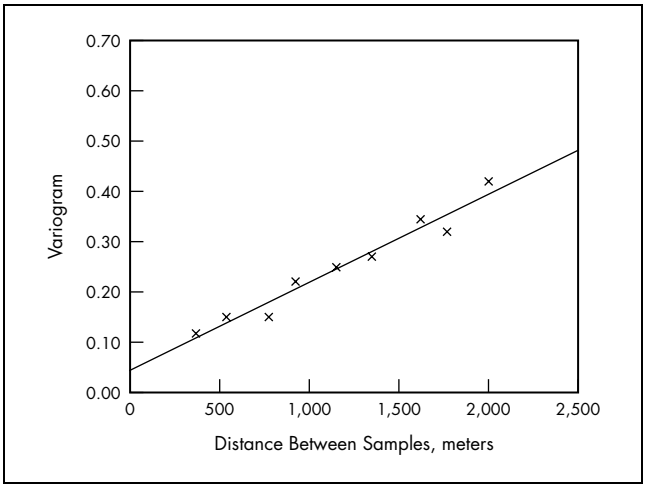


Figure 4.5-9 Linear variogram model

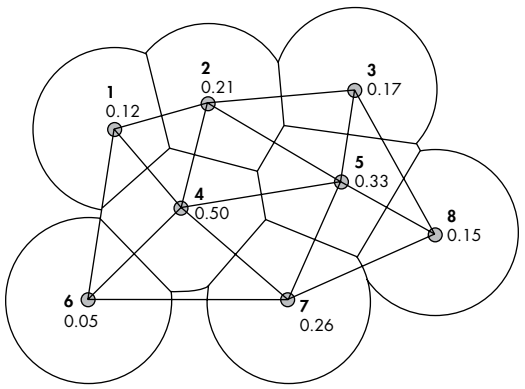
analysis and grade-distribution studies, the variogram may be computed using the logarithms of sample grades. The resulting variogram is often less erratic and more easily interpreted than the variogram of untransformed values. This variogram may be used directly for lognormal geostatistics or may be transformed to a relative variogram as follows:

$$\sigma_R^2 = \text{relative variance} = e^{\beta^2} - 1 \quad (4.5\text{-}7)$$

$$\gamma_R(h) = \text{relative variogram} = \sigma_R^2 - e^{(\beta^2 - \gamma\ln(h))} + 1 \quad (4.5\text{-}8)$$

where $\gamma\ln(h)$ is the log-transformed variogram for distance h.

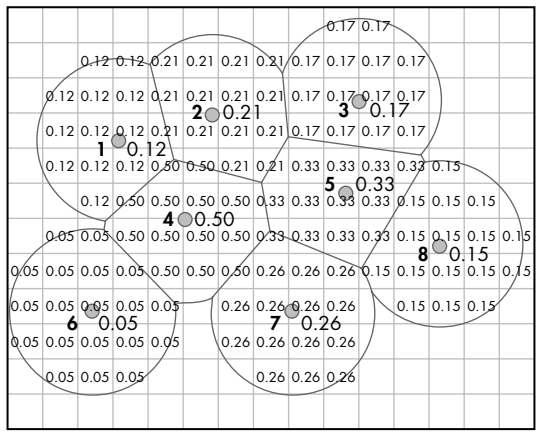
As shown in the above transformation, the lognormal covariances are transformed using the lognormal-to-relative transformation rather than the raw lognormal variogram. Conversion of the lognormal variogram to a relative one is very important, since the nugget/sill ratio is always higher for the relative variogram than for the lognormal variogram. In addition, caution must be exercised when using the lognormal variogram because small deviations from lognormality may have large effects on the transformation to a relative variogram.



Hole	Grade	Area
1	0.12	39.4
2	0.21	37.6
3	0.17	42.0
4	0.50	37.7
5	0.33	33.8
6	0.05	50.1
7	0.26	46.8
8	0.15	46.3

Total Area = 333.7
Total Area × Grade = 71.39
Average Grade = 0.2139

Figure 4.5-10 Estimation example using the polygonal method



Hole	Grade	No. of Blocks
1	0.12	12
2	0.21	14
3	0.17	14
4	0.50	13
5	0.33	12
6	0.05	19
7	0.26	14
8	0.15	16

Total Blocks = 114
Sum Blocks × Grade = 24.21
Average Grade = 0.2124

Figure 4.5-11 Estimation example using the nearest-neighbor method

RESOURCE ESTIMATION (MODELING)

Methods for resource estimation or modeling are generally divided into the traditional geometric methods that are done manually on plans or sections and interpolation methods such as inverse-distance weighting and kriging that require the use of a computer.

Geometric Methods

Manual resource estimations are usually done on plan maps or cross-sectional maps that cut the deposit into sets of parallel slices. Data plotted on the maps include drill-hole locations, assay values, and the geologic interpretation of the mineralization controls. True manual estimates on paper are seldom done anymore because of the widespread availability of computer software for resource estimation. Frequently, the computer performs the same calculations as were used for the original manual methods and the results are comparable. The two basic geometric methods are area averaging and polygonal/cross-sectional estimation.

Area Averaging

The area-averaging method is among the simplest of all reserve estimation methods, involving only a geologic interpretation of the shape of the ore and averaging of the grades within that shape. The tonnage is estimated by multiplying the density of the mineral and the volume of the zone. Volume is estimated by multiplying the thickness of the plan/section and the area defined within the interpreted line.

The method may also be implemented in three dimensions using a wireframed volume, in which case the average grade is estimated based on the samples inside the wireframe and the wireframe volume is calculated directly by the software.

Despite its simplicity, the area-averaging method provides excellent estimates where the drilling pattern is uniform, grades are continuous, and ore boundaries are distinct and sharp. Problems may arise, however, when the drill pattern is not uniform. With a nonuniform drill pattern, a cluster of holes in a high-grade zone will cause overestimation of grade. Area-averaging methods also may be difficult to implement on deposits with discontinuous or spotty ore zones, especially if the ore contacts are gradational and multiple cutoff grades are desired.

Polygonal Methods

Polygonal methods involve drawing a polygonal area of influence around each sample intersection, measuring the area of the each polygon, and then calculating the average grade by weighting each sample grade by the corresponding polygonal area. Tonnage is then computed using the same procedure as was used for the area-averaging method, except that the areas used to compute tonnage are the area of each individual polygon.

The classical manual polygonal estimate was done by drawing polygons on plan maps based on the perpendicular bisectors of the line between each drill hole, as shown in Figure 4.5-10. The current computer-based approximation of the polygonal method is the nearest-neighbor estimation. This method requires superposition of a rectangular grid of blocks over the drilled area, as shown in Figure 4.5-11. The grade of the nearest sample is then assigned to each block. This method will closely approximate the polygonal method if the block size is no more than 25% of the average drill-hole spacing.

The polygonal/nearest-neighbor method has the advantage of simplicity and ease of implementation. It is also independent of interpretation bias and provides an unbiased estimate of the average grade of a deposit at a zero cutoff grade. This unbiased estimate of the average grade of the deposit is very useful for validation of an inverse-distance power, kriged, or other advanced estimate.

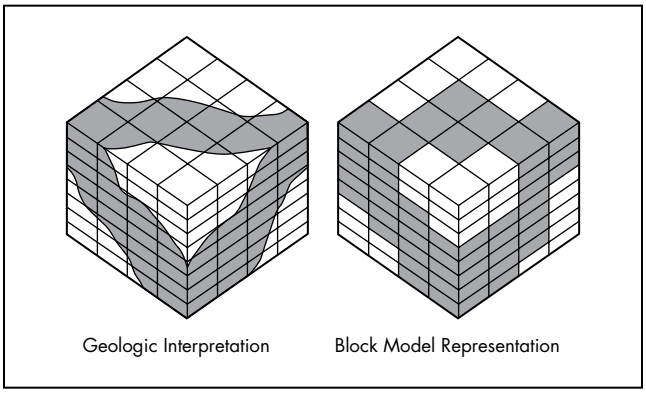


Figure 4.5-12 Geologic feature coded into a block model

The most common problem with geometric methods is that they may imply more selective mining than may be achieved by the mining method. This results from estimating the resource from samples the size of a drill hole but mining larger, less-selective volumes. High-grade blocks usually include lower-grade material when they are mined, and low-grade blocks usually include some higher-grade material. The resulting mined grades are different from the predicted distribution; for cutoff grades below the average grade of the deposit, the mined grade will be lower and the tonnage will be higher. If the cutoff grade is significantly higher than the average grade of the deposit, however, both the mined grade and tonnage can be lower, resulting in a severe overestimation of contained metal.

Moving-Average Methods

The moving-average methods, inverse-distance weighting and kriging, are the most widely used procedures for computer-assisted resource estimation. The basic procedure for both of these methods is as follows:

1. Divide the ore body into a matrix of 3-D, rectangular blocks, as shown in Figure 4.5-12.
2. If geologic controls are present and will be used to control or modify grade assignment, a geologic code must be assigned to each block.
3. Estimate the grade of each block by searching the database for the samples surrounding each block and computing the weighted average of those samples. The weighted average is computed using the following equation:

$$g^* = \text{estimated grade} = \sum w_i g_i, i = 1, 2, 3, \ldots N \quad (4.5\text{-}9)$$

where

w_i = weight assigned to sample i
g_i = grade of sample i
N = number of samples used for estimation

Practical Considerations for Moving-Average Methods

The determination of block size, anisotropies, and sample selection criteria are common considerations for either kriging or inverse-distance estimation. These factors are often the most critical factors in developing a satisfactory resource model because the geologic controls are introduced via these mechanisms. Poor selection of these parameters commonly results in an unsatisfactory resource model with significant overestimation or underestimation of ore tonnage or grade.

Block size. Factors that must be considered when determining block size include the size of the resulting model, drill-hole spacing, mining method, and geologic controls. General rules for block size determination are as follows.

1. The largest block size possible should be used to minimize the size of the block model and reduce computational time and disk storage requirements.
2. The block is normally one-half to one-fourth the average drill-hole spacing. A bigger block may be used if the drilling is extremely dense. Smaller block sizes provide minimal improvement in the estimation unless strong geologic controls are present.
3. The block size must be at least one-half the size of the smallest geologic feature that will be modeled. Larger blocks will destroy the location and/or size of small features.
4. Block size may be related to a proposed mining method. The block height is usually the same as the bench height in a deposit that will be mined by open-pit methods.
5. Most commercial software packages for resource estimation allow each dimension of the block to be a different size and may allow rotation of the entire model to an orientation other than north-south.

Anisotropies. Strong anisotropies, or trends, are often observed in mineral deposits that have ore zones with greater continuity in favorable orientations. Anisotropies can often be identified and are commonly shown by variograms with longer ranges in certain directions.

Sample selection criteria. The purpose of the sample selection step is to provide a subset of the data that is representative of the region around the block. Weighted moving-average methods may be very sensitive to sample selection. The following rules can assist in defining a sample selection search.

1. Samples must be selected from geologic domains similar to that of the block.
2. The maximum radius should be at least equal to the distance between samples to prevent discontinuities in the weighted average as samples drop in and out. (For a square grid, the maximum radius is the diagonal.)
3. The maximum number of samples is usually on the order of 8 to 12. More than 12 samples rarely improves the estimate; fewer than 8 samples may cause discontinuities in the estimated grades.
4. A minimum distance to the nearest sample may be used to prevent excessive extrapolation.
5. A search ellipse or other anisotropic pattern may be used to align the search with trends in the ore, as shown in Figure 4.5-13. The axes of the search ellipse should be oriented parallel to grade trends. The length of the ellipse axes should be proportional to the range of continuity in the respective directions. The variogram ranges and visual appraisal of the grade zones on plans and sections are both used as guides to determining the orientation and length of the search axes.
6. Three composites are usually the maximum required from a single drill hole. More than three provides redundant data and may cause strange kriging weights. For example, the outermost composites in a group of five from the same hole may have larger weights than the inner points.
7. Search patterns may be modified to select data with quadrants or other geometric limits as shown in Figure 4.5-14.

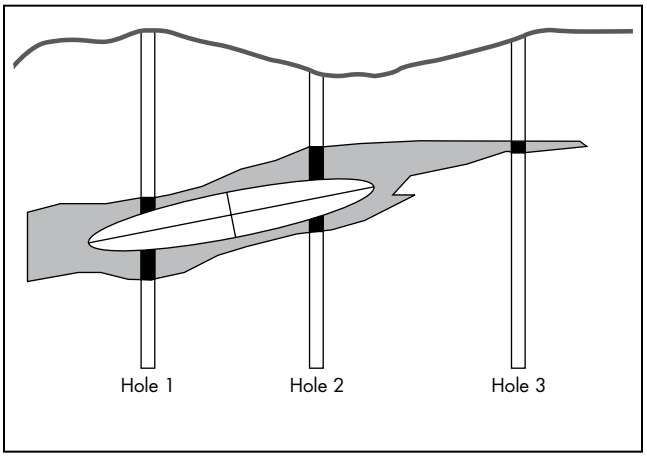


Figure 4.5-13 Anisotropic search ellipse oriented parallel to a mineral zone

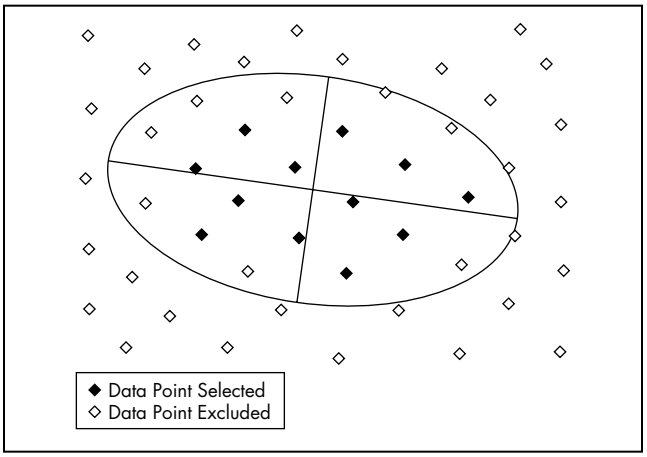


Figure 4.5-14 Oriented, anisotropic search ellipse with a quadrant search limit

Use of a quadrant search will improve estimations if data are clustered.

8. The search pattern should be sketched on plan maps and/or cross-sectional maps in both well-drilled and sparsely drilled areas. This will aid in visualization and ensure that the search pattern is appropriate. Computer software may also provide a direct visualization of the 3-D search ellipse, which is very helpful.

Inverse-Distance Estimation

Inverse-distance weighting, one of the earliest interpolation methods, is based on an empirical observation that the weight of each sample in Equation 4.5-10 is proportional to an inverse power of the distance from the location of the estimate to the sample. The inverse-distance estimate is thus a weighted average with the individual weights computed as an inverse power of distance as follows:

$$w_i = \text{weight for sample } i = \frac{d_i^{-p}}{\sum d_i^{-p}} \qquad (4.5\text{-}10)$$

where

d_i = distance between sample i and the location being estimated
p = weighting power

Care must be taken with an inverse-distance estimate to ensure that none of the distances d_i are very small or equal to zero, resulting in division by zero, or floating point overflow. This problem may be alleviated either by adding a small constant to each distance or by assigning the value of the closest point to the estimate if the distance is less than some threshold, depending on the resource estimation software.

Although it is common to use an arbitrary power like 2 or 3 for inverse-distance weighting, the estimate can be improved by using geostatistical methods to calibrate the estimation as follows:

1. Compute the average variance of samples in blocks that are the same size as the selective mining unit that will be used for mining. The average variance in blocks is referred to as the F-function and is generally available in geostatistical software.
2. Using Krige's relationship, the variance of selective mining units (SMUs) is as follows:

$$\sigma^2_{\text{SMU}} = \sigma^2_{\text{samples in deposit}} - \sigma^2_{\text{samples in SMU blocks}} \qquad (4.5\text{-}11)$$

3. The inverse distance estimate can be adjusted so the variance of estimated blocks is equal to the variance of SMU blocks by adjusting the power, the size of the search ellipse, or the number of points.
4. The variance of estimated blocks is reduced by using a lower power, increasing the radius of the search ellipse, and/or increasing the number of points used in the estimation.
5. The variance of estimated blocks is increased by using a higher power, reducing the radius of the search ellipse, and/or decreasing the number of points used in the estimation.

A certain amount of trial-and-error and experience is required to optimize the estimation parameters. In addition to the above procedure for matching the variance of estimated blocks to the estimated SMU variance, it is also useful to compare the cumulative grade distribution to the grade distribution from production results, and to plot the estimated values and visually verify that the estimation makes sense.

Kriging

Kriging is the geostatistical estimation method developed to provide the "best linear, unbiased estimate" for grade based on a least-squares minimization of the error of estimation, or kriging error. Kriging and its variants have had much theoretical development and are well described in the geostatistical literature. For a more detailed discussion of kriging, the reader may refer to any geostatistical textbook. Similarities and differences between kriging and inverse-distance power are as follows:

- Both methods are weighted moving-average methods and the summation of weights is set equal to 1 so that the estimate is unbiased.
- The kriging weights are determined by solving the kriging equations, which uses as input the covariances between the samples and the estimated block and between each pair of samples. Because the covariances are determined by the variogram, sample weighting is dependent primarily on the variogram rather than trial and error.
- Because the kriging equations consider the covariances among the samples, kriging will reduce the weights of a

cluster of samples, minimizing the effect of variable sample spacing. Inverse-distance-power estimation, however, may be strongly impacted by irregular sample spacing.

Kriged estimates, however, are often "oversmoothed," and the variance of estimates is lower than expected for the variance of SMU blocks. If the estimate is unbiased, the variance of the estimated blocks is the primary factor that determines the shape of the grade distribution, and oversmoothing may cause a significant bias in the tonnage and grade above a cutoff grade. (Generally the tonnage is higher and grade is lower for an oversmoothed estimate.)

Because the kriged estimate is mostly dependent on the variogram estimate, it is difficult to correct for smoothing of the estimates. If the degree of oversmoothing is small, reducing the search radius and the number of samples may be helpful. Although not justified by geostatistical theory, reducing the nugget effect and/or using point kriging instead of block kriging may also be helpful in minimizing oversmoothing of estimates.

Other Types of Kriging

Because of the previously mentioned problems with ordinary kriging, a number of improved kriging methods have been developed in an effort to define not only the local grade, but also to estimate accurately the tonnage–grade distribution. These advance techniques include indicator kriging, multiple-indicator kriging, disjunctive kriging, and uniform conditioning. All of these methods require a highly skilled professional for reliable estimation and are beyond the scope of this handbook.

Dilution and Ore Losses

Dilution and ore losses are a key factor in the conversion of mineral resources to ore reserves. In general, dilution and ore losses are related to either volume–variance effects or geometric effects. Volume–variance effects relate to the decrease in the variance of mining blocks, or SMUs, as the size of blocks becomes larger. The general relationship between the variance of SMUs and samples was shown previously by Krige's relationship (Equation 4.5-11). Some general rules regarding volume–variance effects on resource estimates are as follows:

1. Where the deposit grade distribution is entirely above the mining cutoff grade, volume–variance effects do not need to be considered.
2. Where the estimated variance is higher than the SMU variance, such as with a polygonal estimate, cutoff grades below the median grade tend to underestimate tonnage and overestimate grade. This is corrected by adding dilution tonnage with a grade that is lower than the cutoff grade. The amount of dilution and grade of the dilution is difficult to estimate without production experience with the particular deposit or similar deposits. At cutoff grades higher than the median grade of the deposit, the average grade and tonnage may both be overestimated, and the resource may need to be adjusted with both a dilution and tonnage reduction factor.
3. Where the estimated variance is lower than the SMU variance, such as with a kriged estimate, cutoff grades below the median grade tend to overestimate tonnage and underestimate grade. Where the cutoff is above the median grade, both tonnage and grade may be underestimated. Kriged distributions are difficult to correct for oversmoothing of the grade distribution, which has been the driving force for development of the advanced kriging methods such as multiple indicator kriging and uniform conditioning.

Geometric dilution and ore losses are due to the inability of the mining method to follow accurately and to segregate small isolated pods and small irregular offshoots from the main ore body. Geometric dilution is most significant in deposits with sharp contacts between high-grade ore and barren waste and least significant in deposits with gradational contacts between ore and waste. Dilution tonnage is estimated according to the quantity of waste mined with the ore based on the mismatch between ore body and mining geometry, overbreak in blasting, or lack of accurate location of the ore–waste contact, as shown in Figure 4.5-15. Care must be taken in estimating dilution that the actual ore–waste contact is not more irregular than the model, given that dilution will be underestimated, as shown in Figure 4.5-16. Dilution grade is estimated as the grade of the waste at the ore–waste contact. Mining losses and grades are estimated according to similar procedures.

Selection of Resource Estimation Methods

Selection of an appropriate resource estimation method depends on the geometry of the deposit, the variability of the grade distribution, the character of the ore boundaries, and the amount of time and money available to make the estimate. Deposit geometry determines the amount of detail that must be interpreted and input to the reserve estimation; the variability of the grade distribution determines the amount of smoothing required to estimate mineable blocks; the character of the ore boundaries determines how grade will be estimated at the borders between different grade zones; and the available time and money determine the detail and effort that will be expended on the estimate. Considerations for selection of a resource estimation method are summarized in Table 4.5-3.

Cost

Simple, manual methods such as polygonal and cross-sectional estimations are the least expensive and quickest methods for the estimation of resources when the quantity of data is small. This is usually the case for preliminary evaluations at the exploration stage. As the amount of data increases and a more detailed estimate is desired, computer-assisted methods should be used in order to save time and money. The least expensive computer-assisted methods are automated polygonal or nearest-neighbor methods, and the most expensive methods involve extensive definition of geologic controls in conjunction with the more complex geostatistical methods.

Ore Boundaries

The appropriate reserve and dilution estimation method is determined by the character of the ore–waste contacts. Sharp, simple boundaries are modeled with linear outlines defining discrete mineral zones. Individual estimations are made for each mineral zone; dilution is estimated based on the intersection between the shape of the mineral zones and the shape defined by the geometry of a mining method. A sharp, irregular

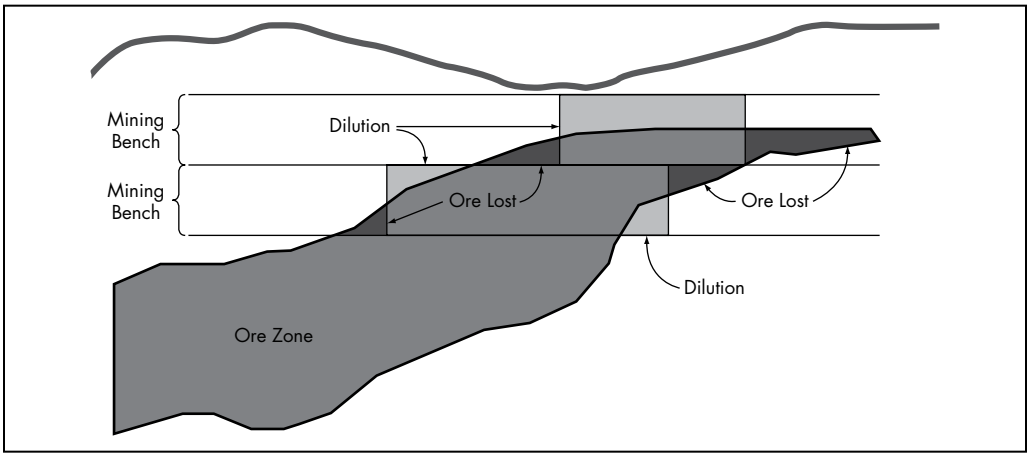


Figure 4.5-15 Dilution and ore losses caused by geometric mismatch between mining geometry and ore-zone geometry

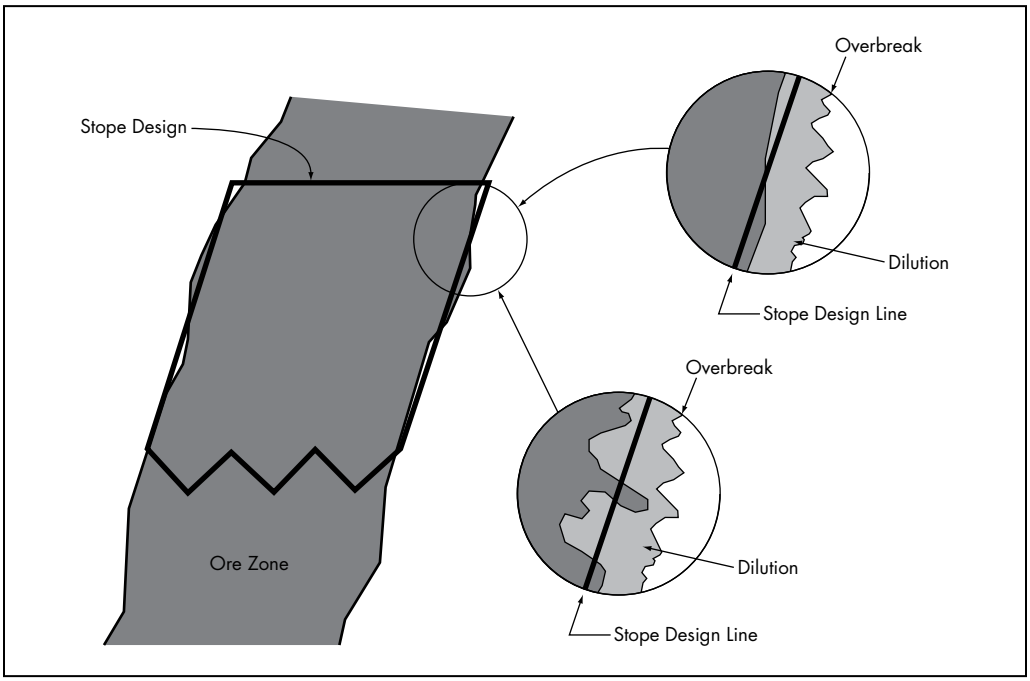


Figure 4.5-16 Underestimation of dilution because of an irregular ore–waste contact

boundary is also described with linear boundaries defining mineral zones; the actual ore–waste contact will be much more irregular than the interpreted boundary, and dilution must be increased accordingly. Geometric methods are usually appropriate for ore bodies with sharp contacts, although kriging or inverse-distance methods may be used within the zones if supported by sufficient data.

Gradational boundaries are handled as transitional between different mineral zones; kriging or inverse-distance methods are most appropriate to model ore bodies with gradational contacts. Sufficient dilution for a gradational contact is usually incorporated in the modeling method.

Extremely erratic, irregular boundaries are difficult to define accurately and are most appropriately estimated using methods such as indicator kriging.

Deposit Geometry

Simple geometry is often found in tabular, stratabound deposits, veins, and structural zones. The geometry of these deposits is easily described using two-dimensional (2-D) methods such as contouring of thickness and elevation. Few additional controls are required other than boundaries to limit the lateral extent of the mineral zones.

Deposits with *moderately complex geometry* include both deposits with simple geometry that have been moderately folded or faulted, and deposits with large, simple, massive shapes such as porphyry copper and molybdenum. Definition of deposit geometry will include definition of fold axes, fault boundaries, and zoning of trends within the deposit. Although these controls are not usually difficult to define, their definition is necessary to provide accurate resource estimates.

Table 4.5-3 Selection of estimation method based on deposit geometry and variability

	Low Variability (COV <0.25)	Moderate Variability (COV 0.25–0.75)	High Variability (COV >0.75)
		Simple Geometry	
Deposit description	Tabular, continuous grade and thickness. Flat or constant dip.	Tabular, large ore pods. Moderately variable grade.	Tabular, small ore pods. Highly variable grade.
Example deposits	• Evaporite • Sedimentary iron • Limestone • Coal	• Stratiform copper • Mississippi Valley lead • Simple porphyry copper, molybdenum	• Gold veins • Gold placers • New Mexico uranium • Alluvial diamond
Estimation methods	Grade and thickness using any 2-D method: polygonal, contouring, inverse distance, kriging. Geometric controls for boundaries of ore zone, faults, and fold axes.	2-D methods. Inverse distance or kriging. Polygonal or cross-sectional with 5% to 15% dilution.	2-D methods. Inverse distance or kriging with recovery functions. Polygonal with 15% to 35% dilution.
		Moderately Complex Geometry	
Deposit description	Simple, bedded. Uniform grade but erratic thickness, gentle folding, or simple faulting.	Simple 3-D geometry. Moderately variable grade.	Simple 3-D geometry. 2-D with smaller, more erratic ore pods. Simple folding, faulting.
Example deposits	• Bauxite (variable thickness) • Lateritic nickel (variable thickness) • Salt dome	• Porphyry copper • Porphyry molybdenum	• Stockwork and Carlin-type gold • Volcanogenic base metals
Estimation methods	Estimate grade, thickness, and elevation using any 2-D method. Must define structural geology (faults, fold axes). Variability of thickness may be difficult to predict.	Inverse distance or kriging with external controls to define the shape and grade trends. Polygonal and cross-sectional methods may be used but will require dilution/volume–variance correction.	Inverse distance or kriging with recovery functions. Polygonal or cross-sectional with 15% to 35% dilution.
		Very Complex Geometry	
Deposit description	Otherwise simple deposits that have been severely folded and faulted.	Complex geometry due to faulting, folding, or multiple mineralization controls. Moderately variable grade.	Deposits with extremely variable grade and highly contorted, complex ore shapes. Typically little continuity between individual ore zones. General mineral envelope definable but with 50% or less ore.
Example deposits	• Talc • Gypsum (deformed)	• Tungsten skarns (folding/faulting) • Base metal skarns (erratic shape) • Copper porphyry combined with local skarns or replacements (multiple controls)	• Archean gold deposits • Roll-front uranium
Estimation methods	Cross-sectional methods with detailed definition of structural geology. Difficult to define geometry for 3-D block models and geostatistical methods.	Cross-sectional methods with detailed input to describe structural geology and ore zones. Geostatistical methods may be appropriate but difficult to implement because of geometric complexity.	Estimation very difficult. Size, shape, and grade not locally predictable. Cross-sectional, area-outline methods, indicator kriging applicable. Errors of 50% to 100% not unusual. Tonnage often overestimated because of incorrect geologic model.

Deposits with *very complex geometry* are usually associated with structural deformation and are folded, faulted, stretched, and twisted to form extremely discontinuous shapes that are difficult to describe and model. Multiple ore controls such as a combination of stratigraphic and structural controls or multiple, overlapping pulses of mineralization also commonly form very complex shapes. Definition of deposit geometry requires detailed examination of structural geology and ore controls to provide cross sections or plan maps that define the shape and location of mineral zones. These sections or maps may then be used directly for manual resource estimation or may be digitized to provide control for a computer block model or 3-D wireframe model. Deposits with complex geometry are prone to large estimation errors because of misinterpretation of deposit geometry and ore controls; order-of-magnitude errors are not infrequent.

Grade Variability

Deposits with low variability may be estimated with many methods. Common methods include automatic contouring and polygonal methods with cross-sectional estimation or area-averaging techniques for more complex geometry.

Weighted averaging methods (kriging and inverse-distance) are most commonly used for estimation of deposits with moderate variability, although polygonal or cross-sectional methods are also used. Weighted averaging methods may require recovery functions and polygonal methods may require dilution to compensate for volume–variance effects, although in most cases the adjustments are small, on the order of 5% to 15%.

Weighted averaging methods are most commonly used for estimation of deposits with high variability. Other appropriate methods may include indicator kriging, polygonal, and cross-sectional methods. Volume–variance effects are often large with these deposits and must be compensated for with recovery functions for weighted averaging methods and large dilution of polygonal and cross-sectional reserves. For COVs above 2.0 or 3.0, local grade estimates are extremely difficult and must be tempered with judgment and caution.

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CHAPTER 4.6

Valuation of Mineral Properties

Bernard J. Guarnera and Michael D. Martin

INTRODUCTION

The valuation of mineral properties or mining companies involves the integration of geology, mining, processing, mineral markets, society, and the environment Accordingly, it is common for a multi-disciplinary team to work on valuation efforts and their findings to be incorporated into the valuation. It is essential, however, that any effort be led by an experienced valuator who assumes responsibility for the valuation report.

What Is a Valuation?

How does a *valuation* differ from an *evaluation*? An evaluation simply focuses on the technical aspects of an asset or assets, whereas a valuation focuses on the worth of the asset. Two major factors are considered:

1. **Highest and best use:** Although all mineral-containing properties have an inherent value, which in itself does not indicate that a valuation of the minerals is required, the valuation performed must be based on the highest and best use of a property. An example would be a mineral deposit suddenly discovered on an undeveloped property in the middle of an area with developed residential or commercial real estate. It is possible that the value of the real estate would exceed the value of the minerals (the highest use) or, if it did not, that real estate development was the only possible use of the property because of zoning or environmental factors (the best use). Therefore, unless the valuation was for a condemnation proceeding specifically to value the mineral interest, the highest and best use would be deemed to be real estate development.
2. **Fair market value (FMV):** The valuation should always be based on the FMV of the asset, which is the price an asset would be exchanged for with the parties being a willing buyer and seller, with both parties having access to the same information about the asset, and with neither party being under compulsion to buy or sell the asset.

Types of Properties

Valuation methods vary in type and effectiveness for both undeveloped properties and properties already in operation. Properties warranting or requiring a valuation can range from raw land, where the presence of minerals is only suspected, to large developed properties that have been mined for many years. The commodities can include metallic minerals, nonmetallic minerals, energy minerals, and gemstones.

Valuation Assumptions

Before a valuation is undertaken, certain basic assumptions must be satisfied:

- Mineral development is the highest and best use of the property (unless the valuation is for condemnation purposes).
- A fair market value is attainable.
- All lands have an inherent value for minerals that might occur on them.
- A market exists for the mineral or minerals that may be on or under the land.
- Economic realism must be employed (e.g., a granite deposit under an ice cap would have no value, whereas one adjacent to a major city could be developed for aggregates or dimension stone).

Existing Mineral Valuation Codes

Although valuations of assets have many things in common, it is recognized that the valuation of mineral deposits, properties, or mining companies requires expertise beyond that offered by the typical appraiser. In recognition of these differences, specific codes governing the valuation of mineral deposits and properties have been developed by professional mining associations in countries where mineral resources significantly contribute to the economy:

- **VALMIN code**—Australasian Institute of Mining and Metallurgy. This code is statutory in Australia.
- **CIMVAL code**—Canadian Institute of Mining, Metallurgy and Petroleum. This code is due to become statutory in Canada.
- **SAMVAL code**—South African mining associations. This code is statutory in South Africa.

The Mining and Metallurgical Society of America is in the process of developing recommended standards for mineral

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Table 4.6-1 Applicable valuation methods

Method	Types of Properties				
	Exploration Stage	Feasibility Stage	Development Stage	Operating Stage	Multi-Property Company
Income (cash-flow) approach		x*	x	x	x
Market-related transaction	x		x	x	x
Market multiples approach			x	x	x
Replacement cost approach	x	x	x	x	
Option/real option pricing					x
Monte Carlo simulation	x	x	x	x	x

Source: Adapted from CIMVAL 2003.
*The income approach may or may not be applicable at the feasibility stage, depending on the reliability of the available information at the time that the valuation is required.

property valuation in the United States. The International Valuation Standards Council is also developing guidelines for the valuation of mineral properties. These are anticipated to focus on market factors and will potentially be in conflict with the above three codes.

Unless specifically requested otherwise, mineral property valuations should be carried out in accordance with one of the VALMIN, CIMVAL, or SAMVAL codes/standards. The choice of code will depend primarily on the reporting location of the company as well as the property, the party requesting the valuation, and the party carrying out the valuation.

TYPES OF VALUATION METHODS

There are three primary methods of valuations:

1. The income (cash-flow) approach, whereby the cash flow resulting from a financial model is discounted at an appropriate rate to yield a net present value (NPV)
2. Market-related approaches, which develop a value based on recent related transactions, and the market multiples approach for publicly traded companies or from recent transactions
3. The replacement cost approach, in which the cost required to duplicate the asset being valued is assessed

Secondary methods include option/real option pricing valuations and Monte Carlo simulations. Table 4.6-1 lists the six valuation methods, together with the types of properties to which they are applicable. The methods themselves are described later in more detail.

In contrast to the other methods listed in Table 4.6-1, the income approach should yield a "true or long-term value" over the life of the asset, provided that the inputs to the cash-flow model are realistic. The market-related transaction or market multiples approach, on the other hand, provides a snapshot value at the time of the valuation; the derived value will likely be higher than the income approach value in prosperous times and lower in difficult times.

The market multiples approach differs from the market-related transaction method in that, rather than comparing the asset against one that was recently sold, it is based on the value ascribed by public markets to units of production of specific commodities. An example would be to base the valuation solely on the pounds of copper or ounces of gold recoverable from the property.

When market valuation methods are used, it is essential that they be adjusted to reflect the realities and characteristics of the asset or company being valued. Failure to allow for these differences will result in incorrect valuations. Thus a property containing 1 million ounces of recoverable gold with the capability of achieving full (cash plus capital) production costs of $200 per ounce is clearly worth much more than another million-ounce property whose full production costs are forecast to be $400 per ounce. Similarly, an underground gold property with a refractory ore would be negatively viewed when compared with an underground gold property with an ore that would only require simple flotation and concentration.

The replacement cost approach can be used as a check on one of the other methods, or alone if none of the other methods is particularly applicable. This method puts a value on finding another similar mineral property and replacing similar infrastructure that previously existed. This method is most commonly used for valuing early-stage exploration properties or properties that have ceased operations but still have resources or reserves. When using this approach, it is essential to consider improvements in technology.

The option/real option pricing valuation approach should be used only to value a company with multiple operations, rather than an individual property. This method is described later in this chapter.

The Monte Carlo simulation approach is a method of analysis based on the use of random numbers and probability statistics to investigate problems with variable potential outcomes. In financial analysis and valuation there is a fair amount of uncertainty and risk involved with estimating the future value of financial numbers or quantity amounts because of the wide variety of potential outcomes (i.e., grade of deposit, reserve tonnage, commodity price, operating costs, capital costs, etc.) The use of Monte Carlo simulation is one technique that can be applied to evaluate the uncertainty in estimating future outcomes and allows for the development of plans to mitigate or cope with risk.

Typically with conventional spreadsheet models, the engineer, geologist, or analyst creates models with the best-case, worst-case, and average-case scenarios, only to find later that the actual outcome was very different. With Monte Carlo simulation, the analyst explores thousands of combinations of the what-if factors, analyzing the full range of possible outcomes—an iterative process yielding much more accurate results with only a small amount of extra work, thanks to the numerous choices of Monte Carlo simulation software that are available. The Monte Carlo simulation cannot eliminate uncertainty and risk, but it does make them easier to understand by ascribing probabilistic characteristics to the inputs

and outputs of a model. The determination of the different risks and factors affecting forecasted variables can lead to more accurate predictions—the desire of all mining managers.

Reviewing Table 4.6-1, one can observe the four stages in the life of a mineral property and the likely applicable valuation methods for each one. Early-stage exploration properties are the hardest to value, whereas operating-stage properties are usually the easiest. In between those two stages, more than one method can usually be employed, with a weighted average value based on the strength of each method used or range of values developed from which a preferred value can be derived. It is also possible for a given property to be in more than one stage at any given time. One such example is a property with undeveloped resources undergoing exploration very near an operating mine.

Valuation Methods for Developed or Operating Properties

Properties that are developed (i.e., ready to operate) or are operating and have a financial history, are usually valued by the income approach. This approach employs the life-of-mine production schedule, forecast or actual operating costs, forecast sustaining and replacement capital costs, and reclamation/closure costs. On the assumption that these have been correctly forecast and projected, the only parameters that would be subject to dispute in this method are the commodity prices and the discount rate used in the valuation.

Some other valuation methods used for developed or operating properties include

- Liquidation value,
- Market-related values,
- Replacement value, and
- The value of a royalty stream if the property is being valued for a lessor.

Income (Cash-Flow) Approach

The income, or cash-flow, method involves constructing a financial model of the cash flow covering the expected life of the mine, generally up to the first 20 years of production. The financial model should be based on constant dollars, where product selling prices, cash operating costs, and future capital requirements are not inflated (varied). It is appropriate to change future operating costs over time by reflecting changing physical conditions, such as longer haul-truck cycles, reduced metallurgical recoveries because of a change in the character of the ore body, and similar measures that the mining professional can predict.

To perform an accurate valuation using this method, the following inputs are required:

- Ore reserves over the life of mine. Resources can be included if factored for their probability of conversion to reserves; however, the valuator should be cognizant of regulatory requirements, such as those of the TSX Venture Exchange (a Canadian stock exchange) that precludes the inclusion of resources in a cash-flow model.
- Production rates
- Operating costs, including on-site general and administrative (G&A) costs, ongoing development costs, and nonincome taxes
- Capital costs—preproduction and sustaining/replacement
- Environmental and reclamation costs
- Commodity prices
- Discount rate

The commodity prices and discount rate utilized in the cash-flow valuation are two critical items that are based on the valuator's experience and judgment. Because of the critical impact these two inputs have on the income approach valuation, they should be developed by the valuator from first principles.

Commodity price selection. While valuations are forward-looking, income approach valuations should normally incorporate a constant commodity price based on long-term historical data. Commodity prices should reflect the up-and-down cycles, which are common to the mineral industry. It is the authors' experience that a 10-year period would normally incorporate both cycles. When valuing an operating property or one near operating status, however, it is acceptable and appropriate to include consensus pricing for the first 2 or 3 years of operation prior to returning to the long-term price. As an example, when an examiner values an operating copper property, if the copper price for the last 10 years has averaged $1.75 per pound, but the current price is $3.50 per pound, the consensus view might be to use $3.50 per pound for year 1 of the cash flow model, $3.00 per pound for year 2, $2.25 per pound for year 3, and then level off at the 10-year average price of $1.75 per pound for the remainder of the mine life.

Discount rate determination. The discount rate essentially reflects the risks present in an investment and is the rate at which the cash flow from a mining property or of a mining company will be discounted. It is never appropriate when conducting a valuation to arbitrarily assign a discount rate; rather the discount rate should be derived from first principles.

Three methods are employed for deriving a suitable discount rate; the method selected is based on the nature of the asset being valued.

1. Weighted average cost of capital (WACC) method
2. Capital asset pricing model (CAPM)
3. Risk buildup method

Weighted average cost of capital discount rate derivation. The WACC method is based on the proportional cost of equity and debt for a particular corporation at a specific time. It should be used as a discount rate only for companies; it is not appropriate for valuing single projects. The key strength of the WACC method is that it incorporates the global risks of all of a company's operations and projects into a single rate, which should reflect the melded risks of the company's assets.

Capital asset pricing model. The CAPM was developed as a valuation tool for shares of publicly traded stocks. It incorporates various elements of an investment, including the risk-free rate of return offered by U.S. Treasury bills and notes, the greater risks inherent in stocks versus other investments, and the volatility of the shares of a company compared to the average company's shares as measured by its beta. (Note: Beta is a measure of a stock's price volatility in relation to the rest of the market. In other words, it is a guide on how a stock's price is likely to move relative to the overall market. Beta is calculated using regression analysis. The whole market, which for this purpose is considered to be the Standard and Poor's 500 (S&P 500), is assigned a beta of 1. Stocks that have a beta greater than 1 have greater price volatility than the overall market and are more risky. Conversely, a beta lower than 1 denotes less volatility than the market and therefore less risk. For example, if the market with a beta of 1 is expected to

return 8% annually, a stock with a beta of 1.5 should return 12%. Young technology stocks will always carry high betas; many utility stocks, on the other hand, carry betas below 1.) The CAPM method is appropriate only for valuing companies; it is not appropriate for establishing the discount rate for individual mining projects or properties. Importantly, the discount rate derived is after-tax for a seller of the shares, and pretax for a buyer of the shares.

Risk buildup discount rate derivation. The risk buildup method is preferred by the authors of this chapter as it reflects the values relevant to the specific properties. In form it is similar to the CAPM method; however, it is differentiated by its inclusion of the technical and other risks associated with the typical mining project. Essentially it adds the components of risk at the project to arrive at an overall risk rate for a given specific property or group of properties. The usual components incorporated are

- The real risk-free rate of return;
- The risk premium expected by an investor who would invest in mining projects which can be assumed to be the same as that for a publicly traded company. There would be additional premium if the project being valued would have a market capitalization of a "small cap" (i.e., less than $200 million);
- Mining industry specific risk; and
- Site-specific risk for individual properties.

The real risk-free rate of return is the difference between the interest rate on U.S. Treasury notes of a maturity approximating that of the project life and the current inflation rate and is measured by the following formula:

$$Rfr = \frac{(1 + Rfn)}{(1 + Ie)} - 1$$

where

Rfr = real risk-free rate of return
Rfn = nominal risk-free rate offered by U.S. Treasury notes
Ie = expected inflation rate

Accordingly, assuming a 10-year mine life and 10-year U.S. Treasury notes yielding 4% with inflation at 1.5%, the real risk-free rate of return is

$$\frac{(1 + 0.04)}{(1 + 0.015)} - 1 = 0.025 \text{ or } 2.5\%$$

With a public company risk premium, investors clearly require a greater return on their investment than that provided by risk-free U.S. Treasury notes. They are willing to accept additional risk for the expectation of a greater return.

If the company involved is a large one (S&P500), the risk premium for such shares can be found at the Ibbotson Associates' Web site. The risk in 2007 was about 7%.

If the company has a market capitalization of less than $200 million (i.e., small cap), an additional risk premium is warranted. In 2007, this was an additional 3% for a total public company risk premium of 10%.

With mining industry risk, based on historic company and industry returns on equity, there is an above-average risk premium for certain industries. These include the aggregate, mining, and petroleum industries, all of which are dependent on the vagaries of natural resources. In 2007, the industry risk premium for the mining industry was 2.5%.

With site-specific project risk, multiple risk factors exist at mining properties ranging from reserve risk through processing, environmental, political, and geotechnical risk. Following are some of the factors that need to be considered:

- **Project status**—This involves exploration, development, or in operation. As a project advances through these stages, the risk factor will normally decrease. For a mature operating property that is performing up to forecasts, the risk will be lowest.
- **Quality of analytical data**—If the quality of the data derived from the drilling, sampling, and assaying of the ore body is suspect, the project risk must reflect this uncertainty.
- **Processing-related risk**—This risk can be high if adequate metallurgical test work has not been performed on samples truly representative of the whole ore body or if new, unproven technology is being employed.
- **Infrastructure-related factors**—Risks can occur if there are unusual circumstances that might cause interruption to the power and water supply or cause access to the property to be lost.
- **Environmental considerations**—In contrast to projects 20 or more years ago, a project located in a sensitive environmental setting must be given a risk rating higher than one that is isolated and insulated from likely environmental damage; government, regulatory, and permitting risks are thus assessed.
- **Operating and capital costs, and working capital**—Poorly predicted figures for these three items introduce substantial risk. The most common of these is an underestimation of total project capital.
- **Prices and markets**—Price projections on which the project economics are based must be realistic, and there must be a market for the product produced.
- **Labor/Management issues**—The availability, education, and trainability of the required labor force in less developed countries is an issue. Union activism poses a risk to some projects. The quality and experiences of the company's management must be considered.
- **Political and social issues, and the social license to operate**—The lack of perceived support from the local inhabitants and government bodies is a major risk.

It is not always possible to secure good information on all of these factors affecting site-specific project risk. If possible, a matrix should be constructed with a ranking from 1 to 10 assigned to each factor. From this, an overall risk factor can be assigned. For an exceptionally low-risk project, a factor of 1% or 2% may be chosen; for one with many uncertainties, the factor is likely to be 5% or higher.

Summary of risk-buildup discount rate. Table 4.6-2 is an example of a risk-buildup discount rate, showing both pretax and after-tax figures. Since the discount rate developed is pretax, it must be converted to an after-tax basis.

Other factors to be considered in the income approach valuation method. Two other factors should be taken into account in an income approach valuation of a property or properties.

The first, and more important of the two, comes into play if an acquisition is involved and if the acquirer will end up being in control of the property, properties, or company. Given that the acquirer will be in charge of his or her own destiny, he or she is not subject to the bad decisions of a senior owner. If

Table 4.6-2 Summary of risk buildup discount rate

Item	Rate, %
Real risk-free rate of return	2.5
Public company risk premium	7.0
Small cap premium	3.0
Industry-specific risk	2.5
Site-specific risk*	3.0*
Total (pre-tax)	18.0
Total (after-tax)	12.0†

*A low-average risk rate of 3% has been chosen for this example.
†From Lerch 1990; the example assumes a tax rate of 33.3%.

the acquirer is in charge, a "control premium" should be added to the total valuation obtained from the income approach method. The amount of this premium cannot be standardized and depends on the type of company and its position in the development/operating chain. During 2007, the control premium for acquisitions of large properties and companies frequently exceeded 30%.

The second factor to be considered is a terminal value of the free cash flow for operations that have a life exceeding that of the financial model. A terminal value is commonly arrived at using the assumption that ongoing operations will mirror the conditions that applied to the last 5 years of the cash-flow valuation, unless there is good reason to expect an ore-grade change or a metallurgical recovery change, and so forth, to occur. The terminal value is measured by the following formula:

$$Tv = \frac{FCF_{N+1}}{(D - G)}$$

where

Tv = terminal value
FCF_{N+1} = annual free cash flow in the residual years after the final year in the financial model
D = discount rate used for the terminal value
G = annualized rate of growth of the enterprise over the life of the financial model

The discount rate used may be higher than that used in the financial model as the inputs to the model would be less certain in the terminal value years.

It is not uncommon for the terminal value to be a significant part of the NPV determined by the financial model.

Market-Related Transaction

On the surface, the market-related transactions or comparable sales approach valuation method should be the simplest to understand and the easiest not to fault. One can simply find several recent transactions with their documented purchase prices and then compare the price paid per pound or ounce at that property with the one requiring the valuation. Unfortunately, it is not that simple. No two mining properties are even remotely identical due to differences in all the parameters that were itemized in the site-specific project risk discourse previously discussed. Even parts of the same mineral deposit can be different. Nevertheless, because of the perceived simplicity of the method, this is a frequently used valuation method and is a preferred technique by the International Valuation Standards Council.

To achieve even relative comparability, all transactions considered must be adjusted in relation to the property being valued. For example, if both are narrow-vein, underground gold properties and one has a grade of 0.6 ounces per ton and the subject property has 0.3 ounces per ton, the value of an ounce at the subject property will obviously be lower than the property it is being compared with. Similar adjustments need to be made for mining costs, processing costs, political factors, geography, and so on.

Market-related transactions, as applied to exploration properties. Generally little information is available about exploration properties due to the early stage of the property in the mine development cycle. Assuming that results are positive, the value of exploration properties increases with the level of work performed. Frequently a "prospectivity factor" is added or deducted to the value based on known results, regional settings, and history.

By the time that a property has either been fully explored, reached the development stage, or started production, there are likely to be other transactions that can be used for developing a market-related transaction valuation, provided that the individual differences between the properties are taken into account.

Market-related transactions, as applied to development or operating properties. When a property is either in development or operating, there will be much credible information available for it, and, unless the commodity is an unusual one, there are likely to be several fairly recent comparable transactions to reference for the valuation. Even so, care must be taken in two areas:

1. The transaction prices for the comparables must be adjusted to present-day conditions when either or both metals prices and costs of production may have changed; and
2. The transaction prices must be adjusted to reflect the different variables that will have affected the price paid for each property, including the relative size of the mineral deposit; differences in ore grade, mining method, and processing recoveries and methods; and the amount and cost of required infrastructure, operating and capital costs, environmental and social issues, tax regimes, and political risk.

Market Multiples Valuation

The market multiples valuation method has similarities to the market-related transactions valuation method and has some of the same drawbacks (principally property or corporate differences). It also has the "advantage" wherein other transactions (comparable existing properties) do not have to be identified and evaluated.

Market capitalization, which is the quoted share price multiplied by the number of issued shares, can be divided by many factors to derive a value per ounce or pound of proven and probable ore reserves or resources, the value per pound or ounce of annual production, the multiplier given to earnings, and so forth. These different metrics constitute a market multiples valuation, and these can then be used to develop a generic value for the company. Such figures are available for many mineral companies, enabling an average valuation per unit of the metric to be established.

A market multiples valuation can also be based on

- A multiple of average annual cash flow, and
- A multiple of earnings before interest, taxes, depreciation, and amortization.

Again, adjustments must be made to ensure that the value developed is truly based on comparable factors. For example, a market capitalization value for a major mining company with several producing mines should not be used to develop a market multiples value for a junior company with only one producing mine.

Replacement Cost Valuation

Replacement cost valuations are simply the expenditure that would be required in current dollars (or other currency units) to duplicate a prior effort. Replacement cost valuations are most commonly used for

- Exploration properties at various stages, and
- Operations that have been shut down with remaining resources or reserves.

For exploration properties, the costs of land acquisition, duplicating any geological, geochemical, or geophysical work, duplicating the prior drilling and assaying performed, and so forth, are determined as the basis of the property's value. Any negative results must be considered, and, using the appraiser's judgment, they may be subtracted entirely or included in a factored manner.

When being applied to operations that have been dormant for a period of time but which still have facilities in place, the replacement cost valuation focuses on the current cost required to replicate the facilities. A factor that must be considered is whether new technology has made the original equipment obsolete. If such is the case, the cost of the new technology must be included, although it is possible that this would overvalue the property. Another factor that should be considered is whether there has been any change in the markets for the commodity that was previously produced.

If the property is being valued by the replacement cost method and resources and/or reserves are still present, the value could be based on the cost of replacing those ounces, pounds, or tons present.

Option/Real Option Pricing Valuation

Although used less frequently than the methods already described, the option/real option pricing valuation method is one that can be used for valuing mining companies with multiple operating properties. The philosophy behind options is based on the formula developed in 1973 by Black and Scholes to be used in the valuation of equities. As currently applied to mineral properties, option valuations are based on the following premises:

- The income approach valuation method may undervalue both producing and nonproducing mining assets. This is generally true in "boom" times, but incorrect in difficult times.
- Mining properties offer the opportunity to be shut down when economics are negatively affecting cash flow and reopened when economic factors are positive. Although this is true in concept, in practice, closing and reopening mines based on volatile economic changes is impractical and would potentially be financially ruinous if attempted by mining companies. The cost of shutting down, maintaining the property on a standby basis, and the time it would take to reopen and ramp-up production is not considered in option theory.
- Mining properties offer a call option on increases in metals prices. (If the gold price is, for instance, $300 per ounce, then a property requiring a price of $350 per ounce to generate a positive cash flow has a finite value.) Note: For readers not familiar with the concept of options, reference is made to puts and calls on 100 shares of a stock on a major stock exchange. Simply, each call gives the call owner the right to purchase 100 shares of the stock in question at a fixed price for a fixed period of time. (The lower the fixed price and the longer the period of time, the higher is the price of buying the call.) For example, if Party A owns 100 shares of a stock currently selling at $100 per share and the calls on a price of $110 per share expiring 2 months in the future are trading at $3 per share, then Party A can sell a call on his or her stock and immediately pocket a check for $300. If the stock does not reach the call price of $110 per share in the next 2 months, Party A will have made $300 and will still have the stock. In the meantime, Party B has bought Party A's call for $300, but if the stock does not reach $110 a share within the 2-month time period, Party B will have lost their $300. However, should the price of the stock rise to, for instance, $116 per share before the 2 months are up, Party B will have doubled their initial investment of $300. (Party B's call gives them the right to buy the stock at $110 per share and they can turn around and immediately sell it for $116 per share, thus realizing a net profit of $300.)

When considering the use of option valuations, it is also important to recognize that

- The longer the option period, the higher the value will be;
- The greater the volatility of the commodity price, the higher the value will be;
- This valuation method will always produce the highest (and probably unrealistic) value; and
- This method is applicable to valuations of companies, not single properties.

Monte Carlo Simulation

The Monte Carlo simulation method can be used for any properties that are at least at the advanced exploration phase. Monte Carlo simulations allow for multiple variables to be changed simultaneously while a specific operation is mathematically performed literally thousands of times. The probabilistic value results from a range of probabilities assigned to each variable in the analysis (i.e., capital and operating costs, and commodity prices) to arrive at a most likely value, or range of values, as based on iterations of cases that sample the distributions of each variable.

Alternative Valuation Methods for Undeveloped Properties

Undeveloped properties include those with blocked-out resources or properties with drill holes that have "ore grade" intercepts. Although the lack of concrete information makes the valuation of such properties more difficult, a "probability" approach, such as the risk-adjusted income approach, can be used. The approach entails the construction of a financial model of the property using likely production rates, ore grades, mining and processing methods, and capital and operating costs.

A justifiable commodity price is chosen, the real risk-free rate of return is used for the discount rate, and the discounted cash flow is calculated. The valuation for an example property then becomes the calculated NPV (say, $100 million), as adjusted for the percentage probability that the items incorporated in the financial model, such as ore reserves, costs, and environmental risks, have been correctly estimated. If the risks for the stated items are, respectively, 80%, 90%, and 50%, the valuation would be $36 million ($100 million × 0.8 × 0.9 × 0.5).

Alternative Valuation Methods for Exploration Properties

Exploration properties include those where no work has been performed and those where some work has been performed.

For properties where no work has been performed, two methods are commonly used:

1. The valuation is a percentage of the surface value of the property. For no work of any kind in a mineralized or unmineralized area, the percentage is 5%. For raw property, but where initial reconnaissance has indicated favorable potential, the percentage is 10%.
2. The valuation is the money that has been spent in staking/leasing and maintaining the property.

For properties where some exploration work has been performed, the following methods are commonly used:

- Modified cost of work performed, with prospectivity factors included
- Geoscience matrix valuation

In the modified cost of work valuation method, the direct costs of work performed are added to valid G&A costs to arrive at a base value. If there have been some highly favorable exploration results, some enhancement of the base valuation is appropriate. Similarly, if results on or at nearby similar properties have been negative, a negative prospectivity factor is applied.

The geoscience matrix valuation method was developed by Lionel Kilburn for the British Columbia Securities Commission to assist them in validating the values being assigned to exploration properties by junior mining companies. Five major criteria are considered, which are divided into nineteen possibilities:

1. The location of the property with respect to off-property mineralization;
2. The presence of any on-property mineralization;
3. The location of the property with respect to off-property geochemical/geophysical/geological targets;
4. The presence of any on-property geophysical/geochemical targets; and
5. Geological patterns on the property associated with known commercial deposits.

The starting point, or base value, for the valuation is the per-acre or per-hectare cost of acquiring the right to a mineral property, usually the cost of staking and maintaining a claim for 1 year. The property is then rated on the basis of its score from the matrix, and this rating is then used to adjust the base value. The value from the matrix is arrived at by assigning points in the five categories, based on whether the property is above or below average. Table 4.6-3 illustrates how the matrix rating is derived.

Table 4.6-3 Categories used in matrix valuation

	Category	Rank	Value Factor
A.	Location with respect to off-property mineralization		
	Sub-ore grade in two horizontal directions	17	1.5
	Ore grade with two horizontal dimensions	13	2.0
	Sub-ore grade with three dimensions known	12	2.5
	Ore grade with three dimensions known	8	3.0
	A past or present producing mine	5	4.0
	A major past or present mine	4	5.0
B.	Location with respect to on-property mineralization		
	Interesting but sub-ore grade with two horizontal directions	13	2.0
	Ore grade with two horizontal dimensions of economically interesting size	8	3.0
	Interesting but sub-ore grade in three dimensions	4	5.0
	An economically interesting ore-grade zone in three dimensions	3	6–8
	Past producer with ore grades measured in three dimensions	2	7–8
	Major past or present producer with ore grades measured in three dimensions	1	9–10
C.	Location with respect to off-property geochemical/geophysical/geological targets		
	One target or two, based on different methods	19	1.3
	Three or more targets	17	1.5
D.	Location with respect to on-property geophysical/geochemical targets		
	One target	13	2.0
	Two or three targets	8	3.0
	Four or more targets	7	3.5
E.	Geological patterns associated with known commercial deposits		
	One or two patterns	13	2.0
	Three or more patterns	8	3.0

Source: Adapted from Kilburn 1990.

RULES-OF-THUMB VALUATIONS

In the rules-of-thumb valuation method, the valuation is based on a percentage of the commodity's price, with the percentage dependent on the state of advancement of the particular property. Table 4.6-4, based on more than 500 transactions analyzed by Frank Ludeman in his publication, *A Decade of Deals*, gives the range of percentages for the different stages of properties (Ludeman 2000).

The rules-of-thumb values provided in Table 4.6-4 should be considered as generic, and the actual percentage a property will value varies with the tenor of the mining industry. The 500 properties studied provided an average value, and the percentage of the commodity price assigned to a property should be based on its characteristics versus that of the "average" property.

REQUIRED QUALIFICATIONS FOR A VALUATOR

The required qualifications for a valuator will depend to some extent on the complexity of the property to be valued, as well as on the type and number of the methods to be employed. The greater the complexity and the number of methods to be

Table 4.6-4 Rules-of-thumb values

Property Stage	Precious Metals (%-per-ounce price)	Base Metals (%-per-pound price)
Early exploration	1.5–2.5	1.0–2.0
Inferred resources	2.5–5.0	2.0–3.0
Measured and indicated resource	5.0–7.5	3.0–5.0
Feasibility	10.0–15.0	5.0–7.50
Production	20.0+	10.0+

Source: Data from Ludeman 2000.

used, the greater must be the knowledge and experience of the valuator (who may be one individual with all the necessary skills and experience or a small team whose combined expertise covers all the skills needed).

The principal qualifications, not necessarily in order of importance, are

- A total lack of bias as to the outcome of the valuation,
- Knowledge of and previous experience in the valuation method(s) to be used, and
- Familiarity with all relevant aspects of the minerals industry.

Sometimes in special circumstances, the valuator or the head of the valuating team may be required to be a member of an accredited appraisal association or professional society. The valuator and the members of the valuation team should hold degrees in geology, mining engineering, or metallurgy; however, overall experience in, and a working knowledge of, the minerals industry is the most important qualification.

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CHAPTER 4.7

Mineral Property Feasibility Studies

Richard L. Bullock

Mine feasibility studies are nearly as old as the industry itself. In the first recorded writing on mining by Agricola (1556), he gave many clues as what to look for in evaluating a mine. Most mineral engineers, geologists, mineral company executives, and mineral development lending agencies think of the feasibility study as the formal methodology that brings the necessary information on a property's raw mineralogical data through the feasibility and preliminary design process and to the point where a comparable economic analysis of the envisioned project can demonstrate financial viability. But the feasibility studies have to be completed by many people evaluating different commodities for many different types of mines and process plants in many different climatic, political, and social environments. Yet, in the end, all of the feasibility studies must accomplish the same thing: demonstrate comparable financial opportunity of investment potential. Thus, the need for a consistent, systematic methodology in performing evaluation and feasibility work is clear.

WHO SHOULD PERFORM FEASIBILITY/ EVALUATION STUDY?

The first focus is who should do the feasibility study. Some might argue that the very people who found the resource obviously know the most about it and therefore those within the exploration group should perform the early feasibility study and even the early bulk sampling or test mining. However, there is no way that the discoverer of a new mineral resource can view the outcome of developing that resource with totally unbiased beliefs, any more than a mother could sit on the jury of her son being tried. This is not to say that the exploration group should not have a large role in the early input, but, from that point on, the project team must be organized with persons of multi-disciplinary backgrounds.

Building this multi-disciplinary project team will be discussed in a later section, but for now, consider the organization that has the responsibility of performing all of the project feasibility and evaluation studies. Sometimes referred to as project development or mine evaluation and development, this organization should perform—or supervise consulting organizations performing—evaluation studies of mineral deposits and mineral processing facilities for projects discovered or acquired, wherever that project is located.

The assignment of evaluation studies of all types to a central headquarters has the following advantages:

- It ensures that all of the projects are studied and evaluated in exactly the same manner for that phase of the study and are treated uniformly and objectively.
- It ensures that all projects will have people of specific disciplines available to work on every aspect (thus each phase will be technically evaluated properly for that depth of study).
- It ensures centralized project planning and scheduling.
- It provides an experienced base group to consider the results of all candidate projects.

STAGES OF PROJECT

Because mining is a business that is constantly depleting assets, mining companies must constantly increase their mineral reserve assets through exploration or acquisition. This generally means that most successful companies will have numerous potential prospects that they are considering, either from a raw exploration point of view or through acquisitions. Thus a growing mining company might have between six and twelve active projects in its portfolio at various stages of exploration, evaluation, or development. These several projects to be evaluated

- May be for different commodities;
- Will probably involve different individuals doing the evaluations;
- Will probably start at different times;
- Will most likely have unequal mine lives; and
- May be located in different countries.

The most important element in doing complete property evaluations for a company is that each one is done exactly like every other evaluation within that company so that a decision is based on comparable economics. *Therefore, a company must put into place a system that will require different evaluators to follow the same procedures on different properties and develop equivalent feasibility studies that can be compared.*

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This is the single most important principle that must be faithfully followed by any company doing property evaluations. Likewise, it would help investment houses if all of their potential clients had projects that had equivalent feasibility studies that were more or less comparable, at least with respect to completeness. One of the primary purposes of this chapter is to instill in each reader the concept that there must be a strictly regimented method of complete property evaluations leading to feasibility reports.

WHAT MUST BE CONSIDERED FOR A PROPER FEASIBILITY STUDY?

For a properly documented property evaluation, quite simply everything must be considered. However, that does not really help much in knowing how to start and what to look for. More specifically, there must be an examination of the potential mineral operation, such as

- Determining the mineral resource (and reserve estimate, if there is one),
- Determining a mining method based on the measured and indicated resource,
- Reviewing the mineral extraction flow sheet,
- Performing a market analysis,
- Determining infrastructure needs,
- Quantifying the environmental and socioeconomic impacts and mitigation required,
- Estimating the costs of these factors, and then
- Performing an economic analysis of the assumed revenues versus the costs to determine if the project meets the company's objectives.

Objectives of Mineral Property Feasibility Study

It is often assumed that the feasibility study's objective is to demonstrate that the project is economically viable if it is developed and exploited in the manner laid out by the study. But this assumes that every mineral deposit evaluated can be profitable. Of course, this is not true; development of most of the earth's mineral deposits is not currently viable.

So what should be the objective of mineral property feasibility study? It should be to maximize the value of the property to the company by determining either to exploit it, sell it, wait for a technology or market change, or do nothing. It should also be the objective to reach that decision as early as possible, with the least amount of money spent. But how can this be done? How does a person know when they have studied each of the hundreds of items of information enough so that they have confidence in the feasibility study and the economic analysis based on that study? One learns to perform a feasibility study by a phased approach to mine evaluation. Several authors (Hustrulid and Kuchta 1995; Gentry and O'Neil 1992; Stone 1997; Taylor 1977) and, in fact, most mineral companies take a similar approach to mineral property evaluation.

INDUSTRY APPROACH TO FEASIBILITY STUDIES

On rare occasions, the activities required in a feasibility study are often described as a single, continuous process—from the time the resource is identified until a decision can be made to develop the property. This one-step approach, in which single feasibility leads directly to development, may sometimes be correct for extremely high-grade ore bodies or if the company requires development for some reason in a specific time frame. But the one-step approach is risky from a technical and an economic point of view. Such methods will usually develop an operation that is, in fact, suboptimal, even though it still may meet the company's needs. Furthermore, it may cost the company far too much money to find out that the project economics prove inadequate. Most companies and books on the subject recommend a phased approach to mineral property evaluation.

Content of Classic Three-Phased Approach

Lee (1984) describes a classic three-phased approach as follows:

> **Stage 1: Conceptual [Scoping] Study**
> A conceptual (or preliminary valuation) study represents the transformation of a project idea into a broad investment proposition, by using comparative methods of scope definition and cost estimating techniques to identify a potential investment opportunity. Capital and operating costs are usually approximate ratio estimates using historical data. It is intended primarily to highlight the principal investment aspects of a possible mining proposition. The preparation of such a study is normally the work of one or two engineers. The findings are reported as a preliminary valuation.
>
> **Stage 2: Preliminary or Prefeasibility Study**
> A preliminary study is an intermediate-level exercise, normally not suitable for an investment decision. It has the objectives of determining whether the project concept justifies a detailed analysis by a feasibility study, and whether any aspects of the project are critical to its viability and necessitate in-depth investigation through functional or support studies.
>
> A preliminary study should be viewed as an intermediate stage between a relatively inexpensive conceptual study and a relatively expensive feasibility study. Some are done by a two- or three-man team which has access to consultants in various fields; others may be multi-group efforts.
>
> **Stage 3: Feasibility Study**
> The feasibility study provides a definitive technical, environmental and commercial base for an investment decision. It uses iterative processes to optimize all critical elements of the project. It identifies the production capacity, technology, investment and production costs, sales revenues, and return on investment. Normally it defines the scope of work unequivocally, and serves as a base-line document for advancement of the project through subsequent phases.

Frequent Problems in Classic Three-Phased Approach

However, some pitfalls are associated with using the classic approach. As used by much of industry, this approach is a nonuniform, nonsystematic, nonstandarized approach to feasibility.

Conceptual/Scoping Study

A conceptual or scoping study can be extremely misleading. Nearly every exploration project that is even slightly submarginal can be shown to be worthy of further development based on casual educated guesses and optimistic, simplified,

or even biased evaluations. Back-of-the-envelope approaches to a mine feasibility study need to stay on the backs of envelopes and out of formal, official-looking reports. At its worst, this type of report can be performed by the exploration firm or project sponsor to try to sell the project to someone else. However, when an independent third party does the conceptual/scoping study, it can be employed as a useful tool for the potential investing company to determine if it wishes to proceed to the next phase of feasibility study or to calculate what the project might be worth on the open market. Also, this approach might be appropriate when looking for commodity targets for the exploration group, but not for further in-house decisions to move the project to the next level, based on the exploration group's mining and milling judgments.

This is not to say that conceptual, unclassified screening studies do not have their place in justifying other types of work, but care and caution are needed so the conceptual study is not dignified beyond its engineering basis. In fact, some countries' security exchange agencies, such as the Canadian Securities Administrators (CSA), allow and specify such a preliminary study, which they call a preliminary assessment. As identified by the CSA, such a report includes a statement that this "assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized; and states the basis for the preliminary assessment and any qualifications and assumptions made by the qualified person." It is completed without substantial engineering studies.

In a conceptual or scoping study, the accuracy of the cost estimates are most often assumed. In fact, the accuracy of all levels of feasibility studies depends on how much good engineering has been performed on the specific project. If none, then the project's related cost and economics are not likely to be accurate, and the study is likely to be misleading. Typically, at this stage, one might do 1% or 2% of the total engineering on the project. Bear in mind that, for a small project, this may amount to 1,800 to 2,400 hours of engineering work. But, for a large project, this percentage may amount to 9,000 to 18,000 hours of engineering. Then, using good engineering judgment, experience, and cost on similar projects, the accuracy of the scoping study feasibility may be in the ±45% range. Other authors claim that an accuracy of 30% is achievable for a conceptual/scoping study (White 1997). However, this accuracy will likely not be achieved unless the project is being developed in an old district where a mine or plant has recently been built and the new installation is similar to the existing one. The 30% accuracy will only be attained after 10% to 12% of the engineering has been completed.

Prefeasibility Study

The problems that have been found with many prefeasibility studies that followed conceptual or scoping studies as outlined is that often this phase simply follows the path set by the conceptual study. There is a reluctance to spend the time and money for a feasibility team to go back and justify the concepts chosen for the mining method, processing method, necessary infrastructure, waste disposal method, and overall size of the operation. Likewise, there is a reluctance to spend the time to optimize any of the functional operations at a time when the project team is small.

Another observed problem is that some of the elements or activities of the prefeasibility study will be taken too far in application, and the project's proponents will invariably proclaim to others in management and the investors that "the study is really more than a preliminary feasibility study." Although this is probably not so, it will give members of management (and possible financiers) some unjustified overconfidence in the project.

Another critical failure that often occurs in this system of feasibility progression is that the preliminary study is the chance to find the "fatal flaws" of the project, if it has any. Sometimes this does not happen or the flaws may be found in the scoping study. One definitely does not want fatal flaw discovery after a large engineering group has been assembled to work on the final feasibility study, because by this time the project's momentum is huge, and it will cost a lot to stop the project.

If the project is being undertaken by a company listed on the Canadian or U.S. stock exchange, then inferred mineral resources may not be used for mine planning purposes, except if a small zone of interburden exists between measured or indicated resource material that must also be mined.

Final Feasibility Study

When using the classic approach, by the time one gets to the final feasibility study, the project direction of each element has usually been set. For all aspects of the project to proceed at the same pace from this point, there is little opportunity to stop and examine the many interrelated operating variables that should have been examined at an earlier stage of the study. Thus it is likely that a nonoptimized design will emerge from this type of study. As a result, the mining industry is full of nonoptimized mines and plants that have been built because those optimization studies did not take place at the proper time, which in this case was during a prefeasibility study. Sometimes toward the end of a final study, the operating management realizes that certain aspects have not been optimized, and subsequently major last-minute adjustments are implemented in an attempt to mitigate these errors. Usually, such actions are based on less than the amount of engineering analysis that went into the original planning, and the accuracy of such last-minute changes and the ripple effect to all other aspects of the project (particularly the environmental and regulatory engineering) damage the credibility of the entire project.

RECOMMENDED APPROACH

Because of the problems outlined previously regarding industry abuses to the nonuniform, ill-defined, classical three-phased approach, a more rigid, uniform, engineered, and systematic three-phased approach to mineral property feasibility is recommended. In a more general way, this has been suggested by Hustrulid and Kuchta (1995) and by Gentry and O'Neil (1992), using the work of Gocht et al. (1988) and Taylor (1977). But what is considerably different as defined here than what has been suggested by others is the sheer magnitude of details enumerated by engineered task. Rigorously following the details—a description of which is contained in the iteration of each phase—makes this method unique and bankable. Nowhere else has this amount of detail of the tasks required in a mineral property feasibility study been documented and published in publicly available literature. Many of the larger mineral groups, such as BHP Billiton, Rio Tinto,

and Anglo Gold, probably have equally as well documented activity lists for each step of the feasibility studies, but they have not been publicly published.

The need for such an approach was imperative because many companies have eight to twelve mineral project feasibility studies to manage at one time. It would not be unusual for such studies to address four or five different mineral commodities located in four or five different countries, all having different starting dates and mine life, and being studied at one time by several different project teams. It is only by formalizing the feasibility study process that management can be assured that items will not be left out or that some activities would not be studied in too much depth. Although some companies may not have a dozen projects going at one time under the conditions described, the established procedure will serve any user of the system well and yield project results that are comparable for financial decisions.

This chapter examines the engineered, systematic three-phased approach to a mine feasibility study. Although not the only system available, many believe it is the safest and most prudent method. As different situations arise on different commodities, the project manager may believe that some steps can and should be omitted. However, one must also be aware of the potential consequences when taking shortcuts, particularly if the company's experience is weak in this type of new project. While looking at the details of the long lists of items that need to be studied in the different phases (described in the next section), the reader may believe there are far too many activities and the time and expense required to accomplish them is too great. Some may choose to combine many of the activities of the preliminary study with the intermediate study. This may be possible and is discussed later. However, one must be careful that this combination does not dilute the preliminary/intermediate study such that a financial decision can't be made with confidence. Some may believe that items can be eliminated or that the study of certain items is not applicable. But a great amount of caution should be used in eliminating any study aspect unless the company has so much experience and data on that particular aspect that the study is simply not necessary.

The three steps of feasibility studies recommended here are

1. Preliminary (or conceptual) feasibility,
2. Intermediate (prefeasibility) feasibility, and
3. Final feasibility.

Although these appear similar to some of the systems previously mentioned, they are not the same. Learning the content of these three studies and how to apply the work from one level of effort to the next are important parts of this chapter. What will be covered is a brief description of the activities at each level of study and how to move a project from exploration through the feasibility phase and then to engineering design—or to the back burner or for sale.

Work Breakdown Structure

Another important aspect is to apply controls to portions of the study. To do this, one must first organize a list of work categories and assign numbers to them. This is known as a work breakdown structure (WBS). No two people will develop identical WBSs; the important thing is to get the work organized so that it can be tracked—both from an accounting and scheduling point of view—and to track it on a computer.

Table 4.7-1 Typical work breakdown structure numbering system

Numbering Sequence	Project Study Level or Project Execution Phase
1XXXX	Preliminary feasibility study
2XXXX	Intermediate feasibility study (may include the test mine/bulk sample)
3XXXX	Final feasibility study (including the design basis document)
4XXXX	Engineering design (includes all preconstruction activities)
5XXXX	Construction/mine development
6XXXX	Mine/plant operations

Within each of the three levels of the feasibility study are 50 to 150 major activities. For each major activity, there are 10 to 20 elements, or work types. A large mining company trying to grow, or even holding on to its depleting asset, may have as many as eight to twelve projects going at any one time, at various levels of study. Because of the complexity of accounting for everyone's time and charging expenses to ongoing work, a numbering system to keep track is essential. In addition to the billing and accounting, a robust WBS ensures that all activities can be handled and scheduled on a computer. This is no small task because many of the activities feed information to other activities before they can begin.

All major projects use such a system, and all U.S. government projects require a WBS. As defined by the American Association of Cost Engineers, a WBS is a product-oriented family tree division of hardware, software, facilities, and other items that organizes, defines, and displays all of the work to be performed in accomplishing the project objectives. An additional advantage is that if the WBS is written in a generic way, all of the projects within a single company can follow the same structure, thus ensuring comparable completeness for any future level of study.

The WBS method outlined in Table 4.7-1 is a generic WBS that could be used on any number of mineral projects. It also uses the project phase as part of the identification. Writing a WBS for each project is possible, but the comparison between all of the projects would be more difficult and possibly less accurate.

The WBS number system carries through the six steps from the preliminary study through the project design, construction, and into operations. The WBS illustrated here is in two parts: (1) in Table 4.7-1 where the first number of each line signifies the phase or step that the project is in when the activity occurred; and (2) in Table 4.7-2 where the numbers on each line refer to the various types of chargeable activities that occur in all of the phases. Thus, for a market investigation and planning activity in the mine/plant operation phase, the WBS number would be 61300, but if the market investment and planning study occurred in the final feasibility study, the WBS number would be 31300. It can be seen that a screening project is not included, because it only officially becomes a project when it passes a screening activity. Within each project phase, a further breakdown of the numbering sequence identifies major areas of work. An example of how this might be broken down is shown in Table 4.7-2.

The feasibility study definitions of each activity serve as a checklist and, with time elements applied to each activity and subactivity, form the basis for building a project schedule. Each project will have unique characteristics that will require

Table 4.7-2 Example WBS numbering system

Work Area Numbering Sequence	Areas of Work
XX100	Preparation for reviews and management approval
XX200	Land and water status and mapping
XX300	Geology and predevelopment bulk sampling
XX400	Environmental and socioeconomic work (excluding permitting)
XX500	Geotechnical and siting studies, and planning
XX600	Agency reconnaissance, government and public relations, and permitting
XX700	Mining, including a test mine
XX800	Mineral processing and metallurgy sampling and testing (upstream)
XX900	Smelting/refining(downstream)
X1000	Surface and ancillary infrastructure facilities
X1100	Personnel
X1200	Labor planning and relations
X1300	Market investigation and planning
X1400	Financial analysis (cost estimates are within elements 0100 to 1300)
X1500	Tax studies and analysis
X1600	Planning, budgeting, project accounting, and reporting
X1700	Preparation of next step of project, of design basis report, or project closure or alternative action by company

changes to the activities listed, but the general logic and activity identifications should apply to most mineral projects to be evaluated. The more consistent the approach, the more accurate the comparison in choosing between the various mineral projects. Using this numbering system and applying time elements to each activity by number allow a schedule network to be built on the computer.

BREAKDOWN OF THE ENGINEERED, SYSTEMATIC, THREE-PHASED APPROACH

This section and Appendices 4.7A–D provide a detailed description of the activities and sequences that are recommended to properly perform a mineral property feasibility study, with the expectation that the property, if developed, will perform at the levels predicted by the feasibility study. In the appendices, some numbers in the sequence appear to be missing to allow for future additions to the WBS system.

Phase I: Preliminary Feasibility Study

Although the objective of each phase of every mineral property feasibility study should be to maximize the value of the property to the company by determining how to proceed with it, more specific to the preliminary feasibility study is to consider those logical mining and processing methods (and other project elements) in just enough detail such that one can

- Determine that they will work together to meet the company's objectives (which are usually financial); and
- Estimate the capital and operating cost, commensurate with the engineering that has been expended.

Depending on the country where the study is to be governed, the product must meet the standards of the U.S. Securities and Exchange Commission *Industry Guide 7* (SEC 2007); *The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code)*, prepared by the Joint Ore Reserves Committee (AusIMM 2004); or Canada's National Instruments 43-101 and 43-101CP.

The preliminary study is based primarily on information supplied through exploration. The company management should tell the exploration group that its report must contain the following information with appropriate maps and cross sections:

- Property location and access
- Description of surface features
- Description of regional, local, and mineral-deposit geology
- Review of exploration activities
- Tabulation of geologic resource material
- Explanation of resource calculation method, including information on geostatistics applied
- Description of the company's land and water position
- Status of ownership and royalty conditions
- History of the property
- Rock quality designation (RQD) values, at the least, and any rock mass classification work that has been done
- Results of any special studies or examinations the exploration department has performed (metallurgical tests, geotechnical work, etc.)
- Report on any special problems or confrontations with the local populace
- Any other pertinent data such as attitude of local populace toward mining, special environmental problems, availability of water and hydrologic conditions in general, and infrastructure requirements

Ideally, a number of mining and processing alternatives will be examined as a screening process. Obviously these need not be in-depth studies, but most experienced mining engineers will quickly be able to determine what mining methods will be applicable and can then place costs on several alternatives for this application. Likewise, an experienced mineral processor can determine the candidate process flow sheets and can place costs on these alternatives.

At the same time, all the other elements of the project must be considered and studied in just enough detail to discover any fatal flaws or problems that need engineering mitigation. Certainly, environmental and socioeconomic issues need to be studied and scoped to the extent that any existing or expected problems will be detected. Then all of these items can be examined for future cost and work plans.

Costs and expenditure schedules will be based on industry-factored historical experience. Major capital costs can be based on telephone quotes from suppliers or canned commercial programs built for this type of application. Usually, no field work or metallurgical testing will be conducted unless a definite metallurgical problem is recognized with the resource and suspected to be a fatal flaw, in which case it should be studied.

Depending on the complexities of the project, approximately 5,000 to 30,000 worker-hours of work is needed to complete these activities during the preliminary study. This description is written for a company or group that is prepared to perform most of the evaluation activities with various contractors. Thus for every task that is to be contracted:

1. A scope of work must be developed;
2. The industry needs to be surveyed for potential contractors;

3. Contractors must be evaluated to ensure they are qualified and likely to perform as expected;
4. A request for bid proposal documents needs to be prepared and sent out;
5. Bid documents must be evaluated and the award made;
6. Negotiations with the winning bidder may be necessary if there have been variances to its bid package, and adjustment made; and finally
7. The contractor can be mobilized.

All of these take a considerable amount of effort. If the work is to be done by experienced in-house engineers on the project team or from that function of an organization, then the contracting procedure does not apply. However, such activities as writing the scope of work should still be carried out by the central project team to make sure that potential challenges/problems are fully identified and that potential impacts are considered for all the other parts of the project.

The results of this preliminary study will be adequate for comparative screening of mining and processing alternatives, while an economic analysis will determine whether to proceed with (or reject) the project. A primary objective of the study is to plan and estimate costs for a further predevelopment program if warranted. Approximately 4% to 8% of the project engineering will need to have been completed, in which case the probable error of cost estimates accuracy should be between 35% and 45%. If 10% to 12% of the total project hours have been completed in the study, the probable error should be between 30% and 35%, while contingencies of between 20% and 35% for capital costs will apply. An economic analysis will be performed, and the preliminary feasibility report will be fully documented. At this point, presentations to management will be made and, depending on the results of the economic analysis, approval to proceed to the next step of the project (or otherwise) follows.

Major Activities of the Preliminary Feasibility Study

A description of these preliminary study activities and tasks can be found in Appendix 4.7A. This generic list applies to all mineral properties but can be adapted with addition and deletions for a particular deposit. Remember, for any of these activities that need to be contracted, the seven contracting steps listed will consume a lot of time.

Phase II: Intermediate Feasibility Study

Based on results of the preliminary study showing that a project has the potential to achieve the desired company goals, the intermediate feasibility study should be initiated. The specific object for this study is significantly different than for the preliminary study. Now that it has been shown, by using at least one mining/processing system, that the mineral resource being examined has potential economic viability, the objective must now focus on methods to optimize each component mine/plant process, while at the same time taking an in-depth look at all of the project parameters briefly studied in the preliminary study.

At this time, accurate topography maps specific to the area must be generated, if not already available. Any shortcomings in the land and water status discovered in the preliminary study must be corrected at this point before investing any more money.

Mine design will be based on information from the early exploration (delineation) drilling program plus any additional exploration sampling done between the two phases. In some cases, bulk sampling may be required. Thus, if permits can be obtained, a test mine may be justified after this phase of the study. If further exploration drilling or trenching takes place during this phase, permits and contractor agreements must be prepared. Under the control of the project team, the sampling program must

- Prepare a sample flow chart,
- Prepare a chain-of-custody security procedure (if not already in place; designed to protect the integrity of the eventual sample analysis), and
- Procure and analyze the new samples.

The new geology and mineral information must be fed into the database and evaluated. After rebuilding and analyzing the new database and documenting the current reserves and resources, new reserve and resource maps can be constructed for mine planning.

Given the shape and character of the ore reserve identified to this point, the mine planning will begin. Only measured and indicated geologic resource material may be used for mine planning in the United States and Canada. Although those mining methods considered in the preliminary study may be reexamined, other methods should also be considered, since the ore body shape, size, character, and grade may have changed. The methods described in this text on mine planning and mining methods should be followed. But this time, after a rough screening of multiple mining methods, two or three of the most probable mining methods (or variations), which are considered safe and environmentally permissible and that will probably yield the lowest cost (or greatest recovery), should be carried through the study until an economic comparison can be made. Likewise, with the latest mineralogical data and mining methods, several mineral processing and waste disposal alternatives should be considered, and those that seem likely to yield the best economics should be carried through the study until a true economic comparison can be made between the methods.

Facilities siting and geotechnical investigations will need to be conducted. If competent personnel are not available within the company, contract preparation to cover the scope of work for the approximately 125 intermediate feasibility activities must be done. The same list of contracting activities must be completed that are shown in Appendix 4.7A for Phase I, the preliminary study, and time must be scheduled for all of this contracting effort. One must not underestimate the time it can take to perform these tasks: scopes of work, requests for proposals (RFPs), survey of industry contractors, obtaining a legal contract, allowing time for bids and evaluating the bids, and negotiating the contract details with the successful contractor.

This must be done for each contractor. It may take several months to get a contractor assessed and the details of their contract accepted. But with contractors on board, work can then begin with environmental baseline studies, impact assessments, and long lead time permit applications. Again, if competent personnel are not available within the company, contract preparation to cover the scope of work for these environmental activities must be done. Bidding lists must be prepared, RFPs issued, bid evaluation criteria written, and the bid evaluation administered. Finally, when negotiated details of the contract are completed, environmental contracts are awarded. Although the baseline studies take time, they should be completed by the time the intermediate feasibility study is

done, which will allow for this information to be submitted, along with the intermediate mining and process planning, to the permitting agencies.

Results of the intermediate study will be adequate for determining economic feasibility and defining additional predevelopment and/or metallurgical testing requirements. In many cases, the benefits and requirements for a test mine or bulk sampling will be fully recognized and defined at this point. In most cases, specific permitting will be required, and this will require time to receive such permits.

The cost estimates for the (two or three) alternatives developed during this phase should be based on detailed functional analysis of the mining and processing methods of each operation, on suppliers' written quotes, and on bench-scale metallurgical testing. By the end of the intermediate study, the engineering on the project should be between 12% and 15% completed. The probable error of cost estimates should be 15% to 20%, while contingencies of between 15% and 20% will apply. Economic analysis will be performed on the favorable sets of alternatives selected. Usually, no more than three sets of alternatives will be evaluated. When the intermediate feasibility report is fully documented, presentations to management will be made and, depending on the results of the economic analysis, approval to proceed to the next step of the project (or otherwise) will follow.

Major Activities of Intermediate Feasibility Study

A description of these intermediate study activities and tasks can be found in Appendix 4.7B. This generic list applies to all mineral properties but can be adapted with addition and deletions for a particular deposit. For any of these activities that need to be contracted, the seven contracting steps listed in the preliminary feasibility discussion apply and will consume a lot of time.

Need for a Test Mine

Many times the feasibility team will not be able to obtain enough ore quality and geotechnical information working with just exploration sampling. In these cases a test mine must be considered. The test mine may come after the intermediate feasibility study or during the final feasibility study. The advantages of a test mine are as follows.

- **From a mining perspective.** Verifies the expected ore continuity, thus eliminating disastrous surprises; accurately assesses the rock strength, allowing prudent planning and sizing of the commercial mine opening; verifies mining efficiency and productivity as it relates to drilling, blasting, and materials handling; determines from reliable water studies the nature of mine water inflows, thus allowing for adequate water-handling procedures to be installed before problems are encountered; better quantifies the mine ventilation friction factors and requirements; and confirms the character of the waste product and how it will be handled in the commercial operation.
- **From a metallurgical perspective.** Verifies and optimizes the metallurgical flow sheet with a pilot-plant process that is continuous lock cycle testing; determines what size and type of equipment will be optimal for the metallurgical recovery; determines what type and amount of reagents will lead to the best recoveries and concentrate grades; determines the required amount of water and how to achieve a water balance; provides a more accurate prediction of concentrate grade, moisture content, and impurities; and provides a much better assessment of the work index from a bulk sample than from small samples.
- **From an environmental perspective.** Demonstrates the ability to control the operation in such a manner that it will not harm the environment; allows the project team to completely study the waste characterization and determine any future problems; and, if water discharge is involved, allows the project team to study the difficulty of settling the discharged water and determine what is necessary to mitigate future problems and determine if zero discharge is possible.
- **From an engineering design perspective.** Improves the ability to make more accurate cost estimates, because of better knowledge of the abrasivity of the rock and of the ground/slope control of the stopes/pit walls, which could lower the cost estimate because less contingency may be needed; improves labor estimates because of a better understanding of the productivity of each unit operation; predicts a more accurate schedule, because of better understanding of the unit productivities; and lowers the overall risk of the project in every aspect.
- **From the perspective of expediting later mine development.** Shortens the overall schedule from the end of the feasibility study to the end of construction, because of early access to develop the commercial mine, and completely utilizes the openings as part of the commercial mine operation; because access to the underground opening already exists, some shafts may be able to be raise bored and then expanded by mechanical excavation rather than by the more expensive conventional shaft sinking methods; and finally, the test mine may be an ideal training facility prior to the commercial mine start-up.

Activities related to the test mine are found in Appendix 4.7C.

Phase III: Final Feasibility Study

The final feasibility study should be initiated when results from the intermediate study show that the project still has the potential to achieve the desired company goals. The objective, as in the first two phases, is still to determine the potential value of the property to its owners—either by determining the optimum method of developing it, by selling it, or by doing nothing further at the moment. However, more specific to the final study, the objective now becomes one of minor refinements to all of the details of the intermediate study that yielded results that met the company objectives. This is designed to optimize the return on the future investment. The final feasibility study will be prepared during the permitting time schedule for the project, since final project features must be reflected in the permits to accurately assess impacts and mitigation for the agencies, commentators, stakeholders, nongovernmental organizations (NGOs), and community. Assuming that this project still shows favorable results at the end of this phase of study, the design parameters set in the final feasibility study will feed into the design basis report, which guides the project into the design and construction phase, and finally into operations.

If test mining with bulk sampling and pilot-plant testing has not been completed, it now becomes part of the final feasibility study. Mine and process facilities will be further studied, and the best alternative developed in the intermediate study will be optimized. Using the latest exploration and

metallurgical test data, probably from the test mine bulk sample, the reserves will be updated and the metallurgical flow sheet will be optimized. Final environmental impacts will be determined following prescribed guidelines. Applications for construction and operating permits will usually be made early in this phase of study (subject to later modification). Mine and process operating cost estimates will again be made by performing a functional analysis. Capital cost will be refined by again soliciting written quotes from vendors. By the end of the final study, the engineering should be 18% to 25% completed. The probable error of cost estimates should be 10% to 15% of the total cost estimate, with a contingency of 10% for most engineered structures. Other less well-defined aspects of the project (e.g., mine development) should have contingencies of at least 15%. An economic analysis will be performed, with the final feasibility report fully documented. At this stage, presentations are made to management, and, depending on the results of the economic analysis, it will give approval to proceed to the design and construction phase of project development and for the associated budget.

Major Activities of Final Feasibility Study Plan and Budget

A description of these final feasibility activities and tasks can be found in Appendix 4.7D. This generic list applies to all mineral properties but can be adapted with addition and deletions for a particular deposit. For any of these activities that need to be contracted, the seven contracting steps listed in the preliminary feasibility discussion apply and will consume a lot of time.

Combining Classic Approach with Recommended Approach

Because there are good reasons (as outlined at the beginning of this chapter) to sometimes utilize the nonuniform classic conceptual/scoping study approach, if the decision is made to move the project to the next level of study, one should convert to the engineered, systematic three-phased approach. In this case, the conceptual/scoping study should be compared with the details of the engineered, systematic, phased preliminary study. Whereas in the classic scoping study only 2% of the engineering may have been done for the first phase, in the engineered, systematic preliminary study as much as 4% to 8% of the engineering would be completed. This means that if one is now going into the intermediate feasibility study in the engineered, systematic three-phased approach, the extra work to be done in the intermediate study must be planned so that by the end of the intermediate study, all of the reasonable functional alternatives of the operation should have been examined, the optimal method selected, and 12% to 15% of the engineering work completed before going into the final feasibility study.

Most mine management people believe that the final feasibility study is the final document of the feasibility process. Unfortunately, with this in hand, most management teams will head for the bank or a joint venture partner and on to a design contractor without fully documenting everything that went into the full feasibility study. This includes not only a description of the geologic reserves, mine and plant facilities, and hardware, along with the positive economic picture, but all of the company design and operating philosophy that is so important when the build-and-operate stage is reached. The plans on how to execute the project and how to operate the completed facility are equally as important to the design contractor, the banker, and the potential joint venture partner. If all important activities through to the final operation have not been examined and then documented, the project is not ready to move forward.

PROJECT DESIGN BASIS REPORT PREPARATION

With project approval, the course of action will either be to go directly into the project execution phase or possibly seek out a joint venture partner. In either case, the design basis report (DBR, sometimes called a design basis memorandum or design basis document) needs to be prepared. The approved project feasibility report will be presented in sufficient detail to produce a DBR, which is the document that will guide the project through the next step: designing the project based on the preceding studies.

Why a DBR Is Needed

The primary purpose of the DBR is to be able to convey to future design engineers a consolidated document in which all the needed information is contained in a condensed version. But it can also serve to inform others, such as financial organizations, construction personnel, or persons who may be interested in a joint venture. Although much of the information is also contained in the final feasibility study, this document is written more for the purpose of documenting (for management) that the project is indeed both feasible and economically viable. In contrast, the DBR is written to convey all of the technical information that will be needed by the architect/engineering (A/E) design organization, which has already been developed by the owner's project feasibility team. It contains all of the drawings prepared during the final feasibility study, plus any others required to convey the needed technical information to the A/E organization for the project, and will be used as a basis for the final bids by the various A/E organizations.

In the introduction, the writer should define the purpose and use of the DBR. At a minimum, the DBR serves several purposes. The DBR

- Defines the technical basis for project design and construction so that basic, detailed engineering downstream can proceed;
- Provides the basis for a coordinated review by the organizational entities involved—the future operations group, the engineering group, management, and the future A/E team;
- Provides documentation for the technical basis and facilities description from the final feasibility study cost estimates; and
- Conveys the company's construction and procurement philosophy to the future A/E team.

The DBR is usually written in several volumes (in the case of the following example, five volumes were produced). Rather than describing in specific detail what should be written under each section and subsection of each volume, a brief description is given here concerning the general content of that volume with a generic outline of items given in Appendix 4.7E.

Volume 1: Management Summary

The management summary, prepared by the project executive or project manager as applicable, summarizes the project objectives, the assumptions that were made, the work that has been completed, the economic analysis and associated risk,

and the recommendations of the project team. Other items that should be covered (if studied by the project team) include project funding and the business plan with market and competition analysis and strategies. Any outstanding major issue involving government agencies related to utilities, transportation, land, royalties, or potential project partnerships must also be mitigated. At the end of the summary are the conclusions and recommendations, with discussions on the reserves, the feasibility of the project, the market, the schedule of the design, construction and start-up as planned in the feasibility study, any preappropriation work contemplated, and needed funding. An example outline of the information that is contained in Volume 1, Management Summary, is shown in Appendix 4.7E.

Volume 2: Project Economics

Prepared by the project executive or project manager, project economics summarizes the capital and operating costs, project schedule, market forecasts, inflation projections (if constant dollar analysis was not used), and other factors that affect the total erected cost and project economics. Identified project risks and the measures needed to mitigate those risks should be documented. An example outline of the information that is contained in Volume 2, Project Economics, is shown in Appendix 4.7E.

Volume 3: Technical Narrative

Prepared by the project team, the technical narrative describes the technical basis for the project and lists the design considerations and constraints. This is the technical meat of the project. The narrative must convey to the future A/E contractor exactly what is to be built and exactly what the A/E constructor is to do and precisely how it will accomplish that. Nothing can be left out. For this reason, all of the drawings prepared during the final feasibility stage, plus whatever drawings are necessary to convey the message to the A/E organization, must be in the DBR. The better defined the project is in the DBR, the more accurate the cost will be to the bid estimates, and the fewer exceptions that will have to be negotiated. An example outline of Volume 3, Technical Narrative, is found in Appendix 4.7E.

Volume 4: Project Execution Plan

This document, prepared by the project team, defines the real and potential problems in the detailed engineering, procurement, and construction of the project. Furthermore, it goes on to describe the best plans to ensure that these problems are mitigated or at least minimized. The recommended contracting plans are spelled out, as are the plans for engineering and design, procurement, and construction. An example content outline of Volume 4, Project Execution Plan, is in Appendix 4.7E.

Volume 5: Operating Plan

Prepared by the company's operations department, the operating plan explains how to minimize the impact for identified potential problems in start-up and continuing operations. Although much of the company's operating philosophy should already have been placed into the design as presented in the final feasibility study, the writers of the operating plan should again emphasize the company's attitude toward mechanization and automation, and what they are willing to pay for it; and its policy on safety and environmental issues and maintenance and contracting. Such issues as labor recruitment and training will be planned, scheduled, and budgeted. Learning-curve estimates will be applied toward the production buildup, so the estimated production will be on schedule and project economics will be preserved. An example outline of Volume 5, Operating Plan, is found in Appendix 4.7E.

It is this DBR document that is used as the basis for the subsequent engineering design. Not only does it contain the technical data and information decided on by the company during the final feasibility study but also the project execution plan for contracting, building, and constructing the mines of the project. It also contains the operating plan, which will guide the engineers and builder to construct the mine/plant so the operating philosophy of the company can be quickly achieved and maintained.

Historic information on the activity duration that goes into the project schedule and the functions that will be performed in the engineering design and final constructed project is included in the next section. It often appears to laypeople that building mines and plants takes much too long and costs too much, but this is not the case. What is true is that the expectations based on most of the final feasibility studies are overoptimistic, and thus the project begins badly. Data from case histories are presented in the next section showing what should be expected.

FEASIBILITY AND PROJECT TIMING, AND SCHEDULE

The time it takes between the discovery of a resource that may be a potential ore body and when the ore body is brought into production can vary significantly. Obviously, with an extremely high-grade ore body, it may take significantly less time to identify enough ore to start mining. Likewise, if money can be made no matter how you mine it, then the company may not want to spend a lot of time optimizing the mining and milling methods (though this could and has proved to be a mistake in the past). On the other hand, it may take many years to define large, marginal mineral resources and to optimize every aspect of the study in order to turn the resource into a viable reserve.

Technological changes over time may also allow the property to be developed after many years of study. The other factor is the ever-changing environmental permitting, which can vary significantly: Although a small punch coal mine in Appalachia can take less than a year, the development of a world-class zinc/copper mineral resource in Wisconsin (United States) was stopped for 20-plus years and finally terminated, even though it can be demonstrated that the underground mine can be built and operated in a manner that would be environmentally acceptable anywhere else. Overall it usually takes from 2 to 6 years just to complete the mineral property feasibility evaluation study.

The overall time from find-to-mine is logically divided between the classical phases of mineral development:

- Preliminary exploration and discovery
- Land and water acquisition commitment
- Exploration
- Feasibility studies/environmental permitting
- Final engineering
- Development and construction
- Start-up to full production

All of these activities vary greatly in length. In this chapter, only the length of the feasibility studies and environmental permitting will be considered.

Table 4.7-3 Estimates for average duration of project evaluations

Project Evaluation Phase	Time Duration
Preliminary	7.5 months (156 working days)
Intermediate	2 years, 8 months (666 working days)
Final	2 years, 10 months (709 working days)
Total	6 years, 1.5 months (1,531 working days)

Nelson & Associates (Nelson 1979) conducted a study for the U.S. Bureau of Mines on the duration of these phases of mineral development for four mining projects in Wisconsin and Minnesota (United States). (Three were operating and one was being considered.) Because these states have strict systems of environmental permitting for new mine development, it is not surprising that the Nelson study found that permitting time for a metal mine in Wisconsin was, by hindsight, more than 100% optimistic. For example, the following information comes from Nelson's Summary of Time table:

- Environmental monitoring: 2.18 years
- Environmental impact report evaluation: 4.85 years
- State permits: 5.25 years
- Local permits: 3.25 years
- Environmental impact report preparation: 1.55 years
- Wisconsin Department of Natural Resources (WDNR) for the Wisconsin Environmental Protection Agency: 3.90 years
- Federal environmental impact statement (EIS): 3.05 years
- Master hearings: 1.16 years
- Other WDNR permits: 3.05 years

In reality, some of these activities can go on simultaneously or overlap. But even the most optimistic schedule to receive permits in Wisconsin for a metal mine was 10 to 12 years.

In addition, in the same report, Nelson also developed a time estimate, based on these four major projects, for feasibility studies as 8 years, which seems too long. However, this information came from four major projects done by four different major mining companies. Although it certainly can take 8 years to do a complete mineral property feasibility study for many projects, it is not necessarily true for all of them. Table 4.7-3 is taken from one mineral company's estimates of the average time expected to complete the feasibility/evaluation on 10 (small to large) projects.

However, this schedule overlaps all activities possible on maintaining a high level of engineering standards (the approximate 300 total activities listed previously, completed mostly by contractors). Depending on the size, grade, location, ownership of the project, and how much financing the owner needs, these times can change radically. The total worker-hours to complete a feasibility study from the conceptual study through the final feasibility study will vary from 50,000 worker-hours to more than 100,000 worker-hours. By the time the final feasibility study is finished, 18% to 25% of the total engineering for the project will have been completed.

In contrast to these long time estimates, Cusworth, (1993) presented the estimates for Australia as typical durations for feasibility studies:

- Scoping study: 7 to 9 months
- Prefeasibility study: 9 to 13 months
- Feasibility study: 12 to 17 months

Therefore, it could be concluded from Cusworth that all projects in Australia vary only from a total of 28 to 39 months. Unfortunately, no details are given as to what was actually covered during these periods. It would have to be assumed that much of the difference between the United States and Australia is the U.S. environmental agencies' red tape. But two other factors may play a significant role: (1) Probably more virgin deposits were being discovered in 1993 in Australia than in the United States, which might tend to be of a higher grade; and (2) Australians tend to turn everything over to contractors, which, with their larger staffs, can usually perform faster than their U.S. counterparts.

Scheduling of each project element must be done from the beginning. This is one of the important reasons to document in advance all of the activities of each feasibility level phase. Then estimated worker-hours of labor time must be assigned to each of these hundreds of project activities and subactivities. Setting up and maintaining the schedule of even a medium-sized project is a major task involving thousands of activities.

ORGANIZING THE PROJECT TEAM

There are many ways to organize a project team, depending on

- Phase of the feasibility study,
- Size and complexity of the project,
- Location of the project, and
- Size and experience levels within the parent company.

First, the talent that is needed either part-time or full-time on a project feasibility team must be considered. Certainly, the team needs people who understand and can perform project management, as well as costing and scheduling for the project. In addition, every technical discipline that has been considered in the evaluation must be populated. This includes the fields of geology, geostatistics, mining, metallurgy, environmental consideration, hydrology, geomechanics, civil infrastructure, and economic evaluation. But there must also be people who can provide and evaluate legal, land, water, public relations, socioeconomic, marketing, tax, and financial information. Depending on the size of the parent company, it must either build the organization within the company structure or depend on the consulting industry to supply the needed talent.

Taking the in-house approach, the company must form a project management and development organization whose staff will be assigned to the project management nucleus of each project. In addition, technical specialists are assigned from a technical support organization on an as-needed basis to perform the multitude of technical activities that will be required. By approaching the problem in this fashion, and using proper labor scheduling, many projects can be handled simultaneously. This approach works well on small- to medium-sized projects up through the intermediate feasibility phase of study; for large or mega-projects, it would probably only work in the preliminary phase.

Taking the consultant approach, the company should still form a project management organization to manage each project but then contract to either one large multidisciplined consultant organization or individual discipline consultants to perform the various technical tasks of each project. The consultant approach will not be discussed in detail, since the A/E consultant normally supplies all the organization's needs.

In-House Project Teams

Two basic management philosophies must be considered: functional matrix or line/staff organizations. This operating philosophy goes beyond the consideration of managing projects but is the means by which all supplied labor report to the company's various functional departments. There will be a considerable difference in how the project teams are made up, depending on whether the company operates as a line/staff organization, a functional/matrix organization, or a hybrid of the two.

It is difficult to generalize, but if the company is running several small- to medium-sized projects, which are in the preliminary or even the intermediate feasibility study phase, it is more cost-effective to organize a core group—consisting of the project manager and a project cost and schedules coordinator—and temporarily assign experts from the matrix technical organization rather than organize several line/staff organizations for each project. By allocating work in this manner, each discipline can usually handle several projects at one time with proper scheduling. It can usually work well through the intermediate level of feasibility, particularly if the projects are located in the same country as the home office mine evaluation and development group. However, when the project is overseas or if it is a large project, the amount of field work required during the intermediate and final studies mandates that it is usually best to move the dedicated project team to a location near the site.

At such time as the magnitude and importance of a project justifies it, a separate project team organization is established. This would normally occur at the end of either the preliminary or intermediate study phase when the cost, duration, level of staffing, or overall importance indicates that a separate project team should be created. At this point, the team would be transferred to a location close to the site of that project but still functioning under the project development group. It is also worth noting that in some countries there is little or no technical mineral engineering base on which to draw for a staff and one must be literally imported.

At the conclusion of the preliminary or intermediate study phase and when it is determined that a separate project organization should be established, an independent project team is organized and works through the local organization as determined by management and coordination with the affiliate or country manager. In such cases, the project may have a project executive who also serves on the local country management committee or staff.

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APPENDIX 4.7A
PHASE I: PRELIMINARY FEASIBILITY STUDY COMPLETE ACTIVITY DEFINITIONS

Activity No. (from WBS) **Activity Title and Description**

10100 **Public Affairs**

- Do in-house determination of public affairs' perceived responsibility and identify company official to serve as spokesperson for the project.
- Inform state government officials if the mineral resource is a new major discovery, prior to official public announcement of the discovery. Then prepare and disseminate initial press release announcing discovery.
- Identify statewide media editors and set up proper liaison and method of briefings.
- Identify concerns of local, regional, and state population, and prepare proactive response demonstrating how each concern will be mitigated. As more data are received, update these proactive responses to the public.

10101 **Review Exploration Report**
Review report prepared at end of exploration phase. Report should contain information on
- Mineral deposit,
- Property location and access,
- Area surface features,
- Exploration activities completed and planned,
- Geology (regional, local, and deposit),
- Potential ore reserves,
- Company's land and water position,
- Property ownership and royalties,
- Property history,
- Special studies performed or environmental problems noted, and
- General data.

Review should include trip to project site to familiarize team members with site and area.

10102 **Prepare Preliminary Study Plan and Budget**
Prepare preliminary feasibility study schedule, with labor and cost budgets necessary to complete preliminary study. Prepare schedule to show activities and time for remainder of project phases:
- Intermediate and final studies
- Design, construct, and develop (through start of production)

10103 **Present Preliminary Study Plan and Budget to Management**
Present schedule, plan, and budget for review.

10104 **Obtain Approval of Preliminary Plan and Budget**
Obtain approval from appropriate levels of management to proceed with preliminary feasibility study outlined in schedule, plan, and budget.

10201 **Review Land and Water Status**
Review land ownership and water rights, control, royalty, and lease situation developed during exploration phase. Project team members should review land status with personnel in the company's land office or its land agent.

10301 **Assemble and Edit Drill-Hole Data**
Assemble drill-hole data pertaining to deposit. Audit data for correctness and completeness.

10302 **Check and Approve Preliminary Reserves**
Check and modify or approve preliminary reserves calculated by exploration group.

10303 **Document and Review Geology and Reserve Data**
Write report documenting drill-hole and reserve data. Report should contain appropriate tables, maps, sections, and written information concerning mineral inventory and reserve data, regional and local geology, and other pertinent information. Review assembled information with appropriate levels of management. Write report in style and format suitable as chapter in preliminary feasibility study report.

10304 **Prepare Intermediate Predevelopment Plan and Budget**
Develop scope of work, schedule, and budget for predevelopment drilling program for next phases of work.

10401 **Develop Environmental Work Specifications**
Define scope of work for an environmental overview of project and surrounding area.

10402 **Develop Environmental Overview**
Develop general environmental plan for protecting quality of water, land, ecology, cultural resources, and socioeconomics of project area during construction and operation. Determine costs, if applicable, to prevent or mitigate environmental damages and return area to near original condition at project end. Costs should have accuracy of +30%.

10403 **Document and Review Environmental Results**
Write report documenting environmental overview. Review results of study with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility report.

10501 **Develop Geotechnical Work Specifications**
Develop scope of work necessary for siting, soils mechanics, rock mechanics, and hydrology studies.

10502 **Perform Geotechnical Overview**
Examine drill cores or send cores for testing, if necessary, to determine unusual characteristics that may impact mining costs. Evaluate potential problems and associated costs. Perform field reconnaissance, with appropriate lab and field tests if necessary, to determine soils and surface hydrology conditions in and around potential mine, mill, tailings, and surface facility sites. Evaluate potential problems and associated costs.

10503 **Establish Tentative Siting Preferences**
Select tentative mine, mill, tailings disposal, and surface facilities sites based on preliminary evaluation of costs, soils mechanics, surface hydrology, and general environmental conditions. Costs should have accuracy of +30%.

10504 **Document and Review Geotechnical Results**
Write report documenting geotechnical and siting overview. Review results with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

10601 **Develop Permit Application Specifications**
Develop scope of work necessary to determine governmental agencies involved and permits required for every stage of project through design/construct and into operation.

10602 **Conduct Preliminary Permit and Agency Overview**
Conduct literature search and telephone conversations to determine permits required to develop, construct, and operate project. Determine local, state, and federal agencies involved. Evaluate the time and cost of permits and bonds needed. These costs should have an accuracy of ±30%.

10603 **Document and Review Permit and Agency Results**
Write report documenting results of permit and agency overview study. Review with appropriate levels of

management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

10701 **Develop Mine Work Specifications**
Develop scope of work necessary for conducting mining evaluation studies.

10702 **Conduct Mining Literature Search**
Review available literature involving mining methods and schemes for deposits of similar nature. This activity could involve visiting similar operations to gather data pertaining to mining methods, equipment, personnel, and costs.

10703 **Identify Possible Mining Methods**
Through literature search, personal knowledge, mine visitations, and discussions with other people, identify technically feasible mining methods applicable to this type of deposit.

10704 **Develop Tentative Layout for Each Mining Method**
Lay out preliminary mine plan for each technically feasible mining method considered.

10705 **Evaluate Alternative Mining Methods**
Perform rough capital and operating cost calculations for each technically feasible method selected. Evaluate several production rates. Perform quick discounted cash-flow analysis and rank methods in order of economic preference. Eliminate alternatives with little or no chance of economic success. Determine production rates that will satisfy market conditions and give best economic rate of return.

10706 **Perform Mine Preliminary Functional Analysis**
Evaluate operational cycles and requirements for labor, equipment, and supply for each mining function and for each alternative selected in Activity 10705 based on mine engineer's experience. The functions include drilling, blasting, loading, hauling, scaling, bolting, ground control, hoisting, primary crushing (if underground), maintenance, supply/debris handling, pumping, and other support services. Prepare cost and operational cycles for each function.

10707 **Develop Mine Capital and Operating Cost Estimates**
Estimate capital requirements necessary to bring mine on stream. Estimate operating costs required to produce ore. Estimate costs for two to four mining methods and production rates selected for study. Costs should have accuracy of ±35% to 40% (depending on method of functional analysis and geologic definition). List costs in format for financial analysis.

10708 **Document and Review Mine Results**
Write report documenting mine study work performed. Review results with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

10801 **Develop Mill Work Specifications**
Develop scope of work necessary for conducting milling evaluation studies.

10802 **Determine Tentative Mill Process Requirements**
Use literature search, company personnel experience and discussions with others to determine feasible process methods. Determine processing requirements for each alternative. Activity may require input from bench tests performed during exploration phase or during preliminary study phase.

10803 **Prepare Mill Flow Sheet**
Prepare flow sheet for each process alternative. Flow sheet should contain sufficient detail to allow selection and sizing of equipment. Perform capital and operating cost calculations for each technically feasible method. Perform quick discounted cash-flow analysis and rank methods in order of economic importance. Eliminate alternatives with little or no chance of economic success.

10804 **Perform Preliminary Mill Functional Analysis**
Evaluate operational cycles and requirements for labor, equipment, and supply for each milling function and for each alternative selected in Activity 10803 based on the metallurgical engineer's experience. The functions include stockpiling/reclaiming, crushing, grinding, screening, concentrating, classifying, clarifying, tailings/waste disposal, concentrate handling at the mill site, maintenance, supply/debris handling, and other support services. Prepare cost and operational schedules for each.

10805 **Evaluate Custom Milling Alternatives**
Investigate opportunities for selling run-of-mine (ROM) material. Determine sales price and charges associated with selling ROM (if a reasonable alternative). Investigate opportunities for tolling ROM material. Determine custom mill capacity, timing, and costs. Costs should include capital and operating expenses for everything associated with tolling. These include transportation of product to custom mill, losses/deducts for processing, tolling charge, sampling methods, transportation of concentrate, and personnel requirements.

10806 **Develop Mill Capital and Operating Cost Estimates**
Determine capital and operating cost estimates for all milling operation alternatives and different production rates. Costs should have accuracy of ±30%. Put costs in format suitable for financial analysis.

10807 **Document and Review Mill Results**
Write report documenting mill study work performed. Review results with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

10901 **Develop Smelter/Refinery Work Specifications**
Determine general requirements for smelting and/or refining mill concentrates. Define and document the scope of work for the custom refining study.

10902 **Evaluate Custom Smelter/Refining Alternatives**
Do preliminary investigation of opportunities for custom smelting and/or refining project concentrate. Determine custom refining capacity, timing, and costs.

Costs should include capital and operating estimates for everything associated with custom refining. These include transportation of concentrate to refinery, transportation losses (where applicable), smelting/refining charge (consider deducts and/or credits), transportation of refined product, and personnel requirements. Costs should have accuracy of ±25%.

10903 **Document and Review Refinery Results**
Write report documenting refinery work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11001 **Develop Surface Facilities Work Specifications**
Develop scope of work and schedule necessary for study of project surface facilities not connected with mine and mill studies.

11002 **Determine Tentative Surface Building Requirements**
Make preliminary estimate of surface buildings required for project operation. Include buildings that serve support function for mine and mill but not buildings directly related to mining and milling activities. Types of buildings include ambulance garage, administration/office, dry/changehouse, guardhouse, security fence, surface shops, and port facilities (if applicable). Include concentrate storage and loading facilities capital and operating costs at shipping docks. These can run into millions of dollars if you have to supply them.

Determine operating and maintenance personnel and equipment requirements (includes shop, office, and dry equipment).

11003 **Determine Tentative Surface Utilities Requirements**
Make preliminary estimate of utilities required for project construction, development, and operation. Study does not include power distribution within facilities included in mining and milling studies. Utilities include electric power (including internal switching and transformers), fuel for buildings and fuel storage for operating equipment, communications (radio, voice/data telephone system, and GPS), potable water, water for dust control, water and system for fire protection, sewage system, and garbage/trash/solid waste removal and disposal area and system.

Determine operating and maintenance personnel and equipment requirements.

11004 **Determine Tentative Surface Transportation Requirements**
Make preliminary estimate of transportation needs for moving equipment, supplies, material, and mine/mill product into and out of project area during project construction, development, and operation. Study should include alternative transportation method such as truck, rail, ship/barge (if feasible), and air haulage (if feasible), combinations of above, and personnel transportation.

Study should determine access road, personnel, and equipment requirements.

11005 **Determine General Surface Facilities Arrangement**
Make preliminary estimate of requirements for
- Internal road for surface facility, plant, and disposal area(s) (does not include haulage roads for open-pit mine but does include equipment for maintaining auxiliary roads);
- Parking areas;
- Construction laydown area(s); and
- Storage area(s).

Prepare preliminary plot plan showing arrangement of all surface facilities, including the listed items, water facilities, mine, mill, and tailings facilities. Determine maintenance material and supply requirements for these areas.

11006 **Determine Surface Mobile and Miscellaneous Equipment Requirements**
Make preliminary estimate of equipment requirements not covered under other activities. This includes equipment for ambulance(s); road and yard area maintenance; supervisor pickups/car(s); maintenance personnel pickups and trucks; loader(s)/backhoe(s); forklift(s); crane(s)/cherry picker(s); portable welder(s), compressor(s), generator(s), and light set(s); small rear-dump truck(s); and crawler tractor(s) with dozer. Determine operating and maintenance personnel and equipment requirements.

11007 **Develop Surface and Ancillary Facilities Capital and Operating Cost**
Determine capital requirements necessary to build surface and ancillary facilities. Determine operating costs associated with surface facilities, including personnel, supplies, office, and safety equipment. List the costs in spreadsheet format suitable for financial analysis. Costs should have accuracy of ±30%.

11008 **Document and Review Surface Facilities Results**
Write report documenting results of surface facilities studies. Review results with appropriate levels of management and other personnel. Write report in format and style suitable as chapter in preliminary feasibility study report.

11101 **Determine General Personnel Requirements**
Determine approximate administrative and management personnel requirements, and operating, maintenance, support, and supervisory personnel requirements developed in preceding activities. Split requirements into salaried exempt, salaried nonexempt, and hourly classifications. Develop labor buildup schedules for each classification.

11102 **Determine Approximate Administrative Costs**
Determine salaries and wages of personnel identified in Activity 11101. Determine payroll burden associated with salaries and wages. Determine cost, type, and quantity of office equipment and supplies required for all offices, including administration, mine, mill, maintenance, and others. Prepare costs in form suitable for financial analysis. Costs should have accuracy of ±20%. Include costs for relocation and salaried personnel.

11103 **Document and Review Organization and Administration Results**
Write report documenting administrative costs and personnel requirements. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11201 **Assess Miscellaneous Labor-Related Cost Factors**
Assess impact and cost of factors affecting labor recruiting, hiring, and retention. Factors include incentive system, labor setting, recruiting, training, retention, performance, and cost. Factors may also include employee housing and company-supplied transportation. List costs in spread format suitable for financial analysis. Costs should have accuracy of ±20%.

11202 **Document and Review Labor-Related Results**
Write report documenting labor-related studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11301 **Conduct Preliminary Market Studies**
Perform market studies to determine selling price of salable products and probable product sales potential. Estimate price ranges for life of project (in terms of constant dollar, not inflation). If changes in product sales potential are identified for the future, they should be included as sensitivities. Prepare expected sales of products in format suitable for financial analysis.

11302 **Document and Review Marketing Results**
Write report documenting marketing studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11401 **Develop Tax Overview Specifications**
Develop scope of work and schedule necessary for conducting study of taxes applicable to project.

11402 **Conduct Tax Overview Study**
Make preliminary study of taxes applicable to project and their cost and impact on construction, development, and operation of project. Prepare tax rates in form suitable for financial analysis.

11403 **Document and Review Tax Study Results**
Write report documenting tax studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11401 **Develop Preliminary Financial Analysis Criteria**
Develop criteria for performing preliminary financial analysis. Criteria should include overall project schedule (includes intermediate and final evaluation, design, construction, development, and startup); mine and product estimated production; capital and operating costs estimates; royalties; escalation factors (only if this is not a constant dollar analysis); corporate overhead allocation; working capital; property acquisition costs; mill recovery; depreciation methods; depletion allowance; tax rates; weighted contingency for unforeseen factors if not included on every capital cost item; capitalization factor; salvage values; working capital; sensitivity analysis; and project alternative comparisons.

11402 **Conduct Financial Analysis**
Conduct financial analysis for total project using Apex or other suitable computer program. Print results of economic analysis.

11403 **Conduct Financial Sensitivity Studies**
Evaluate risk sensitivity of project to key factors such as operating costs, capital costs, reserves, grade, production sales, mill recovery, royalties, taxes, and other items with high degree of uncertainty. Conduct sensitivity analysis using computer program that will perform Monte Carlo simulation, which will assess combined risk sensitivities. Print results of the sensitivity analysis.

11404 **Document and Review Financial Results**
Write report documenting financial analysis and sensitivities. Review results of work with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in preliminary feasibility study report.

11601 **Prepare Intermediate Study Plan and Budget**
Update intermediate feasibility study plan and schedule using formalized scheduling techniques. Update budget for intermediate study, including any additional exploration or sampling needed. Schedule and budget should conform to the activities outlined in Activity 10304 and used in financial analysis (11401).

11602 **Prepare Preliminary Report**
Prepare formal report detailing preliminary study work performed. Prepare report in style and format that is suitable for presentation to management, suitable for use by other project teams, and containing history and results of work performed.

11603 **Present Preliminary Report Plan and Budget to Management**
Present all data generated during preliminary study, plus plan and budget for intermediate study, to management for review. Present information in meeting(s) with hard copies of reports, schedules, and data. Distribute report at least 1 week prior to meeting to allow personnel time to read and review.

APPENDIX 4.7B
PHASE II: INTERMEDIATE FEASIBILITY STUDY COMPLETE ACTIVITY DEFINITIONS

Activity No. (from WBS) **Activity Title and Description**

20100 **Public Affairs**
- As soon as company management approves the intermediate feasibility study plan and budget, notify state government officials that the project will proceed to the next level of examination.
- Brief the media on the decision to proceed to the next level, making sure the media understand that the next level of decision making is many months away and that the final decision of whether or not to build a mine is probably years away.

- Set up in-house means to directly respond to questions from the public by disseminating information as it becomes available and presenting speeches at professional, civic, and labor organization meetings. This may also be done by mineral industry audiovisual presentations tied in with the local settings and issues.
- Conduct meetings to help determine the needs of the area, which will promote sustainable development and lead to a social license to operate.

20101 **Get Management Approval of Intermediate Study Plan and Budget**
Get approval from appropriate levels of management to proceed with intermediate feasibility study outlined in plan, schedule, and budget submitted to management for review at end of preliminary study.

20201 **Review Land and Water Status**
Review land and water ownership, control, royalty, and lease situation developed during exploration phase and updated during preliminary study (if any work done). Project team should review land status and water with personnel in company land office and other appropriate individuals with respect to site locations identified during preliminary study. All related water rights should be determined.

20202 **Develop Topographic Map Work Specifications**
Develop scope of work and schedule necessary for topographic mapping. Prepare request for proposal (RFP) to send to contractors capable of performing the work. RFP should include draft of proposed contract.

20205 **Prepare Topographic Maps**
Notify contractor to proceed with work required by contract. Obtain prepared maps and other data from contractor.

20206 **Negotiate Land and Water Acquisition Requirements**
The company land office should start negotiating options on land and water requirements identified in the preliminary study. Requirements should include alternatives, because generally the final mine, plant, tailings sites, and so forth are not yet identified at this stage of project evaluation. This probably means optioning some land not needed for final selected sites.

20207 **Update Land and Water Acquisition Requirements**
Land and water requirements should be updated as intermediate study progresses and project team learns more about requirements. This may allow land personnel to drop negotiations on certain pieces not needed for project. This updating will not normally select final land sites required for project development, construction, and operation.

20208 **Determine Land and Water Acquisition Costs**
Land personnel should determine approximate costs of buying and/or leasing land and acquiring water necessary to construct, develop, and operate project. Costs will become part of intermediate study financial analysis, so accuracy should be within ±15% to 20%. Put costs in format suitable for financial analysis.

20209 **Document and Review Land and Water Situation**
Write progress report documenting results and costs of land and water negotiations to date. Review results of negotiations with appropriate levels of management and other personnel. Prepare risk analysis and mitigations for land and water satisfactory procurement. Write report in style and format suitable as chapter in intermediate study feasibility report.

20301 **Develop Predevelopment Drilling Work Specifications**
Evaluate exploration report and data. Develop scope of work and schedule necessary to perform predevelopment drilling activities. This activity assumes no exploration drilling was performed during or after the preliminary study and additional drilling is required to bring the reserve accuracy to range to within ±15% to 20%.

20302 **Prepare Predevelopment Drilling Permit Applications**
Gather data required to prepare applications for permits to do predevelopment drilling field work. Prepare applications.

20303 **Procure Predevelopment Drilling Permits**
Submit completed applications for permit to appropriate governmental agencies. Wait for agencies to approve applications. Obtain approved permits.

20306 **Perform Predevelopment Exploration Drilling and Other Field/Lab Work**
Notify contractor(s) to proceed with work required by contract(s). Do other geologic field work as required. What is needed is sufficient drilling and field work to get reserve estimates within ±15%–20% accuracy range. Assay drill-hole samples and/or log drill holes and obtain rock quality designations (RQDs).

20307 **Prepare Test Drill Sample Flow Chart and Chain of Custody**
Prepare flow chart (listing) of drill cores required for various tests (e.g., metallurgy, rock mechanics, porosity, permeability, density, and moisture). Chart should identify size and amount of cores required, location of procured cores (area of deposit), purpose of cores, place cores sent for testing, types of tests, method of assay, number of duplicate tests, number of blind tests, number of tests on standards, and core storage instructions. Establish a mandatory chain-of-custody protocol with proper check points and sign-offs.

20308 **Develop Mineral Inventory Work Specifications**
Develop scope of work and schedule necessary to determine deposit mineral inventory. Determine data requirements for computerized and/or hand-calculated mineral inventory system. In addition to mineral and geologic data, RQD must be developed. Inform data gatherers of required data format.

20309 **Procure Other Drill Samples**
Other core samples are necessary for porosity, permeability, Cerchar abrasivity, density, and moisture

determinations. *Other* means samples needed for tests other than metallurgy and rock mechanics.

20310 **Analyze Other Drill Samples**
Send other core samples to appropriate lab or testing facility(ies). Analyze and test core samples for porosity, permeability, density, and moisture. Send test data to project team and other interested parties.

20311 **Assemble and Edit Drill-Hole Data**
Assemble drill-hole data pertaining to deposit. Edit data for correctness and completeness.

20312 **Build Drill-Hole Computer File**
Prepare mineral inventory data for entry in computer system. Build drill-hole files in computer. Types of data to include: identification; geologic parameters; RQD values for each interval of the mining horizon and 20 ft above and below; collar coordinates and elevation; assay values and intervals; hole depth, dip, and direction; and date hole completed.

20313 **Prepare Geologic Maps**
Prepare necessary drill-hole maps and cross sections, with computer and/or by hand, to help evaluate the mineral deposit.

20314 **Delineate Mineral Zones**
Identify and delineate mineralized zones. Delineate by computer from drill-hole files and/or by hand.

20315 **Compute Potential Mineral Reserves**
Build computer block model and compute mineral reserves with various cutoff grades, mining heights, waste thicknesses, and so forth. Calculate reserves by hand if computer block model is not developed. Prepare a risk analysis and mitigation plan for the mineral reserve tabulated.

20316 **Document and Review Mineral Inventory Results**
Write report documenting results of Phase I predevelopment program. Report should contain data on geology, field work, and reserves. Review results with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report.

20401 **Develop Environmental Work Specifications**
Develop scope of work and schedule necessary for environmental baseline studies, environmental impact analyses, and environmental control plans required for project analysis and costs. Work will serve as database for project permit applications.

20404 **Perform Air Quality Baseline Study**
Conduct literature search to determine amount and value of air quality and meteorological data available in project area(s). Establish project site monitoring stations to gather air quality, including radiological (if required) and meteorological baseline data. Gather data over required period of time.

20405 **Perform Water Quality Baseline Study**
Conduct literature search to determine amount and value of surface and groundwater quality data in project area(s). Gather and analyze samples of surface water in the area(s) over required period of time, generally on performing hydrology studies (Activity 20508).

20406 **Perform Ecological Baseline Study**
Conduct literature search to determine amount and value of ecological data available in project area(s). Gather and/or study samples of life forms (flora and fauna) in area(s) over required period of time.

20407 **Perform Socioeconomic Baseline Study**
Perform literature search to determine amount and quality of labor and housing available in general area. Conduct general area survey to verify and/or update statistics on amount and quality of labor and housing and other socioeconomic data. Study the sustained development options for the affected area. Consider the need for a social license to operate in the area affected.

20408 **Conduct Archeological Investigation**
Conduct project area search for cultural resources or items of historical significance. Concentrate search in areas of potential land disturbance caused by mine, plant, tailings, and surface facilities construction and development.

20409 **Assess Environmental Impacts**
Assess impact to baseline environment caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities.

20410 **Develop Air Quality Plan and Costs**
Develop plan, with one or more alternatives, to maintain air quality in and around project area. Evaluate effects caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans and put costs in format suitable for financial analysis. Cost accuracy should equal ±15%.

20411 **Develop Water Quality Plan and Costs**
Develop plan, with one or more alternatives, to maintain surface water quality in and around project area. Evaluate effects caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans and put costs in format suitable for financial analysis. Cost accuracy should equal ±15%.

20412 **Develop Ecological Plan and Costs**
Develop plan, with one or more alternatives, to maintain quality of ecology in and around project area. Evaluate effects caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans and put costs in format suitable for financial analysis. Cost accuracy should equal ±15%.

20413 **Develop Reclamation Plan and Costs**
Develop plan, with one or more alternatives, to reclaim land disturbed by effects of construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plan and put costs in format suitable for financial analysis. Cost accuracy should equal ±15%.

20414 **Develop Socioeconomic Plan and Costs**
Develop plan, with one or more alternatives, to estimate requirements for community development. Plan

may include requirements for employee housing; medical and dental facilities; schools; community utilities (power, water, sewer, etc.); community services; skills and occupational training other than for the operation; recreational activities; and potential sustained development infrastructure. Estimate capital and operating costs to implement the plans and put costs in format suitable for financial analysis. Cost accuracy should equal ±15%.

20415 **Document and Review Environmental/ Socioeconomic Study Results**
Write report documenting environmental and socioeconomic work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractors should submit report in draft form for project team review before finalizing. Prepare risk analysis and mitigation plan for environmental and socioeconomic concerns.

20501 **Develop Siting and Geotechnical Work Specifications**
Develop scope of work and schedule necessary for siting, rock mechanics, soils mechanics and foundation, and hydrology studies.

20504 **Prepare Hydrology Permit Applications**
Prepare necessary permit application(s) to perform hydrology studies of groundwater and surface water quantities and qualities. Permits are needed primarily to drill test wells.

20505 **Conduct Siting Studies**
Conduct studies to determine suitable locations of all surface facilities for the mine, plant, roads, tailings disposal, and surface facilities. Identify several sites.

20506 **Procure Hydrology Permits**
Submit permit application(s) to appropriate governmental agency(ies). Wait for permit approval. Get approved permit(s).

20507 **Prepare Soils Mechanics Permit Applications**
Prepare necessary permit application(s) to conduct soils and foundation investigations. Permit(s) are likely needed for test pits and test borings.

20508 **Conduct Hydrology Studies**
Conduct studies to collect groundwater quantity and quality data. This usually requires drilling and pump-testing wells to determine amount and quality of water expected during mining and amount and quality of water required for makeup. Conduct studies to determine surface water flow patterns and amounts to expect during possible maximum flood and 100-year-flood periods. Send study data to project team in report form. Contractor should submit report in draft form for review by project team before finalizing.

20509 **Procure Rock Mechanics Samples**
Procure drill-core samples to use for rock mechanics tests. Drilling is usually performed as part of predevelopment drilling program.

20510 **Procure Soils Mechanics Permits**
Submit permit application(s) to appropriate governmental agency(ies). Wait for permit approval. Get approved permit(s).

20511 **Procure Soils Mechanics Samples**
Procure soils samples to use for soils mechanics tests and foundation analyses. Procure samples in areas preferred for plant and surface facilities construction and tailings disposal. Samples usually consist of test borings and test pits.

20512 **Conduct Rock Mechanics Tests**
Ship rock mechanics samples to testing laboratory. Conduct appropriate tests to determine strength of rock. Analyze test results to determine size of openings and pillars in underground mine or pit slope angles (stability) in open-pit mine. Send test results in report form to project team. If rock appears to be applicable to mechanical excavation methods, send rock samples for Cerchar abrasivity tests. Develop rock mass rating designation and/or Barton's Q designation. Contractor should submit report in draft form for review by project team before finalizing.

20513 **Conduct Soils Mechanics Tests**
Ship soils mechanics samples to testing laboratory. Conduct appropriate tests to determine physical and chemical properties affecting building foundations and tailings disposal areas. For open-pit mines, tests are needed to determine slope stability. Conduct appropriate field tests to determine above-soils properties and water flow characteristics. Analyze test results. Field tests could include location of suitable construction materials. Send test results in report form to project team. Contractor should submit report in draft form for review by project team before finalizing.

20514 **Establish Siting Preferences**
Rank sites selected in Activity 20505 in order of preference. Consider factors such as relationship to existing facilities, capital and operating costs, environment, land position, topography, accessibility, capacity, surface water flow patterns, soils mechanics data, and relationship to mineral deposit. Perform ranking using matrix evaluation procedures.

20515 **Document and Review Siting and Geotechnical Results**
Write report documenting siting and geotechnical work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for review by project team before finalizing. Prepare a risk analysis and mitigation plan for the siting, geotechnical, and hydrological information used in the design.

20601 **Develop Agency Reconnaissance, Government, NGO, and Public Relations Work Specification**
Develop scope of work and schedule necessary for agency reconnaissance, government relations, nongovernmental organizations (NGOs), and public relations studies and/or work.

20604 **Conduct Agency and NGO Reconnaissance Literature Search**
Search literature to get overview of permits required by various government agencies, which may affect design, construction, development, or operation of project. Study area projects and identify active NGO groups in the area.

20605 **Identify Involved Government Agencies**
Conduct telephone conversation and personal visits to update data on government agencies having jurisdiction over design, construction, development, and operation of project. Determine all legal and political jurisdictions and all laws, regulations, and legislative codes at the federal, state, and local levels that are applicable to the process of mine/mill planning and operation. The following must also be identified:

- List all political jurisdictions in which the mining operation will exist.
- Obtain copies of all federal and state laws and codes relating to the state and country in which you want to construct a mining operation.
- Obtain a list of all mining permits required and a description of the regulatory processes involved in obtaining the permits.
- For properties within the United States, request determination from the district engineer of the U.S. Army Corps of Engineers (and other federal agencies if federal lands are involved) on its possible involvement and the necessity for a federal environmental impact statement (EIS) under the National Environmental Policy Act.
- If federal EIS is required, determine if state environmental impact report may be used as database or whether operative federal agency will require more, less, or other data than that collected for the state agency.
- Participate in any hearings on any federal statement or actions that occur independent of the state.
- Determine with the state environmental agency which state laws will be applied to the mine development under its jurisdiction. Obtain in writing the rationale for elimination of any potentially applicable laws.
- Inventory other state laws, permits, and permissions applicable to a mine in that state.
- Obtain copies of county zoning codes, sewerage codes, and information on which districts, special assessment districts, or other subjurisdictions of the county may be influenced by the mine.
- Determine each code and jurisdiction that may be encountered and the responsible local administrators. Obtain requirements to be fulfilled under each code.
- Obtain copies of all codes and rules applicable in the township or incorporated area, especially zoning and those related to utilities, waste disposal, and highways.

20606 **Determine Agency Regulations**
Conduct telephone conversations and personal visits to update data on government agency regulations affecting design, construction, development, and operation of project.

20607 **Determine Agency Permit Requirements**
Conduct telephone conversations and personal visits to update data on government agency permit requirements affecting design, construction, development, and operation of project.

20608 **Develop and Implement Government, NGO, and Public Relations Programs**
Develop program(s) to keep government, NGOs, and public informed of nature and status of project. Implement one or more of the programs.

20609 **Document and Review Agency, NGO Reconnaissance Results**
Write report documenting agency and NGO's reconnaissance work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for review by project team before finalizing.

20610 **Document and Review Government, NGO, and Public Relations Programs**
Write report documenting government and public relations programs developed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility report. Develop a risk analysis and mitigation plan for the government, NGO, and public relations programs.

20701 **Develop Mine Work Specifications**
Develop scope of work and schedule necessary for conducting mining evaluation studies.

20704 **Update Mining Literature Search**
Review company, contractor, and general available literature published about mining deposits of similar nature. Search should identify and benchmark mine data related to the mining method, unusual characteristics, types of equipment, potential problems, and so forth. Search may include visits to operating properties of similar nature.

20705 **Identify Possible Mining Methods**
Identify mining methods suitable for developing and producing deposits of this nature. Use combination of literature search and experience of company personnel and contractor personnel. List and define the potential methods. Consider range of production from the various methods.

20706 **Develop Tentative Layout for Each Mining Method**
Prepare tentative mine development and production layouts for each possible mining method identified. Prepare layouts with sufficient detail and accuracy to allow comparisons of capital and operating costs.

20707 **Evaluate Alternative Mining Methods**
Evaluate operational characteristics of each possible mining method. Perform comparative capital and operating cost analyses, using present worth techniques. Identify other factors influencing selection of preferred mining method(s) such as environment, safety, hydrology, recoveries, rock mechanics,

potential for mechanical excavation, dilution, and production limitations.

Rank alternatives in order of preference using matrix system and considering all listed factors and others, if applicable. Select two or three best methods for more detailed evaluations.

20708 **Refine Mine Layouts and Develop Mine Plans**
Prepare mine layouts for chosen alternatives in sufficient detail to allow development of mine plans. Five-year mine plan and life-of-mine plan will be based only on proven and probable reserves for all properties within the United States and Canada or where the ownership stock is listed in those two countries. Mine plans should have an accuracy of ±15% to 20%. Mine plans should include layouts and schedules for head frame, shaft, stations, preproduction development, underground service area development, production, various sizes of underground storage, and the cost benefit of each size and continuing production development.

20709 **Perform Mine Functional Analyses**
Calculate operational and development cycles, labor requirements, equipment requirements, and supply requirements for each mining function and for each alternative selected in 20707 based on mine industry experience in these mining methods. The functions include drilling, blasting, loading, hauling, scaling, bolting, ground control, mine backfill (if needed) hoisting, primary crushing (if underground), maintenance, supply/debris handling, pumping, and other supports services. For each function, develop personnel productivity learning curves and prepare cost and operational schedules for each.

20710 **Develop Mine Capital and Operating Cost Estimates**
Develop cost estimates for total mining operation and alternatives. Costs should have accuracy of ±15% to 20%. Because costs will become part of intermediate financial analysis, put in format suitable for financial analysis.

20711 **Document and Review Mine Results**
Write report documenting mine study work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for review by project team before finalizing. Prepare a risk analysis and mitigation plan on the proposed mining operation. At this point, try to locate and evaluate mining contractors and perform a cost comparison analysis.

20801 **Develop Mill Work Specifications**
Develop scope of work and schedule necessary for process, milling, waste rock storage, and tailings disposal studies.

20804 **Conduct Milling Literature Search**
Review company, contractor(s), and general available literature published concerning processing material of similar nature. Search should identify and benchmark some possible processing methods, types of equipment, potential problems, and so forth. Search may include visits to operating properties using processes of similar nature.

20805 **Procure Metallurgical Samples**
Procure drill-core samples to use for metallurgical testing. Drilling may be performed as part of predevelopment drilling program. Make sure that the samples represent the ore body. Large core samples for autogenous grinding test may be needed.

20806 **Conduct Metallurgical Tests**
Ship metallurgical samples to testing laboratory. Conduct appropriate tests to determine comminuting characteristics (work index); separation and concentration characteristics of all types of ores; reagent consumption; heads, tails, and concentrate analyses; process flow sheet; environmentally harmful gaseous, liquid, and solid products produced; complete waste characterization of all waste products; long-term leachability of metal ions from tailings; and areas of uncertainty. Send test results to project team in report form. Contractor should submit report in draft form for review by project team before finalizing.

20807 **Determine Mill Process Requirements**
Evaluate metallurgical testing results. Use test results, literature search, company experience, and contractor experience to select two or three best process variation methods. Determine processing requirements for each alternative.

20808 **Prepare Milling Flow Sheet**
Prepare flow sheet for each process alternative. Flow sheet should contain sufficient detail to allow selection and sizing of equipment and show material balance.

20809 **Perform Mill Functional Analyses**
Calculate the operational cycles and labor, equipment, and supply requirements for each milling function and for each process alternative selected in 20807 based on the various sizes of production from the mine and industry experience. The functions include stockpiling/reclaiming (size of storage), crushing, grinding, screening, concentrating including heap leaching (if applicable), classifying, clarifying, tailings disposal, concentrate handling, maintenance, supply/debris handling, and other support services. Prepare cost and operational schedules for each.

20810 **Investigate Custom Milling Alternatives**
Investigate opportunities for selling run-of-mine (ROM) material. Determine sale price and charges associated with selling ROM. Investigate opportunities for tolling mine-produced material. Determine custom mill capacity, timing, and costs. Costs should include capital and operating estimates for all associated tolling activities such as mill expansion costs to company (if any), transportation of product to custom mill, losses/deducts for processing, tolling charge, sampling methods, transportation of concentrate, and personnel requirements.

20811 **Determine General Mill Plant Arrangement**
Determine arrangement of mill facilities, including tailings, for each alternative. Prepare design basis and general arrangement drawings.

20812 **Develop Mill Capital and Operating Cost Estimate**
Develop cost estimates for total milling operation alternatives. Costs should have accuracy of ±15%. Because costs will become part of intermediate financial analysis, put costs in format suitable for financial analysis.

20813 **Document and Review Milling Results**
Write report documenting milling and metallurgical work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for project team review before finalizing. Prepare risk analysis and mitigation plan for the metallurgical processing.

20901 **Develop Refinery Work Specifications**
Develop scope of work and schedule necessary for custom smelting/refining studies.

20904 **Investigate Custom Smelting/Refining Alternatives**
Investigate opportunities for custom smelting and/or refining project concentrate. Determine custom refining capacity, timing, and costs. Costs should include capital and operating estimates for everything associated with custom refining such as transportation of concentrate to refinery, refining charge (consider deducts, penalties, and/or credits), transportation of refined product, and personnel requirements.

20905 **Document and Review Custom Smelting/Refining Results**
Write report documenting smelting/refinery work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for project team review before finalizing. Prepare a risk analysis and mitigation plan of the smelting/refining process.

21001 **Develop Surface Facilities Work Specifications**
Develop scope of work and schedule necessary for study of project surface facilities not connected with mine and mill studies.

21004 **Determine Surface Building Requirements**
Determine surface buildings required for project operation. Include buildings serving support function for mine and mill but not buildings directly related to mining and milling activities. Types of buildings include ambulance garage, fire-fighting facility, administration/office, dry/changehouse, guardhouse, and surface shops. Determine operating and maintenance personnel and equipment requirements (including shop, office, and dry equipment). Prepare design basis and general arrangement drawings. Develop capital and operating cost estimates with ±15% accuracy to include in intermediate financial analysis. Put costs in form suitable for financial analysis.

21005 **Determine Surface Utilities Requirements**
Determine utilities required for project construction, development, and operation. Study does not include power distribution inside mine and mill battery limits distribution within facilities included in mining and milling studies. Utilities include electric power; fuel for buildings and fuel storage for operating equipment; communications (radio, telephone, and computer networks required); potable water; fire protection; sewage system; and garbage, trash, and solid waste removal and disposal. Determine operating and maintenance personnel and equipment requirements. Develop applicable piping and instrumentation diagram/drawing (P&ID) and line drawings. Develop capital and operating cost estimates with ±15% accuracy to include in intermediate financial analysis. Put costs in form suitable for financial analysis.

21006 **Determine Surface Transportation Requirements**
Determine transportation needs for moving equipment, supplies, material, and mine/mill product into and out of project area during project construction, development, and operation. Study should evaluate alternative transportation methods such as truck haulage; rail haulage (both off-site and in-plant); ship/barge haulage and port facilities (if needed); air haulage (if feasible); and combinations of these methods.

Study should determine access road requirements and costs, capital and operating costs, personnel and equipment requirements, and other costs associated with each feasible transportation system. Develop capital and operating costs with ±15% accuracy to include in intermediate financial analysis and put costs in form suitable for financial analysis.

21007 **Determine Surface Mobile and Miscellaneous Equipment Requirements**
Determine the surface mobile and miscellaneous equipment requirements not covered under other activities. This includes equipment for emergency medical and safety; road and yard area maintenance; supervisor pickups and car(s); maintenance personnel pickups and trucks; loader(s)/backhoe(s); forklift(s); crane(s)/cherry picker(s); portable welder(s)/compressor(s)/generator(s); small rear-dump truck(s); and crawler tractor(s) with dozer. Determine operating and maintenance personnel and equipment requirements. Develop capital and operating cost estimates with ±15% accuracy to include in intermediate financial analysis. Put costs in form suitable for financial analysis.

21008 **Develop Water Management Plan and Costs**
Determine requirements for total project water management system. This will require evaluating one or more alternatives for mine, mill, tailings disposal, potable, fire protection and other water usage requirements; mine dewatering and pumping requirements; project water balance (identify all water sources and losses); makeup water requirements and source (if water short); water treatment, disposal, or evaporation system (if excess water); potable water system (include source, treatment, storage, and distribution); fire protection system (include source, treatment, storage, and distribution); and dust control water requirements. Determine operating and maintenance personnel and equipment requirements, including buildings or structures to house facilities. Develop P&ID. Develop capital and operating cost estimates with ±15% accuracy to include in intermediate

financial analysis. Put costs in form suitable for financial analysis.

21009 **Determine General Surface Facilities Arrangement**
Determine requirements for surface facility, plant, and disposal area(s) internal road (does not include haulage roads for open-pit mine); parking areas; construction lay-down area(s); and storage area(s) (including mine waste).

Prepare plot plans showing alternatives for arrangement of all surface facilities including listed items, water facilities, mine, mill, and tailings facilities. Rank alternatives in preference order using matrix system to evaluate factors such as facility spacing and location, environment, accessibility, utilization, capital cost of each alternative, and operating costs (if different and applicable).

Choose best alternative. Prepare design basis and general arrangement drawings. Show the capital costs of areas selected from the preceding matrix analysis. Determine operating and maintenance equipment and personnel requirements for areas identified in 21007, and maintenance material and supply requirements for areas listed in 21009.

21010 **Determine Warehouse Requirements**
Determine size of warehouse and storage yard facilities; amount of warehouse inventory; equipment (mobile and stationary) necessary to store warehoused items, load and unload supplies, and move supplies within confines of project area; and operating personnel requirements. Develop capital and operating costs with ±15% accuracy to include in intermediate financial analysis. Put costs in form suitable for financial analysis.

21011 **Develop Surface Facilities' Capital and Operating Cost Estimates**
Operating cost estimates should have accuracy of ±15%. Because costs will become part of intermediate financial analysis, put in format suitable for financial analysis.

21012 **Document and Review Surface Facilities Results**
Write report documenting surface facilities studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Contractor should submit report in draft form for review by project team before finalizing. Prepare risk analysis and mitigation plan for any and all surface facilities or utilities.

21101 **Determine General Personnel Requirements**
Determine administrative and management personnel requirements; and operating, maintenance, support, and supervisory personnel requirements developed in all the preceding project activities. Split requirements into salaried exempt, salaried nonexempt, and hourly classifications. Develop labor buildup schedules for each classification using results of learning curve analysis for each classification.

21102 **Prepare Organization Charts**
Prepare charts showing how each project alternative should be organized. Charts should show lines of authority and responsibility.

21103 **Determine Administrative Costs**
Determine salaries and wages of personnel identified in 21101; payroll burden associated with salaries and wages; and cost, type, and quantity of office equipment and supplies required for all offices including administration, mine, mill, maintenance, and others. Prepare costs in form suitable for financial analysis. Costs should have accuracy of ±15%.

21104 **Document and Review Organization and Administration Results**
Write report documenting administrative costs and personnel requirements. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report.

21201 **Develop Labor Plan**
Develop plan, with alternatives, for attracting and keeping productive, qualified personnel. Items to investigate include recruiting, training, absentee and turnover projections, commuting/fly in–fly out work force, community development, salaries/wages, fringe benefits/payroll burden, incentive system, and union/nonunion considerations.

21202 **Prepare Labor-Related Cost Estimates**
Identify personnel and equipment requirements for plans developed. Prepare capital and operating cost estimates associated with plans developed. Costs should have accuracy of ±15%. Put costs in format suitable for financial analysis.

21203 **Document and Review Labor-Related Results**
Write report documenting labor-related studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Prepare a risk analysis and mitigation plan for all administrative and labor-related issues.

21301 **Conduct Intermediate Market Studies**
Update market studies to determine product requirements, supply and demand forecast, selling price and marketing strategy of salable products, and position relative to competitors. Review metallurgical results of most recent testing against product sales specifications. Estimate price ranges for life of project. Prepare prices in format suitable for financial analysis.

21302 **Document and Review Market Study Results**
Write report documenting marketing studies listed in 21301. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Prepare a risk analysis and mitigation plan for all aspects of marketing the product.

21401 **Develop Intermediate Financial Analysis Criteria**
Develop criteria for performing intermediate financial analysis. Criteria include overall project schedule (includes final evaluation, design, construction, development, and start-up), capital and operating costs, royalties; escalation factors (though the analysis will probably be done in constant dollars), tax rates, working capital, property acquisition costs, mine and mill

recovery, revenues, depreciation methods and depletion allowance, allowance for unforeseen factors, capitalization factors, salvage values, corporate overhead allocation, sensitivity and risk analysis (see 21403), and project alternative comparisons.

21402 **Conduct Financial Analysis**
Conduct analysis for total project using a suitable computer program. Print results of economic analysis.

21403 **Conduct Financial Sensitivity Studies**
Evaluate sensitivity/risk of various key factors. Also, quantify the degree of risk and perform Monte Carlo risk analysis on the collective factors: operating costs, capital costs, reserves, grade, mill recovery, royalties, taxes, and other items with high degree of uncertainty. Print results of sensitivity/risk analysis.

21404 **Document and Review Financial Results**
Write report documenting financial analysis and sensitivities. Review results of work with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Prepare a risk analysis and mitigation plan for the techniques used in the financial analysis.

21501 **Develop Tax Study Specifications**
Develop scope of work and schedule necessary for conducting study of taxes applicable to project.

21502 **Conduct Intermediate Tax Studies**
Update tax studies to determine taxes applicable to project. Analyze taxes to understand how they affect construction, development, and operation of project. Prepare tax rates in format suitable for financial analysis.

21503 **Document and Review Tax Studies**
Write report documenting tax studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in intermediate feasibility study report. Prepare a risk analysis and mitigation plan related to the taxing of future property and production.

21601 **Prepare Final Study Plan and Budget**
Update final feasibility study plan and schedule using formalized scheduling techniques. Update budget for final feasibility study. Schedule and budget should conform to those used in financial analysis (Activity 21401).

21602 **Prepare Intermediate Report**
Prepare formal report detailing intermediate study work performed. Prepare report in style and format that
- Is suitable for presentation to management,
- Is suitable for use by other project teams,
- Contains history and results of work performed, and
- Has a composite table of all the risk factors analyzed in the report.

21603 **Present Intermediate Report, Plan, and Budget to Management**
Present all data generated during intermediate study, plus plan and budget for final study, to management for review. Present information in meeting(s) with hard copies of reports, schedules, and data. Distribute report at least 1 week prior to meeting to allow personnel time to read and review.

APPENDIX 4.7C ADDITIONAL TESTING ACTIVITIES (SUCH AS A TEST MINE OF BULK SAMPLE)

Activity No. (from WBS) Activity Title and Description

30301 **Perform Final Exploration Drilling and Other Field Work**
Continue work outlined under predevelopment drilling contracts (Activity 20306). Do other geologic field work as required. Final exploration means sufficient drilling and field work to get reserve estimates within ±10% accuracy range. Assay drill-hole samples and/or log drill holes.

30302 **Assemble and Edit Drill-Hole Data**
Assemble drill-hole data pertaining to deposit. Edit data for correctness and completeness. Types of data included: identification; geologic parameters; collar coordinates; assay values and intervals; hole depth, dip, and direction; and date hole completed.

30303 **Update Drill-Hole Computer File**
Prepare mineral inventory data for entry in computer system. Add to existing data to update drill-hole files in computer.

30310 **Prepare Geologic Maps**
Prepare necessary drill-hole maps and cross sections to help evaluate the mineral deposit. Prepare maps with computer and/or by hand.

30311 **Delineate Mineral Zones**
Identify and delineate mineralized zones by computer from drill-hole files and/or by hand.

30312 **Compute Potential Mineral Reserves**
Build computer block model and compute mineral reserves with various cutoff grades, mining heights, waste thicknesses, and so forth. Calculate reserves by hand if block model not developed.

30313 **Document and Review Mineral Inventory Results**
Write report documenting results of predevelopment program. Report should contain data on geology, field work, and reserves. Review results with appropriate levels of management and other personnel. Write study in style and format suitable as chapter in final feasibility study report.

30710 **Develop Test Mine Work Specifications**
Develop scope of work and schedule necessary for designing, constructing, developing, and operating test mine (assuming that test mine is needed). Prepare request for proposal (RFP) to send to contractors capable of performing the work. RFP should contain draft of proposed contract.

30711 **Determine Test Mine Permit Requirements**
Identify permits required to design, construct, develop, and operate test mine. Refer to work performed under Activity 20607. Recheck with government agencies for new or different requirements.

30712 **Plan Bulk Sampling Program**
Calculate amount of bulk sample required for pilot-plant testing. Evaluate geologic and reserve data to choose test mine bulk sampling areas representative of deposit. Interface bulk sampling plan with test mining plan, Activity 30713. Modify bulk sampling plan and areas based on realistic mining plan, time schedule, and budget. Select pilot plant to run bulk sample(s). Plan metallurgical tests required. Determine requirements for sample(s) handling and transportation and sampled material disposal.

30713 **Plan Test Mine Program**
Develop a mine plan to select layout and development necessary for the following: metallurgical bulk sampling program (underground [UG] and open pit [OP]); predevelopment drilling program (UG); test mining program; rock mechanics tests (UG and OP); pillar, drift, and stope size evaluations (UG); drilling and blasting or mechanical excavation tests (UG and OP); ground support tests (UG); slope-stability tests (OP); and materials handling tests (UG and OP). Design required test mine surface facilities, access system and development (surface mine stripping or underground mine station[s], and level[s]) needs. Develop schedule and budget for test mine activities. Prepare construction contractor bid package.

30714 **Prepare Test Mine Permit Applications**
Prepare permit application(s) necessary to perform test mining program.

30715 **Procure Test Mine Permits**
Submit permit application(s) to appropriate government agency(ies). Wait for permit approval(s). Obtain approved permit(s).

30716 **Management Approval of Test Mine**
Get approval from appropriate levels of management to proceed with test mining as outlined in budget and schedule developed under Activity 30713.

30717 **Locate, Evaluate, and Select Test Mine Contractor**
Identify contractors capable of performing the work. Send each contractor copy of bid package developed in 30713. Evaluate bids received, contractor's financial status (Dun & Bradstreet report if required) and other pertinent data. Select preferred contractor, preferably using matrix evaluation if low bid is not only selection criterion. Inform contractor(s) of its selection. Give notice to proceed. Revise, if necessary, scope of work and contract to reflect information contained in bids.

30718 **Prepare Test Mine Contract**
Write contract, with assistance from law office and controllers. Get necessary company approvals. Send contract to contractor for signature. Get approved contract from contractor and review for signature correctness. Some delay can occur if contractor wants to negotiate terms before approving.

30719 **Mobilize Test Mine Contractor**
Require time for contractor to arrive on-site and set up once notified to proceed.

30720 **Construct Test Mine Surface Facilities**
Build or erect surface facilities necessary for construction, development, and operation of test mine, including hoisting facilities, shaft collar, and headframe for an underground test mine.

30721 **Perform Test Mine Access and Level Development**
For underground test mine, sink shaft (or other method of access), excavate, and construct station(s) and perform necessary level development. For open-pit test mine, strip necessary overburden and waste material.

30722 **Perform Test Mining**
Conduct mining tests as outlined under Activity 30713.

30723 **Procure Test Mine Bulk Samples**
Procure test mine bulk sample(s) as planned under Activity 30712 in intermediate study. (In many cases, this activity and 30724 occur between the intermediate and final feasibility study.)

30724 **Perform Test Mine Predevelopment Work**
Perform test mine drilling and other geologic work as outlined under Activity 30713.

30725 **Update Potential Mineral Reserves**
Update mineral reserve calculations using drilling, assay, and geologic data gathered during test mining (Activity 30724).

30726 **Update Engineering Data**
Update all previously acquired engineering data with the data gathered during test mine operation.

APPENDIX 4.7D
PHASE III: FINAL FEASIBILITY STUDY COMPLETE ACTIVITY DEFINITIONS

Activity No. (from WBS) **Activity Title and Description**

30100 **Public Affairs**
As soon as the company management approves the final feasibility study plan and budget, notify state government officials that the project will proceed to the next level of examination. Brief the media on the decision to proceed to the final feasibility study, making sure to emphasize that the final decision of whether or not to build a mine is probably many months or even years away. Continue the dissemination of information as it becomes available and continue presenting speeches at local and state meetings. Set up local town meeting where stakeholders can question first-hand what will take place if a mine is built. Address their remaining concerns and the company's plans to mitigate their concerns.

30101 **Management Approval of Final Study Plan and Budget**
Get approval from appropriate levels of management to proceed with final feasibility study outlined in plan, schedule, and budget submitted to management for review at end of intermediate study (Activity 21603).

30201 **Review Land Status**
Review land ownership, control, royalty, and lease situation updated during intermediate study. Project

team should review land status with personnel in company's land office, and other appropriate individuals with respect to site locations identified during intermediate study.

30202 **Finalize Land Acquisitions and Costs**
Land personnel should determine final costs of buying and/or leasing land necessary to construct, develop, and operate project. Commitments for land are probably required at this time. Costs will become part of final study financial analysis so accuracy should be within ±10%. Put costs in format suitable for financial analysis.

30203 **Document and Review Land Situation**
Write report documenting results and costs of land negotiations. Review results of negotiations with appropriate levels of management and other personnel. Complete all land maps. Write report in style and format suitable as chapter in final feasibility study report.

30304 **Develop Core Drilling Work Specifications**
Develop scope of work and schedule necessary for core drilling program to obtain samples for the following tests: metallurgical (may not need if test mine program planned), rock mechanics (may not need if test mine program planned), Cerchar abrasivity, density, porosity, permeability, and miscellaneous. This program may be accomplished with regular predevelopment drilling program or as separate program.

30307 **Prepare Test Core Sample Flow Chart**
Prepare flow chart (listing) of drill cores required for various tests (e.g., metallurgy, rock mechanics, porosity, permeability, density, moisture). Chart should identify size and amount of cores required, from where cores procured (area of deposit), purpose of cores, where cores are sent for testing, types of test, and core storage instructions.

30308 **Conduct Core Drilling Program**
Perform the field core drilling program as planned and scheduled under Activity 30307.

30309 **Analyze Other Core Samples**
Send other core samples to appropriate lab or testing facility(ies). Analyze and test core samples for porosity, permeability, density, and moisture. Send test data to project team and other interested parties.

30401 **Develop Environmental Work Specifications**
Develop scope of work and schedule necessary for environmental impact analyses and environmental control plans required for project analysis and costs. Work will serve as base data for final feasibility costs and probable update data for permit applications. Prepare request for proposal (RFP) to send to contractors capable of performing the work. RFP should include draft of proposed contract.

30404 **Assess Environmental Impacts**
Reassess impact to baseline environment caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. This should include input from the final feasibility study of mine, mill, tailings, and surface facilities.

30405 **Finalize Air Quality Plan and Costs**
Finalize chosen plan to maintain air quality in and around project area. Update effects caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Complete design basis drawings. Estimate capital and operating costs to implement the plans. Put costs in format suitable for financial analysis. Cost accuracy should equal +10%.

30406 **Finalize Water Quality Plan and Costs**
Finalize chosen plan to maintain surface water quality in and around project area. Update effects caused by construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans. Complete design drawings. Put costs in format suitable for financial analysis. Cost accuracy should equal +10%.

30407 **Finalize Ecological Plan and Costs**
Finalize chosen plan to mitigate ecological disturbances caused by effects of construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans. Put costs in format suitable for financial analysis. Cost accuracy should equal +10%.

30408 **Finalize Reclamation Plan and Costs**
Finalize chosen plan to reclaim land disturbed by effects of construction, development, and operation of mine, plant, tailings disposal, and surface facilities. Estimate capital and operating costs to implement the plans. Complete design basis drawings. Put costs in format suitable for financial analysis. Cost accuracy should equal +10%.

30409 **Finalize Socioeconomic Plan and Costs**
Finalize chosen plan to estimate requirements for community development. Plan should include requirements for the following: employee housing, medical and dental facilities, schools, community utilities, community services, and recreational activities. Estimate capital and operating costs to implement the plans. Put costs in format suitable for financial analysis. Cost accuracy should equal +10%.

30410 **Document and Review Environmental Study Results**
Write report documenting environmental work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report. Contractors should submit report in draft form for project team review before finalizing.

30501 **Develop Siting and Geotechnical Work Specifications**
Develop scope of work and schedule necessary to finalize siting, rock mechanics, soils mechanics and foundation, and hydrology studies. Prepare RFP to send to contractors capable of performing the work. RFP should contain draft of proposed contract.

30504 **Prepare Hydrology Permit Applications**
Prepare necessary permit applications to perform hydrology studies of groundwater and surface water

quantities and qualities. Permits are needed primarily to drill test wells.

30505 **Procure Hydrology Permits**
Submit permit application(s) to appropriate governmental agency(ies). Wait for permit approval. Get approved permit(s).

30506 **Conduct Hydrology Studies**
Conduct studies to finalize groundwater quantity and quality data. This usually requires drilling and pump testing wells to determine amount and quality of water expected during mining and amount and quality of water required for makeup. Conduct studies to determine surface water flow patterns and amounts to expect during possible maximum flood and 100-year-flood periods. Send study data to project team in report form. Contractor should submit report in draft form for review by project team before finalizing.

30507 **Prepare Soils Mechanics Permit Applications**
Prepare necessary permit application(s) to finalize soils and foundations investigations. Permit(s) probably necessary to dig test pits and do test borings.

30508 **Procure Soils Mechanics Permits**
Submit permit application(s) to appropriate governmental agency(ies). Wait for permit approval. Get approved permit(s).

30509 **Procure Soils Mechanics Samples**
Procure soils samples to finalize soils mechanics tests and foundation analyses. Procure samples in areas preferred for plant and surface facilities construction and tailings disposal. Samples usually consist of test borings and test pits. Samples will serve as basis for buildings, dams, shafts, and other foundation design specifications.

30510 **Conduct Soils Mechanics Tests**
Ship soils mechanics samples to testing laboratory. Conduct appropriate tests to finalize physical and chemical properties affecting building foundations, tailings disposal areas, and shaft collars. For open-pit mines, tests are needed to determine slope stability. Conduct appropriate field tests to determine above-soils properties and water flow characteristics. Analyze test results. Field tests could include location of suitable construction materials. Send test results in report form to project team. Contractor should submit report in draft form for review by project team before finalizing.

30511 **Procure Rock Mechanics Samples**
Procure drill-core samples to finalize rock mechanics properties. Drilling is part of core drilling program (Activity 30310).

30512 **Conduct Rock Mechanics Tests**
Ship rock mechanics samples to testing laboratory. Conduct appropriate tests to finalize strength of rock. Analyze test results to finalize size of openings and pillars in underground mine or pit slope angles (stability) in open-pit mine. If applicable, test for the application of mechanical excavation. Send test results in report form to project team. Contractor should submit report in draft form for review by project team before finalizing.

30513 **Finalize Siting Preferences**
Determine the final location of all surface facilities. This includes shaft and other mine facilities, mill and processing facilities, tailings facilities and pipelines, and surface ancillary facilities such as roads, buildings, power lines, gas lines, storage areas, waste disposal areas, parking areas, and construction lay-down areas. If an open-pit mine, this would include location of pit. Locate all sites within a few feet of their planned constructed location. Complete design basis drawings. Final facilities location selection should include factors such as the following: facility spacing and location, environment, accessibility, utilization, capital cost, and operating costs (if different and applicable).

30514 **Document and Review Siting and Geotechnical Results**
Write report documenting siting and geotechnical work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report. Contractor should submit report in draft form for review by project team before finalizing.

30601 **Develop Permit Application Work Specifications**
Develop scope of work and schedule necessary for preparation of permit applications. Prepare RFP to send to contractors capable of preparing permit applications. RFP should contain draft of proposed contract.

30604 **Prepare Permit Applications**
Procure necessary forms and formats for all permits required to construct and operate project. Complete all permit applications as required by local, state, and federal agencies. Submit applications to appropriate governmental agencies.

30605 **Procure Construction and Operating Permits**
Wait for various governmental agencies to approve permit applications. Get approved applications from agencies. This task could require some application rewriting or amending if one or more agencies need data not presented in original application.

30606 **Expand Government and Public Relations Programs**
Expand and update program(s) to keep government and public informed of nature and status of project.

30607 **Document and Review Government and Public Relations Programs**
Write report documenting status of government and public relations programs. Review results of programs with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility report.

30701 **Develop Mine Work Specifications**
Develop scope of work and schedule necessary for conducting final mine evaluation study. Prepare RFP to send to contractors capable of performing the work. RFP should contain draft of proposed contract.

30704 **Develop Final Mine Layout**
Prepare final mine development and production layouts. Prepare layouts with sufficient detail and accuracy to develop mine plans and allow estimating capital and operating costs, and development and operating schedules to accuracies of ±10% to 15%.

30705 **Develop Final Mine Plan**
Mine plans should, include layouts and schedules for headframe, shaft, stations, preproduction development, underground service area development, production, and continuing production development. Complete design basis drawings. Develop life-of-mine production plans as well as detailed 5-year mine production plans. Mine plans should have an accuracy of +15%.

30706 **Perform Mine Functional Analyses**
Calculate operational cycles, and labor, equipment, and supply requirements for each mining function. Refine the functions analysis made earlier to include drilling, blasting, loading, hauling, scaling, bolting, ground control, mine backfill (if needed), hoisting, primary crushing (if underground), maintenance, supply/debris handling, pumping, and other support services. For each function, develop personnel productivity learning curves and prepare cost and operational schedules for each.

Use personnel productivity learning curves developed in intermediate feasibility study for each function. Prepare cost and operational schedules for each.

30707 **Develop Mine Capital and Operating Cost Estimates**
Develop cost estimates for total mining operation. Costs should have accuracy of ±10% to 15% and will become part of final financial analysis. Put costs in format suitable for financial analysis.

30708 **Develop Mine Design Specifications**
Develop design specifications for competitive bidding of mine design, construction, and development work. Prepare bid packages. Design specifications for an underground mine should include systems for mine access, materials and personnel handling, ventilation, communications, electrical, mine dewatering, and fuel storage and handling; maintenance and warehousing facilities; explosives handling and storage facilities; crushing facilities; and sewage system.

Design specifications for an open-pit mine normally include communications system(s); sewage system; electrical system; mine dewatering system; fuel storage and handling system; oil, lubrication, and antifreeze system; and explosives handling and storage facilities.

30709 **Document and Review Mine Results**
Write report documenting mine study work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report. Contractor should submit report in draft form for review by project team before finalizing.

30721 **Perform Test Mine Access and Level Development (if not yet completed)**
For underground test mine, sink shaft (or other method of access), excavate and construct station(s), and perform necessary level development. For open-pit test mine, strip necessary overburden and waste material.

30722 **Perform Test Mining (if not yet completed)**
Conduct mining tests as outlined under Activity 30713 in intermediate study.

30723 **Procure Test Mine Bulk Samples (if not yet completed)**
Procure sample(s) as planned under Activity 30712 in intermediate study. (In many cases, this activity and 30724 occur between the intermediate and final feasibility study, in which case these activities would be omitted here.)

30724 **Perform Test Mine Predevelopment Work (if not yet completed)**
Perform test mine drilling and other geologic work as outlined under Activity 30713 in intermediate study.

30725 **Update Potential Mineral Reserves (if not yet completed)**
Update mineral reserve calculations using drilling, assay, and geologic data gathered during test mining (Activity 30724).

30726 **Document and Review Test Mine Results (if not yet completed)**
Write report documenting test mine work performed. Review results of test mining with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

30801 **Develop Mill Work Specifications**
Develop scope of work and schedule necessary for final process, milling, and tailings disposal studies. Prepare RFP to send to contractors capable of performing the work. RFP should contain draft of proposed contract.

30804 **Procure Metallurgical Samples**
Procure drill-core samples to use for metallurgical testing. Drilling is usually performed as part of predevelopment drilling program. This activity may be unnecessary if bulk sample is obtained from test mine.

30805 **Conduct Metallurgical Testing**
Ship metallurgical samples to testing laboratory or pilot mill, if bulk sample for test mine is used. Conduct appropriate tests to determine final metallurgical data and design specifications for comminuting characteristics; separation and concentration characteristics of reagent consumption; heads, tails, and concentrate analyses; process flow sheet; environmentally harmful gaseous, liquid, and solid products produced; and areas of uncertainty.

Send test results to project team in report form. Contractor should submit report in draft form for review by project team before finalizing.

30806 **Finalize Mill Process Requirements**
Evaluate metallurgical testing results. Use test results, literature search, company experience, and contractor experience to determine the best process method. This method, and other technical data gathered, will serve as basis for mill design.

30807 **Prepare Milling Flow Sheet**
Prepare flow sheet for chosen process. Flow sheet should contain sufficient detail to allow selection and sizing of equipment.

30808 **Perform Mill Functional Analyses**
Calculate the operational cycles, and labor, equipment, and supply requirements for each milling function and the process method used in Activity 30807. The functions include the following: stockpiling/reclaiming (size of storage), crushing, grinding, screening, concentrating including heap leaching (if applicable), classifying, clarifying, tailings disposal, concentrate handling, maintenance, supply/debris handling, and other support services. Prepare cost and operational schedules for each.

30809 **Develop Mill Design Specifications**
Develop specifications for competitive bidding of mill and tailings facilities design and construction. Prepare bid packages. Complete design basis drawings and basic engineering drawings.

30812 **Develop Mill Capital and Operating Cost Estimates**
Develop estimates for total milling operation. Costs should have accuracy of +10%. Costs will become part of final financial analysis. Put costs in format suitable for financial analysis.

30813 **Document and Review Milling Results**
Write report documenting milling and metallurgical work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report. Contractor should submit report in draft form for project team review before finalizing.

30901 **Develop Refinery Work Specifications**
Develop scope of work and schedule necessary for finalizing custom refining plans.

30902 **Finalize Custom Refining Contract**
Start finalizing contract terms with custom refinery(ies) suitable for processing mill concentrates. Finalize with custom refinery(ies) quantities of material for processing, timing, and costs. Costs need to include capital (if required) and operating estimates for everything associated with tolling such as transportation of concentrate to refinery, refining charge (consider deducts and/or credits), transportation of refined product, and personnel requirements. Costs should have accuracy of +10%.

30903 **Document and Review Refining Results**
Write report documenting refinery work performed. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31001 **Develop Surface Facility Work Specifications**
Develop scope of work and schedule necessary for final study of project surface facilities not connected with mine and mill studies. Prepare RFP to send to contractors capable of performing the work. RFP should contain draft of proposed contract.

31004 **Finalize Surface Building Requirements**
Finalize surface buildings required for project operation. These include those serving support function for mine and mill but not buildings directly related to mining and milling activities. Types of buildings include ambulance garage, administration/office, dry/changehouse, guardhouse, and surface shops.

Finalize operating and maintenance personnel and equipment requirements (including shop equipment, office, and dry equipment). Complete design basis drawings. Develop capital and operating cost estimates with +10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31005 **Finalize Surface Utilities Requirements**
Finalize utilities required for project construction, development, and operation. Study does not include power distribution inside mine and mill battery limits (distribution within facilities included in mining and milling studies). Utilities should include electric power; fuel for buildings and fuel storage for operating equipment; communications (radio and telephone); potable water; fire protection, sewage system, and garbage/trash/solid waste removal and disposal. Finalize operating and maintenance personnel and equipment requirements. Complete design basis drawings. Develop capital and operating cost estimates with +10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31006 **Finalize Surface Transportation Requirements**
Finalize transportation method chosen in intermediate study for moving equipment, supplies, material, and mine/mill product into and out of project area during project construction, development, and operation. Methods evaluated include haulage by truck, rail, ship/barge (if feasible), and air (if feasible), or combinations of these.

Finalize access road requirements and costs, capital and operating costs, personnel and equipment requirements, and other costs associated with transportation system. Develop capital and operating costs with +10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31007 **Finalize Surface Mobile and Miscellaneous Equipment Requirements**
Finalize requirements not covered under other activities. This includes equipment for medical emergencies, road and yard area maintenance, supervisor pickups/car(s), maintenance personnel pickups and trucks, loader(s)/backhoe(s), forklift(s), crane(s)/cherry picker(s), portable welder(s)/compressor(s)/generator(s), small rear-dump truck(s), and crawler tractor(s) with dozer. Finalize operating and maintenance personnel and equipment requirements. Develop capital and operating cost estimates with

+10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31008 **Finalize Water Management Plan and Costs**
Finalize requirements for total project water management system. This includes the following: mine, mill, tailings disposal, potable, fire protection and other water usage requirements; mine dewatering and pumping requirements; project water balance (identify all water sources and losses); makeup water requirements and source (if water short); water treatment, disposal, or evaporation system (if excess water); potable water system (include source, treatment, storage, and distribution); fire protection system (include source, treatment, storage, and distribution). Finalize operating and maintenance personnel and equipment requirements, including buildings or structures to house the water-related facilities. Complete design basis drawings. Develop capital and operating cost estimates with +10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31009 **Finalize General Surface Facilities Arrangement**
Finalize requirements for surface facility, plant, and disposal area(s); internal road (does not include haulage roads for open-pit mine); parking areas; construction lay-down area(s); and storage area(s).

Prepare plot plans showing final arrangement of all surface facilities, including items in list, water facilities, mine, mill, and tailings facilities. Finalize capital costs of these surface facilities. Determine operating and maintenance equipment and personnel requirements under preceding surface activities (31004 to 31009). Finalize maintenance material and supply requirements for areas identified in list. Complete design basis drawings.

31010 **Finalize Warehouse Requirements**
Finalize size of warehouse and storage yard facilities; amount of warehouse inventory; equipment (mobile and stationary) necessary to store warehoused items, load and unload supplies, and move supplies within confines of project area; and operating personnel requirements. Complete design basis drawings. Finalize capital and operating costs with +10% accuracy to include in final financial analysis. Put costs in form suitable for financial analysis.

31011 **Develop Surface Building Design Specifications**
Develop design specifications for competitive bidding of design and construction. Buildings are identified under Activity 31004. Prepare bid package(s).

31012 **Develop Surface Utility Design Specifications**
Develop design specifications for competitive bidding of design and construction. Utilities are identified under Activity 31005. Prepare bid package(s).

31013 **Develop Surface Transportation Design Specifications**
Develop design specifications for competitive bidding of design and construction. Facilities are identified under Activity 31006. Prepare bid package(s).

31014 **Develop Water Facilities Design Specifications**
Develop specifications for competitive bidding of design and construction. Facilities are identified under Activity 31008. Prepare bid package(s).

31015 **Develop Surface Facilities Capital and Operating Cost Estimates**
Assemble capital and operating cost estimates for surface facilities. Costs should have accuracy of +10%. As costs will become part of final financial analysis, put costs in format suitable for financial analysis.

31016 **Document and Review Surface Facilities Results**
Write report documenting surface facilities studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report. Contractor should submit report in draft form for review by project team before finalizing.

31017 **Develop Environmental Monitoring Plan (EMP) for Proposed Operation**
Now that all of the mine/plant facilities are defined, complete design basis drawings, specify the labor disciplines, the types and requirements of the monitoring program, the inspections to be required, and the method whereby corrective action and compliance will be achieved.

31018 **Define Cost of EMP**
All professional and staff personnel cost and their equipment must be included. Also include allowances for outside testing on a scheduled basis.

31019 **Amend All Permit Applications to Include Aspects of EMP That Pertain to Various Permits**
Return to the various permit applications and insert those actions and plans that the operating company will take to monitor and control all aspects of the operation to remain in compliance with various regulators' requirements.

31101 **Refine General Personnel Requirements**
Finalize administrative and management personnel requirements; and operating, maintenance, support, and supervisory personnel requirements developed in preceding activities. Split requirements into salaried exempt, salaried nonexempt, and hourly classifications. Finalize labor buildup schedules for each classification.

31102 **Update Organization Charts**
Finalize organization charts showing the project organization. Charts should show lines of authority and responsibility.

31103 **Define Administrative Costs**
Finalize salaries and wages of personnel identified in 31101; payroll burden associated with salaries and wages; and cost, type, and quantity of office equipment and supplies required for all offices including administration, mine, mill, and maintenance. Prepare costs in form suitable for financial analysis. Costs should have accuracy of +10%.

31104 **Document and Review Organization and Administration Results**
Write report documenting administrative costs and personnel requirements. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31201 **Refine Labor Plan**
Finalize plan for attracting and keeping productive, qualified personnel. Plan should include items such as recruiting, training, absentee and turnover projections, commuting (including fly in–fly out), community development, salaries/wages, fringe benefits/payroll burden, incentive system, and union/nonunion considerations.

31202 **Refine Labor-Related Cost Estimates**
Finalize personnel and equipment requirements and capital and operating cost estimates for the plan developed. Costs should have accuracy of +10% and be in format suitable for financial analysis.

31203 **Document and Review Labor-Related Results**
Write report documenting labor-related studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31301 **Conduct Final Market Study**
Update market studies to determine selling price of salable products. Check product specifications of final metallurgical test with required product specifications. Estimate price ranges for life of project and prepare prices in format suitable for financial analysis.

31302 **Document and Review Market Study Results**
Write report documenting marketing studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31401 **Develop Financial Analysis Criteria**
Develop criteria for performing final financial analysis. Criteria should include overall project schedule (design, construction, development, and start-up), ore production and final production schedule, capital and operating costs, royalties, escalation factors, tax rates, working capital, property acquisition costs, mill recovery, depreciation methods, depletion allowance, allowance for unforeseen, capitalization factors, sensitivity and risk analysis (see 31403), salvage values, and corporate overhead allocation.

31402 **Conduct Financial Analysis**
Conduct financial analysis for total project using a suitable computer program. Print results of economic analysis.

31403 **Conduct Financial Sensitivity Studies**
Evaluate sensitivity/risk of various project key factors. Also, quantify the degree of risk and perform Monte Carlo risk analysis on the collective factors, including operating costs, capital costs, reserves, grade, mill recovery, royalties, taxes, and other items with high degree of uncertainty. Conduct sensitivity analysis using a suitable computer program. Print results of sensitivity analysis.

31404 **Document and Review Financial Results**
Write report documenting financial analysis and sensitivities. Review results of work with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31501 **Develop Tax Study Work Specifications**
Develop scope of work and schedule necessary for finalizing study of taxes applicable to project.

31502 **Conduct Final Tax Study**
Update tax studies to finalize taxes applicable to project. Analyze taxes to understand how they affect construction, development, and operation of project. Prepare tax rates in format suitable for financial analysis.

31503 **Document and Review Tax Studies**
Write report documenting tax studies. Review results of studies with appropriate levels of management and other personnel. Write report in style and format suitable as chapter in final feasibility study report.

31601 **Prepare Design, Construction, and Development Schedule and Budget**
Update and expand plan and schedule using formalized scheduling techniques. Update budget for project design, construction, and development. Schedule and budget should conform to those used for final study financial analysis (Activity 31401).

31602 **Prepare Final Study Report**
Prepare the design basis memorandum documenting all of the technical parameters in a single document. Prepare formal report detailing final study work performed. Prepare report in style and format that is
- Suitable for presentation to management,
- Suitable for use by other project teams, and
- Containing history and results of work performed.

31603 **Present Report, Plan, and Budget to Management**
Present all data generated during final study, plus plan and budget for design, construction, and development phase to management for review. Present information in meeting(s) with hard copies of reports, schedules, and data. Distribute report at least 1 week prior to meeting to allow personnel to read and review.

APPENDIX 4.7E
OUTLINE OF DESIGN BASIS REPORT SECTIONS

VOLUME 1: MANAGEMENT SUMMARY

INTRODUCTION AND SUMMARY
Technical feasibility
- Ore deposit
- Facilities
- Viability

Economic feasibility
- Investment and capital cost
- Economic analysis
- Sensitivities

- Operating cost
- Sales price and operating profit
- Construction and life of mine schedule

Product market analysis
- Competitive situation
- Market development activities
- Execution plan
- Project execution responsibility

Division of project execution (if applicable)
- Line of reporting
- Planned method of contracting
- Construction plan
- Construction labor buildup

Business plan
- Objectives
- Demand for finished or refined product

Product market analysis
- Competitive situation
- Market development activities

Market strategy
Outstanding issues
- Country mining law and code
- Potential project partner (if being considered)
- Land purchase from the local government

Purchase of private land and rights-of-way
- Interface with country agencies
- Reestablishment of contractual basis
- Project mobilization
- Construction and operating permits
- Power supply agreements
- Concentrate transport rail agreement concentrator water supply agreement

CONCLUSIONS AND RECOMMENDATIONS
- Conclusions
- Reserves
- Feasibility
- Market
- Preappropriation work funding
- Recommendations

VOLUME 2: PROJECT ECONOMICS

FOREWORD
SECTION I: OVERVIEW
Project schedule
Capital cost estimate
Operating cost estimate
Marketing
Business climate and investment outlook
Economic analysis
Finance/funding
SECTION 2: SCHEDULE
Project schedule
Schedule basis and assumptions
SECTION 3: CAPITAL COST ESTIMATE
Capital cost estimate summary
Initial facilities
Deferred/replacement capital
Owner's costs
Facilities cost estimate basis
- Schedule basis
- Sources of cost information
- Escalation basis (if used)
- Direct materials basis
- Direct labor basis
- Construction indirect costs
- Contractor engineering cost basis
- Contractor's fee basis
- Project contingency cost basis

SECTION 4: OPERATING COST ESTIMATE
Summary
Operating cost estimate basis
- Production schedule
- Sources of cost information
- Escalation (if used)
- Operating labor
- Operating supplies
- Repair and maintenance material
- General and administrative cost
- Other costs
- Operating cost contingency

Projected first year operating costs
- Production factors
- Project timing
- Business factors

Position of this property in world seriatim of industry
- Comparison with other mine candidates for development
- Comparison with existing and potential mine producers

SECTION 5: MARKETING
Summary
Overview of the commodity market
Commodity demand
Commodity mine supply/demand balance
Commodity price
Market analysis
Commodity concentrate market
Commodity finished product market
By-product market
Marketing and business strategy
SECTION 6: BUSINESS CLIMATE AND INVESTMENT OUTLOOK
Summary
Political outlook
Economic outlook
Investment climate
Microeconomic outlook
SECTION 7: ECONOMIC ANALYSIS
Summary
Basis of analysis
- Capital costs
- Operating costs
- Working capital

Production data
- Revenue
- Escalation
- Tax

Economic results
Sensitivities
- Revenue and cost

SECTION 8: FINANCIAL STRATEGY
Financial strategy objectives
Financing plan

VOLUME 3: TECHNICAL NARRATIVE

(For subcategories in Volume 3 marked with an asterisk, there should be a summary, the technical design basis for that item, the design considerations and assumptions that were made for

that item, the technical system description of the components of that system within a subcategory, environmental control systems, and, finally, the equipment list for the subcategory. Although these items are not repeatedly listed under each subcategory, each must be documented.)

SECTION 1: OVERVIEW
Introduction
- Mine
- Primary crushing
- Concentrating/cleaning
- Waste disposal
- Offsites
- Waste dump leaching (if applicable)
- Leachate recovery plant (if applicable)
- Plans for future expansion (if applicable)

Design basis
- Production rates
- Start-up scheduled
- Production buildup schedule
- Objectives of each operational function in design

SECTION 2: MINE AND PRIMARY CRUSHING
General summary
Geology, exploration, resource, and reserve description
Mining*
- Mining plan

Layout of mining facilities
- Description and site conditions
- Design considerations
- Plot plan
- Type of building and construction

Civil works related to mining
- Summary
- Site investigations
- Site preparations
- Miscellaneous civil works

Primary crushing and storage facilities*
Maintenance facilities*
Auxiliary mine buildings*
Utilities
- Summary
- Water supply*
- Fire protection*
- Power*
- Compressed air*
- Fuel oil and lubrication handling facility*
- Communications*

Industrial wastewater collection, treatment, and disposal*
Other environmental control systems
Warehousing and supplies handling*
Discussion of pre-engineering trade-off studies
SECTION 3: ORE CONVEYANCE SYSTEM
(Whatever system is to be used must be fully described. Whether a mine hoisting shaft, a slope conveyor system, overland conveyor system, slurry pipeline, truck or rail system, etc., the design basis must be given.)*
SECTION 4: CONCENTRATOR OR PROCESS CLEANING
General summary (battery-limits)
Layout and civil considerations
- Location map
- Plot plan
- Site considerations

Course product storage*
Communitions circuit(s)*
Mineral extraction circuit(s)*
Thickening, filtering, and drying*
Chemical storage, preparation, and distributions*
Sampling and process control*
Utilities and yard facilities*
Yard and plant piping*
Maintenance facility for process plant*
Warehousing and handling of supplies of processing plant*
Auxiliary processing buildings *
Environmental control systems
Any product expansion plans*
SECTION 5: WASTE DISPOSAL AND WATER RECOVERY/TREATMENT
General summary (battery limits)
Waste system pipeline*
Waste disposal area description
- Summary
- Site selection
- Regional topography and geology

Local site geology
- Hydrology (groundwater)
- Meteorology
- Hydrology (surface water)
- Seismicity

Operation of waste disposal area
- Summary
- General features
- Description of proposed deposition system
- Completion plans
- Seepage mitigation plans
- Dust control plans

Waste dam construction
Summary future work
- Waste dam design basis (specify waste compaction if required)
- Waste dam details
- Stability analysis
- Construction materials specification and placement procedures
- Quality control plans
- Staged dam construction sequence
- Equipment list

Reclaim water system*
Seepage water recovery*
Waste utilities and services*
Waste pond area civil works and buildings*
Maintenance of waste facilities*
Other environmental control systems
SECTION 6: OFFSITES
General summary
- Facilities
- Product storage, transport, and shipping
- Freshwater supply
- Electric power supply
- Access roads
- Communications
- Fire protection
- Mine area drainage and waste treatment plant
- Solid waste collection and disposal plans
- Plant security
- Product transport, storage, and shipping*

- Water supply*
- Electrical power supply*
- Access roads*
- Communications system*
- Security facilities
- Mine area drainage treatment plant*
- Solid waste collection and disposal*

Other environmental control systems
SECTION 7: DUMP OR PAD LEACHING (if applicable)
Summary
Design basis
- Metallurgical process
- Operating schedule
- Projected tonnages and analysis
- Process flow sheet and mass balance

Leaching parameters
- Reagent requirements
- Environmental requirements

Design considerations
- Process design support documents
- Factors considered
- Environmental considerations

System description
General description
- Leach area preparation
- Pregnant leach solution collection
- Leach solution distribution
- Raffinate and pregnant leach
- Solution pumping

Emergency discharge handling
- Electrical system description
- Plant heating, ventilation, and air conditioning
- Control and instrumentation
- Sampling and analytical control
- Environmental safeguard description

Equipment list
- Mechanical process equipment
- Electrical equipment

Solution diversion system*
Waste diversion system*
SECTION 8: DOWN STREAM EXTRACTION
(Any and all downstream extraction processes, such as SX/EW or metals smelting and refining must be fully described.)*

VOLUME 4: PROJECT EXECUTION PLAN

SECTION 1: INTRODUCTION
Objectives and purpose
- Clearly convey to company management how project will be executed.
- Clearly convey to future A/E and construction contractors how project will be executed.
- Provide organizational structure and divisional responsibility for project.
- Complete safe, operable mine/plant, on schedule, within budget.
- Complete mine/plant, meeting all country and government regulations.
- Identify major outstanding issues and action that must be addressed prior to execution.
- Define complete basis to enable project to mobilize and accelerate critical early activities to achieve earliest project completion.

Conclusions
SECTION 2: BACKGROUND
Project history
Project general description
- Mine
- Process plant
- Infrastructure/offsites
- Other facilities

Project milestones
Guidelines to use of country resources
SECTION 3: PROJECT ENVIRONMENT CONTROLS AND BUSINESS ENVIRONMENT
Concerns and interest of country government
- Environmental protection required and permits needed
- Water supply (construction)
- Water supply (operations)
- Waste disposal impoundments
- Roads, electrical power, and communication
- Concentrate transportation
- Mine
- Process plant

Land acquisition
Country taxes
Labor market
- General
- Market mechanism
- Subcontracting labor supply
- Direct hire

Employment requirements
Competition for resources during project period
Country economy
Resources of concern
Public relations
Company public relations plan
Project team public relations plan
Contractors and subcontractors public relations
SECTION 4: PROJECT EXECUTION ORGANIZATION
Overall project organization
Engineering, procurement, and construction coordination task force organization
Company project organization
Project executive's organization
Contractor(s) project organization expected
Deputy project director in home country
Deputy project director in country of project
Deputy project director of engineering
Deputy project director of procurement
Deputy project director of construction
Project control director
Finance director
Human resource director
Turnover and replacement organization
SECTION 5: SCHEDULES AND LABOR REQUIREMENTS
Schedules
- Mine engineering
- Mine procurement
- Mine construction
- Process plant engineering
- Process plant procurement
- Process plant construction
- Infrastructure/offsites engineering
- Infrastructure/offsites procurement
- Infrastructure/offsites construction

- Labor distribution to all areas in all phases
- SECTION 6: PROJECT ENGINEERING EXECUTION
- BASIS (assuming appropriation approval)
- Objectives
- Detailed work plan for contract engineering
  - Methodology execution
  - Division of work
  - Amount of work in the home country
  - Amount of work in the foreign project country (if applicable)
  - Engineering personnel orientation
  - Orientation meetings
  - Site orientation and description
  - Mine description
  - Process plant description
  - Infrastructure/off-site
  - Other facilities
  - Planned organization
  - Project basic documentation and references
  - Engineering documents
- Standards and criteria to be used
- Engineering quality control
- Quality assurance achievement expected
  - Purpose
  - Scope
  - Audit methodology expected
  - Contractor
  - Company
- SECTION 7: PROJECT PROCUREMENT EXECUTION
- BASIS
- Procurement organization, functions, and responsibilities
  - Scope and policy
  - Organization
  - Responsibilities and functions
- Procurement procedures and documentation
  - Procedures expected
  - Documentation expected
  - Country vendor survey information
- Available materials in country or nearest available
- Available fabrication facilities in country or nearest available
- Available subcontracting services in country or nearest available
- SECTION 8: PROJECT CONSTRUCTION EXECUTION
- BASIS
- Construction management
  - Project organization
  - Construction management procedures
  - Reporting
  - Construction expected
  - Construction management interfaces project management
- Preappropriation activities (as applicable)
- Labor
  - Craft supply plan
  - Logistics of labor source
  - Supervision required
  - Safety organization
  - Expatriate housing (if applicable)
- Construction equipment, tools and consumables in general
  - Major equipment
  - Procurement sources
- Transportation of equipment, tools and consumables
  - Maintenance program expected
  - Tools
  - Consumables
  - Fuel
  - Aggregate
  - Concrete
- Construction facilities

VOLUME 5: OPERATING PLAN

- SECTION 1: INTRODUCTION
- SECTION 2: OWNER
- Local organization
  - Location
  - Description of staff
  - Type of management
- Management committees (if applicable)
- Business interfaces
  - Transportation company
  - Utility companies
  - Adjoining property agreements
  - Operating consulting agencies (if applicable)
  - Government regulatory, licensing and permitting agencies
- SECTION 3: OPERATING DEPARTMENTS
- General
  - Operating schedule
  - Mine camp (if applicable)
  - Personnel transportation
    - Staff
    - Hourly
  - Food service (if applicable)
  - Medical facilities
- Functional department and interrelationships
  - Mining department
    - Mine operation
    - Geological/surveying
    - Mine engineering
    - Mine and field maintenance
    - Electrical
    - Primary communition
    - Other functional operating group specific to this property
  - Concentrator/cleaning plant department
    - Plant operation
    - Metallurgical/process engineering
    - Maintenance
    - Electrical
    - Process control and instrumentation
  - Central maintenance and fabrication department
  - Transportation department
- Other operating department specific to this property
- SECTION 4: RECRUITING
- Labor needs
  - Staff (by function)
  - Hourly (by function)
- Manning plan buildup estimate
  - Staff (by function)
  - Hourly (by function)
- Availability of personnel labor pool
  - Staff (by function)
  - Hourly (by function)
- SECTION 5: TRAINING
- Training objectives
  - Initial training
  - Ongoing training
- Management and professional development

- Training program
  - Job positions to be trained to match job descriptions
  - Training organizations
    - Use of outside institutions
    - Vendors training
    - Inside training
  - Initial training time estimated
- SECTION 6: START-UP
- Basis of start-up philosophy
  - Who will participate
  - Who will be in charge of start-up plan
  - Mining department's plan
  - Concentrator/process plant's plan
  - Transportation facilities' plan (if applicable)
- Organization for start-up
  - Mine
  - Concentrator/process plant
  - Transportation
- Start-up assistance
  - Operational staff
  - Architect/engineering contractor
  - Vendors and other consultants (if applicable)
  - Time and budget estimate for start-up
- SECTION 7: INFRASTRUCTURE AND SUPPORT SERVICES (any auxiliary operations that support the main production operations)
- Housing (if applicable)
- Food service (if applicable)
- Personnel transportation (if applicable)
- Power system
- Power generation
- Acquired power
- Water system
- Tailings/waste disposal system
- SECTION 8: MAINTENANCE
- Company philosophy and policies
- Maintenance control programs
  - Work control system
  - Preventive maintenance
  - Maintenance planning and scheduling
  - Maintenance management reports
  - Job priorities
  - Downtime analysis philosophy
  - Backlog reporting system
  - Numbering control system
  - Warehouse and inventory control system
- Other tasks
  - Maintenance work requirements
  - Collection and cataloging of equipment information
  - Develop equipment
  - Identification codes
  - Develop preventive maintenance schedules
  - Develop maintenance forms
  - Execute contracts for rebuild and repair components
- SECTION 9: ROAD MAINTENANCE
- Description of road system to be maintained
- Responsibilities for specific areas to be maintained
- SECTION 10: ENVIRONMENTAL
- Company policy and objectives
  - Present conditions by areas
  - Source of pollution by areas
  - Assessment of hazards from above sources
  - Objectives of monitoring program
- Monitoring program recommended responsibilities
  - Internal responsibilities
  - Consultant responsibilities
- SECTION 11: ADMINISTRATION AND SUPPORT SYSTEM
- Purchasing department
- Controllers department
  - Financial control
    - Registration and depreciation of property and materials
    - Accounting system
  - Financial reporting
  - Cost accounting and cost distribution
  - Capital and expense budgets
  - System development
- Marketing
  - Marketing philosophy of major products
  - Marketing philosophy of by-products
- SECTION 12: COMMUNICATIONS
- Company philosophy
- Organization
- Central database system
- Computer applications support
  - Ore resource management and information
  - Operation production modeling and automation
  - Personnel
  - Management
    - Capital/financial
    - Equipment
    - Communication systems
  - Mine systems support (including GPS)
    - Plant systems support
    - Office systems support
- SECTION 13: SAFETY
- Company philosophy
  - Organization
  - Training
  - Safety protective equipment policies
  - Fire protection
  - Mine fire protection
  - Plant fire protection
  - Other surface area protection
- Interrelationships between operations and safety/health and first-aid clinics
- SECTION 14: SECURITY
- Company philosophy (this section dependent on geographic and political location of operation)
- Organization required
  - Internal organization
  - Contracted organization
  - Function of organization
  - Areas of security concern
    - Mine
    - Plant
    - Transportation of product
    - Other surface facilities
- APPENDICES
- Organization charts
- Condensed job descriptions of all jobs
- Maintenance management control system
- Description and forms
- Business control system description and forms
- Environmental monitoring programs details
- Personnel training module details

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CHAPTER 4.8

Cost Estimating for Underground Mines

Scott A. Stebbins

INTRODUCTION

Estimating the costs of mining is often referred to as an art. Unfortunately, this definition turns many would-be evaluators away because of this understandable misconception. Cost estimating, as with any predictive process, requires an evaluator to envision and quantify future events—in other words it requires one to be creative. A better description is that estimating the costs of mining is a creative endeavor. Fortunately in mining, most of the values that an evaluator must predict either stem from measurable entities, such as the configuration of a deposit, or from well-understood and accepted engineering relationships. In actuality, mine cost estimating is a process of matching values obtained through simple engineering calculations with cost data, a process made easier in recent years thanks to readily available printed and electronic information databases.

Mine cost estimating is also referred to as an art because no widely accepted rigorous approach to the process exists. Unlike the process of estimating costs in the building construction industry, in mining, the process varies noticeably from one evaluation to the next, not only in approach but also in scope.

A complete mine cost estimate cannot be fully detailed in the few pages available here. The information presented in this chapter is primarily aimed at minimizing the intimidation felt by many geologists and engineers when they undertake a cost estimate. The basic premise is that anything can be estimated. And the approach detailed here is one in which more or less complete listings of labor, supply, and equipment requirements are based on information about the deposit and the proposed mine. These listings are then used in conjunction with documented salaries, wages, supply costs, and equipment prices to produce estimates of mine capital and operating expenditures. This method, most often referred to as an abbreviated itemized approach, is much easier than it might initially appear. Although there are several other methods available, including parametric equations, factoring, cost models, and scaling, itemized estimates have the advantage of providing thorough documentation of all of the assumptions and calculations on which the estimated costs are based. As a consequence, the results are much easier to evaluate and adjust, and for this reason, they are more useful. Because they rely on much of the same information required to do a proper job using any of the other methods, evaluators are often surprised to find that engineering-based, itemized estimates can be accomplished with some expedience.

Early in any mine cost estimate, long before the evaluator begins to worry about the cost of a scoop tram, the scope of the evaluation must be determined. To accomplish this, the purpose of the estimate must first be defined. If it will be used to select which one of several deposits should be retained for future exploration expenditures, then the estimate will be less thorough than one used to determine the economic feasibility of a proposed mine or one used to obtain funding for development. Coincidently, the level of information available with regard to deposit specifics also plays a part in determining the scope of the estimate. As the level of information increases, so do the scope of the estimate and the reliability of the results.

Accuracy is a measure of predicted (or measured) value versus actual value. It cannot really be quantified until well after the project is under way and the estimated costs can be compared with the actual expenditures. So, cost estimators instead work more in terms of reliability, which is a measure of the confidence in estimated costs. Reliability is determined by the level of effort involved in the evaluation and by the extent of the available deposit information. Simply, the more information that is available (specifically geologic and engineering information), the greater the reliability of the estimated costs. If an evaluator has a firm grasp on the deposit specifics and works diligently to estimate all the costs associated with development and production, then a highly reliable estimate should result.

Estimators determining the potential economic success of developing a mineral deposit must undertake an iterative process of design and evaluation. After settling on an initial target production rate, the process can be broken down into the following four steps:

1. Design the underground workings to the extent necessary for cost estimating.
2. Calculate equipment, labor, and supply cost parameters associated with both preproduction development and daily operations.

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3. Apply equipment costs, wages, salaries, and supply prices to the cost parameters to estimate associated mine capital and operating costs.
4. Compare estimated costs to the anticipated revenues under economic conditions pertinent to the project (using discounted cash-flow techniques) to determine project viability.

After the estimator evaluates the results, he or she will make adjustments to the design and the production rate as necessary and then repeat the process.

PRELIMINARY MINE DESIGN

The goal of the mine planner is to optimize economic returns from the deposit (or to otherwise achieve the corporate goals of the project's owners). The objectives of evaluators as they design a mine for the purpose of estimating costs is to determine the equipment, labor, and supply requirements both for preproduction development work and for daily operations. The extent to which the evaluator takes the design is important—the process is one of diminishing returns. Roughly speaking, 10% of the engineering required for a complete mine design probably provides the data necessary to estimate 90% of the costs. More detailed final engineering aspects of mine design (such as those needed to ensure adequate structural protection for the workers and sufficient ventilation of the underground workings) seldom have more than a minor impact on the overall mine costs.

At the initial stages of an estimate, the key element is distance. In the preliminary design, engineers need to establish the critical distances associated with access to the deposit, whether by shaft, adit, or ramp. Most of the costs associated with preproduction development are directly tied to the excavations required to access the deposit. The length (or depth) of these excavations, along with their placement, provide several cost parameters directly, such as those needed to determine preproduction consumption of pipe, wire, rail, and ventilation tubing. These distances also provide an indirect path to estimating preproduction consumption values for items such as explosives, drill bits, rock bolts, shotcrete, and timber. And finally, they impact many subsequent calculations that the evaluator must undertake to estimate the required sizes of pumps, ore haulers, hoists, and ventilation fans.

Engineers let the configuration of the deposit and the structural nature of the ore, footwall, and hanging wall dictate the stoping method used to recover the resource. Stoping method selection is discussed in great detail in other chapters of this book.

The underground development openings necessary to access and support the stopes are as important as the stoping method itself. Engineers rely on some basic calculations to estimate the lengths of the drifts, crosscuts, ramps, and raises associated with each stope. After they determine the amount of ore available in the stope, they use those lengths to approximate the daily advance rates needed to maintain the desired ore production rate. While it is true that the use of average rates can be quite misleading (particularly in the first 5 years of operation), for the purposes of estimating costs (in particular in estimating preliminary costs), the overall costs per ton will not change much through the process scheduling of these activities in detail. And while the timing of the costs will have some impact on project economics, the extent to which they are detailed vs. the overall impact on project economics refers back to the statement about diminishing returns in the first paragraph of this section.

In the process of determining stope development requirements, estimators rely on stope models in conjunction with the deposit dimensions. For example, the diagram in Figure 4.8A-1 (Appendix 4.8A) provides the basis of the stope design for the room-and-pillar method. From that basis, the evaluator can then move on to use relationships similar to Example 1 to establish the design parameters for the stope.

Example 1. Stope Design Parameters

1. **Stope length:** The maximum suggested stope length, L_{ms}, is estimated by

$$L_{ms} = [(S_o + S_{hw}) \div 1{,}732{,}000] + 5.4$$

where
S_o = ore strength (equal to the ore compressive strength, kPa, times the ore rock-quality designation, %)
S_{hw} = hanging wall strength (equal to the hanging wall compressive strength, kPa, times the hanging wall rock-quality designation, %)

If the actual deposit length, L_{ad}, is greater than the maximum stope length, then the suggested stope length, L_s, is as follows:

$$L_s = L_{ad} \div \text{rounded integer of } [W_{ad} \div (L_s \times 0.75)]$$

where
L_{ad} = projected deposit length (plan view) ÷ cos(deposit dip)

If the actual deposit length, L_{ad}, is less than maximum stope length, then the stope length, L_s, is equal to the actual deposit length, L_{ad}.

2. **Stope width:** The maximum suggested stope width, W_{ms}, is estimated by

$$W_{ms} = L_s$$

If the deposit width, W_d, is greater than the maximum stope width, W_{ms}, then the suggested stope width, W_s, is calculated as follows:

$$W_s = W_d \div \text{rounded integer of } [W_d \div (W_{ms} \div 0.75)]$$

If the deposit width, W_d, is less than the maximum stope width, W_{ms}, then the suggested stope width, W_s, is equal to the deposit width.

3. **Stope height:** The suggested (vertical) stope height, H_s, is estimated by

$$H_s = T_d \div \cos(D_d)$$

where
T_d = measured deposit thickness
D_d = deposit dip (degrees)

4. **Resource recovery:** The suggested resource recovery, R_r (%), is provided by:

$$R_r = [(S_o + S_{hw}) \div 1{,}055{,}865] + 48.857$$

5. **Pillar size:** The plan view area of the pillars, A_p, is estimated by

$$A_p = \{L_s \times W_s \times [1 - (R_r \div 100)]\} \div 25$$

6. **Pillar width:** The pillar width, W_p, is provided by

$$W_p = W_{pr} \times \sqrt{[A_p \div (W_{pr} \times L_{pr})]}$$

where
W_{pr} (pillar width ratio) = $W_s \div (W_s + L_s)$
L_{pr} (pillar length ratio) = $L_s \div (W_s + L_s)$

7. **Pillar length:** The pillar length, L_p, is provided by

$$L_p = L_{pr} \times \sqrt{[A_p \div (W_{pr} \times L_{pr})]}$$

8. **Face height:** If the stope height, H_s, is greater than 7.6 m, then the estimated face height, H_f, is provided by

$$H_f = H_s \div \text{rounded integer of } (H_s \div 7.6)$$

If the stope height, H_s, is less than 7.6 m, then the estimated face height, H_f, is equal to the stope height, H_s.

9. **Face width:** If $W_s - W_p$ is greater than 12.2 m, then the suggested face width, W_f, is estimated as follows:

$$W_f = (W_s - W_p) \div \text{rounded integer of } [(W_s - W_p) \div (12.2 \times 0.75)]$$

If $W_s - W_p$ is less than 12.2 m, then the suggested face width, W_f, is provided by

$$W_f = W_s - W_p$$

10. **Advance per round:** The suggested advance per round is provided by

$$0.952679 \times [(W_f \times H_f)^{0.371772}]$$

11. **Development requirements (advance per stope for room-and-pillar stopes in deposits that dip less than 25°):**
    - Haulage drifts
      - length = stope length
      - location = ore
    - Haulage crosscuts
      - length = stope width
      - location = ore

Evaluators apply relationships such as those shown in Example 1 to a multitude of deposit configurations to arrive at a stope design. By building similar relationships for any stoping method, they can determine the pertinent associate stope development requirements and, subsequently, the pertinent cost parameters.

Sketches (usually a three-view drawing) of the deposit access headings, stopes, and underground excavations (shops, pump stations, lunch stations, hoist rooms, etc.) provide much of the preliminary mine design information that an evaluator needs for a cost estimate. Only the lengths of the excavations are needed early in the analysis. Values determined by evaluators as they calculate the subsequent cost parameters provide the information necessary to define the cross-sectional areas of these openings.

COST PARAMETERS

Engineers find that the process of defining the parameters necessary for a cost estimate is a wonderful (perhaps only for an engineer) progression of simple mathematical calculations in which one value seems to always lead to, and interconnect with, the next. These calculations branch in ways that create many logical paths to a complete compilation of the needed cost-estimation parameters, but all paths do eventually lead to such a compilation. One generalized path is illustrated in this section. Be aware, however, that most of these procedures are interchangeable, and many paths exist. Also, it is not the intent here to work through a complete step-by-step estimate, because such an example would apply only to a finite number of deposit types. The intent is instead to provide insight into the process that estimators use and to remove some of the mystery that might create a hesitation to proceed. Successful estimators need to show a willingness to suggest values for as-of-yet unknown parameters. For example, if one is working at an operating mine, then it is possible to know all of the required parameters and to calculate, as opposed to estimate, the costs. But for undeveloped projects, it is not possible to know parameters such as the ore and waste powder factors and the amount of water that must be pumped on a daily or hourly basis.

Most (if not all) parameters required for a cost estimate fall into one of three categories in that they define labor, supply, or equipment requirements. These categories represent the items that cost money, that is, the items for which funds must be expended. Consequently, evaluators work in this phase of the estimate to specify the equipment, the supplies, and the work force necessary to mine the deposit. They find that the key to specifying these factors lies in the process of determining how much time (how many hours) it takes to perform the individual tasks of mining.

Operations in an underground mine are, for the most part, either cyclic or continuous in nature, and most are designed to transport materials such as ore, waste, air, water, workers, and supplies. Operations that do not transport materials (such as equipment repair or rock support installation) are typically in place solely to service operations that do.

The rate at which ore is produced provides evaluators a good place to start as they begin to define the cost parameter values. This rate is typically based on the desired life of the mine and the size of the resource.

Resource size is known (or it has at least been approximated, hence the evaluation) and estimators often begin with the following relationship, a variation of which is known as Taylor's rule, to approximate a possible project life:

$$\text{project life, yr} = 0.2 \times \sqrt[4]{\text{resource size, t}}$$

With values for the mine life and the resource known, estimators can then determinate the daily production rate as follows:

$$\text{production rate, t/d} = \text{resource, t} \div [\text{mine life, yr} \times \text{operating schedule, d/yr}]$$

Of course, many factors influence the rate of ore production (such as market conditions, deposit configuration, and profit maximization), so many evaluators use more sophisticated approaches to determine the initial production rate (Tatman 2001). As the evaluation of a project proceeds, however, the production rate

needs to be altered from one iteration to the next as the economic ramifications of each development scenario become clear.

After an initial production rate is determined, evaluators can use it in conjunction with the ore haul distances (gleaned from the mine design) to estimate the capacities of the machines used to collect ore in the stopes, transport it through crosscuts and drifts, and then finally haul it (through an adit, ramp, or shaft) to the surface. The heights and widths of these machines (rear-dump haulers, scoop trams, rail cars, conveyors, etc.) provide the basis for the cross sections of all the openings through which they must travel. Evaluators multiply the products of the heights, widths, and lengths of the openings by the density of the rock through which they pass to determine the amount (in metric tons) of rock that must be removed during their excavation. When they apply a powder factor (kilograms of explosive per metric ton blasted) to this amount, the result is the amount of explosives needed to liberate the rock. As stated earlier, evaluators find that as one design parameter is determined, its value usually provides the information needed to determine many more.

After they determine the size of the haulers, engineers can refer to manufacturer's literature (often available through their Web sites) to ascertain the speeds of those machines in relation to various haul conditions. And with those speeds, they can calculate how many hours the machines need to operate each day to meet production goals and, in turn, the required number of machines and operators. Evaluators rely on cycle-time calculations (Example 2) to supply the basis for most such values, and as such these calculations represent one of the more important concepts of any cost estimate. Cycle-time calculations are used whenever an estimator needs to determine the number of machines required to perform a cyclic operation.

Example 2. Cycle-Time Calculations

Consider a case where a 20-t capacity, articulated rear-dump truck hauls ore to the surface. Ore is placed in the truck by a 6.1-m^3-capacity remotely operated loader near the entrance of the stope. The truck hauls the ore 550 m along a nearly level drift, and then it hauls the ore 1,450 m up a 10% gradient to the surface. After reaching the surface, the truck travels another 200 m to the mill, where the ore is dumped into a crusher feed bin. (If the truck is not loaded to capacity, it is primarily because the capacity of the loader bucket in conjunction with the number of cycle either under- or overloads the truck.)

First, the speeds of the machine must be ascertained over each segment of the haul route. This information is gleaned from technical manuals supplied by equipment manufacturers. In this example, the approximate speeds over the following gradients are as follows.

1. Haul speeds:
   - Up a 10% gradient, loaded ≈ 6.4 km/h
   - Over a level gradient, loaded ≈ 16.1 km/h
   - Over a level gradient, empty ≈ 20.3 km/h
   - Down a 10% gradient, empty ≈ 15.8 km/h

   These values include an allowance for a rolling resistance equivalent to a 3% gradient.
2. Travel times:
   - Haul travel times are

   [550 m ÷ (16.1 km/h × 1,000 m/km)] × 60 min/h = 2.05 min
   [1,450 m ÷ (6.4 km/h × 1,000 m/km)] × 60 min/h = 13.59 min
   [200 m ÷ (16.1 km/h × 1,000 m/km)] × 60 min/h = 1.04 min
   - Return travel times are

   [550 m ÷ (20.3 km/h × 1,000 m/km)] × 60 min/h = 1.63 min
   [1,450 m ÷ (15.8 km/h × 1,000 m/km)] × 60 min/h = 5.51 min
   [200 m ÷ (20.3 km/h × 1,000 m/km)] × 60 min/h = 0.59 min
3. Total travel time:

   2.05 + 13.59 1.04 + 1.63 + 5.51 + 0.59 = 24.41 min

   Evaluators may wish to tune the above estimate further by considering delays attributable to altitude duration, acceleration, and deceleration. However, the effort spent should be proportionate to the purpose of the estimate and the reliability of the available information. Specifically, if acceleration and deceleration were to increase the overall cycle time by 30 seconds (or 2%) but the mill had not been firmly sited (which might change the overall haul distance by as much as 10%), then the effort spent fine-tuning the estimate would be futile because it would do nothing to increase the reliability of the results.

   In addition to travel, the truck's cycle also includes time spent in loading, in dumping the load, and in maneuvering into position for each of these tasks. In this example, one also needs to estimate the cycle time for the loader to figure the amount of time that the hauler spends in the loading portion of its cycle. For this example, it can be assumed that the weight capacity of the loader is 13.44 t and that, for any given load, the bucket is typically 85% full. It can also be assumed that the ore in its blasted condition weighs 2.85 t/m^3. If the round trip from the dump point to the active face and back takes the loader 2.40 min (1.40 min to haul and 1.00 min to return), the loader takes 0.80 min to collect a load of ore and 0.40 min to dump that load, and the truck spends 2.65 min maneuvering and dumping during each cycle, then the following series of calculations provides the time necessary to load the truck as well as the overall cycle times of both vehicles.

   Given a production rate of approximately 4,000 t/d, a shift length of 10 h and a production schedule of about two shifts per day, the following are calculated.
4. Loader volume capacity:

   6.1 m^3/load × 2.85 t/m^3 × 0.85 = 14.78 t/load

5. Loader weight capacity: Because the weight capacity of the loader is 13.44 t, the load is limited by weight.
6. Loader cycle time:
   - Collect load ≈ 0.80 min
   - Haul load ≈ 1.40 min
   - Dump load ≈ 0.40 min
   - Return time ≈ 1.0 min
   - Total cycle time ≈ 3.60 min
7. Truck load time:
   - 20 t ÷ 13.44 t/load = 1.49 loads or two cycles per truck
   - Two cycles per truck × 3.60 min/load = 7.20 min to load truck
8. Total truck cycle time:
   - Load = 7.20 min
   - Travel = 24.41 min
   - Maneuver and dump = 2.65 min
   - Total cycle time = 34.26 min
9. Daily truck productivity:
   - 2 shifts/d × 10 h/shift × 60 min/h = 1,200 min/d
   - (1,200 min/d ÷ 34.26 min/cycle) × 20 t/cycle ≈ 700 t/d

10. Truck requirements:

    4,000 t/d ÷ 700 t/truck = 5.71, or 6 trucks

11. Hourly truck productivity:

    (20 t/cycle ÷ 34.26 min/cycle) × 60 min/h ≈ 35 t/h

12. Daily truck use:

    4,000 t/d ÷ 35 t/h ≈ 114 h/d

Next, evaluators need to determine the work force required to operate the truck fleet. Typically, equipment operators work noticeably less than the total number of hours for which they are paid. After evaluators account for the time workers spend at lunch and on breaks (in addition to time they lose traveling to and from the working face), they find that, on average, about 83% of the operator's time is actually spent in productive activities (the actual value of course varies from one operation to the next). Consequently, the total amount of time for which workers must be paid to achieve the 4,000-t/d production rate is as follows:

114 h/d ÷ 0.83 = 137 h

Because each shift is 10 hours long, the number of truck drivers is determined by

137 h ÷ 10 h/shift = 13.7 or 14 workers

Estimators now need to reexamine the number of trucks that they initially selected. In this case, it is apparent that, after accounting for worker efficiency, more trucks will be needed, that is

14 workers ÷ 2 shifts/d = 7 trucks

Therefore, in examining the trucks as they operate over the designed haul profiles, evaluators approximate values for several key cost parameters. The number of trucks, the number of operators, daily truck use (hours per day), and the number of hours that the drivers must work all become clear. Because the two latter values differ and because each is used to determine a different cost, evaluators also find that each must be estimated separately.

Maintenance-labor requirements can be estimated either as a factor of the number of operating units or, in some cases, as a factor of the number of operators. Supervision and technical staffs can then be determined from the sum of the results.

If evaluators determine the hourly work force properly, they have gone a long way toward ensuring an acceptable level of reliability. Wages often account for more than half the total operating cost, so if the work-force estimate is solid, the cost estimate is probably more than halfway complete. Conversely, the cost of operating underground mining machinery typically represents a far less significant portion of the total underground operating cost. But because the size and configuration of the work force is closely tied to the equipment requirements (and because the equipment purchase costs can be significant), evaluators should strive to properly determine those requirements. The results have a direct impact on the reliability of estimated costs.

Costs parameters for other cyclic operations (drilling, mucking, loading, hauling, hoisting, etc.) can be ascertained in a manner similar to that used in the truck example above. As is evident, cycle-time calculations are not difficult. Evaluators find the task of locating the rates (or speeds) at which machines operate (drill penetration rates, mucker transport speeds, hoist velocities, etc.) much more troublesome, but even this information is often readily available. The most common sources include literature from manufacturers, references such as this handbook, information databases such as *Mining Cost Service* (InfoMine USA 2009b), or statistical compilations such as those contained in the *Mining Source Book* (Scales 2009). Exclusive of these, speeds or rates of advance are very often easy to estimate through observation. And with the machine speed and a bit of imagination, any evaluator can provide a perfectly reasonable estimate of the cost parameters associated with almost every cyclic operation. It may also be useful to maintain a database of advance rates and machine productivity for different situations.

Machines that facilitate a continuous movement of materials (ore, waste, air, water, workers, etc.) are considered noncyclical, and the associated cost parameters can be estimated accordingly. Conveyors, generators, pumps, and ventilation fans fall into this category. In Example 3, a method that can be used to approximate the parameters associated with draining the mine and pumping the water to the surface is presented. This example illustrates the estimation process as it applies to continuous-flow operations.

Example 3. Continuous-Flow Calculations

Consider a case where a mine produces water at a rate of 400 L/min. Common engineering references indicate that a flow rate of roughly 1.0 m/s represents a reasonable value for the velocity of liquid pumped through a conduit. With this information, the following series of calculations provides an estimate of several of the required cost estimation parameters.

1. Pipe diameter:
   - The volume of water that flows through a meter of pipe each second is approximately

     (400 L/min ÷ 60 s/min) ÷ 1 m/s = 6.66 L/m

     6.66 L/m × 0.001 m^3/L = 0.0066 m^3/m

   - The diameter of pipe that enables the desired flow rate is approximately

     cross-sectional area of a pipe = ($\pi \times d^2$) ÷ 4

     $[(0.0066\ m^3 \times 4) \div \pi]^{0.5}$ = 0.092 m or 9.2 cm

   Therefore, the shaft must be fitted with a 9.2-cm inside-diameter pipe to remove water from the mine.

   Although no longer accepted, units of horsepower (hp) provide a useful visualization to cost estimators. The definition of the term (1.0 hp = 33,000 ft·lb/s) incorporates three primary contributors to costs: weight, distance, and time.

   To state that relationship in words, if the weight of the material, the distance that it must be moved, and the speed at which it is moved are all known, then the energy required for the task can be roughly estimated, as can the size of the motor and the subsequent costs. The more accepted unit of watts correctly incorporates mass into the equation, but to provide insight into the estimation

process, the following relationship can be used to approximate power requirements for the drainage problem illustrated earlier.

2. Pump horsepower:
   - The volume of water in the pipe at any one point of time is approximately

     $250\ \text{m} \times 3.281\ \text{ft/m} \times \{[\pi \times (9.2\ \text{cm} \times 0.03281\ \text{ft/cm})^2] \div 4\} = 58.70\ \text{ft}^3$

   - The velocity of the water is approximately

     $1.0\ \text{m/s} \times 3.281\ \text{ft/m} = 3.281\ \text{ft/s}$

   - The weight of the water in the pipe is approximately

     $58.70\ \text{ft}^3 \times 62.4\ \text{lb/ ft}^3 = 3{,}663\ \text{lb}$

   - Therefore, the horsepower required to move the water up the shaft is approximately

     $(3.281\ \text{ft/s} \times 3{,}663\ \text{lb}) \div 550\ \text{ft·lb/s} \approx 22\ \text{hp}$

     $22\ \text{hp} \times 0.7457\ \text{kW/hp} = 16.4\ \text{kW}$

For cost-estimating purposes, evaluators can use this value to approximate the size of a pump and the power required to move the water. However, it is important that evaluators differentiate between actually specifying the equipment—as an engineer would do in the advanced stages of a mine design—and approximating representative parameters. There is, of course, more to selecting a pump and determining the required power than has been illustrated. And pipes come in a limited range of standard diameters. But evaluators must keep in mind the purpose of the estimate and the reliability of the available information.

Well-established equipment selection procedures for items such as pumps are available elsewhere in handbooks such as this, and their use is, of course, encouraged whenever appropriate. But estimators can rely on the basic principles just presented, or variations of those principals, to approximate the cost parameters associated with almost any underground continuous-flow system, whether it is pumping water, conveying ore, blowing ventilation air, or transporting backfill. Each moves a specific weight over a specific distance at a specific speed. The basic premise of the approach presented here is that anything can be estimated. In the early stages of a deposit evaluation, specific requirements for tasks such as draining the mine are sketchy at best, so the evaluator must keep in mind that additional complication does not necessarily lead to additional reliability.

But that is not to say that cost-estimating approaches, such as the one shown in the previous example, are unreliable. For comparison purposes, the actual power required to transport water up a shaft is typically determined by using relationships similar to the following:

$$\text{power} = \text{pressure} \times \text{volume}$$

$$\text{pressure} = \text{fluid density} \times \text{gravity} \times \text{height}$$

Most importantly, the calculations must account for pump and motor efficiency, which is most often in the range of 65% and 75%.

3. Pump horsepower:
   - Flow rate ≈ 400 L/min ≈ 6.667 L/s
   - Pumping height ≈ 250 m
   - Pressure = $1{,}000\ \text{kg/m}^3 \times 9.81\ \text{m/s}^2 \times 250\ \text{m} = 2{,}452.5\ \text{kPa}$
   - Power = 2,452.5 kPa × 6.667 L/s = 16,350 W or 16.35 kW
   - 16.35 kW ÷ 0.7457 kW/hp = 22 hp
   - If the pump efficiency is 68%, then the pump power requirement is approximately 16.35 kW ÷ 0.68 efficiency = 24.4 kW.

In comparing the two relationships and the associated results, evaluators can see that the primary difference is in the pump efficiency value, which of course should be the case unless the head loss due to friction is excessive.

The previous series of calculations provides the size of the drain pump and the diameter of the associated pipe, which are both needed to determine the associated costs. This series also represents a process that estimators can use to determine the cost parameters associated with almost any continuous-flow operation. Of course, the equipment use value (in terms of hours per day) for continuous-flow systems is usually apparent because these types of systems either operate for the entire shift or the entire day.

Evaluators often find it difficult to determine mine ventilation requirements and the associated cost parameters. To determine these parameters, evaluators must have an understanding of both the energy required to move air through the mine and the volume of air that must be moved. Typically, the nature (length, perimeter, and roughness) of the openings that provide access to the deposit are examined along with the same qualities of the stopes to approximate the energy required to deliver air to the underground workings. Flow rates are based on the number of workers, the amount of air required to dilute diesel fumes, and any volume losses through rock structures or abandoned workings. The energy and volume values, when considered along with the natural ventilation properties of the designed workings, are then used to approximate the size and horsepower requirements for the fans, or in other words, the parameters needed to estimate costs.

Ventilation calculations are really beyond the scope of most early-stage feasibility studies. However, they must be considered in a cost estimate because they can represent the one item that can profoundly change the size (or number) of the deposit access openings. If ventilation horsepower requirements are excessive (for instance, if large volumes of air must be pushed through a very limited number of small openings), then operating costs increase dramatically, and either the size of the openings should be enlarged or there need to be more openings. Total ventilation requirements can be estimated quite reliably using curves based on the annual tonnage mined and whether or not the mine is fully diesel mechanized.

After most of the equipment and associated labor parameters have been approximated, evaluators need to begin to determine supply consumption. Evaluators may notice that the equipment operating parameters provide the basis for consumption rates of several supplies, such as diesel fuel, electricity, repair parts, lubricants, and tires. However, it is critical to note that the more popular cost services, such as *Mine and Mill Equipment Operating Costs Estimator's Guide* (InfoMine USA

2009a) and the *Cost Reference Guide* (EquipmentWatch 2009) include these as equipment operating costs. Evaluators are cautioned not to include these values twice in their estimates.

When engineers define supply cost parameters, they find that the explosives consumption rate is a good place to start. For estimating purposes, the amount of explosives consumed is reflected in the powder factor. At operating mines, powder factors are determined through trial and experience and are dynamic. Because such information does not exist for a proposed operation, a historic value from a mine that relies on a similar stoping method to recover comparable rock provides a reasonable starting point. Powder factors at operating mines can be found in case studies contained in handbooks such as this or in statistical compilations such as the *Mining Source Book* (Scales 2009). In addition to the explosives consumed in the stopes, estimators also need to consider that explosives are consumed in each development heading and that powder factors will vary from one to the next in relation to the face area, the configuration of the opening, and the rock characteristics. Powder factors for blasts in the development headings are typically higher than those for the production blasts in the stopes.

Estimators can use the explosives consumption rate (typically reported in terms of kilograms of explosive per metric ton blasted) in conjunction with the density of the explosive, the diameter of the blastholes, and a hole-loading factor to approximate the drilling requirements (in terms of meters drilled per day). They can then divide that value by a drill penetration rate to arrive at drill use (in terms of hours per day) and, subsequently, the minimum required number of drills and drillers. Estimators can also use drilling requirements to approximate the number of blasting caps and the lengths of fuse consumed each day. And when viewed in relation to drill bit and steel wear rates, the daily drilling requirements can also provide the consumption rates for these supplies.

Consumption rates for many of the supplies needed at a mine are directly tied to the advance rates of the development openings in which they are placed. The following are obviously used up at a rate that mimics the rates of advance of the associated development openings (both prior to and during production): compressed air pipe, freshwater pipe, drainage pipe, electrical cable, ventilation tubing, and rail. Rock bolt, timber, shotcrete, and rock bolt matt requirements also vary in proportion to the advance rates of the development openings.

Evaluators quickly notice that the tasks of estimating the consumption rates for supplies and the daily use requirements for the mining machinery are not difficult. They do, however, often find it difficult to fully understand and incorporate all of the implications of the differing specifications for all these supplies and machines (tapered vs. rope-threaded drill rods, ANFO vs. emulsion explosives, friction-set vs. resin-set rock bolts, hydraulic vs. pneumatic drifters, etc.). Although a full knowledge of all of these implications is not necessary for a reliable estimate, engineers who fully understand them will most certainly have increased confidence in their results.

COST ESTIMATES

After an evaluator has established all of the cost parameters, the estimation process is one of simple calculations and tabulations. Because most costs, both capital and operating, are tied to average daily equipment use, supply consumption, wages, or salaries, estimators from this point forward need only identify the most reliable source of cost information, apply the costs to the previously derived parameters, and then tabulate the results. This process is well demonstrated in the cost models found in Appendices 4.8A through 4.8C. (Daily equipment use varies greatly according to the mine development schedule, so the estimate must be based on "snapshots" of the schedule at representative times; that is, this snapshot is the goal.)

Evaluators calculate equipment operating costs by multiplying use (in terms of hours per day) by the hourly operating costs for the machine, which they typically glean from sources that include the *Mine and Mill Equipment Operating Cost Estimator's Guide* (InfoMine USA 2009a) and the *Cost Reference Guide* (EquipmentWatch 2009). Costs from these sources are categorized as to repair parts and labor, fuel, electricity, lubricants, tires, and ground-engaging components (bucket teeth, tracks, etc.). Those preproduction development costs associated with machine use are really just summations of equipment operating costs over the period of time needed to excavate the development openings. When the mine is in production, daily equipment use varies according to the mine development schedule, so estimates should be based on a representative "snapshot" (or series of snapshots).

Labor costs are determined in a similar manner. To arrive at daily labor costs, estimators need only multiply the number of workers assigned to any one discipline by the number of hours worked per shift and then multiply the result by the associated hourly wage (factored for burden). Wages from mines found throughout North America can be found in labor surveys published by InfoMine USA. The factors shown in Table 4.8-1, when applied to average wages for the United States, are sometimes used to roughly estimate wages in other parts of the world.

These factors are based on mandated minimum wages in the respective countries, and as a consequence, they provide only a rough guideline and should be used with some caution. As an evaluation progresses, estimators should attempt to gather actual salary and wage data for the region in which the project is located. A case can be made that that labor efficiency is proportional to wage rates, so that more people are required to achieve the same result in lower-wage environments. Consequently, lower wage rates rarely result in proportionally lower operating costs. Wages must be factored for the additional expenses incurred by the employer for each employee.

These expenses, commonly referred to as burden, include contributions to Social Security taxes, worker's compensation and unemployment insurance, retirement plans, and medical benefit packages. Additionally, evaluators must factor either the wages or the work force to account for the expenses associated with vacation and sick leave, shift differential allowances, and overtime pay. Publications available from InfoMine USA contain extensive details of the costs of these benefits at more than 300 active operations. Estimators calculate costs for salaried workers in a manner similar to those that they use for hourly workers, and the sources for salaries are the same as those for wages.

Finally, evaluators calculate supply costs by multiplying daily consumption rates by the prices of the consumables. These are typically gleaned from individual vendors or from *Mining Cost Service* (InfoMine USA 2009b). As with those associated with equipment operation and labor, the expenses associated with supply consumption contribute both to preproduction development and to operating costs. Estimators need to tally the costs of items such as pipe, rail, ventilation tubing, electric cable, rock bolts, and shotcrete for each development

Table 4.8-1 Adjustments for wages worldwide

Country	Percentage of Average U.S. Wage
Australia	141.1
Brazil	29.4
Cambodia	13.9
Chile	30.3
Croatia	55.5
Czech Republic	55.2
Ecuador	41.3
Indonesia	8.1
New Zealand	131.6
Peru	34.8
Russia	14.4
South Africa	22.8
Sri Lanka	12.1
Thailand	18.7
Venezuela	36.7
Zambia	9.4

opening, both before and during production, along with expenses of drill bits, explosives, caps, and fuses.

Up to this point, costs have been estimated in terms of dollars per day. The utility of this approach now becomes apparent. Operating costs are most often reported in terms of dollars per ton of ore, and capital costs are typically reported as annual expenditures. To report operating costs in the appropriate terms, evaluators need only divide the sum of the daily operating costs by the total amount of ore mined each day. For capital costs, evaluators can simply multiply the daily costs for a specific task by the number of days it takes to complete that task (for instance, the number of days needed to complete an adit) or, if the task takes more than a year to complete, by the number of days spent on the task each year.

Operating costs typically include a miscellaneous allowance for expenses too small or too numerous to list separately, or for expenses associated with unscheduled and unanticipated tasks. Evaluators sometimes account for such uncertainties by always faulting to the generous side when they calculate each cost estimation parameter. However, it is preferable to include and list the allowance as one separate value so that those who rely on the estimate can judge its impact for themselves.

Capital costs should include a contingency fund. As opposed to the function of the miscellaneous allowance that was included with the operating costs, the contingency fund is an actual expense that represents an account set aside for any additional, unforeseen costs associated with unanticipated geologic circumstances or engineering conditions. The contingency fund is not in place to cover inadequacies in the cost estimate or failings in the mine design, but the amount of the fund is typically proportional to the amount of engineering that has gone into the project. The money is almost always spent.

Evaluators also need to account for several other expenses in the capital-cost tabulation. These include costs associated with efforts expended on project feasibility, engineering, planning, construction management, administration, accounting, and legal services. For lack of better information, estimators commonly factor values for these from the overall (equipment purchase plus preproduction development) capital cost. A variety of sources report an equivalent variety of factor values, but some of the more commonly used factors include the following:

- Feasibility, engineering, and planning: approximately 4% to 8%
- Construction supervision and project management: approximately 8% to 10%
- Administration, accounting, permitting, and legal services: approximately 8% to 14%

As an alternative, evaluators can base these values on estimates of the time spent on each in conjunction with the salaries of the suitable personnel because most of the expenses are attributable to their work (along with the associated office overhead). However, many of these preproduction tasks are often outsourced, and if such is the case, the associated expenses should be adjusted accordingly.

To permit a mine, engineers are typically required to submit the results of much of the work that they undertake during the feasibility, engineering, and planning process to the appropriate permitting agencies. Estimators are cautioned not to include these expenses twice in their evaluations, once as part of the feasibility, engineering, and planning cost and again as part of the permitting cost.

ECONOMIC EVALUATION

To determine the economic viability of a proposed mine, evaluators must compare estimated costs to anticipated revenues under the economic conditions linked to the project (taxes, royalties, financing, etc.). As mentioned previously, costs are categorized as either capital or operating so that they may receive the appropriate treatment in an after-tax analysis. Operating costs are those that can be directly expensed against revenues as they accrue and include funds that an organization spends operating the equipment, purchasing supplies, and paying wages and salaries. Capital costs are those that cannot be fully expensed in the year incurred and include items such as the following:

- Exploration
- Property acquisition
- Engineering and construction management
- Mine and mill equipment purchase
- Infrastructure
- Preproduction development
- Buildings
- Contingency fund
- Working capital
- Postproduction reclamation

Estimators categorize operating costs in several ways. Production-oriented evaluators are typically most comfortable with results that reflect costs in terms of dollars per unit of development (e.g., dollars per meter of drift) or dollars per unit of production (e.g., dollars per metric ton mined). Because operators primarily write checks to the supply vendors, the equipment manufacturers, or the workers (wages and salaries), many evaluators prefer to see costs broken down accordingly. The choice is really just a matter of preference tempered with intended use. Because most early-stage economic evaluations intend only to estimate overall operating costs, the breakdown is not critical, only the results.

The process of an after-tax discounted-cash-flow economic evaluation is beyond the scope of this discussion. Reliable results are based on many factors in addition to the

estimated costs. Project revenues, for instance, are not simply the product of the commodity price, the production rate, and the resource grade. The recovered grade, for instance, must be factored for losses and dilution at the mine, and for concentration inefficiencies at the mill. Charges that the operator must pay for smelting and refining must be considered, as must penalties for deleterious minerals. Federal and state income taxes, as well as sales, property, and severance taxes, reduce anticipated revenues. And if operators rely on external financing to back their project or if royalties must be paid to partners, property owners, or other entities, then project economics are further diminished.

In closing, one should keep the estimate in perspective. There is no way to exactly predict the costs of a proposed mine, and all evaluators know that their estimate will ultimately be proven wrong. However, evaluators must do their best to minimize the extent to which their estimated values differ from the actual project costs.

UNDERGROUND MINE COST MODELS

Appendices 4.8A through 4.8C present three cost models that evaluators can use to make preliminary, order-of-magnitude estimates for projects for which there is limited deposit information. These models are based on theoretical engineering parameters and do not represent any specific mine. They include the following techniques: room-and-pillar mining, block-cave mining, and mechanized cut-and-fill mining.

Engineers do not rely on models to make significant economic decisions. A cost model, no matter how carefully the estimator prepares it, is only a representation of a hypothetical set of resource parameters and cannot be expected to represent costs for a specific deposit with the degree of reliability necessary for investment. Models can, however, be quite useful as comparative tools, and evaluators often rely on them to establish cutoff grades for preliminary reserve estimates.

The figures in the appendices are idealized sketches of the stope layouts for each model.

Model Construction

The models presented in Appendices 4.8A through 4.8C were developed by evaluating sets of hypothetical resource parameters using standard engineering-based cost-estimating techniques (such as those described in the preceding paragraphs) to approximate capital and operating costs for underground mine designs based on specific deposit parameters. Some of the selected salary, wage, and supply costs on which the program relies are listed in Tables 4.8-2 through 4.8-4. These are the most recent values from *Mining Cost Service* (InfoMine USA 2009b).

Cost estimates for the modeled projects list all of the labor, material, supply, and equipment operating expenses accrued at the mine site, including those associated with supervision, administration, and on-site project management. Also listed are the costs of purchasing and, if necessary, installing all of the necessary machinery, as well as those associated with preproduction development work and constructing the surface facilities. Costs not included in the estimates are as follows:

- Exploration
- Off-site roads, power lines, or railroads
- Taxes (except sales tax)
- Depreciation
- Off-site product transport

Table 4.8-2 Annual salaries for professionals (2009 dollars)

	Annual Salary, US$	
Job Title	**Small Mines**	**Large Mines**
Mine manager	93,300	153,000
Superintendent	72,000	104,700
Foreman	62,000	71,500
Engineer	75,500	83,100
Geologist	62,500	71,000
Shift boss	54,000	66,000
Technician	40,000	50,000
Accountant	52,000	60,800
Purchasing agent	60,000	64,800
Personnel manager	68,000	99,900
Secretary	30,000	36,200
Clerk	32,000	35,800

Source: Salzer 2009.

Table 4.8-3 Hourly wages for workers (2009 dollars)

	Hourly Wage, US$	
Worker	**Small Mines**	**Large Mines**
Stope miner	23.50	24.00
Development miner	23.50	25.00
Equipment operator	21.69	21.30
Hoist operator	18.02	21.70
Locomotive operator	16.00	19.00
Support miner	22.00	22.25
Utility operator	19.52	19.17
Exploration driller	19.25	20.20
Crusher operator	22.00	22.90
Backfill plant operator	18.70	18.91
Mechanic	19.64	21.88
Electrician	24.38	23.18
Maintenance worker	16.25	18.90
Helper	16.00	17.28
Underground laborer	15.70	18.30
Surface laborer	14.00	16.40

Source: Salzer 2009.

Table 4.8-4 Supply prices (2009 dollars)

Item	**Price per Unit, US$**
Emulsion explosives (cap sensitive)	3.11/kg
Watergel explosives (non-cap sensitive)	1.52/kg
Primers (0.23/kg)	3.49 each
Blasting caps (nonelectric, 3.65-meter lead)	1.97 each
Fuse	0.814/m
Diesel fuel	0.719/L
Lubricants	2.171/L
Cement	112.36/t
Electricity	0.110/kW·h
Timber	300.00/m^3
Lagging	254.24/m^3
Steel	4.24/kg

Source: InfoMine USA 2009b.

- Overtime labor costs
- Milling, smelting, and refining costs
- Permitting
- Home office overhead
- Insurance
- Town site construction and operation
- Incentive bonus premiums
- Sales expenses
- Interest expense

Each modeled mine includes at least two routes of access to the deposit. For mine models that produce less than 4,000 t/d through a single shaft, a secondary access raise provides emergency egress and completes the ventilation circuit.

In the models, ore and waste rock densities are 0.367 and 0.401 m^3/t, respectively. Ore swells to 155% of its in-place volume on excavation, and waste swells to 145% of its in-place volume. Rock-quality designations and compressive strengths vary from one model to the next. Values for several methods from a variety of mines are listed in Table 4.8-5.

Preproduction development work blocks out enough ore to initiate operations at the design production rate. And the level of production development work is designed to maintain that rate throughout the life of the mine. All shop, office, worker changehouse, warehouse, and mine plant buildings are constructed on the surface. Working capital allows for 2 months of project operation, and a sales tax rate of 6.75% is applied to all equipment and nonfuel supply purchases. Capital costs do not include the expenditures associated with outside contractors, infrastructure, home office overhead, insurance, or project startup (except working capital). Costs are in late 2008 and early 2009 dollars.

Unit Costs

Wages and salaries used in the models represent U.S. national averages as reported in *U.S. Metal and Industrial Mineral Mine Salaries, Wages and Benefits, 2009 Survey Results* (Salzer 2009). In keeping with the results of that survey, lower wages and salaries are used for the smaller mines, and higher wages and salaries are used for larger mines. In the models, the cutoff point between small and large mines is set at 100 employees. Equipment and supply prices are, for the most part, taken from *Mining Cost Service* (InfoMine USA 2009b).

In the models, the salaries shown in Table 4.8-2 are adjusted upward to account for a 38.0% burden rate at the small mines and a 44.0% burden rate at the larger mines. Hourly wages used in the cost models are shown in Table 4.8-3. The wages shown in Table 4.8-3 are adjusted upward to account for a 38.0% burden rate at the small mines and a 44.0% burden rate at the larger mines. Supply prices used in the models are shown in Table 4.8-4.

Table 4.8-5 Rock characteristics

Mining Method	Rock Quality Designation, %	Rock Compressive Strength, kPa
Cut-and-fill models		
Ore	50	68,950
Waste	35	51,700
Shrinkage model		
Ore	65	103,425
Waste	80	172,375
End-slice model		
Ore	75	137,900
Waste	80	172,375
VCR model		
Ore	75	137,900
Waste	80	172,375
Room-and-pillar model		
Ore	75	155,135
Waste (footwall)	65	137,900
Waste (hanging wall)	55	120,660
Sublevel long-hole model		
Ore	55	82,700
Waste	75	137,900
Block-cave model		
Ore	65	103,420
Waste	65	103,420
Sublevel cave model		
Ore	55	82,740
Waste	45	68,950

ACKNOWLEDGMENTS

The author thanks Otto Schumacher, founder of Western Mine Engineering, Inc., and developer of *Mining Cost Service*, for his unmatched contribution to the craft of mine cost estimating and for years of sage advice.

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APPENDIX 4.8A
COST MODELS FOR ROOM-AND-PILLAR MINING

These models represent mines on flat-lying bedded deposits that are 2.5, 5.0, or 10 m thick, respectively, with extensive areal dimensions. Access is by two shafts that are 281, 581, or 781 m deep and a secondary access/vent raise. Ore is collected at the face using front-end loaders and loaded into articulated rear-dump trucks for transport to a shaft. Stoping follows a conventional room-and-pillar pattern, with drilling accomplished using horizontal drill jumbos. A diagram of the development requirements for room-and-pillar mining is shown in Figure 4.8A-1, and cost models are shown in Table 4.8A-1.

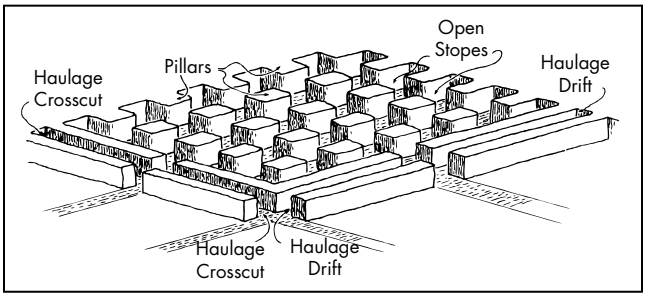


Figure 4.8A-1 Development requirements for room-and-pillar mining

Table 4.8A-1 Cost models for room-and-pillar mining

	Daily Ore Production, t		
Cost Parameters	**1,200**	**8,000**	**14,000**
Production			
Hours per shift	8	8	8
Shifts per day	3	3	3
Days per year	350	350	350
Deposit			
Total mineable resource, t	5,080,300	43,208,000	86,419,000
Dip, degrees	5	5	5
Average maximum horizontal, m	1,000	2,000	2,400
Average minimum horizontal, m	700	1,500	1,500
Average thickness, m	2.5	5.0	10
Stopes			
Stope length, m	59	59	60
Stope width, m	43.5	44.4	45.3
Stope height, m	2.5	5.0	10.0
Face width, m	4.3	4.4	4.5
Face height, m	2.5	5.0	10.0
Advance per round, m	2.3	3.0	3.9
Pillar length, m	6.9	6.9	7.0
Pillar width, m	5.1	5.2	5.3
Pillar height, m	2.5	5.0	10.0
Development openings			
Shafts			
Face area, m^2	15.1	33.4	39.1
Preproduction advance, m	281	581	781
Cost, shaft 1, $/m	9,760	15,430	14,520
Cost, shaft 2, $/m	9,800	15,490	14,570

(continues)

Table 4.8A-1 Cost models for room-and-pillar mining (continued)

	Daily Ore Production, t		
Cost Parameters	**1,200**	**8,000**	**14,000**
Drifts			
Face area, m^2	12.5	17.8	19.9
Daily advance, m	6.1	20.0	17.2
Preproduction advance, m	490	1,748	1,501
Cost, $/m	1,130	1,310	1,410
Crosscuts			
Face area, m^2	12.5	17.8	19.9
Daily advance, m	4.5	15.0	12.9
Preproduction advance, m	360	1,311	1,125
Cost, $/m	1,060	1,220	1,310
Ventilation raises			
Face area, m^2	3.9	16.3	27.2
Daily advance, m	0.3	1.53	2.19
Preproduction advance, m	250	550	750
Cost, $/m	880	1,750	1,800
Hourly labor requirements, workers/day			
Stope miners	16	56	96
Development miners	12	24	24
Equipment operators	2	6	14
Hoist operators	8	8	12
Support miners	2	2	2
Diamond drillers	2	4	6
Electricians	5	7	8
Mechanics	12	25	36
Maintenance workers	5	14	19
Helpers	5	14	21
Underground laborers	6	18	25
Surface laborers	5	14	19
Total hourly personnel	**80**	**192**	**282**
Salaried personnel requirements, workers			
Managers	1	1	1
Superintendents	2	4	4
Foremen	4	10	21
Engineers	2	5	7
Geologists	2	6	8
Shift bosses	6	16	27
Technicians	4	10	14
Accountants	2	5	7
Purchasing	3	8	11
Personnel managers	4	10	14
Secretaries	5	14	19
Clerks	6	18	25
Total salaried personnel	**41**	**107**	**158**
Supply requirements, daily			
Explosives, kg	959	5,975	10,208
Caps, no.	389	1,591	1,582
Boosters, no.	357	1,497	1,510
Fuse, m	1,529	6,643	7,773
Drill bits, each	8.65	43.59	58.02
Drill steel, each	0.62	3.15	4.19
Freshwater pipe, m	10.6	35.0	30.1

(continues)

Table 4.8A-1 Cost models for room-and-pillar mining (continued)

	Daily Ore Production, t		
Cost Parameters	**1,200**	**8,000**	**14,000**
Compressed air pipe, m	10.6	35.0	30.1
Electric cable, m	10.6	35.0	30.1
Ventilation tubing, m	10.6	35.0	30.1
Rock bolts, each	61	309	455
Buildings			
Office, m^2	1,047	2,734	4,037
Changehouse, m^2	929	2,230	3,275
Warehouse, m^2	269	657	748
Shop, m^2	536	1,409	1,614
Equipment requirements, number and size			
Stope drills, cm	5 each 3.49	17 each 4.13	20 each 5.72
Stope front-end loaders, m^3	5 each 1.1	16 each 1.1	18 each 1.5
Stope rear-dump trucks, t	1 each 15.0	4 each 35.0	6 each 35.0
Development drills, cm	3 each 3.49	6 each 3.81	5 each 4.13
Development front-end loaders, m^3	2 each 1.1	4 each 1.1	3 each 1.5
Development rear-dump trucks, t	2 each 15.0	4 each 35.0	3 each 35.0
Raise borers, m	1 each 2.4	1 each 3.7	1 each 3.7
Production hoists, cm	2 each 152	2 each 203	2 each 305
Rock bolters, cm	1 each 3.81	1 each 3.81	1 each 3.81
Freshwater pumps, hp	4 each 0.5	4 each 0.5	4 each 0.5
Drain pumps, hp	8 each 25	14 each 164	18 each 288
Service vehicles, hp	7 each 82	20 each 210	28 each 210
ANFO loaders, kg/min	2 each 272	5 each 272	6 each 272
Ventilation fans, cm	1 each 122	1 each 244	1 each 274
Exploration drills, cm	1 each 4.45	1 each 4.45	1 each 4.45
Equipment costs, $/unit			
Stope drills	1,041,000	1,041,000	1,043,800
Stope front-end loaders	102,900	102,900	111,200
Stope rear-dump trucks	291,900	548,200	548,200
Development drills	702,000	1,041,000	1,041,000
Development front-end loaders	102,900	102,900	111,200
Development rear-dump trucks	291,900	548,200	548,200
Raise borers	4,180,500	6,737,100	6,737,100
Production hoists	1,171,900	2,047,200	3,508,000
Rock bolters	690,000	690,000	925,000
Freshwater pumps	15,000	59,900	82,500
Drain pumps	7,200	7,200	7,200
Service vehicles	270,000	378,200	293,900
ANFO loaders	41,600	41,600	41,600
Ventilation fans	113,300	184,100	184,100
Exploration drills	72,000	72,000	72,000

(continues)

Table 4.8A-1 Cost models for room-and-pillar mining (continued)

	Daily Ore Production, t		
Cost Parameters	**1,200**	**8,000**	**14,000**
Cost Summary			
Operating costs, $/t ore			
Equipment operation	2.86	2.74	3.14
Supplies	7.42	4.96	3.47
Hourly labor	14.96	7.32	4.87
Administration	7.70	3.33	2.81
Sundries	3.29	1.83	1.43
Total operating costs	**36.23**	**20.18**	**15.72**
Unit operating cost distribution, $/t ore			
Stopes	8.22	6.47	5.50
Drifts	4.86	2.16	1.02
Crosscuts	3.58	1.63	0.77
Ventilation raises	0.16	0.27	0.22
Main haulage	3.25	1.63	1.91
Services	5.92	3.03	2.55
Ventilation	0.16	0.10	0.06
Exploration	0.38	0.15	0.10
Maintenance	0.76	0.47	0.29
Administration	5.65	2.44	1.87
Miscellaneous	3.29	1.83	1.43
Total operating costs	**36.23**	**20.18**	**15.72**
Capital costs, total dollars spent			
Equipment purchase	19,759,200	52,562,200	60,707,100
Preproduction underground excavation			
Shaft 1	2,738,000	8,964,200	11,346,600
Shaft 2	2,754,000	9,000,200	11,379,500
Drifts	554,900	2,283,200	2,120,100
Crosscuts	379,500	1,595,400	1,477,900
Ventilation raises	221,100	962,500	1,347,700
Surface facilities	2,463,300	5,336,300	6,986,100
Working capital	2,319,400	9,419,600	12,839,700
Engineering and management	3,753,100	10,491,500	12,397,400
Contingency	2,887,000	8,070,400	9,536,500
Total capital costs	**37,829,500**	**108,685,500**	**130,138,600**

Source: Data from InfoMine USA 2009b.

APPENDIX 4.8B
COST MODELS FOR BLOCK-CAVE MINING

These models represent mines on large, bulk deposits, roughly 450, 525, and 600 m to a side. Access is through three to five shafts that are 430, 530, or 630 m deep and by secondary access/ventilation raises. Ore is collected using slushers, and haulage from the stopes is by diesel locomotive. Stope development includes driving drifts (haulage, slusher, and undercut) and raises (stope draw, orepass, and boundary weakening). Caving is initiated by blasting on the undercut level. A diagram of the development requirements for block-cave mining is shown in Figure 4.8B-1, and cost models are shown in Table 4.8B-1.

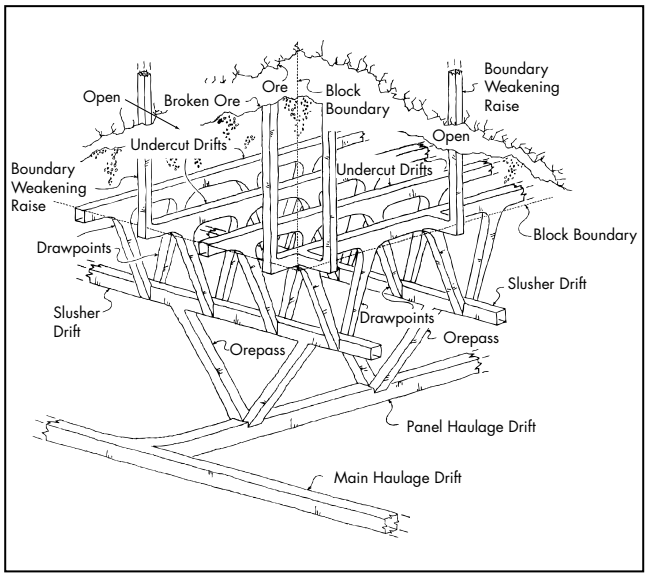


Figure 4.8B-1 Development requirements for block-cave mining

Table 4.8B-1 Cost models for block-cave mining

	Daily Ore Production, t		
Cost Parameters	**20,000**	**30,000**	**45,000**
Production			
Hours per shift	8	8	8
Shifts per day	3	3	3
Days per year	365	365	365
Deposit			
Total mineable resource, t	84,000,000	147,000,000	252,000,000
Average maximum horizontal, m	450	500	600
Average minimum horizontal, m	450	500	600
Average vertical, m	150	175	250
Blocks			
Block length, m	150	165	200
Block width, m	150	165	200
Block height, m	150	175	250

(continues)

Table 4.8B-1 Cost models for block-cave mining (continued)

	Daily Ore Production, t		
Cost Parameters	**20,000**	**30,000**	**45,000**
Development openings			
Shafts			
Face area, m^2	43.2	48.4	54.2
Preproduction advance, m	1,290	2,120	3,150
Cost, shaft 1, $/m	14,580	15,740	16,970
Cost, shaft 2, $/m	14,630	15,790	17,030
Cost, shaft 3, $/m	14,670	15,830	17,050
Cost, shaft 4, $/m	—	15,880	17,080
Cost, shaft 5, $/m	—	—	17,110
Drifts			
Face area, m^2	9.6	9.6	9.9
Daily advance, m	2.6	3.1	3.4
Preproduction advance, m	1,350	1,800	2,400
Cost, $/m	1,480	1,510	1,520
Crosscuts			
Face area, m^2	9.6	9.6	9.9
Daily advance, m	3.4	4.3	4.9
Preproduction advance, m	1,800	2,400	3,400
Cost, $/m	1,430	1,410	1,470
Drawpoints			
Face area, m^2	100	100	100
Daily advance, m	2.3	3.0	3.1
Preproduction advance, m	1,220	1,615	2,160
Cost, $/m	4,780	5,520	5,740
Orepasses			
Face area, m^2	18.8	27.9	41.5
Daily advance, m	0.91	1.22	1.41
Preproduction advance, m	480	700	990
Cost, $/m	1,790	2,510	3,790
Boundary raises			
Face area, m^2	13.6	19.3	27.8
Daily advance, m	11.4	15.7	18.6
Preproduction advance, m	6,000	8,400	13,000
Cost, $/m	1,140	1,240	1,750
Ventilation raises			
Face area, m^2	38.1	56.3	83.7
Daily advance, m	0.19	0.20	0.21
Preproduction advance, m	400	455	600
Cost, $/m	2,520	3,730	5,750
Hourly labor requirements, workers/day			
Undercut miners	66	96	144
Development miners	30	42	50
Motormen	5	6	8
Hoist operators	18	24	30
Support miners	4	4	4
Diamond drillers	6	10	18
Electricians	7	8	9
Mechanics	28	36	46
Maintenance workers	24	30	38

(continues)

Table 4.8B-1 Cost models for block-cave mining (continued)

Cost Parameters	Daily Ore Production, t 20,000	30,000	45,000
Helpers	17	24	33
Underground laborers	30	38	48
Surface laborers	24	30	38
Total hourly personnel	**259**	**348**	**466**
Salaried personnel requirements, workers			
Managers	1	1	1
Superintendents	4	4	4
Foremen	30	42	63
Engineers	8	10	12
Geologists	9	11	14
Shift bosses	21	27	39
Technicians	16	20	24
Accountants	9	11	14
Purchasing	14	17	22
Personnel managers	13	17	23
Secretaries	24	30	38
Clerks	30	38	48
Total salaried personnel	**179**	**228**	**302**
Supply requirements, daily			
Explosives	545	726	754
Caps	429	595	740
Boosters	412	575	717
Fuse	3,407	4,923	6,913
Drill bits	18.70	26.78	35.87
Drill steel	1.042	1.481	1.915
Freshwater pipe	17	23	27
Compressed air pipe	17	23	27
Electric cable	17	23	27
Ventilation tubing	17	23	27
Rock bolts	30	37	48
Shotcrete	1	1	1
Concrete	5	10	17
Rail	12	15	17
Buildings			
Office	4,573	5,825	7,515
Changehouse	2,961	3,983	5,342
Warehouse	369	462	597
Shop	761	971	1,274
Mine plant	222	265	265

(continues)

Table 4.8B-1 Cost models for block-cave mining (continued)

Cost Parameters	Daily Ore Production, t 20,000	30,000	45,000
Equipment requirements, number and size			
Undercut drills	14 each 4.76	20 each 5.08	29 each 5.72
Production slushers	13 each 213.4	18 each 213.4	27 each 213.4
Horizontal development drills	2 each 3.17	2 each 3.17	3 each 3.49
Vertical development drills	5 each 4.76	7 each 5.08	8 each 5.72
Development muckers	3 each 0.3	5 each 0.3	5 each 0.3
Locomotives	3 each 31.8	5 each 31.8	5 each 31.8
Production hoists	3 each 3,176	4 each 3,979	5 each 5,518
Rock bolt drills	3 each 3.81	3 each 3.81	3 each 3.81
Shotcreters	1 each 53	1 each 53	1 each 53
Freshwater pumps, hp	6 each 0.5	8 each 0.5	10 each 0.5
Drain pumps, hp	9 each 593	16 each 550	20 each 781
Service vehicles, hp	16 each 210	24 each 210	33 each 210
Compressors, m^3/min	1 each 142	1 each 227	1 each 227
Ventilation fans, cm	1 each 152	1 each 183	1 each 213
Exploration drills, cm	1 each 4.4	2 each 4.4	3 each 4.4
Equipment costs, $/unit			
Undercut drills	7,680	7,680	7,680
Production slushers	82,000	82,000	82,000
Horizontal development drills	702,000	702,000	702,000
Vertical development drills	8,300	8,300	8,300
Development muckers	65,000	65,000	65,000
Locomotives (with cars)	1,404,000	1,404,000	1,404,000
Production hoists	1,825,000	1,845,600	2,313,900
Rock bolt drills	7,680	7,680	7,680
Shotcreters	81,800	81,800	81,800
Freshwater pumps	128,100	128,100	155,700
Drain pumps	7,200	7,200	7,200
Service vehicles	378,200	378,200	378,200
Compressors	149,200	149,200	209,700
Ventilation fans	113,300	113,300	113,300
Exploration drills	72,000	72,000	72,000

(continues)

Table 4.8B-1 Cost models for block-cave mining (continued)

Cost Parameters	Daily Ore Production, t 20,000	30,000	45,000
Cost Summary			
Operating costs, $/t ore			
Equipment operation	1.65	1.91	2.19
Supplies	1.02	0.82	0.81
Hourly labor	3.41	3.43	3.16
Administration	1.88	1.64	1.43
Sundries	0.80	0.78	0.76
Total operating costs	**8.76**	**8.58**	**8.35**
Unit operating cost distribution, $/t ore			
Stopes	1.07	1.03	0.98
Drifts	0.24	0.18	0.14
Crosscuts	0.32	0.26	0.21
Drawpoints	0.31	0.28	0.20
Boundary raises	0.78	0.77	0.77
Orepasses	0.07	0.08	0.07
Ventilation raises	0.02	0.02	0.02
Main haulage	1.32	1.31	1.48
Services	2.07	2.34	2.41
Ventilation	0.01	0.01	0.00
Exploration	0.07	0.08	0.10
Maintenance	0.25	0.21	0.18
Administration	1.43	1.23	1.03
Miscellaneous	0.80	0.78	0.76
Total operating costs	**8.76**	**8.58**	**8.35**
Capital costs, total dollars spent			
Equipment purchase	24,807,400	36,241,100	47,323,900
Preproduction underground excavation			
Shaft 1	6,274,400	7,946,900	10,700,100
Shaft 2	6,291,100	7,975,100	10,725,800
Shaft 3	6,308,200	7,993,700	10,742,800
Shaft 4	—	8,016,800	10,757,800
Shaft 5	—	—	10,778,600
Drifts	1,990,600	2,604,500	3,651,100
Crosscuts	2,573,500	3,429,500	4,996,600
Drawpoints	5,806,100	6,379,300	12,393,000
Boundary raises	6,809,700	11,856,800	22,715,600
Orepasses	860,000	1,882,100	3,751,600
Ventilation raises	1,009,500	1,770,600	3,449,300
Surface facilities	6,163,100	7,961,400	10,028,900
Working capital	10,652,600	15,644,500	22,870,500
Engineering and management	8,956,200	13,527,500	21,062,000
Contingency	6,889,400	10,405,800	16,201,500
Total capital costs	**95,391,800**	**143,635,600**	**222,149,100**

Source: Data from InfoMine USA 2009b.

APPENDIX 4.8C
COST MODELS FOR MECHANIZED CUT-AND-FILL MINING

These models represent mines on steeply dipping veins, 3.5, 4.0, or 4.5 m wide, respectively, and 500, 1,400, or 1,900 m along the strike. Access is via a shaft that is 524, 719, or 863 m deep. Haulage to the shaft is by scoop tram. Stoping includes drilling and blasting with jumbos, ore collection and haulage from the stopes by scoop tram, and sand filling. A secondary access/vent raise extends to the surface. A diagram of the development requirements for mechanized cut-and-fill mining is shown in Figure 4.8C-1, and cost models are shown in Table 4.8C-1.

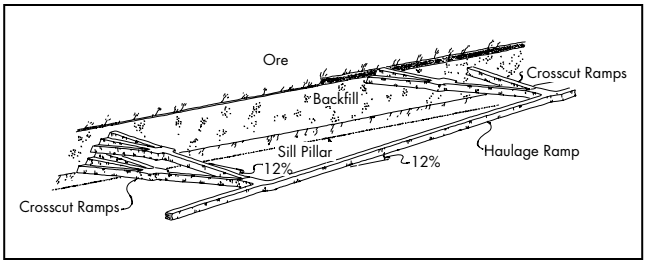


Figure 4.8C-1 Development requirements for mechanized cut-and-fill mining

Table 4.8C-1 Cost models for mechanized cut-and-fill mining

Cost Parameters	Daily Ore Production, t 200	1,000	2,000
Production			
Hours per shift	8	8	10
Shifts per day	1	2	2
Days per year	320	320	320
Deposit			
Total mineable resource, t	704,000	4,231,700	9,874,000
Dip, degrees	75	75	75
Average strike length, m	500	1,400	1,900
Average vein width, m	3.5	4.0	4.5
Average vertical, m	150	295	425
Stopes			
Stope length, m	100	300	400
Stope width, m	3.6	4.2	4.6
Stope height, m	48.3	47.5	45.6
Face width, m	3.6	4.2	4.6
Face height, m	2.9	3.2	3.4
Advance per round, m	2.5	2.8	2.8
Sill pillar length, m	100	300	400
Sill pillar width, m	3.6	4.2	4.6
Sill pillar height, m	5.8	6.4	6.8

(continues)

Table 4.8C-1 Cost models for mechanized cut-and-fill mining (continued)

Cost Parameters	Daily Ore Production, t 200	1,000	2,000
Development openings			
Shafts			
Face area, m^2	11.9	18.6	22.6
Preproduction advance, m	475	665	791
Cost, $/m	7,730	9,210	11,740
Ramps			
Face area, m^2	9.0	11.8	13.9
Daily advance, m	1.8	2.9	3.5
Preproduction advance, m	625	1,229	1,771
Cost, $/m	1,050	1,180	1,260
Crosscuts			
Face area, m^2	9.0	11.8	13.9
Daily advance, m	2.7	4.3	5.3
Preproduction advance, m	950	1,844	2,625
Cost, $/m	920	1,040	1,070
Orepasses			
Face area, m^2	0.9	1.6	2.5
Daily advance, m	0.1	0.6	1.0
Preproduction advance, m	145	285	411
Cost, $/m	640	800	1,000
Ventilation raises			
Face area, m^2	2.0	3.5	5.3
Daily advance, m	0.1	0.5	0.6
Preproduction advance, m	445	635	761
Cost, $/m	940	1,190	1,530
Hourly labor requirements, workers/day			
Stope miners	4	12	20
Development miners	8	10	8
Equipment operators	1	3	5
Hoist operators	2	4	4
Support miners	2	2	3
Diamond drillers	2	6	6
Backfill plant operators	1	2	4
Electricians	4	5	6
Mechanics	5	10	13
Maintenance workers	2	4	6
Helpers	3	4	5
Underground laborers	2	5	8
Surface laborers	2	4	6
Total hourly personnel	**38**	**71**	**94**
Salaried personnel requirements, workers			
Managers	1	1	1
Superintendents	1	1	2
Foremen	1	2	4
Engineers	1	2	3
Geologists	1	2	3
Shift bosses	2	6	6
Technicians	2	4	6
Accountants	1	2	2
Purchasing	1	2	4
Personnel managers	2	4	5
Secretaries	2	4	6
Clerks	2	5	8
Total salaried personnel	**17**	**35**	**50**
Supply requirements, daily			
Explosives, kg	194	723	1,222
Caps, each	92	289	439
Boosters, each	82	259	396
Fuse, m	351	1,337	2,289
Drill bits, each	1.05	4.17	7.03
Drill steel, each	0.07	0.3	0.50
Backfill pipe, m	1.8	2.9	8.8
Freshwater pipe, m	4.5	7.2	8.8
Compressed air pipe, m	4.5	7.2	8.8
Electric cable, m	4.5	7.2	8.8
Ventilation tubing, m	4.5	7.2	8.8
Steel liner plate, kg	25	155	322
Rock bolts, each	27	128	238
Cement, t	10.6	56.5	114.6
Buildings			
Office, m^2	434	894	1,277
Changehouse, m^2	441	824	1,092
Warehouse, m^2	239	435	592
Shop, m^2	469	909	1,263
Mine plant, m^2	111	111	111
Equipment requirements, number and size			
Production drills, cm	3 each 3.490	4 each 3.490	6 each 3.810
Production scoop trams, m^3	3 each 1.9	4 each 6.1	6 each 6.5
Vertical development drills, cm	3 each 2.870	2 each 2.870	2 each 2.870
Horizontal development drills, cm	2 each 3.175	2 each 3.490	2 each 3.490
Development scoop trams, m^3	2 each 0.19	2 each 6.1	2 each 6.5
Production hoists, cm	1 each 152	1 each 152	1 each 152
Rock bolt drills, cm	1 each 3.81	1 each 3.81	1 each 3.81
Drain pumps, hp	6 each 10	8 each 43	9 each 87
Freshwater pumps, hp	2 each 0.5	2 each 0.5	2 each 0.5
Backfill mixers, hp	1 each 3	1 each 15	1 each 15
Backfill pumps, hp	2 each 5.3	2 each 8.3	2 each 8.3
Service vehicles, hp	5 each 75	5 each 82	7 each 130

(continues)

Table 4.8C-1 Cost models for mechanized cut-and-fill mining (continued)

(continues)

Table 4.8C-1 Cost models for mechanized cut-and-fill mining (continued)

Cost Parameters	Daily Ore Production, t 200	1,000	2,000
Compressors, m^3/min	1 each 23	1 each 23	1 each 23
Ventilation fans, cm	1 each 122	1 each 122	1 each 122
Exploration drills, cm	1 each 4.45	1 each 4.45	2 each 4.45
Equipment costs, $/unit			
Production drills	702,000	702,000	702,000
Production scoop trams	295,000	714,000	734,000
Vertical development drills	8,300	8,300	8,300
Horizontal development drills	702,000	702,000	702,000
Development scoop trams	295,000	714,000	734,000
Production hoists	1,230,200	1,319,900	1,429,400
Rock bolters	690,000	690,000	690,000
Shotcreters	35,600	63,400	63,400
Drain pumps	12,700	17,800	20,000
Freshwater pumps	7,200	7,200	7,200
Backfill mixers	28,600	57,500	57,500
Backfill pumps	10,800	10,900	10,900
Service vehicles	265,500	270,000	284,500
Compressors	61,700	61,700	61,700
Ventilation fans	113,300	113,300	113,300
Exploration drills	72,000	72,000	72,000
Cost Summary			
Operating costs, $/t ore			
Equipment operation	5.61	3.92	4.28
Supplies	26.75	17.09	14.74
Hourly labor	42.91	15.99	13.30
Administration	19.95	7.87	5.57
Sundries	9.52	4.49	3.79
Total operating costs	**104.74**	**49.36**	**41.68**

(continues)

Table 4.8C-1 Cost models for mechanized cut-and-fill mining (continued)

Cost Parameters	Daily Ore Production, t 200	1,000	2,000
Unit operating cost distribution, $/t ore			
Stopes	13.11	9.73	10.62
Drifts	13.31	3.27	1.97
Crosscuts	20.57	5.05	3.05
Orepasses	0.90	0.62	0.78
Vent raises	0.97	0.60	0.49
Main haulage	5.31	3.22	2.85
Backfill	7.13	6.83	7.00
Services	12.50	7.34	5.16
Ventilation	1.14	0.19	0.10
Exploration	2.19	1.35	0.85
Maintenance	1.79	0.73	0.68
Administration	16.29	5.94	4.34
Miscellaneous	9.53	4.49	3.79
Total operating costs	**104.74**	**49.36**	**41.68**
Capital costs, total dollars spent			
Equipment purchase	10,004,000	13,143,300	16,995,700
Preproduction underground excavation			
Shafts	3,673,400	6,130,800	9,289,300
Drifts	656,100	1,453,500	2,232,100
Crosscuts	877,000	1,925,000	2,816,500
Orepasses	92,100	228,100	447,400
Ventilation raises	416,400	758,800	1,161,800
Surface facilities	1,661,000	2,774,100	3,587,600
Working capital	1,117,600	2,633,100	4,446,600
Engineering and management	2,259,400	3,433,800	4,749,000
Contingency	1,738,000	2,641,300	3,653,000
Total capital costs	**22,495,000**	**35,121,800**	**49,379,000**

Source: Data from InfoMine USA 2009b.

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CHAPTER 4.9

Cost Estimating for Surface Mines

Scott A. Stebbins and Jennifer B. Leinart

INTRODUCTION

It is obvious that costs vary from one mine to the next. So, although it is of interest to know the costs associated with surface mines in general terms, it is also important to understand how to estimate the costs of a proposed operation in a way that considers the unique development and operational parameters, and subsequently costs, of each deposit. Although focusing primarily on how to estimate costs, this chapter also includes general operating expenses for typical surface mine configurations.

There are probably as many ways to estimate mining costs as there are cost estimators. Because of the lack of a standardized approach, evaluators are left to estimate costs as best they can, so almost everyone uses a slightly different method. A standardized method that suits every situation would be extremely difficult to develop, given that each proposed mine is unique and conditions can be so variable. Although no such approach exists, many well-documented methods are available. For example, there are the tried-and-true, broad-brush approaches, one of which is the parametric method, where costs are derived from general algorithms (or curves) of the following form:

$$\text{cost} = x(\text{parameter})^y$$

The parameter in these algorithms can be almost anything, but most often it is the production rate. The x and y values are derived through statistical evaluations of known or estimated cost data. The U.S. Bureau of Mines Cost Estimating System, also known as CES (USBM 1987), can be considered a parametric approach, as can methods developed by O'Hara (1980) and Mular (1982).

Another example of a broad-brush method is the factored approach. Usually with this technique, one primary cost (such as the cost of the purchased equipment) is subjected to a series of factors to estimate all the other pertinent costs of the project (Vilbrandt and Dryden 1959). This method has fallen out of general use because it is, in light of subsequent approaches, considered too general.

Evaluators also commonly rely on a comparative approach. With this method, estimators examine costs at similar projects and make adjustments, often through the use of scaling factors (Schumacher and Stebbins 1995), to account for differences in operating parameters. This may be the most comforting of the broad-brush approaches, but it can also be the most misleading. Conditions simply vary too much from one project to the next to rely too heavily on comparative costs. If conditions were the same at every deposit, then assigning costs from a past or similar project would be acceptable, and the approach would be widely used. But it is the differences in the operating parameters from one project to the next that dictate the differences in costs, so these must be fully considered.

Cost models are a form of the comparative approach. These consist of a compilation of cost estimates along with the parameters on which those estimates are based. Evaluators find the example from within the compilation that most closely resembles their project, and they then use the costs associated with the example as an indication of the costs at their project. Example cost models for typical surface mine configurations can be found in Appendix 4.9A.

Significant effort went into the derivation of the specific variations of the aforementioned methods, and each represents an invaluable source of useful, reliable information. In particular, the CES curves (USBM 1987) enable evaluators to estimate costs for a multitude of mining and mineral processing activities for which no other source exists. But, arguably, the concern with each of these approaches is the lack of transparent detail. Evaluators are left to wonder if results truly represent their project. Even though broad-brush methods are often used because much of the information needed for more detailed analyses is difficult to obtain, evaluators still continually strive for more verifiable, and hence reliable, results.

In the past, the broad-brush approaches also maintained their popularity in part because more detailed analyses were time-consuming. Over the past 20 years, however, things have changed. Most evaluators now use a more detailed, engineering-based approach to estimating costs at almost every stage of

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project evaluation. Two events have led to this eventuality: The first was the development, publication, and distribution of *Mining Cost Service* (InfoMine USA 2009b), along with an increase in the availability of information similar to that contained in *Mining Cost Service* through the Internet. This annually updated document is a comprehensive compilation of current mine and mineral processing cost information. The second event was an improvement in spreadsheet and application-based calculation modeling capabilities, which enabled evaluators to handle the significant increase in the amount of work associated with engineering-based estimates in a timely manner. Evaluators now conduct engineering-based estimates in time frames previously achievable only when they used the broad-brush approaches.

ENGINEERING-BASED, ITEMIZED COST ESTIMATING

The method detailed in the next few paragraphs is best described as an engineering-based, abbreviated, itemized approach. It consists of three major steps, along with a highly variable number of minor steps.

In the first step, estimators design a mine to the maximum extent possible given the available information. For a deposit that can be mined using surface techniques, even a general pit outline, an overall depth, and a delineation of the routes to the processing plant and the waste stockpiles provide a great deal of information pertinent to the cost estimate.

In the next step, evaluators estimate or calculate all the parameters associated with the things that cost money: the workers, the equipment fleet, and the consumable supplies. This step is where an estimator expends the most effort, although the first design step previously outlined is the most important in achieving reliable results.

The final step is the simplest, thanks to publications such as *Mining Cost Service*. Evaluators need only apply known unit costs for labor, equipment operation, and supplies to the projected and calculated development and operating parameters to arrive at estimates of the operating costs (in addition to estimates of many of the preproduction development costs). They then need to apply equipment purchase prices along with the costs of some common mine facilities to the previously determined parameters to arrive at the primary components of a capital cost estimate.

The advantages of the engineering-based, itemized approach are many. It can be applied at almost any stage of a project evaluation, from the initial phases when information is scarce to the final stages when almost all pertinent resource and project characteristics have been established. It is reliable in that it concerns itself almost exclusively with parameters specific to one deposit. It lends itself well to computerization because so much of the work involves simple calculations (albeit a lot of them) that are easily encoded on a spreadsheet or a Windows-based application. It is easily adjusted and updated as more information becomes available. As such, the reliability of the estimate increases as the information base expands. And when the evaluation stage is complete, the final computerized product is (in essence) a dynamic cost model that engineers can use to examine operational alternatives throughout the life of the mine.

Traditionally, and logically, evaluators have kept the level of detail in their cost estimates comparable with the amount of information available for the deposit. Unfortunately, it is sometimes tempting to reduce the level of detail in an effort to reduce the amount of time spent on the estimate. Ignoring detail by procedures such as averaging site parameters or combining cost components can reduce the representativeness of the estimate. For instance, if haul distances and gradients for individual haul segments can be gleaned from maps and plans, the cycle time associated with the haul may be significantly different than the cycle time for a more convenient but less reliable overall distance and average gradient (over the entire distance). Example 4 (presented later in this chapter) helps to illustrate this point.

Just as significantly, combined cost values, such as those presented for equipment operation in various publications, can also lead to estimates that are not fully representative. If such costs are broken down into individual components, (i.e., fuel, lubricants, repair parts, tires, and wear parts), then each component can be adjusted individually to suit conditions. For instance, in a situation where a mobile loader is used to collect extremely abrasive rock, the evaluator might adjust the tire and wear part consumption rates upward. If these components were not treated separately, the evaluator might simply adjust the entire composited operating cost upward. The significance of avoiding such an approach is this: If you increase the tire consumption rate by 100% (i.e., multiply the tire operating cost by 2) and the tire cost is initially 10% of the overall operating cost, then the impact on the overall machine operating cost is minimal, as would be any error in the evaluator's assumption of the increase. And, because equipment operating costs may only represent 25% of the overall operating cost, the impact of any error would be even less. In essence, a 100% error in a cost component that comprises only 2.5% of the overall cost is much less significant than a similar error in a cost component that comprises 25% of the overall cost.

GETTING STARTED

Often, where to start is the question. It is sometimes a difficult question to answer when an estimator is trying to figure out how much a deposit will cost to mine. However, when evaluators begin the process of approximating the costs of a mining project, they soon notice a synergy. As one parameter is determined, the value of another is often defined. For instance, as the number of trucks needed to haul the ore is determined, the number of drivers required to operate the trucks and the number of mechanics needed to maintain them are also determined. An evaluator can then use those values to begin the process of estimating the sizes of the shop, the parking lot, the living quarters (if needed), and the workers' changehouse.

A few things must always be known to estimate the costs of a surface mine. The first four things to look for are a target production rate, a stripping ratio, the ore and waste haul profiles, and an estimated powder factor. Just those four items provide a good start.

The target production rate is most often based primarily on the overall size of the resource, although the assets of the operator play a secondary role. In very general terms (from a strictly economic perspective), the more revenues that the project generates early in its life, the better. To that end, operators in a perfect world prefer to maximize the production rate and initiate operations as soon as possible. On the other hand, operators often rely on revenues from the project to fund further development and expansion and thereby minimize early expenditures and the associated economic risks. For the purposes of early-stage feasibility analyses, estimators often aim for somewhere in the middle. A variation of Taylor's rule (Hoskins 1977) provides a reasonable value and is expressed as follows:

$$\text{capacity, t/yr} = (\text{metric tons resource}^{0.75}) \div 70$$

In comparison to production rates at active mines, results from this equation (which was in use as far back as the 1970s) are now conservative. Operators now try to drive economic conditions in their favor by taking advantage of economies of scale. Doing so also shortens the duration of their projects. As an added economic benefit, revenues (and hopefully profits) are maximized as early as possible in the project. A current and more representative equation that works in a manner similar to the variation of Taylor's rule just mentioned is as follows:

$$\text{capacity, t/yr} = (\text{metric tons resource}^{0.69}) \div 20.12$$

For the other items, a sketch or two can be of great help. A plan view of a proposed pit, the surrounding terrain, and the location of the mill and waste dump sites along with a few cross sections through the pit and along the main haul routes furnish the information needed to roughly estimate the stripping ratio and to define the haul-route profiles.

DRILL AND BLAST

To estimate the costs of drilling and blasting, engineers can glean a great deal of information from just a powder factor. Such a factor (which is most often reported in terms of kilograms of explosive per metric ton blasted) of course differs from one project to the next and is typically determined through experimentation, observation, and adjustment over time at an active operation. Consequently, the value will not be known ahead of time. But reported powder factor values are plentiful in books such as this handbook, in case studies contained in periodicals, and in publications such as the *Mining Source Book* (Scales 2009).

A powder factor from a mine in rock similar to that of a proposed project should supply an initial value that is within reason. From this one value, engineers can of course estimate the cost of explosives in terms of dollars per metric ton of ore. But in addition, they can also estimate how much (in terms of meters) to drill each day, which in turn provides the number of blastholes that must be drilled each day, and that value in turn furnishes the number of caps and boosters consumed each day.

With the daily drilling requirements in hand, estimators can approximate values for daily drill use (in terms of hours per day), drill bit and steel consumption, and (with all this previous information) they can proceed to gauge the required number of drillers and blasters. All of this is a lot to derive from just a powder factor and it is important to remember that for an early-stage cost estimate, precise values are not necessary (nor can they be expected). Reasonable, representative values are required, but highly precise values are simply not obtainable at the early stages of a cost estimate unless the information needed for such precision exists.

To illustrate the process, Example 1 works from a powder factor to estimate consumption rates (and subsequently costs) for explosives, caps, detonation cord, and drill bits and steel. From there, estimated values are further used to suggest drill use (in terms of hours per day) as well as labor requirements.

Example 1

Consider the case where the following have been determined:

- Production rate = 5,000 t/d
- Stripping ratio = 2.5:1 t waste to t ore
- Ore powder factor = 0.305 kg/t ore
- Waste powder factor = 0.331 kg/t waste
- Explosive (ANFO) specific gravity = 0.80
- Hole diameter = 15.24 cm
- Bench height = 12.20 m
- Subdrilling = 1.43 m
- Stemming = 4.27 m
- Drill bit penetration rate = 1.10 m/min
- Drill bit consumption = 2,500 m/bit
- Worker efficiency = 83%
- Drill relocation and setup = 2 min/hole

Based on this information, the following can be calculated.

1. Explosive consumption:

   Ore: 5,000 t/day $\times$ 0.305 kg/t ore = 1,525 kg/d

   Waste: 5,000 t ore/d $\times$ 2.5 t waste/t ore $\times$ 0.331 kg/t waste = 4,138 kg/d

   total = 1,525 kg/d (ore) + 4,138 kg/d (waste) = 5,663 kg/d

2. Daily drill-hole volume:

   5,663 kg/d $\times$ (0.80 $\times$ 1,000 kg/m^3)) = 7.08 m^3/d

   unit volume of blasthole = [π $\times$ (15.24 cm $\times$ 100 cm/m)^2] $\times$ 4 = 0.01824 m^3 per meter of depth

3. Daily drilling requirements:

   total drilling (explosives only) = 7.08 cm^3/d $\div$ 0.01824 m^3/m drilled = 388 m/d

   hole loading factor = ((12.20 m + 1.43 m) – 4.27 m) $\div$ (12.20 m + 1.43 m) = 0.687

   total drilling requirement = 388 m $\div$ 0.687 = 565 m/d

   holes drilled each day = 565 m $\div$ (12.20 m + 1.43 m) $\approx$ 42 holes

4. Drill use:

   daily drill use = (565 m $\div$ 1.1 m/min) $\div$ 60 min/h $\approx$ 8.56 h/d

5. Worker requirements for drilling:

   daily drilling = 8.56 h/d + ((2 min $\times$ 42 holes) $\div$ 60 min/h) = 9.96 h/d

   worker requirement = 9.96 h/d $\div$ 0.83 (worker efficiency) = 12.0 h/d

6. Worker requirements for blasting:

   blasthole loading = (4 min/hole $\times$ 42 holes) $\div$ 60 min/h = 2.80 h/d

This example points out the one real difficulty of engineering-based itemized cost estimates—values for several parameters are not always readily apparent or available. Powder factors, drill penetration rates, drill bit consumption rates, and bench heights have yet to be determined in the early stages of project evaluation, and subdrilling depths and stemming requirements have yet to be calculated. In the engineering-based itemized approach, evaluators determine most such parameters using one of two processes: statistical analysis of reported data or calculations based on established engineering relationships.

In Example 1, various mine operating information sources are searched to compile a series of data points. As mentioned earlier, these sources include case studies printed in periodicals, data in publications such as the *Mining Source Book* (Scales 2009), and compilations in handbooks such as this. For example, if engineers need to estimate a powder factor, they might first collect and compile as many reported values as possible. At the same time, they would identify a parameter related to the powder factor and gather associated, representative data points. Powder factors are often listed along with a rock type, for instance, and rock types can be roughly related to compressive strengths (discussed in other chapters of this handbook). With these two data strings (powder factors and rock compressive strengths), evaluators can develop an empirical relationship that they can then use to estimate a powder factor based on the rock type (as represented by an estimate of the compressive strength of the rock). Typically, such a relationship must be derived through geometric regression of the data. One such compilation (based on information primarily from the *Mining Source Book* and this handbook) provided the following algorithm:

$$\text{powder factor, kg/t} = 0.0240 \times (\text{compressive strength, MPa})^{0.4935}$$

Geometric regression analysis is outside the scope of this discussion but is detailed in most statistical analysis textbooks.

For a specific deposit, values for the compressive strength of the rock become clearer as more information becomes available and actual testing begins, but the value gained through the regression should provide a reasonable basis for early estimates. And, while for illustrative purposes in this chapter, the powder factor is related to rock strength, it may be more closely tied to other factors such as the production rate, explosive type, or rock quality. When a relationship such as this has been derived, it can be used to estimate values at other properties.

Evaluators also rely on standard, established engineering relationships to determine some of the project cost parameters. In Example 1, for instance, if the bench height is 15 m, an engineer can rely on well-established blasting design algorithms (Olofsson 1997) to determine subdrilling and stemming requirements as shown in Example 2.

Example 2

1. Maximum blasting burden:

   $\text{maximum burden} = 1.36 \times (L_b^{0.5}) \times R_1$

   where

   L_b = charge concentration ~ 14.5 kg/m (Olofsson 1997)

   R_1 = correction for vertical drilling ~ 0.95 (Olofsson 1997)

   $\text{maximum burden} = 1.36 \times (14.5^{0.5}) \times 0.95 = 4.92 \text{ m}$

2. Subdrilling:

   $\text{subdrilling} = 0.3 \times 4.92 \text{ m (maximum burden)} = 1.48 \text{ m}$

3. Error in drilling:

   $\text{error in drilling} = [152 \text{ mm (blasthole diameter)} \div 1{,}000] + (0.03 \times (15 \text{ m} + 1.48 \text{ m})) = 0.65 \text{ m}$

4. Adjusted burden:

   $\text{burden} = 4.92 \text{ m (maximum burden)} - 0.65 \text{ m} = 4.27 \text{ m}$

5. Stemming:

   stemming = 4.27 m (equivalent to burden)

Although Example 2 relies on algorithms from Olofsson (1997), there are several such sources for drilling and blasting engineering calculations, and each may be more or less detailed in its approach. It is only important that, in the early stages of an evaluation, estimators arrive at reasonable, reliable numbers.

As is evident, both of these values (stemming and subdrilling depths) are needed to provide a reasonable estimate of daily drilling requirements (in terms of meters drilled). When an evaluator analyzes these values in conjunction with the bench height, the result is the number of holes that must be drilled each day and, subsequently, the number of caps and boosters consumed on a daily basis.

EXCAVATE AND HAUL

Estimators find that most of the expense of any surface mine is attributable to excavating the rock, loading it into some sort of conveyance, hauling it somewhere (either a mineral processing plant or a stockpile), and then dumping it. Consequently, a representative estimate hinges on the reliability of the excavating and hauling costs.

As with the cost estimates of all the other surface-mining tasks, the basis for the costs of excavating and hauling begins with the design. It is crucial to know the routes over which the ore and waste will be hauled. The more that is known about these routes, the more reliable the estimates will be. Distances and gradients are the key components. And while average gradients over total haul distances can be used, much more reliable results are achieved if the routes are split into segments at each significant change in gradient. The importance of carefully defining the distances and gradients of each segment increases with the stripping ratio. Evaluators find that large projects with high stripping ratios can become, in essence, waste bound, in that the space needed to stack and store waste is at a premium. At such deposits, operating costs are more sensitive to waste haul distances and gradients than to any other factor.

To estimate excavating and hauling costs, evaluators must first determine cycle times for both the excavators and the haul trucks. Evaluators use these cycle times in conjunction with respective machine capacities to gauge the size of the required fleet and to eventually estimate operating costs and purchase prices. If the purpose of an evaluation is to estimate the average costs of production for the project, then the haul profiles should be defined at a point halfway through production. In other words, they should be based on the pit profile at that point in time when about half the resource has been extracted. When engineers structure the cost-estimating process on a spreadsheet or through a Windows application (or any number of other computerized approaches), it is entirely possible for them to estimate the costs associated with haul profiles from any bench, in fact from any point on any bench, in the pit. This is, of course, pertinent when an evaluator is optimizing a resource with software that asks for production costs from various benches as part of the optimization process.

Cycle times for excavators are, for the most part, fixed and related to machine size. Wheel loaders are the exception

in that they are sometimes called on to travel a short distance from the active face to the loading point. Most tracked excavators simply pivot after they collect a load of broken rock to transfer that load to the truck. Truck cycle times are more complicated. Although some of the time components are fixed (spot, load, dump, and turn), travel times typically represent the largest component of a truck's cycle. It is also the component that typically has the greatest impact in distinguishing costs at one project from those at another.

Engineers attempt to achieve the following goals as they design the excavator and hauler segments of their mine plan:

- Three to six loader cycles should completely fill the truck bed.
- Loader bucket capacities should be selected so that, whatever the number of cycles, the truck is full or close to full after loading is complete. For instance, a 7.0-m^3 bucket could be used to fill a 21.0-m^3-capacity truck, but it would be inefficient if used to fill a 17.0-m^3-capacity truck. Two loads would not fill the 17.0-m^3-capacity truck completely, but three loads would overfill it.
- The number of trucks and the number of loaders should be determined to minimize both the amount of time that any loader must wait for a truck and the amount of time that any truck must wait in a queue to be loaded.

To meet these goals, engineers rely on a multistep process. First, they estimate the loader cycle time and use it in conjunction with the loader's bucket capacity to determine the number needed to meet production goals. This first step is straightforward and proceeds as shown in the following example.

Example 3

Estimate daily excavator and truck use for the following situation:

- Shift length ≈ 8 h
- Production schedule ≈ 2 shifts/d
- Waste production capacity ≈ 18,000 t/d
- Front-end-loader bucket capacity (volume) ≈ 11.5 m^3
- Front-end-loader bucket capacity (weight) ≈ 21.7 t
- Average bucket fill factor ≈ 90%
- Material weight ≈ 2,400 kg/m^3
- Material swell ≈ 55%
- Cycle time
  - Load ≈ 12 s
  - Lift and swing time ≈ 12 s
  - Dump time ≈ 8 s
  - Return and lower time ≈ 10 s
- Rolling resistance ≈ 3%
- Haul profile
  - *Segment 1:* From working face across pit floor, 200 m at 0% gradient
  - *Segment 2:* From pit floor to pit entrance, 1,200 m at 12% gradient
  - *Segment 3:* From pit entrance to waste stockpile, 1,600 m at –6% gradient
  - *Segment 4:* From base of stockpile to top of stockpile floor, 800 m at 12% gradient
  - *Segment 5:* From top of stockpile across to dump point, 200 m at 0% gradient
- Operator efficiency ≈ 83%

1. Bucket load:

   2,400 kg/m^3 ÷ [1 + (55% swell ÷ 100)] = 1,550 kg/m^3

   [11.5 m^3 × 1,550 kg/m^3 × 0.90 (fill factor)] ÷ 1,000 kg/t = 16.0 t

2. Total cycle requirement:

   18,000 t/d ÷ 16.0 t/cycle = 1,125 cycles/d

   [1,125 cycles/d × (12 s + 12 s + 8 s + 10 s)] ÷ 60 s/min = 787.5 min/d

3. Loader operators:

   [787.5 min/d ÷ 0.83 (efficiency)] ÷ 60 min/h = 15.8 h/d

   15.8 h/d ÷ 8 h/shift ≈ 2 operators

Next, an average truck cycle time is determined (see Example 4). Evaluators base both haul and return times (travel times) on the length of the haul and the average gradient over that length. Gradient is defined as the change in elevation divided by the length over which that change takes place. A downhill gradient is typically reported as a negative value, and an uphill gradient is reported as a positive value.

Engineers typically obtain the speeds of the trucks over these haul distances through rimpull/speed/gradeability curves and retarder curves, which are specific for each vehicle. Examples of these curves can be found in the *Caterpillar Performance Handbook* (Caterpillar 2009). Travel speeds (with the vehicle either loaded or empty) for down-gradient segments can be gleaned from the retarder curves, and travel speeds for up-gradient segments (loaded or empty) are taken from the rimpull/speed/gradeability curves.

On examination, it is apparent that the data in these curves can also be subjected to geometric regression analyses to provide relationships that estimators can then use to approximate speeds based on the specific gradients. These relationships do not provide precise results, but they are more than adequate for early-stage cost-estimating purposes. Estimators typically ignore increases in travel times due to acceleration or deceleration over haul lengths of any significance. Over very short hauls, these need to be considered.

Evaluators also need to adjust the travel gradients for rolling resistance. Because of the flexibility inherent in roadbeds and the weight on the tires, trucks always "sink" into the road surface just a little as they travel along. One way to visualize rolling resistance is to view it as the gradient that the tire must continually overcome to drive out of the slight depression that it creates in the roadbed because of the weight that it carries.

Example 4

Consider the following situation:

- Bed capacity (volume) ≈ 60 m^3
- Bed capacity (weight) ≈ 90 t
- Material weight ≈ 2,400 kg/m^3
- Material swell ≈ 55%
- Turn and spot time ≈ 15 s
- Dump time ≈ 8 s
- Return and lower time ≈ 12 s

First, the time to load the truck is estimated from the previous example.

1. Load time:

   60 m^3 bed capacity ÷ [11.5 m^3 bucket capacity × 0.85 (fill factor)] ≈ 6 cycles to load

   [6 cycles × (12 s + 12 s + 8 s + 10 s)] ÷ 60 s/min ≈ 4.20 min/truck

2. Travel time—fully loaded:

   Segment 1: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 43 km/h] × 60 min/h = 0.54 min

   Segment 2: 12% gradient resistance + 3% rolling resistance = 15% total resistance

   [(1,200 m ÷ 1,000 km/m) ÷ 9 km/h] × 60 min/h = 8.00 min

   Segment 3: –6% gradient resistance + 3% rolling resistance = –3% total resistance

   [(1,600 m ÷ 1,000 km/m) ÷ 64 km/h] × 60 min/h = 1.50 min

   Segment 4: 12% gradient resistance + 3% rolling resistance = 15% total resistance

   [(800 m ÷ 1,000 km/m) ÷ 9 km/h] ÷ 60 min/h = 5.33 min

   Segment 5: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 43 km/h] × 60 min/h = 0.54 min

   total haul time loaded = 15.37 min

3. Turn and dump time = 1.20 min.
4. Travel time—return empty:

   Segment 5: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 63 km/h] × 60 min/h = 0.38 min

   Segment 4: –12% gradient resistance + 3% rolling resistance = 9% total resistance

   [(800 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 1.50 min

   Segment 3: 6% gradient resistance + 3% rolling resistance = 9% total resistance

   [(1,600 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 3.00 min

   Segment 2: –12% gradient resistance + 3% rolling resistance = 9% total resistance

   [(1,200 m ÷ 1,000 km/m) ÷ 32 km/h] × 60 min/h = 2.25 min

   Segment 1: 0% gradient resistance + 3% rolling resistance = 3% total resistance

   [(400 m ÷ 1,000 km/m) ÷ 63 km/h] × 60 min/h = 0.38 min

   total haul time for return = 7.51 min

5. Turn and spot to load = 0.80 min
6. Total cycle time:
   - Load = 4.20 min
   - Travel loaded = 15.37 min
   - Turn and dump = 1.20 min
   - Return time = 7.51 min
   - Turn and spot to load = 0.80 min
   - Total cycle time = 29.08 min
7. Required number of trucks:

   6 cycles/truck × 11.5 m^3/cycle × 0.9 (fill factor) × 1,550 kg/ m^3 = 96,225 kg/load

   18,000 t/d ÷ (96,225 kg/load ÷ 1,000 kg/t) = 187.1 loads/d

   187.1 loads/d × 29.08 min/load = 5,440.9 min/d

   5,440.9 min/d ÷ (2 shifts/d × 8 h/shift × 60 min/h) ≈ 6 trucks

8. Time spent in queue:

   29.08 min/cycle ÷ 6 trucks = 4.85 min available to load truck

   Because 4.85 minutes > 4.20 min/load, then

   time the loader spends waiting for a truck = 4.85 min – 4.20 min = 0.65 min

   Had the time that the loader spent waiting for a truck been negative (i.e., trucks have to wait for the loader), it would have been necessary to increase the size of the loader.
9. Truck drivers:

   5,440.9 min/d ÷ 0.83 (efficiency) ÷ 60 min/h = 109.3 h/d

   109.3 h/d ÷ 8 h/shift ≈ 14 operators

Because of the efficiency of the truck drivers, it is apparent in these estimates that at least one (and probably two) utility operators will be needed to drive the trucks while the regular drivers take their breaks. And if this estimate is to represent the costs at a point in time halfway through the project, then more trucks may be needed later in the project as the pit deepens and haul distances increase. The six trucks indicated at this point are operating close to their maximum capacity.

If, instead of estimating the travel times over each haul segment, evaluators had calculated an average haul gradient over the entire haul distance, the results would have been noticeably different because the average gradient is about 5.0%. The significantly slower segment where the gradients are 12% would not have the same impact on the analysis and the haul-cycle time would be reduced by more than 1.5 minutes. This is one example where a high level of detail, even at a very early stage of an analysis when information is scarce, has a significant impact on the reliability of the results.

Most of the machine specifications stated in the excavator and truck examples were taken from the *Caterpillar Performance Handbook* (Caterpillar 2009). Machine specifications are very often available from equipment manufacturers. Similar information can also be found in *Mining Cost Service* (InfoMine 2009b).

ANCILLARY SYSTEMS

To produce a complete cost estimate, much work remains for the evaluator even after the drilling, blasting, excavating, and hauling costs have been determined. However, these previously determined costs (along with the parameters derived during the estimation process) do provide a basis for estimates of the remaining costs.

To begin with, the costs associated with many of the machines typically found at any surface mine have yet to be estimated. For most such projects, costs for bulldozers, graders, dust suppressant tankers, equipment-maintenance trucks, pumps, lighting plants, personnel movers, and, in some cases, generators, crushers, and conveyors, may all need to be estimated and included.

For each of these machine types, the determining factors that provide the basis for the estimated costs are (as with the drills, excavators, and haulers) the capacity of the machine and how many hours it must operate each day. Consequently, the techniques that evaluators rely on to estimate the capacity and daily use parameters for each are similar to those they use to gauge the same parameters for the drills, excavators, and haulers.

For instance, at almost every surface mine, a fleet of bulldozers manages blasted rock at the working faces and dumped waste rock at the stockpiles (in addition to performing a host of other tasks). The process that estimators use to determine the number and operating requirements of these machines is rarely as straightforward as the process that they use to determine the excavator and loader needs, but it is still based on a very similar approach.

At all but the smallest operations, bulldozers work continuously at each dump site. They also often work at each active face, moving scattered, broken rock to the excavator. The size requirements for these machines are based on the amount of material that they handle each shift and the distance that the material must be moved. Specifically, each blade load carries with it a volume that will be moved over a distance at a speed typically specified in the manufacturer's documentation. With these three parameters (speed, distance, and capacity), an engineer can approximate productivity through the following general relationship:

$$\text{productivity, t/h} = (\text{volume, m}^3 \times \text{density, t/m}^3 \times \text{velocity, m/h}) \div \text{distance, m}$$

And with that, the engineer can estimate daily use as follows:

$$\text{daily use, h/d} = \text{production rate, t/d} \div \text{productivity, t/h}$$

When hourly costs (available from the sources mentioned earlier) are applied to the daily use (hours per day) values, the results report in terms of dollars per day. To arrive at a final value in terms of dollars per metric ton of ore, evaluators need only to divide the dollars per day value by the metric tons of ore mined each day.

Evaluators can use a similar process to arrive at daily use values for the graders and dust suppressant tankers. For instance, consider a road built from friable material in a wet environment that would need to be graded twice daily. If the length and width of the road are known, then an estimator can use the following general relationship to approximate the amount of time spent grading the road each day:

$$\text{productivity, m}^2\text{/h} = \text{velocity, m/h} \times \text{blade width, m}$$

And with this, the evaluator can estimate daily use (in terms of hours per day) as follows:

$$\text{daily use, h/d} = \text{grading requirements, m}^2\text{/d} \div \text{productivity, m}^2\text{/h}$$

As previously demonstrated, when hourly costs are applied to the daily use values, the results report in terms of dollars per day and subsequently in dollars per metric ton of ore.

CAPITAL AND OPERATING COSTS

For the purpose of project evaluation, costs are typically categorized as either *operating* or *capital* (as opposed to *fixed* or *variable*) so that they can be subjected to after-tax, discounted cash-flow analyses. In short, operating costs are those that can be fully expensed in the year incurred. The expenses of the consumables (including those associated with equipment operation), wages, and salaries are typically all considered operating costs and are most often estimated either in terms of dollars per metric ton of ore or dollars per year. Capital costs are those that cannot be fully expensed in the year incurred and include items such as the following:

- Mine and mill equipment purchase
- Development
- Engineering and construction management
- Infrastructure
- Working capital
- Postproduction reclamation
- Preproduction stripping
- Property acquisition
- Exploration
- Buildings
- Contingency fund

Although this chapter deals primarily with costs and cost estimation, it is worth mentioning that from an after-tax economic-viability standpoint, it is best to minimize the preproduction capital expenses and incur them as close to start-up as possible. Because of the time value of money, capital expenses accrued later in the operation have a lesser impact on the overall project net present value. So, during the preliminary mine-design process, evaluators find that it is worth the effort to structure the project in a way that expedites production.

COST COMPONENTS

While expenses at a mine can be categorized as either *capital costs* or *operating costs*, both are comprised almost entirely of labor, supply, and equipment components. Whether building a processing plant, constructing a tailings impoundment, or mining an ore deposit, evaluators find that most of the money spent on the project goes to either the workers (laborers, skilled tradesmen, equipment operators, supervisors, technicians, managers, etc.), the supply vendors (to purchase wood, drill bits, concrete, steel, explosives, tires, diesel, etc.), or to equipment manufacturers (to purchase machines or buy parts).

LABOR

Wages and salaries, and the "burdens" associated with each, very often represent the largest expenditures at any mineral-development project. In fact, wages, benefits, mandated employment taxes, and bonuses can sometimes account for

more than half of the total operating costs. Depending on the size of the mine, labor costs can account for anywhere from 15% to 60% of the total operating costs. These costs can escalate if the mine is situated in a remote area without a local source of skilled labor.

Wages also tend to be one of the more variable components of an evaluation. Project location has a significant impact, and evaluators are urged to examine wages on a regional level to properly account for the associated expenses in their evaluation. Wages and salaries for miners in several countries are tracked and reported in publications such as *U.S. Metal and Industrial Mineral Mine Salaries, Wages and Benefits: 2009 Survey Results* (Salzer 2009).

Base wages are "loaded" with mandated employment taxes, including Social Security, Medicare, unemployment taxes, and workers' compensation taxes. Other items add to the burden factor, such as shift differentials; overtime; medical, dental, and vision benefits; retirement plans; short- and long-term disability insurance; life insurance; accidental death and dismemberment insurance; sick leave; vacation and holiday pay; and other benefits. To retain employees, companies often use creative benefits such as paid tuition, transportation to remote mine sites, attendance bonuses, safety bonuses, family and individual assistance plans, and paid fitness-club memberships.

Many mines pay their production miners a bonus based on meeting development or production goals. These bonus systems are sometimes modified to include safety, ground conditions, and other factors. Safety violations can reduce or even eliminate a production bonus. Other criteria sometimes used to calculate bonuses are individual performance, safety performance, commodity price, profit, recovery, ore grade, production, and cost savings to calculate bonuses.

SUPPLIES

Supply prices are less volatile than wages and salaries, but they still vary from one region to the next and from one vendor to another. While it is always preferable to obtain local prices from established vendors, it is often impractical to do so during the early stages of project evaluation. *Mining Cost Service* (InfoMine 2009b) provides an extensive array of supply costs that are reliable for early-stage feasibility work.

In the evaluation process, some supply costs are commonly reported as equipment-operating costs because their consumption rates are directly tied to machine use. Diesel fuel, gasoline, electricity, tires, and lubricants all fall into this category. And, as demonstrated earlier, many of the project's labor requirements and subsequent costs are also directly dictated by daily machine use. In addition to individual equipment-operator requirements, mechanic, electrician, machinist, and equipment-maintenance worker requirements also vary in proportion to machine use.

EquipmentWatch's *Cost Reference Guide* (EquipmentWatch 2009), InfoMine USA's *Mine and Mill Equipment Costs: An Estimator's Guide* (InfoMine USA 2009a), and equipment manufacturer publications such as Caterpillar's *Caterpillar Performance Handbook* (Caterpillar 2009) provide invaluable equipment operating parameter and cost information. As such, they also serve as indirect guides to mechanic requirements and equipment-related supply consumption rates (fuel, electricity, tires, and lubricants). Some recent prices for supplies commonly consumed at surface mines are listed in Table 4.9-1.

Table 4.9-1 Supply prices (2009 dollars)

Item	Price per Unit, US$
Ammonium nitrate fuel oil (ANFO)	1.06/kg
Extra-gelatin dynamite	4.50/kg
Primers (0.45 kg)	4.58 each
Blasting caps (nonelectric, 6-m lead)	3.15 each
Detonation cord (25 grain)	0.063/m
Detonation cord (40 grain)	0.073/m
Rotary drill bits (17.15 cm)	2,746 each
Rotary drill pipe	81.58/m
Percussion drill bits (4.45 cm)	58.80 each
Percussion drill steel	78.63/m
Dust suppressant	0.82/L
Diesel fuel	0.720/L
Lubricants	2.171/L
Cement	112.36/t
Tailings pipe (20.3-cm abrasion-resistant steel)	56.98/m
Water return pipe (10.2-cm polyvinyl chloride)	15.88/m
Synthetic liner (36-mil Hypalon)	11.04/m^2
Geotextile	4.16/m^2
Soil stabilizer	3.20/L

EQUIPMENT

As with the supply prices, equipment purchase prices are typically obtained from vendors. However, in the early stages of an evaluation, it is even more difficult to obtain these values than it is the supply costs because the necessary machines can only be specified in the most general terms. InfoMine USA's *Mining Cost Service* (InfoMine 2009b) and *Mine and Mill Equipment Costs: An Estimator's Guide* (InfoMine USA 2009a) also contain extensive purchase price lists for machines commonly used at surface mines. For early-stage feasibility work, equipment prices are usually based on list prices as suggested by the manufacturers, with no discounts assumed and no options added.

Early in the evaluation process, most evaluators specify new machines for all production-related project requirements, and purchase prices reflect this. If anticipated use is minimal, estimators may specify previously owned (used) machines for some of the secondary support equipment, such as water tankers and road graders. If previously owned machines are relied on for production work, the equipment productivity and availability (and the associated operating costs) should be adjusted accordingly in anticipation of increased maintenance and repair requirements.

COST MODELS

When evaluators have limited deposit information, they can use mine models for order-of-magnitude estimates. In addition, models can be used to provide insight into the nature of mining costs in general. The impact of changes in operating parameters can be easily understood when presented in a format that compares costs associated with one configuration directly to those associated with another.

Appendix 4.9A presents three cost models that evaluators can use to make preliminary estimates for projects. The cost models include surface mines of 1,000, 10,000, and 80,000 t/d.

Each cost model compares stripping ratios of 1:1, 2:1, 4:1, and 8:1 (waste to ore). In these cost models, note the

Table 4.9-2 Hourly wages for personnel (2009 dollars)

Worker	Hourly Wage, US$
Driller	22.20
Blaster	22.36
Excavator operator	22.80
Truck driver	19.97
Heavy equipment operator	21.98
Utility operator	18.17
Mechanic	22.53
Electrician	23.90
Maintenance worker	18.20
Laborer	17.30

Source: Salzer 2009.

Table 4.9-3 Annual salaries for professionals (2009 dollars)

Job Title	Annual Salary, US$
Mine manager	147,600
Superintendent	92,600
Foreman	71,700
Engineer	79,500
Geologist	73,000
Technician	48,700
Accountant	64,600
Purchasing agent	63,100
Personnel manager	89,800
Secretary	35,700
Clerk	39,700

Source: Salzer 2009.

unexpected changes dictated by increases in stripping ratios at the larger operations, and also note the ratio of labor costs to equipment operating costs as production rates increase.

These models are theoretical and are not representative of any existing mine. Note the pit and haul parameters, which provide the basis for each design. These should be one of the key points of comparison if the models are to be used to provide estimates for any proposed operation. Costs associated with each model account for all pertinent labor, material, supply, and equipment operating expenses accrued at the mine site. Costs for supervision, administration, and on-site project management are all included. Expenses associated with preproduction development, equipment purchase and installation, and building and facility construction are also included. In these models, costs for the following operations and facilities are considered:

- Ore and waste drilling, blasting, and excavation
- Ore haul from the active face to the mill site
- Overburden and waste haul from the active face to the dumpsite
- Constructing and operating the facilities required for equipment maintenance and repair, electricity and fuel distribution, drainage, explosives storage, and sanitation
- Constructing a mine office, a warehouse, and a worker changehouse plus all associated site work

The mines in the cost models are located in areas of moderate relief with warm summers and temperate winters. Wages and salaries used in the cost models are from *U.S. Metal and Industrial Mineral Mine Salaries, Wages and Benefits: 2009 Survey Results* (Salzer 2009).

Representative hourly wages used in the models are listed in Table 4.9-2. In the models, the above wages are adjusted upward to account for a 39.0% burden rate. Salaries for professionals that were used in the models are shown in Table 4.9-3, and in the models, the salaries are adjusted upward to account for a 39.0% burden rate. Models were constructed with Sherpa Cost Estimating Software for Surface Mines (Stebbins 2009).

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APPENDIX 4.9A

The following tables present cost models for a surface mine producing ore at waste-to-ore strip ratios of 1:1, 2:1, 4:1, and 8:1.

Table 4.9A-1 Surface mines: 1,000 metric tons of ore per day

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Ore production, t/d	1,000	1,000	1,000	1,000
Waste production, t/d	1,000	2,000	4,000	8,000
Total resource, million t	3.12	3.12	3.12	3.12
Final pit dimension				
Pit depth, m	84	97	115	141
Pit floor length, m	154	177	210	256
Pit floor width, m	77	88	105	128
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	87	91	98	110
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	414	564	770	1,031
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	342	394	468	571
Gradient, %	2.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	117	130	150	180
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	112	173	251	343
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	171	197	234	286
Gradient, %	2.0	2.0	2.0	2.0
Stockpile base to surface				
Distance, m	213	269	339	427
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	109	137	172	217
Gradient, %	0.0	0.0	0.0	0.0
Hours per shift	8	8	8	8
Shifts per day	2	2	2	2
Days per year	312	312	312	312
Bench height—ore, m	3.66	3.66	3.66	3.66
Bench height—waste, m	4.88	4.88	4.88	4.88
Powder factor—ore, kg/t	0.35	0.35	0.35	0.35
Powder factor—waste, kg/t	0.31	0.31	0.31	0.31
Development				
Preproduction stripping, t	30,000	60,000	120,000	240,000
Haul road construction, m	1,565	1,955	2,483	3,165
Equipment, number and size				
Hydraulic shovels, m^3	1 each 2.3	1 each 2.3	1 each 2.3	1 each 2.3
Front-end loaders, m^3	1 each 2.3	1 each 2.3	1 each 3.8	1 each 12.2
Rear-dump trucks, t	4 each 32.0	5 each 32.0	4 each 41.0	6 each 54.0
Percussion drills, cm	2 each 6.35	3 each 6.35	4 each 6.35	6 each 6.35
Bulldozers, kW	2 each 60	3 each 60	4 each 60	3 each 110
Graders, kW	1 each 105	1 each 105	1 each 115	1 each 115
Water tankers, L	—	—	1 each 9,500	1 each 9,500
Service/tire trucks, kg GVW*	2 each 1,800	3 each 1,800	2 each 6,800	3 each 6,800
Bulk trucks, kg GVW	1 each 2,000	1 each 2,000	1 each 2,000	1 each 2,000
Light plants, kW	4 each 7.8	4 each 7.8	4 each 7.8	4 each 7.8
Pumps, kW	2 each 3.7	2 each 7.5	2 each 11.2	2 each 14.9
Pickup trucks	3	3	3	5
Buildings				
Shop, m^2	266	332	337	583
Dry, m^2	157	209	232	313
Office, m^2	204	230	256	383
Warehouse, m^2	167	167	174	224
Hourly personnel requirements				
Drillers	3	4	5	9
Blasters	2	2	2	2
Excavator operators	3	3	3	3
Truck drivers	5	8	7	11
Equipment operators	5	7	7	7
Utility operators	1	1	3	3
Mechanics/electricians	3	4	5	7
Laborers/maintenance	5	7	8	12
Total hourly personnel	**27**	**36**	**40**	**54**
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	0	0	0	0
Foreman	2	2	2	2
Engineer	1	1	1	1
Geologist	0	0	0	1
Supervisor	1	1	1	2
Technician	2	2	3	4
Accountant	0	0	0	0
Clerk	0	1	1	1
Personnel manager	0	0	0	0
Secretary	1	1	1	2
Warehouse	0	0	0	1
Total salaried personnel	**8**	**9**	**10**	**15**
Primary supply requirements				
Diesel fuel, L/d	1,502	2,195	3,047	5,225
Powder, kg/d	660	970	1,590	2,830
Caps, units/d	50	69	106	180
Drill bits, units/d	0.101	0.149	0.244	0.434
Detonation cord, m/d	445	653	1,071	1,906

(continues)

Table 4.9A-1 Surface mines: 1,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	4.21	6.17	10.09	17.92
Labor	4.84	6.23	7.61	10.01
Equipment operation	1.67	2.31	4.05	7.56
Administration	2.20	2.46	2.67	3.75
Sundry items	1.29	1.72	2.44	3.92
Total operating costs	**14.21**	**18.89**	**26.86**	**43.16**
Capital costs, $				
Equipment	2,725,700	3,493,500	4,575,100	8,027,800
Haul roads/site work	534,600	642,900	1,241,100	1,600,600
Preproduction stripping	164,100	281,100	502,300	880,800
Buildings	691,200	780,700	1,349,100	1,985,200
Electrical system	68,700	95,000	101,900	111,500
Working capital	417,200	565,200	793,800	1,324,600
Engineering and management	342,300	433,900	636,000	1,036,600
Contingency	452,700	572,700	840,600	1,364,200
Total capital costs	**5,396,500**	**6,865,000**	**10,039,900**	**16,331,300**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Ore production, t/d	10,000	10,000	10,000	10,000
Waste production, t/d	10,000	20,000	40,000	80,000
Total resource, million t	37.44	37.44	37.44	37.44
Final pit dimension				
Pit depth, m	193	222	264	322
Pit floor length, m	352	404	481	587
Pit floor width, m	176	202	241	293
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	198	208	225	253
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	947	1,290	1,760	2,360
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	783	901	1,072	1,308
Gradient, %	2.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	268	298	345	412
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	254	394	572	782
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	392	451	536	654
Gradient, %	2.0	2.0	2.0	2.0

(continues)

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Stockpile base to surface				
Distance, m	489	616	776	977
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	249	313	395	497
Gradient, %	0.0	0.0	0.0	2.0
Hours per shift	10	10	10	10
Shifts per day	2	2	2	2
Days per year	312	312	312	312
Bench height—ore, m	4.60	4.60	4.60	4.60
Bench height—waste, m	6.72	6.72	6.72	6.72
Powder factor—ore, kg/t	0.33	0.33	0.33	0.33
Powder factor—waste, kg/t	0.29	0.29	0.29	0.29
Development				
Preproduction stripping, t	300,000	600,000	1,200,000	2,400,000
Haul road construction, m	3,580	4,470	5,681	7,244
Equipment, number and size				
Hydraulic shovels, m^3	1 each 8.4	1 each 8.4	1 each 8.4	1 each 8.4
Front-end loaders, m^3	1 each 12.2	2 each 16.1	2 each 19.9	4 each 19.9
Rear-dump trucks, t	11 each 54.0	11 each 77.0	20 each 91.0	34 each 100.0
Rotary drills, cm	3 each 20.00	2 each 25.08	2 each 27.94	4 each 31.12
Bulldozers, kW	4 each 110	5 each 140	6 each 180	9 each 180
Graders, kW	1 each 115	1 each 140	2 each 140	2 each 140
Water tankers, L	1 each 19,000	1 each 19,000	1 each 26,500	1 each 30,000
Service/tire trucks, kg GVW*	5 each 6,800	5 each 11,000	9 each 11,000	15 each 11,000
Bulk trucks, kg/min	1 each 450	1 each 450	1 each 450	2 each 450
Light plants, kW	4 each 8.9	4 each 10.1	5 each 10.1	7 each 10.1
Pumps, kW	3 each 37.3	3 each 74.6	4 each 74.6	5 each 93.2
Pickup trucks	7	8	12	17
Buildings				
Shop, m^2	908	1,168	2,404	5,030
Dry, m^2	394	441	738	1,168
Office, m^2	587	715	1,047	1,533
Warehouse, m^2	363	643	696	1,159
ANFO storage bin, m^2	64	80	130	234
Hourly personnel requirements				
Drillers	4	3	4	5
Blasters	2	2	2	4
Excavator operators	4	6	6	10
Truck drivers	19	19	36	61
Equipment operators	9	11	13	19
Utility operators	3	3	4	5

(continues)

Table 4.9A-2 Surface mines: 10,000 metric tons of ore per day (continued)

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Mechanics/electricians	10	12	24	37
Laborers/maintenance	17	20	39	60
Total hourly personnel	**68**	**76**	**128**	**201**
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	1	1	1	1
Foreman	2	2	4	4
Engineer	2	2	3	5
Geologist	1	2	3	4
Supervisor	3	3	6	9
Technician	5	6	8	11
Accountant	1	1	2	3
Clerk	2	3	4	7
Personnel manager	1	2	2	4
Secretary	3	4	5	8
Security	1	1	2	3
Total salaried personnel	**23**	**28**	**41**	**60**
Primary supply requirements				
Diesel fuel, L/d	11,307	18,396	32,654	62,939
Powder, kg/d	7,250	9,102	14,903	26,505
Caps, units/d	72	55	67	92
Primers, units/d	68	51	63	88
Drill bits, units/d	1.570	1.253	1.654	2.372
Detonation cord, m/d	861	688	907	1,301
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	1.69	2.07	3.33	5.74
Labor	1.91	2.42	3.68	6.33
Equipment operation	1.44	2.49	4.79	9.44
Administration	0.69	0.84	1.20	1.74
Sundry items	0.57	0.78	1.30	2.33
Total operating costs	**6.30**	**8.60**	**14.30**	**25.58**
Capital costs, $				
Equipment	13,956,400	22,375,800	45,083,900	88,465,500
Haul roads/site work	2,183,300	3,050,600	6,007,700	7,942,200
Preproduction stripping	824,200	1,438,700	2,731,400	5,609,500
Buildings	3,217,500	3,803,900	6,191,900	10,826,200
Electrical system	179,200	190,200	406,100	428,000
Working capital	1,631,900	2,035,800	3,282,600	5,567,800
Engineering and management	2,105,500	3,252,400	6,310,300	11,877,200
Contingency	2,246,600	3,411,200	6,673,100	12,514,900
Total capital costs	**26,344,600**	**39,558,600**	**76,687,000**	**143,231,300**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day

	Stripping Ratio, Waste/Ore			
Cost Parameters	**1:1**	**2:1**	**4:1**	**8:1**
Ore production, t/d	80,000	80,000	80,000	80,000
Waste production, t/d	80,000	160,000	320,000	640,000
Total resource, million t	350.4	350.4	350.4	350.4
Final pit dimension				
Pit depth, m	407	468	557	679
Pit floor length, m	740	852	1,014	1,237
Pit floor width, m	370	426	507	618
Final pit wall slope, degrees	50	50	50	50
Haul profile—ore				
Face to pit ramp				
Distance, m	418	438	474	533
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	2,000	2,720	3,708	4,973
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to mill				
Distance, m	1,650	1,899	2,260	2,756
Gradient, %	0.0	2.0	2.0	2.0
Haul profile—waste				
Face to pit ramp				
Distance, m	565	628	726	868
Gradient, %	0.0	0.0	0.0	0.0
Ramp entrance to pit exit				
Distance, m	533	831	1,204	1,648
Gradient, %	12.0	12.0	12.0	12.0
Pit exit to waste stockpile				
Distance, m	825	949	1,130	1,378
Gradient, %	2.0	2.0	2.0	2.0
Stockpile base to surface				
Distance, m	1,030	1,298	1,635	1,960
Gradient, %	12.0	12.0	12.0	12.0
Across stockpile to dump				
Distance, m	524	660	831	1,247
Gradient, %	0.0	0.0	0.0	0.0
Hours per shift	8	8	8	8
Shifts per day	3	3	3	3
Days per year	365	365	365	365
Bench height—ore, m	5.49	5.49	5.49	5.49
Bench height—waste, m	8.53	8.53	8.53	8.53
Powder factor—ore, kg/t	0.28	0.28	0.28	0.28
Powder factor—waste, kg/t	0.23	0.23	0.23	0.23
Development				
Preproduction stripping, t	2,400,000	4,800,000	9,600,000	19,200,000
Haul road construction, m	7,544	9,423	11,969	15,364

(continues)

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Equipment, number and size				
Cable shovels—ore, m^3	1 each 26.0	1 each 35.2	1 each 61.2	1 each 61.2
Cable shovels—waste, m^3	1 each 26.0	2 each 35.2	2 each 61.2	4 each 61.2
Rear-dump trucks, t	32 each 181	47 each 218	61 each 327	134 each 327
Rotary drills, cm	3 each 38.10	5 each 38.10	6 each 38.10	10 each 38.10
Bulldozers, kW	4 each 305	6 each 305	6 each 305	9 each 305
Graders, kW	2 each 160	3 each 160	5 each 160	6 each 160
Water tankers, L	1 each 30,000	1 each 53,000	2 each 53,000	3 each 53,000
Service/tire trucks, kg GVW*	13 each 20,500	19 each 20,500	28 each 20,500	45 each 20,500
Bulk trucks, kg/min	2 each 450	3 each 450	4 each 450	6 each 450
Light plants, kW	4 each 10.1	5 each 10.1	5 each 10.1	7 each 10.1
Pumps, kW	6 each 93.2	7 each 93.2	8 each 93.2	9 each 186.4
Pickup trucks	26	33	47	75
Buildings				
Shop, m^2	5,621	8,892	15,172	33,071
Dry, m^2	1,122	1,803	3,096	6,184
Office, m^2	2,325	2,964	4,318	6,820
Warehouse, m^2	1,488	3,533	5,519	9,199
ANFO storage bin, m^3	358	682	844	1,492
Hourly personnel requirements				
Drillers	7	12	15	26
Blasters	8	14	16	28
Excavator operators	6	9	9	15
Truck drivers	89	129	165	365
Equipment operators	13	19	19	28
Utility operators	9	10	16	20
Mechanics/electricians	65	112	225	449
Laborers/maintenance	93	161	334	661
Total hourly personnel	**290**	**466**	**800**	**1,592**

(continues)

Table 4.9A-3 Surface mines: 80,000 metric tons of ore per day (continued)

Cost Parameters	Stripping Ratio, Waste/Ore 1:1	2:1	4:1	8:1
Salaried personnel requirements				
Manager	1	1	1	1
Superintendent	1	1	1	1
Foreman	6	6	6	6
Engineer	8	11	17	27
Geologist	7	10	17	30
Supervisor	12	12	12	12
Technician	14	17	22	29
Accountant	6	8	14	24
Clerk	12	17	28	50
Personnel manager	6	9	15	27
Secretary	12	16	23	36
Security	6	8	13	24
Total salaried personnel	**91**	**116**	**169**	**267**
Primary supply requirements				
Diesel fuel, L/d	79,815	129,236	310,370	670,455
Electricity, kW·h/d	57,262	87,550	164,508	280,565
Powder, kg/d	40,808	77,605	96,021	169,638
Caps, units/d	92	154	184	307
Primers, units/d	88	150	180	303
Drill bits, units/d	2.436	4.632	5.731	10.124
Detonation cord, m/d	1,366	2,541	3,144	5,554
Cost Summary				
Operating costs, $/t ore				
Supplies and materials	1.11	2.10	2.60	4.59
Labor	0.88	1.40	2.39	4.55
Equipment operation	1.70	3.32	6.22	12.14
Administration	0.28	0.36	0.52	0.83
Sundry items	0.40	0.72	1.17	2.21
Total operating costs	**4.37**	**7.90**	**12.90**	**24.32**
Capital costs, $				
Equipment	93,345,400	235,141,700	441,435,900	844,119,800
Haul roads/site work	10,946,400	27,513,100	35,694,800	43,404,000
Preproduction stripping	4,553,100	23,065,300	21,687,700	45,707,400
Buildings	12,079,400	26,577,200	33,517,000	65,059,900
Electrical system	1,485,900	1,870,200	2,698,700	3,526,400
Working capital	8,818,800	16,957,500	22,500,200	40,011,200
Engineering and management	15,375,000	39,536,200	67,666,100	126,902,900
Contingency	13,778,500	35,370,400	60,270,000	112,872,000
Total capital costs	**160,382,500**	**406,031,600**	**685,470,400**	**1,281,603,600**

Source: Data from InfoMine USA 2009b.
*GVW = gross vehicle weight.

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PART 5

Management and Administration

CHAPTER 5.1

Mine Economics, Management, and Law

Michael G. Nelson

INTRODUCTION

Finding an ore deposit and putting it into production as a mine requires the execution of many complex tasks, such as exploration and evaluation, planning, development, production, processing, and reclamation. Also required is a team of technical experts with skills in many areas—drilling, mapping, modeling, equipment selection, explosives and blasting, ground control, ventilation, power systems, waste disposal, and so forth. Much of the content of this handbook is devoted to these types of tasks and skills, almost all of which are specifically technical, based on various disciplines of science and engineering.

However, the technical aspects of mine development and production are not sufficient by themselves. Even when all the science and engineering are done correctly, there are other key components to a successful project. Capital and start-up funds are required; the mine must make a profit; day-to-day operations must be efficiently managed; and operations must be in conformance with all local and national laws, which may in some cases change unexpectedly.

This chapter deals with mine economics, management, and law, and specifically with aspects of those three topics that directly influence the ability of a company to find a mineral deposit, develop a mine, and operate the mine profitably. The disciplines and practices of mine economics, mine management, and mining law are largely devoted to controlling risk and maximizing returns from the exploitation of mineral deposits. Those risks and returns accrue to investors, mining company shareholders, governments, and residents of the area around the mine.

Every investment carries risks, and successful investors are adept at assessing and accommodating those risks. Management and control of the risks associated with mining investments require special methods, because the assets in a mining investment are more difficult to quantify than those in many other investments. Furthermore, some investors have a perception that mining investments can deliver high rates of return. The term *gold mine* is frequently used in a colloquial sense to describe an unusually good investment.

Unfortunately, under the right combination of these circumstances, large investments can be made and lost in mining ventures that are valued incorrectly or unscrupulously. A good historic example is the case of the Emma mine, which was located near Salt Lake City, Utah (United States) and whose history is described in detail by Jackson (1955). The original Emma claims were staked in 1864 by two prospectors, Chisholm and Woodman, who were described as "rough in manners and character."

As was common in that time and place, the original owners tried to work the prospect but were compelled to bring in partners to finance the venture. The ownership rapidly became unclear, but by 1868, ore was being shipped to Great Britain for reduction. At this point, two skilled promoters entered the scene. General George Baxter, former president of the New York Central Railroad, and Trenor W. Park, former receiver for the famous Mariposa estate in California, succeeded in gaining control of the mine.

By 1870, London capitalists had overcome the shock of their earlier, disastrous losses in the quartz-gold ventures of California, and registration of British companies intending to invest in American mining ventures boomed. Baxter and Park enlisted the assistance of a notable group, including Professor Silliman of Yale University, who prepared a favorable report; William Morris Stewart, U.S. senator from Nevada; and Major General Robert C. Schenck, the U.S. minister to the Court of St. James. This group was assisted in its efforts by one Baron Grant, a promoter who, in return for providing introductions to British financiers, would receive 10% nominal capital of any new company. A prospectus for the Emma Silver Mining Company was issued in 1871, offering £1,000,000 in shares at £20 per share. In addition to Schenck, Stewart, Baxter, and Park, the company's board included three members of Parliament. Professor Silliman's report was taken at face value, and no independent evaluation was made. The shares were soon selling at £3 to £4 above the offering price.

At first, the mine was highly productive, yielding daily a hundred tons of ore containing 5.7 to 20.10 kg (200 to 700 oz) of silver per ton, and within a few months $1,500,000 in ore had been mined (Rickard 1932). However, there was considerable skepticism in the United States. The *Engineering and Mining Journal* commented, "We do not see in the prospectus

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of the company any justification for these high figures, except a historical one. The mine appears to be valued at five million dollars because it has produced some two million already, and no positive proof is offered as to the amount of ore actually known to be in reserve—at least, none that we have seen" (Anon. 1871).

At the end of 1872, the company director "made the astounding discovery that their stock of available ore was exhausted, everything in sight worth taking had been extracted without any new ore bodies having been discovered" (Jackson 1955). In the annual report issued on March 1, 1873, the directors expressed their "regret and disappointment" that the balance sheet was much less favorable than they had until recently anticipated. The shareholders were incensed, the mine in Utah was closed, and most of the work force dismissed. The share price dropped rapidly from £23 to one-tenth that amount.

By this time, Schenck had resigned from the board of directors, but as the lawsuits proliferated, back and forth across the Atlantic, the U.S. House of Representatives appointed a special committee to investigate his connection with the Emma mine promotion. Months of testimony resulted in 879 pages of proceedings and the finding that Schenck's conduct had been "ill-advised, unfortunate, and incompatible with the duties of his official position" (Jackson 1955).

Until 1892, exploration continued, companies were formed and dissolved, and sporadic but generally unprofitable production at the Emma mine continued. In 1894, the directors of the company that held the original Emma claims voted to devote their remaining small capital to the investigation of West Australian gold properties.

Unfortunately, even today, some of the errors made by investors in the Emma mine have not disappeared. Stock is issued and investment decisions are made on the basis of a single "expert" report, without seeking independent verification. Investors are impressed by the political, social, or economic standing of company officers and promoters, and fail to determine whether any of those individuals have the training and experience needed to operate a mining company. Past production is all too often taken as demonstrable evidence that reserves are still in the ground.

MINE ECONOMICS

This section addresses four topics related to the basic premises of the profitable exploitation of a mineral deposit:

1. Evaluation of mineral properties
2. Mineral property feasibility studies
3. Cost and cost estimation methods
4. Investment analysis

Financing of mining projects is addressed here but is discussed more thoroughly elsewhere in this handbook.

Mineral Property Evaluation

The deposit must be characterized adequately and with the required degree of certainty. The extent of the deposit must be defined, usually in terms of the amount of ore present. The quality or tenor of the deposit must also be defined, usually expressed for metal deposits as grade: the fraction of metal present, in milligrams per ton for precious metals, or as a percentage for base metals. For coal, quality is characterized by the impurities present, specifically sulfur and ash; the heating value; and (for metallurgical coal) the coking qualities. For industrial minerals, user requirements are often specific in terms of impurities and other properties, and standard expressions for quality are less common.

In all cases, the entire makeup of the deposit must be considered. The presence of certain impurities may render valueless an otherwise attractive deposit. For example, a limestone deposit that contains too much silica may not be suitable for the manufacture of cement. In other cases, the presence of an otherwise valuable constituent may lead to complications in the ore processing. For example, a gold–silver ore that has too high a silver content may require more expensive methods for recovery of the precious metals from the leach solution. Thus the characterization of a deposit must be carried out by a diverse team, whose members thoroughly understand geology, mining, mineral processing, metallurgy, and chemistry.

Besides characterizing the quality of the deposit, the evaluation will also include an analysis of the legal status of the mineral holdings. This will include determination of who owns the mineral or mining rights, who owns or controls the surface rights, who controls access to the property, and who owns the required water rights. The legal definitions of these rights vary widely among countries and even among states or provinces within some countries.

In many cases, the evaluation or characterization of a deposit will include a preliminary assessment of the feasibility of mining that deposit. That assessment will focus particularly on any characteristics of the deposit (and its locality) that may be problematic. Examples include deposits

- In particularly remote locations;
- Where permitting may be difficult;
- Where local opposition may be extreme;
- Where water, fuel, or electric power are expensive or unavailable in sufficient quantities; or
- Where the political climate is unstable.

These preliminary assessments are often called audits—environmental, regulatory, social and cultural, and so forth—and are described extensively in the literature.

Inaccurate or incomplete characterization of a mineral deposit can have serious consequences. Such inaccuracies may or may not be intentional, but once the numbers are in print, the tendency to use them for raising money may be difficult to resist. Errors in reserve estimation, intentional or not, continue to occur. Though such cases rarely become the subject of congressional hearings, as the Emma mine did, they do have serious consequences.

The requirements for accurate characterization of deposits are given in various national standards, as described in detail by Bourassa et al. (2003). These standards have been adopted by stock exchanges, government agencies, and professional societies to ensure uniformity and accuracy in reports describing mineral deposits.

In 1988, at the request of members of the Society of Mining Engineers (later changed to the Society for Mining, Metallurgy, and Exploration, or SME), the president of SME formed Working Party No. 79, Ore Reserve Definition. Its mission was to develop guidelines for the public reporting of exploration information, resources, and reserves. In 1989, the *Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves* (the JORC Code) was published and was immediately incorporated into the Australian Stock Exchange listing rules. In 1991, SME's guidelines were first published, and the Institution of Mining and Metallurgy in the United Kingdom revised its standards for reporting of mineral

resources and reserves (SME 2007). The U.S. guide and U.K. revisions were both based on the 1989 JORC Code.

In the collapse of the Bre-X project in Indonesia (Danielson and Whyte 1997), private and institutional investors suffered huge losses when it was revealed that exploration results had been incorrectly reported. After that, international standards took on increasing importance (Cawood 2004). *The SME Guide for Reporting Exploration Information, Mineral Resources, and Mineral Reserves* was updated in 1999 when the reporting of mineral resources and reserves was required to be made by a "competent person," as defined therein (SME 2007). The *SME Guide* was then recommended for use by SME members. However, the U.S. Securities and Exchange Commission (SEC) did not recognize "resources," as defined in the *SME Guide* and other documents, in its evaluation of proposed mining projects (Kral 2003). When SME became a recognized overseas professional organization, it instituted its registered member category in 2006. Those who qualify can obtain this membership upgrade through SME. Applicants must first meet strict educational and professional standards and undergo a vetting process by the Society's Admissions Committee (Gleason 2007).

In an effort to resolve the differences between the *SME Guide* and the SEC rules and regulations for 2007, a revised version of the *SME Guide* was issued, which included improved definition of the term *mineral resources* and its subdivisions (measured, indicated, and inferred mineral resources), and clarification of the technical, economic, legal, and permitting requirements that must be satisfied before a reserve can be declared. A section was added defining the commodity prices that can be used for reserve estimation and reporting, and how price sensitivity should be measured during periods of low prices. Documentation requirements were clarified, including the requirement for a Mineral Reserves Declaration Report. The role of the competent person was reemphasized. However, the position of the SEC with respect to public reporting remains that stated in *Industry Guide 7* as interpreted by SEC staff (SEC 2007). Consequently, at any given time, some key aspects of the *2007 SME Guide* may be inconsistent with SEC requirements. The SME document, *The Guide for Reporting Exploration Results, Mineral Resources, and Mineral Reserves* (SME 2007) is available at the SME Web site.

When a big financial scandal like that of Bre-X is exposed, the details of the how the property was incorrectly valued are closely scrutinized. In contrast, companies and individuals are understandably reluctant to publicly discuss projects that were undertaken in good faith but fail nonetheless (SME 1998). Thus, it is difficult to assess the role of incorrect property valuations in those failures. Guarnera (1997) sums up the effect of geological risk in recent projects as follows: "No single feature has caused so many mining projects to fail as have reserves not being what were originally estimated by the mining company. A few examples of projects which have had notable reserve problems are:…Hayden Hill…Cove-McCoy…[and] Grouse Creek."

A brief discussion of these projects is instructive.

The McCoy property in Nevada (United States) was acquired in 1986, and the nearby Cove prospect was discovered in 1987. By 1988, the owner reported proven and probable mineable reserves at Cove of 65.3 t (metric tons) of gold and 3 kt of silver, with drill-indicated possible mineralization of 83.9 t of gold and 4.5 kt of silver (Emmons and Coyle 1988). In 1990, the owner reported that proven and probable gold reserves at Cove were cut by 11% to 4.6 million troy oz, and estimates of other mineralization at depth had dropped from 4.5 million oz of gold to about 900,000. The error was attributed to downhole contamination of cuttings in reverse circulation drilling (Gooding 1990). In 1993, Amax Gold reported a US$64.1 million write-down of the carrying value of Hayden Hill (*Globe and Mail* 1993). In 1996, the owner incurred a $30 million expense to stabilize the pit wall (*Globe and Mail* 1997). By 1998, production had been scaled back (Bagnell 1998), and by the middle of 2002, no production from Cove-McCoy was reported (*National Post Canada* 2002).

In 1992, the Grouse Creek mine in Idaho (United States) was permitted, with initial annual production expected at 3.1 t/a of gold and 12.4 t/a of silver, changing later to 2.2 t/a of gold and 77.8 t/a of silver (Anon. 1992). By 1995, the owner announced it was writing down its entire $US95-million investment in the mine because it had encountered significant shortfalls in both grade and tonnage of the ore being mined (*Globe and Mail* 1995).

Completion of permitting for the Hayden Hill project in California (United States) was announced by Haddon (1992). In 1993, the owner reported a US$64.1-million write-down of the carrying value of Hayden Hill (*Globe and Mail* 1993). Schwab et al. (1994) assert that some of the tonnage and grade shortfall that was experienced at Hayden Hill may have resulted from an absence of adequate drilling, obscured by variogram analysis that supposedly showed the grade variations within the range of low variance over distances which, on close inspection of the geology, could not be supported.

Finally, it is important to remember that reserve risk is not always a function of resource risk. Modifying factors in the various standards are meant to account for the conditions that should be considered when converting a mineral resource to an ore reserve.

Mineral Property Feasibility Studies

A feasibility study of a mining project is an appraisal of the commercial viability of that project, which accounts for engineering, economic, permitting, and environmental variables. Like mineral property evaluations, feasibility studies require the dedicated participation of a diverse group of skilled professionals. In addition to an understanding of geology, mining, mineral processing, metallurgy, and chemistry, the feasibility study team must also comprehend cost estimation, construction and project management, civil engineering, electrical engineering, permitting and other legal requirements, economics, and finance.

To control costs and to cater to the progressive increase in geological confidence and understanding of the modifying factors, feasibility studies are usually conducted in stages. If a study at the first stage produces favorable results, indicating the property may be exploited profitably, a more detailed (and costly) study will be conducted, resulting in an increased certainty of the project economics. Some mining companies have internal experts who conduct feasibility studies, while others rely on engineering firms or consultants to fulfill this function. In all cases, feasibility studies are conducted to defined standards, stating the requirements for a study at a given level of certainty. Although these standards vary somewhat, in all cases the notion is the same: increasing the detail increases the certainty but costs more money. For example, Vancas (2003)

Table 5.1-1 Criteria for feasibility studies

Type of Study	Completion of Engineering Documents*, %	Probable Error, %
Rough order-of-magnitude	2	±35–45
Preliminary (conceptual or scoping)	5	±25–35
Budget appropriation (prefeasibility)	15	±15–25
Project control (feasibility)	27	±10–20
Definitive	40	±5–15

Source: Adapted from Vancas 2003.
*Drawings, specifications, procedures, etc.

describes the scheme used by Bateman Engineering for defining types of feasibility studies, as shown in Table 5.1-1.

Although not specified in the scheme of Table 5.1-1, the steps needed for permitting and reclamation should also be considered, and are often included, in the various stages of a feasibility study.

It is important to remember that any feasibility study is only as good as the information used to prepare it. Although this may seem obvious, many errors can lead to inaccuracies in a study. Gypton (2002) reviewed 60 projects and found only 15 came in under budget. Of the 45 over budget, 35 were more than 15% over. Of all the projects, 25 were within the ±15% criteria. Similarly, Bertisen and Davis (2007) reviewed 63 mining and smelting projects completed over four decades and found that as-built capital costs averaged 25% higher than estimates at the bankable feasibility study stage. About half of the projects had as-built capital costs outside the expected ±15% of the feasibility study capital cost estimate, and cost overruns of 100% or more occurred in roughly 1 of 13 projects.

What could lead to such errors? Vancas (2003) gives a list of project pitfalls:

- Being forced into unrealistic deadlines
- Not defining the scope of the project clearly at the beginning of the project
- Allowing changes of the scope without documenting them or determining their impact to the schedule, resources, and project budget
- Getting senior management's attention too late for them to help
- Inability to say "no" (even when obvious that what is being requested is impossible)
- Not establishing communication channels from the beginning of the project
- Not establishing a control mechanism to track and monitor the project
- Deserting control mechanisms when the project starts getting into management by crisis
- Continually reorganizing the project team
- Committing to arbitrary dates with no real basis for setting those dates
- Building up staff too quickly when work is not ready and/or disbanding support staff too quickly
- Not having a person in charge of the project with responsibility, accountability, and authority
- Not freezing the specifications and other baseline definitions

To this list could be added the too-common errors of confusing precision with accuracy and not understanding the inherent risks associated with mineral resources, particularly at the inferred resource category. Bertisen and Davis (2007) asserted that the persistence of bias is intentional, driven by a scarcity of project financing and the need to inflate the project economics in a bid to secure financing.

Danilkewich et al. (2002) provide guidelines for the project owner in preparing for a feasibility study. They suggest the owner be responsible for preparing a complete scope of work, a full delineation of assumptions and constraints, a well-defined execution plan, and a comprehensive bid checklist.

Feasibility studies are an indispensable tool in the decision-making process that leads up to the development of a mineral deposit. However, their preparation and interpretation must be carefully managed to ensure satisfactory results. Northcote (2007) provides an excellent discussion of how to minimize functional risks during a project evaluation and summarizes it thus:

> To reduce project failures, the foundations need to be properly laid. This starts with the stakeholders understanding the project life cycle versus the value of the impact of change and having a quality project evaluation report.
>
> A competent project evaluation manager needs to be appointed who will select appropriately skilled and managed study groups to assist in identifying the opportunities and the risks. Setting up of the study at the outset so that all participants know the systems, schedules and objectives is crucial to a focused and on timely outcome. Once the project evaluation has commenced, retention or access to key people will reduce revisiting concepts and strategies that more than likely have been addressed in the evaluation process. This can reduce cost and schedule impacts.
>
> Project evaluation management is challenged in keeping study groups focused and addressing the risk issues in a timely manner. Regular scheduled meetings, competently chaired, generate synergies that keep the study groups focused. Special review meetings are to be scheduled throughout the project where peer reviewers are invited to test the other's findings.
>
> Before a commitment can be made by the stakeholders, in addition to the usual documentation covering the mining, technical, budget estimates, construction schedule and market aspects, there needs to be a design criteria and a project execution plan.
>
> Short cuts during project evaluation will result in a weak foundation and increase the risk of project failure.

Cost Estimation

Cost estimation is a part of every feasibility study. A mineral deposit should not be considered for development unless the estimated annual operating profit after taxes and other costs is sufficient to recover, with interest, the cost of developing the mine and of closing and reclaiming it. Such considerations should also include sufficient income to provide for mine closure and final reclamation costs. This is often neglected, and poor performance at closure can lead to bad public relations, denial of future permits, and even bankruptcy.

As a project progresses through the types of feasibility study previously described, the cost estimates are successively

Table 5.1-2 Specifications for capital cost estimates in feasibility studies

Category	Conceptual or Scoping Study	Prefeasibility Study	Feasibility Study
Basis, to include the following areas: civil/structural, architectural, piping/HVAC, electrical, instrumentation, construction labor, construction labor productivity, material volumes/amounts, material/equipment, pricing, infrastructure	Order-of-magnitude, based on historic data or factoring	Estimated from historic factors or percentages and vendor quotes based on material volumes	Detailed from engineering at 15% to 25% complete, estimated material take-off quantities, and multiple vendor quotations
Contractors	Included in unit cost or as a percentage of total cost	Percentage of direct cost by area for contractors; historic for subcontractors	Written quotes from contractor and subcontractors
Engineering, procurement, and construction (management) (EPC(M))	Percentage of estimated construction cost	Percentage of detailed construction cost	Calculated estimate from EPC(M) firm
Pricing	FOB mine site, including taxes and duties	FOB mine site, including taxes and duties	FOB mine site, including all taxes and duties
Owner's costs	Historic estimate	Estimate from experience, factored from similar project	Estimate prepared from detailed zero-based budget
Environmental compliance	Factored from historic estimate	Estimate from experience, factored from similar project	Estimate prepared from detailed zero-based budget for design engineering and specific permit requirements
Escalation	Not considered	Based on company's current budget percentage	Based by cost area with risk
Working capital	Factored from historic estimate	Estimate from experience, factored from similar project(s)	Estimate prepared from detailed zero-based budget
Accuracy	±50%	±25%	±15%
Contingency	25%	15%	10% (actual to be determined based on risk analysis)

Courtesy of M.A. Holden.

Table 5.1-3 Specifications for operating cost estimates in feasibility studies

Category	Conceptual or Scoping Study	Prefeasibility Study	Feasibility Study
Basis	Order-of-magnitude estimate	Quantified estimates with some factoring	Describes the basis of the estimate; detailed from zero-based budget; minimal factoring
Operating quantities	General	Estimates with some factoring	Detailed estimates
Unit costs	Based on historic data or factoring	Estimates for labor, power, and consumables; some factoring	Letter quotes from vendors; minimal factoring
Accuracy	±35%	±25%	±15%
Contingency	25%	15%	10% (actual to be determined based on risk analysis)

Courtesy of M.A. Holden.

refined and made more accurate. At higher levels of certainty, more detailed drawings and more directly quoted prices for major equipment are required. One approach defines three types of feasibility study: conceptual or scoping, prefeasibility, and feasibility. Tables 5.1-2 and 5.1-3 show the specifications for the capital and operating cost estimates, respectively, in each type of study.

The correct completion of a cost estimate requires consistency, attention to detail, and good sources of cost information. Many equipment suppliers have proprietary software for estimating capital and operating costs, which may often be used at no cost by prospective customers. For example, Caterpillar equipment costs can be determined directly from the company's Build and Quote Web site (Caterpillar 2010). In addition, regularly updated cost data are available by subscription at the *CostMine* Web site, maintained by InfoMine (2010). Inaccurate cost estimates will result in cost overruns and may result in the expenditure of large amounts of capital funds on a project later found to be unprofitable, or conversely, the rejection of a project that would have resulted in significant profits.

Investment Analysis

An investment analysis may be carried out as part of a feasibility study or as a separate effort. In any case, the purpose of the analysis is to determine whether or not development of the project will provide sufficient economic returns to justify the required initial and ongoing investment required. The analysis must consider the cost of the capital funds employed and the risk involved in the project.

For any development project, investment of the required funds must be justified to the funding source. When a company considers investing its own funds in a project, that alternative will be compared with other available investment opportunities. Those may include other new projects, improvements to existing facilities or equipment, or additional exploration for new prospects. If funding is to be sought from outside investors, those investors will make the same sort of comparisons.

Several criteria may be used for analysis of investments. Some are relatively easy to calculate, such as the accounting rate of return and the payback or payout period. Other criteria are calculated using discounted cash-flow methods, which are more complex. These include present value, future value, annual value, net present value (NPV), benefit–cost ratio, and internal rate of return (IRR).

NPV is the most commonly used, single criterion for comparing investments, but some analysts also use the IRR. There is some controversy over the use of these two criteria. Torries (1998) states that "both NPV and IRR have valid uses as merit measures for practical application of investment evaluation methods [and] IRR has no greater number of faults than does NPV, even when multiple root problems are included," whereas Hajdasinski (2000) believes that "the IRR is a conceptually flawed and operationally dysfunctional project evaluation criterion."

Discounted cash-flow (DCF) methods require detailed and extensive calculations, and executing them by hand requires considerable skill and patience. The development of personal computers has made DCF analysis much easier. Commonly available programs, such as Microsoft Excel, include built-in functions for many of the parameters associated with DCF analysis, and more sophisticated programs are readily available. Unfortunately, the relative ease with which DCF analysis may now be conducted has not altered the fact that the results of such an analysis are only as good as the input data. In some cases, when a complex analysis is done by computer, it is tempting to simply assume that the results are valid without rigorously reviewing the input data and assumptions.

Other drawbacks to DCF methods are summarized by Clevenger (1998) and Lawrence (2000). For example, it is difficult to estimate some of the key parameters, such as future cash flows and discount rates, and the practice of subtracting the cash flows of one project from those of another that is mutually exclusive (before discounting) can produce incorrect results. In addition, DCF measures do not directly recognize the value of future opportunities, unless the uncertainty regarding the execution of those opportunities is estimated and included in the analysis. Finally, small changes in the discount rate used can dramatically change the results of the analysis. Regarding this last point, Lawrence notes that "whilst it is preferable for valuations by DCF/NPV modeling techniques to give as much detail as possible in the derivation of the technical basis of the inputs used and the Discount Rate selected, it is more important for it to contain a table or graph showing the impact on the valuation of a change of 1% in the Discount Rate, from say zero to 15% per year (in real terms). This allows the reader to truly test the reasonableness of the valuation by estimating a value based on other Discount Rates."

Statistical simulation methods are often used to more accurately quantify the range of error associated with cost estimates and investment analyses. These methods are discussed in subsequent chapters in this handbook.

MINE MANAGEMENT

The topics of leadership, employee relations, and training, discussed in subsequent chapters of the handbook, are all important components of mine management, which will be discussed in general here.

Engineers are often skeptical of management experts. An anonymous cynic defined a manager as "someone who can always tell you what you've done wrong, but never tell you what to do next." Nonetheless, the operation of a single mine or a mining company requires the expert and careful management of dozens, if not hundreds, of functions and tasks.

Historic Approaches

The scope of topics included under the heading "Mine Management" has grown considerably as the industry has progressed and adapted to changing conditions. Early handbooks were often directed specifically to the country in which they were published and addressed practical matters such as mine organization; business and technical management; accounting principles; cost-keeping; mine records; wage schemes; contract work; bonus, cooperative, and leasing systems; methods of paying wages; accident compensation; pensions and benefit funds; labor relations; arbitration and conciliation boards; changehouses; mine communities; miners' dwellings; potable water supply systems; sanitation and diseases encountered in mining; and worker health and safety.

Contemporary Management Values

It is interesting to compare the historic topics with those in the public statement of the Rio Tinto Group, a large, multinational mining company. Rio Tinto first published *The Way We Work—Our Global Code of Business Conduct* in 2003; it was last updated in 2009 and is available in print and on the Internet (Rio Tinto 2009). Although it is intended to provide the company's employees with guidance on how to conduct themselves at work and when representing Rio Tinto, the document by implication describes Rio Tinto's management approach to corporate responsibility, sustainability, and integrity. All employees are strongly encouraged to report any violations of law and are provided with the means to do so. Strong commitment is expressed for important values:

- Incident- and injury-free workplaces
- Protection of health and well-being
- Excellence in environmental performance and product stewardship
- Respect for the rights and dignity of Rio Tinto's employees and those of its business partners
- Respect for human rights consistent with the Universal Declaration of Human Rights
- Strong relationships with communities and indigenous peoples
- Avoidance of conflicts of interest
- Prohibition of bribes and corruption, in all forms
- High ethical standards in dealing with governments
- Accurate and consistent communication with the media and investors
- Maximum transparency consistent with good governance and commercial confidentiality

Clearly, the preceding list does not include all the issues managed by Rio Tinto and its employees. Rather, the company must deal with issues covered in the historic handbooks plus those described in *The Way We Work*. That second set of issues may be thought of as *higher values*, which must now be rationally and consistently managed by all mining companies. Management of higher values is important first because it is simply the ethically and morally correct thing to do. Second, because mining companies continue in business by public consent, when values like those expressed in *The Way We Work* are not upheld, a company loses credibility and may eventually lose its license to operate in a given location.

The International Organization for Standardization (ISO) has prepared standards for environmental management, ISO 14001, and quality management, ISO 9001 (ISO 2010). Full discussion of the management of these values is beyond the scope of this chapter. However, some brief examples are instructive.

Labor Relations Management

Companies that manage higher values have an approach to labor relations that is significantly different from the historic norm. Zanolli (1972) notes that the United Mine Workers of America has been "involved in widely publicized and bitter battles with the coal industry employers in collective bargaining…[and]…has even battled with the government and in 1947 had the experience of collectively bargaining with the Government when the coal mines were taken over by the Federal Government." This adversarial relationship of miners' unions with employers and governments was common in many countries until the 1980s. The author recalls being told in 1987, by a West Virginia mine superintendent, "Anything I can do to get rid of a union miner is good. Every union miner is just a problem." Contrast this with a statement made by Leigh Clifford, then-CEO of Rio Tinto, in 2007: "Do you remember how anarchic labour relations nearly throttled WA's [Western Australia's] iron ore industry in the 1970s and drove our chief customer to encourage supply from Brazil? Today, work practices in our mines are more rational and everyone—employees, customers, companies and governments, is better off as a result" (Clifford 2007). Progressive mine managers recognize that their employees constitute a resource equal in value to their ore reserves and treat them accordingly.

Safety Risk Management

The concepts of risk management appeared in the 1970s (Field 2003) and were first applied in the petrochemical, nuclear, military, and aerospace industries. This proactive approach to improving risks, as opposed to a reactive "fix it when it breaks" mentality, was in most cases triggered by a major public disaster such as the Flixborough (England) chemical plant disaster in 1974, the Three Mile Island (Pennsylvania, United States) nuclear plant event in 1979, the *Piper Alpha* offshore oil platform disaster in the North Sea in 1988, and others (Joy and Griffiths 2007).

The management approach to a given issue can be either proactive or reactive. Proactive change involves actively attempting to make alterations to the work place and its practices. Companies that take a proactive approach to change are often trying to avoid a potential future threat or to capitalize on a potential future opportunity. Reactive change occurs when an organization makes changes in its practices after a threat or opportunity has already occurred (Reference for Business 2010). Risk management methods enable the systematic application of a proactive approach.

For many years, the management of safety in the mining industry was reactive. Breslin (2010) notes that "most of the Federal safety and health legislation has followed major mining disasters that received significant public attention." In some cases, a reactive response still occurs. As recently as 2006, three mine disasters in the United States (with a total of 19 fatalities) resulted in the passage by the U.S. Congress of the Mine Improvement and New Emergency Response (MINER) Act. Popovich (2010) observed that, since 2006, U.S. coal companies have spent almost $800 million on new mine safety technology and equipment, much of which was required for compliance with the MINER Act. Unfortunately, although reactive management can control many conditions, it cannot control them all. In addition, behavior control, whether individual or corporate, is achieved more successfully by proactive methods.

Risk management has been applied in many areas of the mining industry. As early as 1974, Matthews (1974) recognized that "most of the contractual problems related to underground construction are associated with risk and its management. Unless all of the ramifications of this subject are understood, it will be difficult to employ contracting practices best suited to the needs of a particular project. It is hoped that a detailed study of the nature of risk will assist in this understanding." Hebblewhite (2009) describes the use of risk management techniques for the control of geotechnical hazards in Australian mines. Assessment of risks, including political risk (Gavelan and Dessureault 2004), is also a standard part of almost all mineral property evaluations and project feasibility studies.

The application of risk management techniques to mine safety has been notably successful. The Australian mining industry initially identified the use of risk-based management techniques during the 1980s. Using research studies performed by the coal sector, which evaluated techniques used in the nuclear industries of various northern hemisphere countries, the industry identified the scope of applying a risk-based approach to mining. Primarily because mining has many uncertainties and a large number of variables, a clear-cut answer cannot always be defined for every situation. The value of risk management became clear in the mid-1990s, shortly after the 1994 Moura coal mine explosion in Australia, in which 11 miners died (Hopkins 2000). As a result, the mining industry began using risk analysis methods to mitigate certain key hazards.

By 1997, regulatory bodies in Australia began to require safety management plans for principal hazards. Western Australia passed the Mines Safety and Inspection Act (Western Australia 1994), and in New South Wales, the chief inspector of coal mines published a risk management handbook (NSWDPI 1997). Queensland issued standards the next year (QDME 1998; QMC 1999). These regulations require mines to perform major hazard risk assessments on a regular basis to address the possibility of unwanted events such as spontaneous combustion, gas outbursts, explosions, air blasts, inundations, and roof falls.

A comparison of fatality rates (number of fatal injuries per million hours worked) for underground mining from 2004 to 2006 indicates that the risk management approach to mine safety is having a marked effect (Table 5.1-4).

In 2001, the Minerals Council of Australia initiated a national project to promulgate a good practice guideline for risk assessment in the minerals industry. The Minerals Industry Safety and Health Centre (MISHC) at the University of Queensland was commissioned to draft guidelines, working closely with a representative cross section of the industry, which included seven large mining companies and nine government agencies. The resulting document, National Industry Safety and Health Risk Assessment Guideline, Version No. 7 (Joy and Griffiths 2007), is an exhaustive discussion of risk management as applied to mine safety. It includes descriptions

Table 5.1-4 Underground mine fatality rates 2004–2006, United States and Australia

Country and Commodity	Average Fatality Rate (fatal injuries per million hours worked)
U.S. coal	0.25
U.S. metal/nonmetal	0.14
Australia coal	0.04
Australia metalliferous	0.07

Source: Adapted from Iannacchione et al. 2007.

of methods and procedures, examples of forms, worksheets, reports, and other valuable resources. MISHC provides an on-line resource of information on mining industry risk management through its Minerals Industry Risk Management Gateway (MIRMgate 2010).

The use of risk management methods to manage mine safety has spread rapidly. Safety standards and practices in Great Britain, which apply to all industries and workplaces, are described in *A Guide to Health and Safety Regulation in Great Britain* (HSE 2009). They were developed in 1992 and are quite similar to those in Australia. Many large mining companies, including Alpha Natural Resources, Anglo American, Barrick Gold, BHP Billiton, Newmont Mining, Rio Tinto, and others, have embraced an approach to safety, health, and environmental quality that uses the principles of risk management.

MINING LAW

Mining law traditionally refers to the body of law governing access to mineral deposits, the right to mine those deposits, and the taxes or royalties assessed on the products of mining. In the last half century, many other laws that affect mining operations have been enacted. Those laws vary considerably from one country to another, and even within countries, but it is useful to consider them in six general (if somewhat overlapping) categories:

1. Access to the land
2. Right to mine
3. Taxes, leasing fees, and royalties
4. Employment, work conditions, and compensation
5. Environmental protection
6. Cultural and social issues

Access to the Land

These laws will govern which lands may be accessed for exploration and mining. Restrictions may be placed to protect forests, parks, and cultural resources. Regulations may specify access or leasing fees for exploration or mining. Companies may be required to attain more than one access right. For example, in some jurisdictions, the surface and mineral estates may be severed; that is, a company or individual may own surface and mineral rights but may be denied vehicular access to its property through a national park or other protected area. In other jurisdictions, such as Great Britain and South Africa, the state owns all minerals. The concept of state ownership of minerals leads to a system of mineral leases, or concessions, which requires some effort to understand. The distinction between real property and chattel, or personal property, is significant with respect to the distinctions between ownership of a mineral reserve in place and the mineral product after it has been mined or the wastes that are stored on the mine property after beneficiation.

Right to Mine

Unless the lands containing a mineral deposit are purchased in fee simple, the right to mine begins with permission from the property owner to enter the property. If this right is granted by means of a lease, then the lessor may require evidence that a social license to mine can be obtained.

In most jurisdictions, the right to mine requires one or more permits from government agencies. In some cases, the mining permit may be secured with relative ease, but mining cannot begin until several environmental and other permits are also in place. In some jurisdictions, the permitting process may seem opaque and difficult, especially to outsiders.

The right to mine may be forfeited if a company violates any of the laws to which it is subject. In some cases, this forfeiture may be a direct result of government actions; in other cases, it may be the *de facto* result of large fines or other penalties. In the worst case, a national government may confiscate or nationalize the property, mining claims, equipment, and all other assets of a mining company.

Taxes, Leasing Fees, and Royalties

Taxes and fees paid by a mining operation can be complicated. In many cases, a mining operation may be taxed by several entities. For example, a mine in the United States may include lands where ownership is divided among the federal and state governments and private holders. That mine could well be required to pay leasing or claim fees to the federal and state governments, corporate income taxes to the federal and state governments, severance taxes or royalties to the state government, real property taxes to the county government, and royalties to the private landholder. Some government agencies require the payment of lease fees and royalties in advance, and this may significantly increase the capital investment required to place a property in operation. During the feasibility analysis, it is important to consider the likely tax and royalty liabilities for a project.

When taxes, fees, and royalties are set by government agencies, they may be subject to change at short notice. In some countries, laws may be changed with little or no regard for existing contracts and agreements, and these changes can suddenly and drastically alter the economic viability of a project. For example, although *ex post facto* protection is taken for granted in the United States and many other countries, this is not the case everywhere, and the local situation should always be carefully investigated.

Employment, Work Conditions, and Compensation

Laws may specify wage rates and required benefits packages in a given location; limit work hours; determine the conditions for operation of labor unions; or specify the makeup of the work force in terms of ethnic or gender diversity, percentage of local or native residents to be employed, and so forth. Other laws may include provisions to protect the safety and health of workers, including required personal protective equipment, safe working conditions, safety requirements for equipment and machinery, and compensation for workers injured or made ill by working conditions.

Environmental Protection

Detailed discussion of environmental assessments, environmental impacts, and other environmental issues may be found in Part 16 of this handbook. This chapter discusses only general considerations.

Environmental laws cover many areas. They usually govern the use and contamination of surface and groundwater, the discharge and storage of solid waste (including dust, waste rock, and tailings) or domestic waste (garbage), the disposal of radioactive and other hazardous materials, and the control of gaseous emissions from equipment and processes. In most locations these laws also cover the protection of endangered and/or protected species or ecosystems (including parks, forests, rivers, and lakes) and the protection of *viewsheds* (including the night sky). Entities protected under environmental laws may be widely distributed in space and time—for example, a caribou herd or a salmon fishery.

It is important for mining companies and mining engineers to be aware of some of the land-use planning concepts under which lands may be designated under a certain usage category and thus rendered unsuitable for mining. Unsuitability may be afforded to areas with natural hazards, renewable resources, fragile ecosystems, historic sites, and for which reclamation is technically or fiscally not feasible. This land-use concept is also called *legal sterilization of lands*. Mine designers who are unaware of the designation concept and its application in environmental protection laws may find their pet projects stalled when they least expect it.

Environmental laws usually govern the restoration or reclamation of sites and features disturbed or altered by mining activities. Reclamation requirements may be specific and detailed, and extend far into the future, representing a liability that is difficult to quantify. Many government agencies require the posting of a bond to guarantee compliance with reclamation requirements. In some circumstances (when risk cannot be adequately quantified), bonding companies are unwilling to issue a bond. In such cases, the mining company is required to post the full amount required. In the worst case (usually for a smaller company), this will effectively halt the project; in the best case, it will increase the upfront capital cost of the project.

Cultural and Social Issues

Laws relating to cultural and social issues may require protection of archaeological or historic heritage sites, protection of cultural heritage sites, and control of traffic from material haulage or employee travel. They may also regulate the construction of employee housing; specify the steps to be taken when local residents are relocated to accommodate mining operations; and prescribe compensation for property, water, subsidence, and access rights. Some local jurisdictions may require or request a mining company to make investment in local infrastructure as part of a mine development program.

Cultural and social issues are difficult to quantify, and laws regarding them can be subject to widely varying interpretations. Organizations opposed to a mining project often use these issues as the basis for objecting to the project, even after the required permits have been issued.

Mining Law in a Global Business Climate

It is often said that "gold is where you find it." Mining companies have historically operated in diverse global locations, and that is still the case. Companies must be prepared to conduct exploration, development, and production almost anywhere in the world. Although the technical requirements for these activities will differ in various locations, much greater differences will be found in the legal, political, and cultural requirements. Thus, it is important for a company to have employees who are well trained and experienced in the laws and practices of each country in which the company operates.

Almost all companies now state clearly their intention to operate in full compliance with all applicable laws and regulations. In almost all cases, mining companies are also committed to operating in compliance with the highest and best environmental standards, often those of the ISO or a similar organization, even if the local laws are less stringent.

Mining companies must be fully aware of all laws that apply to their operations. This will, of course, include local laws, but laws of the company's home country may also apply. For example, the U.S. Foreign Corrupt Practices Act of 1977 sets forth standards for accounting transparency and prohibits bribery of foreign officials (U.S. Department of Justice 2004). It applies to any U.S. or foreign corporation that has a class of securities registered or that is required to file reports under the Securities Exchange Act of 1934; to any individual who is a citizen, national, or resident of the United States; and to any corporation and other business entity organized under the laws of the United States or having its principal place of business in the United States.

Estimating Legal and Political Risk

The preceding discussion of mining law has alluded to instances when changes in political regime or local laws and regulations can seriously affect the viability of a development project or mining operation. The assessment and quantification of legal and political risk is one of the biggest challenges for a mining company, which must rely on its experience, internal expertise, and the advice of qualified consultants.

An annual survey of mineral development potential is conducted by the Fraser Institute of Vancouver, British Columbia, Canada (McMahon and Cervantes 2009). In 2008, 658 mining and exploration professionals responded to the survey, which calculates the policy potential index and the current mineral potential index. The policy potential index measures the effects on mineral exploration of government policies, including uncertainty over the administration, interpretation, and enforcement of existing regulations; environmental regulations; regulatory duplication and inconsistencies; taxation; native land claims and protected areas; infrastructure; socioeconomic agreements; political stability; labor issues; geological database; and security.

The current mineral potential index is based on whether or not a jurisdiction's mineral potential under the current policy environment encourages or discourages exploration. There is considerable overlap with the policy potential index, probably because good policy will encourage exploration, which in turn will increase the known mineral potential.

These indices provide a useful assessment of the risks associated with mineral exploration in the areas included in the survey: 7 states in Australia, 12 provinces in Canada, 14 states in the United States, and in 34 other countries. The Fraser Institute survey also includes comments made by respondents, which provide valuable insights based on their experiences. Because some of these comments illustrate how the legal and regulatory climate in a country can affect exploration projects, they are reproduced here to emphasize those effects. Because this handbook will be a reference for many years, and because political conditions in many locations can change unexpectedly, the names of countries and political leaders are not given.

> In [location A], title and laws mean nothing. The law is what [the government] says it is at any given time—and [the government] is an amorphous political party. After spending $10 million on exploration in [location A] we were stonewalled by [the government] for four years as a means to deny us the final production permit.
>
> —Exploration company, company president

> [Location B] is actively seeking independence and looking towards minerals and petroleum to fund the country.
>
> —Exploration company, manager

> In [location C], the landowners status is entrenched in law; therefore, tenure is secure, with the…government responsible for allocating royalties to their citizens. The government can choose to participate in a mining venture but, they purchase their position at fair market value and, only BEFORE, the project becomes revenue producing, operating mine.
>
> —Producer company with less than US$50 million revenue, corporate secretary

> In [location D], if you build it, [the president] will steal it. [Location E] is a close second for similar reasons along with tribal claims.
>
> —Producer company, company vice-president

> [Location F] introduces without discussion [and] unilaterally a royalty on an industry weeks away from opening a diamond mine after capital expenditure of $1 billion.
>
> —Exploration company, company president

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CHAPTER 5.2

Economic Principles for Decision Making

Ian Runge

Mining used to be a business primarily focused on the technical aspects of getting valuable ore out of the ground and extracting the minerals in a metallurgically efficient way. Without denying the importance of these skills, a narrow focus on technical issues is no longer sufficient to guarantee success, even in rich ore bodies. Skill in economics is an essential partner to technical skill in every step of the mining process. The economic way of thinking starts before the first drill hole is put in the ground and includes not just the most economic way of mining but also the most economic method of assessing mining projects. Economics directs mining strategy and takes into account changes in worldwide demand for mineral products.

This chapter introduces the economic approach to decision making and focuses on how individuals engaged in mining enterprises make decisions using sound economic principles. While the better decisions and the more successful enterprises are adopted and copied by others, the less-successful enterprises fail or are taken over, and outdated practices disappear. Thus the structure of the industry evolves. This chapter presents these economically based decisions in three parts.

The first section, "Mining Economics and Strategy," looks at the industry from a strategic perspective—the broad trends in the mining world and the way that much of the world is interconnected—and highlights and challenges some common misconceptions of mining economics. For long-term success, practitioners must at least be cognizant of these broader influences.

The second section, "Costs," follows a narrower approach. Fortunately, most economic assessments in mines do not require a broad understanding of the whole financial world but can achieve reliable answers focused on what is happening in the company, the mine site, or just one part of the mine. This section focuses on costs and how the understanding of costs is critical for decision making—from day-to-day choices at the mine face to long-term life-of-mine planning.

The third section, "Time Value of Money," examines one of the most important aspects of assessment in all but the simplest of mines: understanding what is happening in the mine over an extended period of time. Investing in new, bigger machinery at the start can reduce costs and provides benefits over the rest of the mine life. Mining the richest ore first can increase profits and reduce debt but can also impact mine life. Understanding these issues requires economic tools to assess the time value of money.

MINING ECONOMICS AND STRATEGY

Most approaches to economics assume that one person or one company's actions do not affect the whole market. This assumption is usually quite reasonable, because even the biggest mining companies usually only account for a small proportion of world production. A single extra ton of copper produced at one mine will not change the world price of copper. Yet the world is interconnected, and the results for one participant in such an interconnected marketplace often depend on the choices (strategies) of all the other participants. Perhaps that extra ton of copper will not affect the world price, but if the changed economic forces that allow this mine to expand production also apply to its competitors, then maybe they too will expand their production. If everyone expands production, then perhaps the world price will change.

For many decisions in operating mines, the standard (non-strategic) approach of ignoring what competitors are doing and using conventional economic models is quite appropriate. If an electric rope shovel is more economical than a diesel-powered hydraulic excavator in a mine, this choice is unlikely to be wrong even when the outside world changes quite substantially, or even if other mines elsewhere in the world favor a different choice.

Nevertheless, there are many areas of mining and mining-specific applications where the use of conventional models must be used cautiously. One such area is in the understanding of risk and return. With financial investments (in the stock market, for example), low-risk, lower-return opportunities form part of a continuous spectrum to higher-risk and higher-return opportunities. In this style of market, the choice of a higher (expected) return opportunity is synonymous with greater uncertainty and higher risk. With individual mine investment, this risk/return trade-off is not so clear-cut. The uniqueness of each ore body and the idiosyncratic risk attached to each project means that higher returns are not necessarily synonymous with higher risk. This creates opportunities for operators who understand the characteristics of the ore body, and who

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understand the risks associated with mining it, to potentially make higher-than-normal returns without exposing the company to any higher risk. It also inhibits some decision making when projects are evaluated using conventional models that are founded on efficient market risk/return characteristics.

Strategic considerations are also important because of high sunk costs in mining—a characteristic of mining that is less important in most other industries. If a company has high sunk costs but low cash costs and the market price declines quite substantially, the company is unlikely to go out of business or even to reduce production. With lower prices, cash flow is still likely to be positive, and even though the company might be reporting losses (from an accounting perspective), reducing production will result in even larger losses.

This raises two important strategic issues. If potential *competitors* have knowledge of a company's capital structure, acting rationally, they might be deterred from entering the same market, knowing that the company will not easily be put out of business. With less competition, the company may be able to gain a price premium. However, this comparative advantage can also work against the company if *customers* also know about the company's capital structure. If they believe the company will continue producing so long as price exceeds the company's cash cost, the company is vulnerable to exploitation. What should the company's strategy be? Can the company assume that its customers and competitors will act rationally? One must ask if one's own corporate structure promotes or inhibits rational choices for one's actions?

Conventional but incorrect economic thinking pervades many other areas of mining, often in critical areas such as reserve definitions and the change in costs as reserves are depleted. With few exceptions, mines commence at the shallowest and/or highest-grade sections of the ore body and progress to parts that are less economically attractive. Moreover, new discoveries are generally deeper and lower grade than existing mines. This suggests that the cost of mining and, with it, the price of mineral commodities must rise over time. Thus, for example, renewable sources of energy (solar, wind) are commonly seen as the inevitable successors to today's coal, oil, and uranium supplies.

Yet even this intuitively obvious trend hasn't proven correct over the last century, and economics provides much of the explanation. For example, Baumol and Blackman (1993) describe work by themselves and others demonstrating that "the real cost (price) of extraction for a sample of thirteen minerals had declined for all but two between 1870 and 1956" and that "the price of fifteen resources for the period 1900 to 1986 until the 'energy crises' of the 1970 [showed] negligible upward trend in the real (inflation-adjusted) prices." Practitioners who have spent a long time in the industry know this well.

The extraordinary growth in living standards since the industrial revolution has driven the demand for minerals, but the same technological developments that have underpinned this growth have also expanded the effective stocks of natural resources at a rate faster than their rate of exploitation. There is no guarantee that this historical trend will prevail in the future, but a long-term strategy that simply adopts the common view rather than one founded on the economics of supply and demand risks incorrect choices and over- or underinvestment.

COSTS

To most people the concept of *cost* is subject to no ambiguity. It is the amount of money one has to take out of their billfold, purse, or bank account to buy something. Day-to-day transactions seldom require any further discernment, because purchasing a small item on its own does not evidently preclude the purchase of anything else later on. For larger expenditures, the real cost is more evident. An overseas vacation might mean delaying the purchase of a new car for another year; in deciding to spend the money on the vacation, one must be comfortable with the idea of driving the old car for longer. The decision—the *real* cost of the vacation and the one that influences choice—is based on the imagined loss of enjoyment and utility from driving around in the new car that much sooner.

In economics, the cost of anything is the highest-valued opportunity necessarily forsaken. This chapter looks at cost from this economic perspective. Unlike accounting costs, which are historical, the economic view of costs is a forward-looking one. Costs in this sense inform decision making, choosing between the imagined value a person will enjoy from following one path and the value of an alternative path. The tools described in this chapter and in the following economics chapters are aimed at understanding this value in order to help make these decisions. This chapter introduces the concepts of marginal, average, variable, fixed, and sunk costs. It illustrates why the marginal cost calculation is such a vital one in pit optimization and in determining the scale of investments and why certain costs are excluded or included in cash-flow calculations.

Types of Costs

To make sensible business decisions, every business needs to know its costs to produce its products. Of the variety of ways to measure costs, some cost concepts are more appropriate for certain problems than others. This section explores these different cost concepts and some subtleties in understanding them.

Every business incurs costs that do not vary with output and costs that do. A *fixed cost* is an expense that does not vary with the level of output—for example, an annual payment to maintain a mining lease (assuming the payments are independent of production). The portion of a fixed cost that is not recoverable is a *sunk cost*. Sunk costs should not affect subsequent decisions and are excluded in preparing a cash flow of a mining property

Example 1. You have spent $15 million evaluating a mining property over a long period of time, and the project looks (almost) viable. Your accounting policy requires you to allocate the $15 million across the proven reserves, but when this cost is included, the project fails to meet your required investment return. Should the exploration costs be included or excluded?

Solution. The exploration costs should not enter into the decision to proceed or not. If you proceed with the project, your accountants will report a loss on the project (because they will write off the cost of exploration and assign it to the project), but if you *do not* proceed, your accountants will *still* report a loss. The $15 million is common to all alternatives because it has already been expended and is unrecoverable.

Are these costs truly unrecoverable? The exploration costs in this example may have already been spent, but they do not automatically become sunk costs. You might not be able to recover your whole $15 million of expenditure, for example, but the property might be salable for $10 million. Only $5 million of the original $15 million is a sunk cost, and $10 million is a *recoverable cost*.

How does one treat the cost of what is already owned, such as the exploration property in the preceding example or one's existing equipment? Such choices are common in operating mines, particularly when underutilized equipment is owned and can potentially be used in new, though less-than-ideal ways. In these cases, the company must use the economic cost to help make the decision, defined by the value of the opportunity that is forsaken. The choice is between *using* the equipment and doing *without* the equipment, and each alternative has various money costs and benefits. In the finance literature, these true economic costs are referred to as *opportunity costs*.

Example 2. You have some older equipment that cannot be used for overburden removal, and you propose to use it for reclamation. You already own it, so there is no purchase price and no cash flow. If you do not use it for reclamation, you could sell it for $1 million. Should the $1 million be included in the cash-flow analysis and in the decision to use it for reclamation?

Solution. Yes. The alternative case has its costs, plus a revenue of $1 million (minus taxes) from the sale of the equipment, so this potential revenue is lost in the first case.

Lost revenues (from the alternate scenarios) are called opportunity costs because, by accepting the project, other opportunities for using the assets are foregone. These true economic costs should be used regardless of the value that the accounting system places on the asset in question.

Variable costs change with the level of output. Typically, as output increases, so does the need for labor, fuel, electricity, and materials, so variable costs depend on the wages and prices that a firm must pay for these inputs. Although variable costs are commonly called *operating costs*, in the day-to-day decisions one cannot assume a one-to-one correspondence between what an accountant calls an operating cost and what is truly a variable cost. Whether a cost is a fixed or a variable cost depends on the time frame of the decision. For yearly budgeting, labor costs are a variable cost, because labor requirements can usually be increased or decreased in line with yearly production requirements. But on a day-to-day basis, most labor costs are fixed. If a truck driver has reported to work and there is no truck available, then this labor cost cannot be avoided.

Thus, even if the production manager (making day-to-day decisions) and the planning engineer (making life-of-mine decisions) both have the same objective—to maximize company profit—the way they make their choices might be quite different. For a production manager, almost all costs are fixed on a day-to-day basis, so from that perspective the lowest cost per ton is achieved when the production is at a maximum. Thus the normal and entirely rational objective for a production manager is to forget about costs and just maximize production. From the perspective of a long-range planning engineer, few costs are fixed. There is scope to buy and sell equipment, change equipment mix, and change the extraction sequence. Decisions must be made not only on production requirements but also on optimization of costs, both capital and operating.

Any new mining development also includes costs that the decision maker does *not* take into account. For example, following commencement of a new mine, the increased traffic might require higher costs of local road maintenance. Dust and noise pollution might impose costs on people quite removed from the project. This style of cost is termed an *externality*. Externalities can be both positive and negative. A supermarket valued at $0.5 million before a mine commences might be valued at $1 million after the mine starts due to the increased business it enjoys from mine personnel. Externalities are changes in value incurred by others outside of the company that are not explicitly taken into account in any decision within the company.

There is a risk that economic assessments which do not take into account significant externalities might lead to incorrect choices. Many large firms already extend their assessments to include externalities on social responsibility grounds even if no legislated requirement has mandated this.

Marginal Costs

In economics, few concepts are more important than the concept of marginal cost.

The marginal cost is the change in total cost. The counterpart to marginal cost is marginal revenue. Marginal revenue is the change in total revenue.

One can consider almost any production process, which will involve some fixed and some variable costs. As production expands, the fixed costs are unchanged, so the average per-unit cost of production attributable to this component declines. If this were the only trend, then the highest production case would be the lowest overall cost of production. However, few production processes work this way. The fixed parts of the process can only service a limited range of variable parts. As production expands for the same fixed components, the efficiency of the system declines. Each increment of production incurs a little more variable costs than the previous increment.

For example, a loader/truck system is the archetypal system in mining. The haul road establishment costs, loader capital costs, and most of the loader operating costs are the fixed components, and the trucks are the variable component. When only one truck is paired to the loader, the average cost of production is high because the fixed costs of owning and operating the loader are spread over a relatively small production. The loader spends a lot of time idle while the one truck is hauling the material to be dumped. When two trucks are allocated, production will increase—but not quite to double the previous amount, because there will be queuing at the start of the shift. The only extra costs are for the extra truck. As additional trucks are added, production will increase further but by a declining amount as the increasing numbers of trucks interfere with each other.

Figure 5.2-1 shows the idealized situation for this style of production process using sample production and production cost numbers for illustration.

The average cost of production is initially high at low levels of production, and each increment of production has a low but increasing marginal cost. If the marginal cost is less than the average cost, the average cost declines with increases in production. The production rate that yields the lowest average unit cost of production occurs where the marginal cost curve crosses the average cost curve (30 units of production in Figure 5.2-1).

Although the lowest average unit cost of production is certainly a desirable objective, usually the objective is to maximize profits (or minimize losses). If the selling price is $2.33, for example, production can be expanded to 40 units and the additional production still yields a return higher than the marginal cost. Indeed, this is the rule: Expand production until the marginal cost equates to the (marginal) revenue. If in

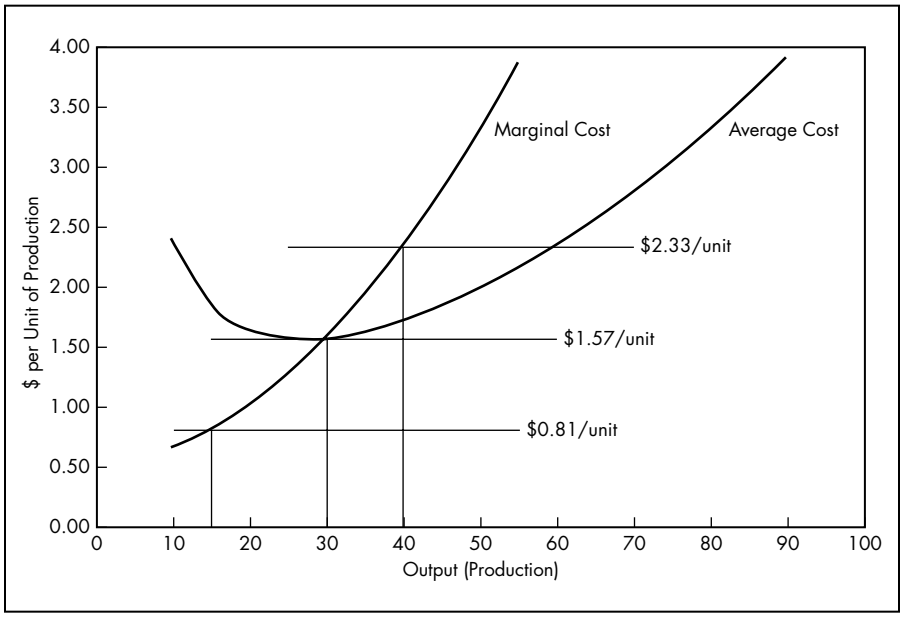


Figure 5.2-1 Average and marginal cost curves

this pricing scenario the production was expanded to 60 units, with an *average* cost of $2.33/unit, then the mine would still be profitable. However the production from the first 40 units would be subsidizing the last 20 units of production. Profits cannot be improved by increases in production rate where the marginal cost exceeds the marginal revenue.

A similar situation occurs for mines in a loss-making situation. In Figure 5.2-1, if the selling price is $0.81/unit, for example, the mine cannot avoid losses at any output level. But the losses are minimized at the point where the marginal cost equates to the selling price (about 15 units of production), not at the lowest point on the average cost curve (30 units of production).

For some simpler assessments (selecting loader/truck fleets, for example), reliable computer simulation packages are available. For most assessments, however, the compilation of a graph similar to Figure 5.2-1 is a lengthy process. The marginal cost curve normally cannot be calculated directly, because as the production expands, the process employed to undertake it frequently changes. If this example involved a loader/truck fleet, for example, the size of loader and truck selected might be quite different for an output of 60 units of production compared to an output of 20 units. Thus, a desired production rate of 60 units requires design of a system appropriate to this rate and its attendant costs, whereas a desired production rate of 30 units requires a system design appropriate to this lower rate—presumably using smaller equipment. The marginal cost is the change in total cost—calculated by multiplying the average cost at the higher production rate by this output and subtracting the equivalent total cost at the lower production rate.

In this example, the selling price was assumed to be independent of production. For many mineral commodities such as gold and silver, this is an appropriate assumption because gold and silver are readily transportable and freely traded on world markets, and the production from any one mine—even the biggest mines—is small compared to the size of the world market. But for many other commodities, price is dependent on production because the cost of transport to supply into larger and more distant markets reduces the mine gate price. Most industrial minerals are in this category, as well as bulk commodities such as coal. How then can the optimum output be determined?

In these cases (where price cannot be assumed constant), the same logic and the same rule applies: Select an output level where the marginal cost equates to the (marginal) revenue. In this case the marginal revenue is not a constant; it too must be calculated. Example 3 illustrates how the marginal revenue calculation can be addressed.

Example 3. Consider a mine currently producing 8 Mt (million metric tons) of coal per year under a mix of spot sales and contracts to a variety of regional customers. At any one time customers are all paying slightly different prices for the same coal, but in due course prices become more widely known and these influences reduce. Long-term contracts have price variation clauses that adjust to market conditions. The average selling price for the existing output is $10/t. You can expand production by 1 Mt/yr at an operating cost (for this extra coal) of $6/t with only a small amount of capital. Repaying this extra capital, you can still make your required return on investment at a selling price of only $9/t. You believe you can find additional customers who will purchase the extra 1 Mt/yr at $9/t. Should you proceed?

Solution. This seems to be a clear-cut case. If the selling price exceeds the marginal cost, profits increase with each increase in production. The risk is that, lacking any ability to keep prices secret (never a good strategy, in any case) for any length of time or to differentiate the new coal from the old coal, *all* of the output from the mine will be priced downward. Your customers themselves have incentives to do this. For instance, what is to stop your new customer from selling some of your new coal to one of your old customers (at perhaps $9.50/t), with both of them being better off?

In this example, prices are not independent of production. Selling some coal below average price makes it harder to

maintain the price of your existing supply. Perhaps you only risk downward price revision by 5%, but this reduced price applies to all of your output, if not immediately, then certainly in the near future. The marginal revenue is the change in total revenue, and this is not the same as the proposed $9/t selling price that the new customer is initially prepared to pay. Expansion is only viable to the point where marginal cost equates to marginal revenue. Table 5.2-1 shows this calculation.

The additional output has an effective selling price (marginal revenue) of just $5.50/t—a price at which the expanded production is not viable because it is less than the marginal cost of $9/t. Of course, real-life cases always have additional complexities that are not included in this example, but the principle applies. The change in total cost and total revenue should be applied to the change in production, and this result should guide the decision of whether or not to proceed, not the narrowly defined costs and revenues associated just with the extra production.

The confusion between average, incremental, and marginal payoffs can work the other way, too. Most managers naturally hesitate to throw good money after bad, but if an existing project is already making a loss, this may be irrelevant in the decision regarding incremental expenditure on it. Sometimes an existing project is yielding poor returns because of a bottleneck in the production chain, and small incremental investments to remove such bottlenecks can yield large marginal returns.

Whenever optimization is the objective, marginal costs should be the focus.

Procedures aimed at pit optimization (in open-pit mines) and cutoff grade calculations (in all types of mines) apply this identical principle. Starting from an initial ore body, they examine extensions to the ore body in all dimensions to ascertain whether the marginal revenues from the extension exceed the marginal cost of extracting the additional ore and waste. The optimum limit of mining is where, at the margins, the return equates to the cost.

TIME VALUE OF MONEY

Money tomorrow is not as valuable as money today. Given the choice of having the same amount of money in the future or right now, everyone would prefer to have it now. Money received in the future has some risk that it might not materialize, but even if there was no such risk, it is still worth more if it is available for use now. If it is available now, the things that one might conceivably do with the money are as broad as possible. If it is not available until some time in the future, then the opportunity set is limited to a smaller subset of this first set. Maximum freedom to choose is always worth something—particularly in more uncertain environments. Therefore, money to be received in the future must include a premium if it is to be considered equivalent to money in hand today. Future cash flows (money) must be discounted in order to compare with present cash flows.

Almost every economic decision in mining involves cash flows (spending money, receiving money) occurring at different points of time. Consequently, economic evaluations must incorporate a way for equating these money values at some constant point in time (usually, now). For simple calculations, future values are derived by taking current values and multiplying by the interest rate (compounded). Or, equivalently, future values (anticipated cash flows) are turned into the equivalent present value by discounting—that is, dividing by the compounded interest rate.

Table 5.2-1 Marginal revenue calculation

Production Scenario	Total Production, Mt/yr	Estimated Average Selling Price per t	Total Annual Revenue, millions
Current mine	8	$10.00	$80.0
After expansion	9	$9.50	$85.5
Extra output	1	$5.50 Unit marginal revenue	$5.5 Marginal revenue

For meaningful calculations of mining investment propositions, a complete tabulation is normally prepared for all of the cash flows through each year of the project's life. The aggregate cash flow (the sum of the expected positive and negative cash flows) in each year is calculated first, and this value is turned into a present value via the applied discount rate. To account for uncertainty and other factors, the discount rate is usually greater than the long-term interest rate.

Valuation at Constant Point in Time

The first series of time-value calculations apply simple formulas to bring anticipated cash flows to an equivalent time reference basis for calculation. Simple calculations are grouped into two categories:

1. How to turn a future value into the equivalent present value and vice versa, and
2. How to turn a regular series of equal values occurring over several years into an equivalent single amount in the present and vice versa.

The two functions used to relate present values to future values and vice versa are

1. Compound amount function (future value), and
2. Present value function.

The future value is determined by the following formula:

$$FV = PV \times (1 + i)\, n$$

where

FV = future value
PV = present value
i = interest rate (in the time period)
n = number of time periods (years)
$(1 + i)^n$ = compound factor

Example 4. Your company has to pay a reclamation bond to the government for each hectare (ha) of disturbed land. The funds are held in trust, earning interest at 6% compounded annually until reclamation is complete, whereupon they are returned. If you disturb 40 ha of land this year and the bond is $50,000/ha, how much do you expect to get back when reclamation is completed in 3 years' time?

Solution.

Present value: $50,000 × 40
(of money paid out now) = $2.0 million

Compound factor: $(1 + 0.06)^3$
= 1.191

Therefore,
Future value: $2,382,000

The present value function is used to move a future value estimate back to the present and is the inverse of the future value function:

$$PV = \frac{FV}{(1+i)^n}$$

where

$$\frac{1}{(1+i)^n} = \text{present value function}$$

Example 5. You have received bids from two manufacturers for purchase of a new dragline. The first bid (A) is competitive but from a company that requires payment in full on placement of order. The second bid (B) is for a higher price, but no payment is required until the machine starts digging in 3 years. Which is the preferred option?

Dragline A bid price (payment today):	$30,000,000
Dragline B bid price (payment 3 years' time):	$40,000,000
Required return on capital (discount rate):	15%
Time *n*:	3 years

Solution.

Present value (of purchasing Dragline B 3 years into future)	$40,000,000
	× 0.6575
	$26,300,000

Dragline B has a lower cost than Dragline A in present value terms.

Functions are also used to relate a regular series of equal values occurring over several years into an equivalent single amount. The most useful of these is the capital recovery function. Used to spread a present value amount evenly over a period of *n* years, it produces a series of equal values occurring at the end of each year for the time period specified.

$$\text{capital recovery factor} = \frac{i}{1 - \frac{1}{(1+i)^n}} = \frac{i(1+i)^n}{(1+i)^n - 1}$$

Example 6. The expected life of a rope shovel is 16 years, after which time the mine will close and the salvage value will be effectively zero. What is the annual owning cost, including allowance for return on your capital invested in the rope shovel? If the shovel works 6,000 hours per year, what is the hourly cost?

Required return on capital (discount rate):	15%
Cost of rope shovel:	$7,000,000
Time (*n*):	16 years

Solution.

Capital recovery factor:	$0.15/[1 - (1/1.15)^{16}]$
	$= 0.15/[1 - 0.1069]$
	$= 0.1679$
Equivalent annual cost of shovel over 16 years	$= \$7{,}000{,}000 \times 0.1679$
	$= \$1{,}175{,}300$/yr
Therefore, hourly cost	$= \$1{,}175{,}300/6{,}000$
	= $195.88 per hour

Almost all scientific or engineering calculators and spreadsheets now include functions capable of undertaking these calculations directly.

Discounted Cash-Flow Analysis

Although all of the functions discussed in the previous section are important in determining values for activities occurring over time, their usefulness is limited because they do not take taxation effects into account and need regular cash flows. Since almost all real-life cases involve taxation, and operating costs and revenues vary over time, an alternative evaluation method must be used. The method universally used for almost all mining and other business evaluations is the discounted cash flow (DCF) technique. (This technique is only briefly introduced for a simple case; however, Chapter 2.4 in the handbook addresses this technique from a broader perspective.) In *Mining Economics and Strategy* (Runge 1998), comprehensive examples of the DCF technique are set out for a wide range of mining applications.

A big difference exists between corporate finance (i.e., costing, economics, and capital investment decisions) and financial accounting, which stresses incomes and earnings. While accounting procedures document what has happened, mining economics aims at informed decisions on what to do. For accounting purposes, all expenditures are normally apportioned over the period that the expenditure translates into useful work. For planning and operating a business, there is no apportionment—allowance has to be made when the expense actually occurs.

Example 7. Consider the purchase of a dozer for $600,000 paid for today. The entire $600,000 is an immediate cash outflow. An amount of $600,000 has to be available at the time the dozer is delivered—before it has done any useful work. However, assuming straight-line depreciation over the 6-year life of the dozer, only $100,000 is considered an accounting expense in each year. Current earnings (reported profits) for this year are reduced only by $100,000. The remaining $500,000 is expensed (counted as an operating cost) over the following 5 years.

To run the business, what is important is cash flow, not accounting profit. In Example 7, the company supplying the dozer requires the full purchase price to be paid now, not just the amount of depreciation that the accountant attributes to this year's cost of production. Furthermore, capital expenditures always occur before any production, whereas accounting conventions only assign their costs (and revenues) during or after production has taken place.

The objective of a cash-flow analysis is to simulate all of the anticipated cash flows over the project's life (and express them in present value terms) to help make a decision to proceed or not. The most obvious cash flows are

- Revenues from sale of the products,
- Expenses incurred in producing the products, and
- Capital expenditures necessary to bring about production.

Capital expenditures are tabulated in the cash flow in the year prior to their use. The plant or equipment must be operational before any production takes place (the start of the

Table 5.2-2 Sample discounted cash flow

	Year*					
	0	1	2	3	4	5
1. Production, oz	—	32,000	50,000	50,000	50,000	45,000
2. Operating revenue at $500/oz	—	$16,000	$25,000	$25,000	$25,000	$22,500
3. Operating expenses	—	$10,598	$17,762	$19,339	$21,073	$20,882
4. Operating profit (2 – 3)	—	$4,402	$7,238	$5,661	$3,927	$1,618
5. Capital expenditure	$15,000	—	—	—	—	—
6. Tax depreciation this year at 27.5% (declining balance)	—	$4,125	$2,991	$2,168	$1,572	$1,140
7. End-of-year written-down value for tax purposes	—	$10,875	$7,884	$5,716	$4,144	$3,005
8. Salvage value (= 7)	—	—	—	—	—	$3,005
9. Taxable profit (4 – 6)	—	$277	$4,247	$3,493	$2,355	$478
10. Income tax payable at 35% tax rate	—	$97	$1,486	$1,223	$824	$167
11. After-tax profit (9 – 10)	—	$180	$2,760	$2,270	$1,531	$311
12. Net cash flow (4 – 5 + 8 – 10)	($15,000)	$4,305	$5,751	$4,439	$3,103	$4,455
13. Discount factor (at 15% ROI†)	1.0000	0.8696	0.7561	0.6575	0.5718	0.4972
14. Discounted cash flow (12 × 13)	($15,000)	$3,744	$4,349	$2,919	$1,774	$2,215
15. Net present value	$0	—	—	—	—	—

*All annual cash flows are in thousands, rounded to nearest thousand.
†ROI = return on investment.

period). With the end-of-year convention, capital expenditures are therefore placed at the end of the preceding year.

Cash-flow tabulations should normally commence with production tabulated on the top or near to the top of the table, because almost all of the revenue and many of the operating expenses are related to production. Revenues (the primary cash inflow) are also tabulated at the top of the table.

To obtain the operating profit, operating costs are subtracted from the operating revenues. All operating expenses are included in a cash-flow calculation, even if some of the costs pertain to production in following years. This differs from the way operating costs are treated for accounting purposes, where expenses that pertain to production in succeeding time periods (e.g., advance stripping) are apportioned to the period in which they directly relate to production.

A sample discounted cash-flow tabulation for a mining project is set out in Table 5.2-2. In this tabulation, a hypothetical gold mine with a 5-year life produces up to 50,000 oz/yr and expects to sell all of the output at $500/oz. The main background data needed for this tabulation (or any discounted cash-flow tabulation) are set out in Table 5.2-3.

In Table 5.2-3 the data has been deliberately chosen for illustrative purposes so that the selling price of $500/oz yields a net present value of zero at a discount rate of 15%. This discount rate is just a guideline and is usually determined by senior company finance personnel weighing the relative risks and the cost of capital between this project and any alternative projects that the company might otherwise choose to apply its resources to. Thus, in this example, if the gold price exceeded $500/oz, the internal rate of return would exceed this (opportunity) cost of capital, and the project could proceed.

Table 5.2-3 Base data for discounted cash-flow calculation

Item	Value
Initial capital cost	$15,000,000
Life of project	5 years
Salvage value at end of life	At written-down value
Production per year	Varies
Selling price	$500/oz
Annual operating expenses	As shown
Depreciation rate for tax purposes (declining balance)	27.5%
Tax rate	35%
Discount rate	15%

Although the format for cash-flow tabulations in real-life applications—even for large projects—is similar to Table 5.2-2, even simple assessments include many more elements than shown. Elements commonly built into the DCF tabulation include exchange rate factors, expected escalation and de-escalation of cost and revenue components (inflation adjustments), and finance charges.

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CHAPTER 5.3

Management, Employee Relations, and Training

G. Aubrey Lee

INTRODUCTION

Demands for resources that must be accessed by mining are increasing exponentially. Energy demands in emerging industrialized nations such as China and India are putting a strain on the availability of coal and other resources throughout the world. In some areas, labor shortages are a tremendous hindrance to mining operations and are driving up prices of coal and metals because a stable, trained work force cannot be sustained.

As a result, recruiting, developing, and retaining a skilled work force must be incorporated into every mining organization's strategic planning process. At the same time, maintaining effective employee relations is becoming increasingly complex in today's globalized mining operations and must also be approached strategically. Components of these strategies must include

- Providing at all levels of the organization's effective leadership that recognizes maintaining good labor relations as essential to mining operations;
- Viewing training, employee development, and organizational learning as an integral aspect of company strategy, not merely a cost that adds no value;
- Managing employees' work/life balance, giving particular attention to shift work and other working conditions;
- Understanding that safety, like productivity and quality, can be managed to effectively protect the safety and health of miners working in potentially hazardous conditions;
- Managing contract workers who have become a critical component of the mining labor force;
- Effectively managing both union and nonunion mining work forces, as well as the legal, ethical, and practical implications confronting each scenario; and
- Understanding cultural differences and adapting management styles and practices to effectively manage the issue which may be foreign/counter to the manager's own culture.

The approach to managing each of these components will, of course, vary widely throughout the world. Culture, regulatory environments, and union versus nonunion status will all impact upon the way employee relations and training are managed and conducted. As a result, much of this chapter will be general in nature. Readers will need to assess what is written in light of their own specific circumstances and adjust their practices accordingly.

Another point to be emphasized that impacts the approach to this chapter is that many mining organizations are multinational and thus have managers working outside of their home country. Insights are offered about ways to approach employees from cultures whose norms and behaviors differ drastically from those of the "foreign" manager. Failure to understand how to manage cross-culturally is recognized as a frequent source of underperformance or even failure for expatriate managers and engineers. Sometimes this will be as critical as deficiencies in "hard" or technical skills. Thus, training and orientation in this area is essential.

Leadership is often described as achieving the goals of the organization through the efforts of others. This means that, although the primary responsibility is achieving the organization's goals, the only way to do that is by providing effective leadership for the employees.

FIRST PRIORITY FOR MINING LEADERS—HEALTH AND SAFETY

Mining has historically been difficult and sometimes dangerous work, although technology and safer work practices have improved working conditions in recent decades. Safety is a larger issue than ever before, however. The protection of the health and safety of miners is essential and should never be considered only in light of public opinion. In other words, workers must be protected because it is the right thing to do.

Managers in the mining industry must strive to manage health and safety issues in the same manner, regardless of the country in which they are doing business. Being lax in nations that have less stringent safety and health regulations is unfair to employees and can damage a company's reputation worldwide. Shareholders, the media, activist groups, unions, and governmental agencies now closely watch the safety, health, and environmental performance of multinationals and have

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the capacity to take actions that can severely damage a company's ability to operate.

Although the top management and/or corporate boards (or governments) of mining organizations set policy, operating supervisors and managers on the front line have ultimate control over working conditions. They are responsible for promoting safe working practices by effectively training and managing miners and working conditions in the operations under their control.

BASIC PRINCIPLES OF MANAGEMENT

One key to successful management is to follow a consistent pattern of interaction with one's team. Effectively handling difficult situations while increasing the performance of the work group can make the difference between success and mediocre performance, particularly in a mining environment. Following are five key principles for successful leadership:

1. **Lead by example.** In most cultures, "do as I say, not as I do" will not be effective. Supervisors or managers with poor work ethics or sloppy or unsafe work practices tend to engender the same type of performance among their co-workers.
2. **Maintain self-esteem.** Berating employees in front of their peers, discounting their suggestions or complaints, and projecting a superior, "know-it-all" attitude will result in demotivated individuals who will never live up to their potential. In some cultures, such tactics can result in the total demoralization of the person. Mining leaders must do all they can to make employees feel they are valued and not just a means to an end. Employees need to understand that their job, regardless of where it stands on the company's organizational chart, is vital to the success of the mining operation where they are employed.
3. **Establish and maintain constructive relationships.** Adversarial relationships on the job are counterproductive and can even be dangerous to health and safety, particularly in mining operations. Mining operations supervisors and managers can influence employees' opinions about their jobs and the company more than any other group. Even in union environments with entrenched negative attitudes toward management, making a conscious effort to avoid conflict and engender positive interaction is critical.
4. **Always focus on the situation, the problem, or behavior—not the person.** People make mistakes. Except in situations where a person is intentionally trying to sabotage the organization, most poor performance is unintentional. Most employees do not consciously set out to do a bad job. This is not to say that some sort of corrective or disciplinary action should not be taken based on policy and past practice. Either way, the supervisor should address the issue, not the person, and make every effort to reinstruct, remind, or do whatever is needed to prevent a reoccurrence of the problem. Berating the person while offering little help to encourage better behavior or performance going forward is counterproductive and should be avoided. Furthermore, people are part of a system in any mining operation, that is, a system made up of other miners, equipment, tools, processes, procedures, and so forth. Supervisors and managers must recognize that if a system exists where employees have difficulty performing properly, perhaps the system is the root cause of the issue, not the miners. Often modifying a procedure, repairing equipment, or reconfiguring staffing will improve performance.
5. **Take initiative to make things better.** "Continuous improvement" is sometimes an overused expression, but it is still valid in any industry, particularly when supervisors and managers on the front line of a mining operation have opportunity to spot those seemingly little things that can make a big difference in quality, productivity, or safety. Miners under direction of a supervisor may have an even better perspective to see potential improvement. Much can be gained from soliciting and paying close attention to suggestions from employees. In addition to improving the operation, morale will be boosted if employees see that their ideas are heard and, where possible, implemented.

Good management, founded on the principles outlined above, must be supported by good habits. The characteristics seen in Figure 5.3-1, if integrated into the manager's behavior, can lead to more productive, profitable, and safer mining operations. As stated previously, cultural differences can arise when management practices are proposed that are more or less Anglo-American. However, many of the suggestions given will have universal application. Cultural differences are addressed more fully later in this chapter, with particular reference to assisting multinational managers who are working outside of their native culture.

MANAGING UNION OPERATIONS

Union membership is identified by the United Nations as a universally recognized right, as stated in Article 23, subsection 4, of the United Nations Universal Declaration of Human Rights (UNGA 1948). The same subsection acknowledges the right to *not* join unions. This ambiguity and the friction such a position creates are nothing new to the mining industry, where companies and unions have experienced difficult times in the past. Bloody and costly strikes, work stoppages, and other issues are still a possibility for many mining companies. Nevertheless, as has been proven by mining companies throughout the world with various unions, it is possible to manage in a way that creates favorable financial returns for the company while offering good compensation and working conditions to unionized miners (SESCO 1979).

Managers and supervisors should be aware that transnational union strategies have been globally invigorated by the consolidation of existing trade union organizations in the mining, chemical, and energy sectors. This resulted in the formation of the International Federation of Chemical, Energy, Mine and General Workers' Unions (ICEM) in 1996. The union's strategy is to consolidate local union activity with the entire global network of union members and outside resources. Experts say that this is an example of the coming trend of transnational union relations, resulting in unions forming alliances with human rights, environmental, and other activist groups to pressure transnational companies to agree on codes of conduct regarding alleged violations in areas such as civil rights, employment law, safety, and environment. The international media have also been mobilized to report on such alleged violations as a result of more concerted and high-profile global union activities. What does all this mean for mining supervisors and managers? Front-line mining leaders must understand that their role is

1. **Communicates Effectively**
   - Shares information with employees about job performance, about the company, and about how to do the job
   - Is a good listener; listens without interrupting or disagreeing
   - Asks for opinions and encourages employees to take initiative
2. **Provides Clear Performance Expectations**
   - Communicates both positive and negative feedback to employees
   - Addresses problems when they arise; does not wait for review time; addresses individual problems in private
   - Evaluates on performance, not personality
   - Communicates about "getting the job done" in meetings
   - Lets employees know that they are expected to seek solutions to their problems
   - Gets employees' buy-in on performance expectations by explaining the "whys" of the situation when appropriate
3. **Develops Employee Relationships**
   - Has genuine concern for employees—is not superficial or patronizing
   - Builds relationships and respects employees' individuality
   - Is caring about employees' personal lives without prying or invading privacy
   - Does not have "favorites"
4. **Does Not Micromanage**
   - Is careful not to overcontrol the employees
   - Delegates fairly
   - Gives employees a chance to grow
   - Keeps in touch about progress—asks the right questions
   - Responds enthusiastically to ideas and suggestions
   - Praises employees
   - Helps employees learn from their mistakes
5. **Has a Sense of Humor**
   - Admits mistakes
   - Remains consistently upbeat
   - Sets an example in professionalism, work ethic, integrity, tone, and morals
6. **Provides Consistent Direction**
   - Does not keep changing agendas
   - Sets priorities
   - Has a clear and consistent direction for the departmental operation and shares it regularly with employees
7. **Looks for Ways to Learn from Others**
   - Is open to change
   - Strives to present logic and thought, yet remains conscious of the fact that it requires people to accomplish goals
   - Recognizes that success primarily depends on employees

Figure 5.3-1 Characteristics of an effective manager

critical in addressing the mounting resources that organized labor is amassing to organize miners. Although there are no silver bullets to ensure good labor relations, leadership that promotes fair and consistent treatment of workers and safe working conditions goes a long way in avoiding issues with organized labor, including costly strikes, labor slowdowns, and the negative publicity such events can create.

The same skills required to effectively manage people in nonunion operations can be employed in union mines. However, all actions taken on behalf of or against employees must be within the tenets of the union contract. All who have ever managed union employees understand that there are stringent contracts that are inflexible, and there are those that permit managers some latitude. Labor laws that govern how contracts are written vary widely from nation to nation and can even fluctuate depending on the political party or group that is in power at the time. The relative strength or weakness of company and union leadership in a given situation also greatly influences how union miners can be managed. In nations that do not feature state-sponsored labor unions, many companies make a concerted effort to maintain a nonunion status because of the inefficiencies sometimes encountered in managing within the confines of a union contract.

Managers employing union miners must, of course, make every effort to avoid strikes, work stoppages, or slowdowns. Such events can be precipitated by any number of issues, including financial disagreements, disputes over working conditions, and the treatment of workers by management. Many labor relations experts contend that the last factor, particularly how people on the front line are managed, is a critical factor that can be positively impacted by training managers to practice effective employee relations skills. The following section includes some practical guidelines for managing a union work force and begins with a brief overview of labor law.

Managing Union Employees to Avoid Legal Issues

Although the following is based primarily on Anglo-American labor laws, some of the requirements will apply to the laws of other nations. Because labor relations are complex and location specific, expert counsel should always be sought when managing unionized employees, particularly in times of labor–management conflict.

- **Good-faith bargaining.** The labor laws of most Anglo-American countries and some other nations require employers and labor unions to bargain in "good faith." What constitutes good faith on the part of the employer

or the union is difficult to define and is constantly being interpreted based on the facts at hand. However, the basic requirement of good-faith bargaining is that both the employer and the union must negotiate with the view of trying to reach an agreement. The bargaining obligation is not satisfied by merely going through the motions without seeking to adjust differences. Should there be a dispute on the part of either party as to whether "good faith" is being exercised, it is up to the courts or whichever governmental agency regulates union–employer relations to determine the "totality of conduct" by looking at all relevant facts of the issues within the case at hand. There are numerous factors to consider, but, as mentioned previously, the determining factor will be the perceptions of both the union and the company as to whether the other side is trying to reach an agreement or is simply stalling or complicating the process.

- **Mandatory bargaining issues.** Some terms and conditions of employees must be on the bargaining table. Again, these vary from nation to nation, but wages, compensation for nonworked time, certain nonwage benefits, working conditions, management rights, and union management or leadership rights generally must be subject to negotiation.
- **Nonmandatory bargaining issues.** Certain subjects are nonmandatory and include the definition of the bargaining unit; conditions affecting supervisors; performance bonds; clauses that are illegal, such as harassment or discrimination, based on the laws and regulations of the nation; and right-to-work laws as may exist in certain nations, states, or provinces. Management should seek the appropriate legal counsel when determining what is mandatory and what is not, to avoid unnecessary conflict in the bargaining process, since this will vary widely and is, of course, subject to interpretation.

Management Actions During Contract Negotiations or Organizing Attempts

The behavior of management and supervisory personnel is critical during contract negotiations or during union organizing drives. Again, from an Anglo-American perspective, there are certain statements or actions that management cannot say or do lest they fall foul of the labor laws governing their country or province. Such prohibitions, as identified by SESCO Management Consultants (1979), include the following:

- **Bargaining individually with employees.** Because unions are organized collectives of people, management cannot bargain individually with employees on matters—even minor ones—pertaining to wages, benefits, or working conditions.
- **Threats, interrogations, promises, and spying (TIPS).** Management cannot threaten to close, move, or reduce operations to avoid being unionized, or coerce employees to accept part of a contract that is perceived as favorable to management and unfavorable to union workers. On an individual basis, management cannot threaten to discharge employees for engaging in union activity. Interrogating employees about their involvement or the involvement of other union activities could also result in the violation of labor laws. Spying on union activities to see who is supporting the union or attending union functions is also prohibited. Promising a pay increase or benefit improvement if employees do not join the union or if they support contract provisions that are unfavorable to the company, in the case of contract negotiations, is also illegal, as is making false statements about the positions of unions.

The TIPS model is an easy and effective way to remember what managers and supervisors can and cannot do during a union organizing campaign or contract renewal period. Remembering this simple acronym can save managers and supervisors much grief and legal problems when dealing with union employees in potentially volatile and confrontational times.

Behaviors Management Must Avoid:

Threaten employees
Interrogate employees
Promise employees
Spy on employees

What Management Can Do

The key to making comments to employees during organizational attempts or contract renewals is simple: Tell the truth. Managers can talk to employees about the company's position in negotiations or the status of the negotiation process as long as their comments are factual and accurate and do not threaten to reduce or promise improvements in pay, benefits, or working conditions.

Managers are free to impartially enforce rules in ways that are fair and consistent with past practices (and within the confines of the contract, if the company is unionized). In other words, management can continue to manage the business as long as union supporters or union employees are not singled out for harsh treatment. Most importantly, management can tell employees that the mine plans to keep operating to serve its customers during labor negotiations or strikes.

GLOBAL MANAGEMENT

Managing internationally outside of one's country of origin is complex and difficult. Most managers are trained in their home country and therefore adopt a management style based on their own culture. Since multinational ownership is prevalent in the mining industry and most companies rely on expatriate managers in various levels of mining operations, understanding how to adapt to different cultures is essential.

Management, according to Hofstede (2001), is primarily an invention of the West and, as such, carries the baggage of Western culture that often flies in the face of practices that are considered acceptable in other countries. The following discussion identifies four management competencies—leading, decision making, controlling, and motivating—and offers suggestions regarding the issues Western management style creates and how that style might be modified and adapted to fit a different cultural context. These are presented through Hofstede's framework of national value systems: power distance, individualism (as opposed to collectivism), masculinity, uncertainty avoidance, and long-term/short-term orientation. These five dimensions were found to exert strong influence on the relationships between employees and the acceptability of management behavior (Hofstede 2001).

1. **Small versus large power distance** describes the extent to which the less powerful members of institutions and

organizations expect and accept that power is distributed unequally. Small power distance (e.g., Austria, Israel, Denmark, and New Zealand) expect and accept power relations that are more consultative or democratic. People relate to one another more as equals regardless of formal positions. Examples of countries that value high power distance are Malaysia, the Philippines, and Panama. This means inequality is readily accepted and leaders are in fact expected to exert strong influence. Participative management styles that seek input from subordinates may not be effective and can be viewed as weakness on the part of managers.

2. **Individualism versus collectivism** refers to the extent to which people are expected to stand up for themselves and to choose their own affiliations, or alternatively to act predominantly as a member of a lifelong group or organization. Latin American cultures rank among the most collectivist in this category, including Guatemala and Ecuador, as well as China and certain other Eastern cultures. Western cultures, as seen in the United States, Great Britain, and Australia, are some of the most individualistic.
3. **Masculinity versus femininity** refers to the value placed on traditionally male or female values (as understood in most Western cultures). So-called "masculine" cultures value competitiveness, assertiveness, ambition, and the accumulation of wealth and material possessions, whereas feminine cultures place more value on relationships and quality of life. Because of taboos about sexuality in many cultures, particularly masculine ones, and because of the obvious gender generalizations implied by Hofstede's terminology, this dimension is often renamed by users of Hofstede's work as, for example, "quantity of life versus quality of life." Societies with strong masculine values are Japan, Austria, Mexico, the United States, and West Germany. China and Indonesia are also moderately high in masculine values. Countries with feminine values are Sweden, Norway, Denmark, and France. It is helpful to understand that both men and women tend to adhere to the dominant value in masculine and feminine cultures.
4. **Uncertainty avoidance** reflects the extent to which members of a society attempt to cope with anxiety by minimizing uncertainty. Cultures high in uncertainty avoidance prefer rules (e.g., about religion and food) and structured circumstances, and employees tend to remain longer with their present employer. Mediterranean cultures, Latin America, and Japan rank the highest in this category. Countries with low uncertainty avoidance values include Singapore and Jamaica.
5. **Long- versus short-term orientation** describes a society's "time horizon," or the importance attached to the future versus the past and present. In long-term-oriented societies, values include persistence (perseverance), ordering relationships by status, practicing thrift, and having a sense of shame. In short-term-oriented societies, values include normative statements, personal steadiness and stability, protecting one's face, respect for tradition, and reciprocation of greetings, favors, and gifts. China, Japan, and the Asian countries rank especially high in long-term orientation, with Western nations ranking rather low (or short-term), and many of the less developed nations rank very low.

Table 5.3-1 Cultural dimension comparisons among nations*

Country	Power Distance	Uncertainty Avoidance	Individualism	Masculinity
Australia	7	7	2	5
Costa Rica	8(tie)	2(tie)	10	9
France	3	2(tie)	4	7
West Germany	8(tie)	5	5	3
India	2	9	6	6
Japan	5	1	7	1
Mexico	1	4	8	2
Sweden	10	10	3	10
Thailand	4	6	9	8
United States	6	8	1	4

Source: Hofstede 2001, with permission from Geert Hofstede BV.
*1 = highest; 10 = lowest.

All of these cultural factors were found to exert strong influence on the relationships between employees and the acceptability of management behavior (Hofstede 2001). In addition, concepts of time, characterized by Trompeanaars and Hampden-Turner (1998) as *sequential* versus *synchronous*, can affect U.S. and Far Eastern business relationships and can have a strong impact on mining operations in particular.

It is important to note that these cultural differences describe averages or tendencies and not characteristics of individuals. A Japanese person, for example, can have a very low uncertainty avoidance compared to a Filipino, even though their national cultures point strongly in a different direction. Consequently, a country's scores should not be interpreted as deterministic. Table 5.3-1 provides other examples of cultural dimension comparisons.

Flexing Competencies to Avoid Cultural Conflicts

Conflicts relating to cultural differences often arise from the management responsibilities of leading, decision making, motivating, and controlling. The following are examples of the nature of certain conflicts and what might be done to eliminate or at least mitigate them.

Leading

Collectivist societies prefer tight social networks in which individuals care for one another; they desire organizations that are sensitive to the members' interests (Hofstede 2001). In societies high in collectivism (Asia, Middle East, Latin America), "warm" personal approaches are critical. In China specifically, all relationships are personal. This often frustrates Western managers who must, in their opinion, spend an inordinate amount of time developing relationships with officials in Chinese businesses before meaningful business discussions can begin. Time spent visiting with workers and discussing morale and health is also often viewed as unnecessary by Western managers but is essential to Chinese and other Eastern employee groups. Correcting mistakes or poor performance must be handled delicately. Extreme care must be used when criticizing in these societies. Loss of self-respect brings dishonor to workers and their families. Trompeanaars and Hampden-Turner (1998) described a frank discussion between a Dutch doctor and Chinese subordinate that resulted in the subordinate's suicide. This is a prime example of how "saving face" is extremely important in collectivist culture.

Perhaps the strongest impact of collectivist culture that often impedes and can even ruin American business relationships in China is *guanxi*. The most direct translation of the term is "relationship," but the concept is much more complex. It describes an intricate relationship built over time through the exchange of favors and social exchanges. It effectively creates an unspoken obligation that can even be banked. It could be likened to a currency that, to many Westerners, can be foreign and burdensome. However, it is deeply imbedded in the culture, and a failure to understand or a failure to participate can break deals (Xin and Pearce 1996).

Decision Making

Power distance reflects the tendency of people to accept inequality between individuals, organizations, and institutions (Hofstede 2001). In the United States, with low power distance, middle managers do not hesitate to take recommendations to the chief executive officer (CEO). In contrast, South Asian CEOs make the decisions and then provide detailed instructions to subordinates, reflecting high power distance (Javidand and Dasmalchian 2003). Participative management does not work in high-power-distance contexts, so top managers must be decisive. In Mexico, for example, participatory management is apt to be misunderstood. Managers are expected to be powerful in issuing orders.

At the other extreme, Arab and African leaders, exemplifying low power distance, often use a consultative approach in decision making, almost to a fault (Schuler et al. 1996). Uncertainty avoidance also impacts decision-making behavior. Germany ranks high in uncertainty avoidance compared to Singapore or to their European neighbor Denmark. Germans plan carefully, and this is reflected in their society, which is heavily dependent on rules and laws. The stereotypical German trend for excellence in engineering reflects the extreme desire to be precise and to avoid a poor design that could create risk. Americans with entrepreneurial tendencies often conflict with Germans who want no part in ventures that are not a sure thing (Baum et al. 2006.)

Motivating

Motivation techniques must fit culture. Intrinsic factors such as recognition, challenge, and work itself feature poorly in high-power-distance societies. Workers may see this as manipulative and thus not be motivated. In the United States, however, and in the United Kingdom, intrinsic motivation can work. Tangible reward must be offered, but appropriately (Huang and Van de Vliert 2003). From the collectivist perspective, tangible individual motivational techniques can be counterproductive (Persaud 2003). Groups are motivated in Japan and other collectivist societies as opposed to individuals. Although individual rewards for exemplary performance may not work in this situation, a group reward offered to everyone in the work group will most likely be successful (Caudron 1995).

Time also comes into play in motivating employees. The Western concept of sequential time describes time as linear like an arrow, as opposed to time being synchronous, recurrent, and cyclical in nature in Asian cultures. This often exasperates American and other Western managers who tend to operate on a schedule with the concept that "time is money" and "we must get this project done and get on to the next," which is an essential mindset to the mining industry.

Controlling

In the United States, termination for poor work performance or behavior that is against company policy is acceptable. However, some cultures avoid employee terminations for poor performance if at all possible (Nisbett 2003). When asked what to do about a 15-year employee with poor performance for a year, who previously had had outstanding performance, managers in the West vary considerably from those in the East in terms of their response. For example, 75% of Americans and Canadians, countries high in individualism, said to terminate; only 30% of Singaporean and Korean managers said to terminate, reflecting collectivist values.

Employee termination is also reflective of cultures high in masculinity. Feminine tendencies lean more in the direction of compassion, caring, and working with problem employees, as opposed to termination (Nisbett 2003). Managers in foreign countries often try to manage the wrong actions. In Hong Kong, an American Sears manager told employees to report to work on time, not late. They did, but left on time instead of working late hours, as they had been doing. This is another example of sequential versus synchronous time. Time to the Hong Kong workers was cyclical; start time did not matter. They would be there, as long as they are needed. However, tardiness and the linear time of the West was all the American manager understood. He would have gotten more productivity if tardiness had been tolerated.

The previous scenarios mostly describe Western managers working in foreign cultures. Managing multicultural work forces means that managers may encounter various dimensions of national cultures in the same work group. For example, in areas with highly diverse populations, managers may have to be sensitive concurrently, which is a challenge. As a result of this dynamic, managers sometimes create team leaders from each nationality in order to address concurrent multicultural issues, with particular emphasis given to language differences.

Ethnocentrism

One of the most damaging cultural issues multinational managers must avoid is an attitude called ethnocentrism. This means that people tend to regard their own culture as superior and downgrade the culture and practices of those from different cultures. American managers, for example, are often accused of ethnocentrism when they assume American business practices are best and degrade local customs and practices. Failing to learn about the language, history, religion, social practices, and legal/political structures will damage credibility, particularly if expatriate managers project a superior, ethnocentric attitude. Such behavior can cause managers to fail and create issues that may result in irreparable harm to a company's ability to manage and operate effectively.

Modifying to Management Competencies

Unless strategies to effectively change local cultures can be implemented, which is difficult and should not be taken lightly (Hofstede 2001), managers must be trained to manage outside their home culture. Trompeanaars and Hampden-Turner (1998) note that before behaviors and management styles can be modified, values must be reconciled. Western managers must understand that behaviors are rooted in deeply held values that may conflict with their training, which is rooted in Western management practices. The previous scenarios

illustrate concrete, visible conflicts; each is rooted in values, not merely superficial behavior patterns that can be easily modified. As a result, competencies must be enhanced with cross-cultural style flexing, based on a deep understanding of the values that dominate the other culture. This could be accomplished by the following:

- Proper selection of those intended for international assignments. Intensive interviews and selection devices must be employed to avoid sending those with latent prejudices that might later come forward and result in conflict.
- Deep immersion in the new culture prior to assignment and an understanding of the values and history of the country's culture. Hofstede's (2001) dimensions offer one approach to identifying cultural issues that managers must be prepared to address. Working through case studies similar to the examples described earlier can help the manager to develop cross-cultural competence.
- Assignment of a culture coach, preferably a bilingual member of the new culture, to work with the expatriate manager. It's important, however, that this individual not become a de facto manager simply by having proximity to the "boss."
- Having proper mechanisms in place to provide feedback about the manager's performance.
- Taking measures to ensure that the managers assigned to lead in another culture are ready for culture shock, separation from home (and possibly family) along with the different foods, language, beliefs, and all the components of culture that will confront them on a personal level. If the family is to be relocated with a manager, extensive efforts must be made to ensure they are prepared for the new culture and environment. All too often an international assignment fails, not because of the employee, but because the family was not adequately prepared for the drastic lifestyle changes required for living in a foreign country.

RECRUITMENT AND RETENTION

A shortage of qualified miners in all types of positions is a critical issue in many countries and regions of the world. During the last decades, as mining declined, the work force was not replaced. Now, economic growth in China, India, and other nations has driven demand for metals and coal. The coal mining industry's current boom cycle has created greater demand and higher prices, but many companies are unable to meet demands because of the severe labor shortage. People currently employed in mining are retiring, and there is a lack of younger people to fill the vacancies. In Canada, it is estimated that between 60,000 and 90,000 mining jobs will open up in the nation during the next decade with not enough applicants to fill them (CIM 2009). The Minerals Council of Australia estimates that by 2015, Australia alone would need 70,000 employees on top of the 120,000 it has now to keep up with demand (MCA 2008). In the United States, layoffs, outmigration, and an aging work force in the Appalachian region have created a shortage of experienced miners. Labor officials put that deficit at more than 6,000 miners in West Virginia and Kentucky alone—the second and third highest coal-producing states. Wyoming, the nation's top coal-producing state, faces similar problems (BLS 2008).

Effective employee relations and leadership skills go a long way to enhancing employee recruitment and retention. However, managers must also undertake specific employee recruitment and retention strategies in order to keep many mining operations fully staffed with competent employees. Recruitment and retention must thus be considered a key organizational objective. Such strategies might include the following:

- Develop retention plans (e.g., employee-friendly hours of work and rosters) to increase the attractiveness of the industry. Long hours and difficult shifts with hours that do not match lifestyles can create recruitment and retention issues, particularly for employees with families. Although cultures and employee preferences vary, a concentrated effort should be made to evaluate shifts to find a schedule that is preferable to employees and yet still meets the company's production needs.
- Upskill the workers remaining in the industry to cover skills obsolescence. If employees who are leaving take essential skills with them, often the only choice is to train those left behind in the skills being lost. This can have several positive effects. First, it can provide a more flexible work force, since employees who are able to do multiple, high-skilled jobs can cover for absences or provide extra help when needed. It can also motivate workers who prefer to do a variety of jobs, thus enhancing retention.
- Increase the emphasis on multitasking among existing employees. As with upskilling, having cross-trained employees, even in lower-skilled positions, increases work-force flexibility and can improve worker satisfaction. It can also, in some circumstances, reduce the number of miners needed in certain types of mining operations. Usually pay incentives are offered to employees who are cross-trained, but extra payroll expense is usually worth the additional skills and the flexibility it provides.
- Have recruitment strategies that encourage young people to enter the industry. Since the past economic slump, some mine operators have stopped recruiting and now older miners are retiring in large numbers. Special emphasis needs to be placed on targeting young people and pointing out the pay benefits and other advantages the mining industry can offer, particularly during times when employment in other industries may be shrinking. An important consideration here is that younger generations are motivated in different ways than older generations. Generally they do not expect to work for the same company all their lives, so "company loyalty" may be lacking. They are often motivated by the latest technology. Personal time is a priority, and excessive overtime requirements may cause them to seek employment elsewhere. They might also need more direction and feedback and expect working relationships that are less formal.
- Introduce paid training or subsidized training in or out of work time. All training offered in today's highly competitive work environment should be offered on paid time. Except in rare situations, new or present employees will not look favorably on unpaid training and might in fact be driven to seek employment elsewhere. The exception to this is when the company is paying tuition for college courses, which may or may not be undertaken during working hours.
- Develop partnerships with education. Partnering with colleges, universities, technical school, and other educational institutions is now being successfully practiced throughout the industry. Subsidizing training programs that are

tailored to the company's specific needs can attract young people who want to make sure they have a job after completing their degree or technical training program.
- Provide compulsory training commensurate with industry competencies, particularly for training required by law. It's necessary to ensure that employees have the basic skills needed for the job to which they are assigned. This may seem obvious, but often basic skill training is neglected, particularly when experienced employees are hired from other companies, as it is often assumed they have certain required skills and knowledge. This may be an incorrect assumption that can be both costly and dangerous. Training varies widely from company to company; just because a miner has performed the same or a similar job at another company does not mean the miner meets the standards for the new job. All newly hired experienced miners must be assessed and trained in areas where they are deficient. Legally required training, including safety and emergency procedures, must *never* be neglected and should always be undertaken for new hires or employees assigned to new positions with different safety requirements.
- Treat contractors as company employees for training purposes. Contractors are an increasingly important fillip to the company's own work force and provide essential human resources to fill various types of positions. Mining operators must ensure that contactors are trained to the same level as employees on the company's payroll, particularly in safety and operating policies and procedures that are specific to the mining operation.
- Multicultural and bilingual training may be necessary. Different cultures approach training and education differently. Nothing can frustrate workers and their supervisors more than being unable to effectively perform their assigned jobs after being trained. This can lead to poor work performance and can cause employees either to be terminated or to leave through frustration. It is imperative that managers be sensitive to cultural and language difficulties that can impede effective training so that they can make the necessary adjustments.

TRAINING AND EMPLOYEE DEVELOPMENT

Training and developing employees and managers is more critical than ever in today's mining industry. Competent workers who can work effectively and safely provide the company with a competitive advantage. In regions where employee recruitment and retention is an issue, having a comprehensive training program can help retain employees by giving them the opportunity to stay current with new procedures and technology. Additionally, prospective employees are more attracted to companies that have a training program that can make them competent miners in a planned, systematic manner.

Training Needs Assessment

No company can afford training that does not contribute to the productivity, quality, or safety of its mining operations. Training needs assessments are an excellent tool to identify programs that are needed and that will contribute to both long- and short-term objectives for the company. Following are some of the benefits of conducting a training needs assessment:

- After completion, the assessment can be used as a tool to evaluate the direction and purpose of training and how training decisions link to performance.
- This tool can help managers and employees complete individual training plans to ensure that employee development is in line with company-wide training efforts.
- The process can give an objective, businesslike basis for decisions when providing employees with the justification for training decisions is difficult.
- Because the mining industry is experiencing many changes in its work environment, it is important to assess the impact of these changes on performance and determine what skills and tools are needed to meet new challenges.
- It has become vitally important for the industry to ensure that training dollars are spent on specific skill needs that will positively impact performance.
- A training needs assessment is the first step in determining whether training is the appropriate response to a performance problem.

When the assessment is completed, a master training plan can be developed with the following elements: The training mission is the purpose or reason for doing the training. Training goals are developed next. They are driven by the mission statement. They are the specific objectives for conducting training in an organization. Once the training mission and training goals are developed, a training strategy is needed. The strategy articulates how to implement the mission and accomplish the goals. The training strategy is followed by a training plan. The training plan outlines the types of training that will be included in the process of meeting the strategy, the mission, and the goals. An implementation plan will help coordinate the timing and resources needed for the training.

Learning Organizations

Advancements in mining technology and a globalized mining industry require that employees at all levels of an organization become lifelong learners to ensure that their skills and knowledge do not become obsolete. Mining engineers, after graduating from college, must immediately begin to acquire new industry- or company-specific skills and knowledge in order to be productive. Senior engineers must also constantly learn to avoid becoming outdated. As a result, lifelong learning has to be deliberate, relevant, planned, and continuous. Organizational learning involves individual learning, and those who make the shift from traditional organization thinking to learning organizations develop the ability to think critically and creatively, which is easier said than done.

Senge (1990) describes five disciplines required for a learning organization:

1. **Personal mastery.** This applies to individual learning; organizations cannot learn until their members begin to learn. Personal mastery has two components: first, to define what one is trying to achieve (a goal), and second, to obtain a true measure of how close one is to that goal. This is critical to mining organizations that have a number of new employees. Knowing how competent an employee is at the job assigned is critical to safe and productive mining.
2. **Mental models.** A mental model is one's way of looking at the world. It determines how a person thinks and acts. A simple example of a mental model comes from Senge's description of arm wrestling (1990). In traditional arm wrestling, the goal is for participants to attempt to force the arm of the competitor to the table as many times as

possible, with each participant attempting to be the one who "flips" rather than be the one who is "flipped." This is a flawed mental model. If each participant could simply stop resisting and take it in turn to flip or be flipped as many times as possible, both could win. In other words, the "theory in use" of arm wrestling is flawed; with cooperation rather than competition, both arm-wrestlers come out ahead. Argyris (1990) suggests that in organizations, this can manifest itself as the desire to

- Remain in unilateral control;
- Maximize winning and minimize losing;
- Suppress negative feelings; and
- Be as rational as possible, by which people mean defining clear objectives and evaluating their behavior in terms of whether or not they have achieved them.

3. **Teams.** A workplace team is "a number of persons, usually reporting to a common superior and having some face-to-face interaction, who have some degree of interdependence in carrying out tasks for the purpose of achieving organizational goals" (French and Bell 1995). Successful teams usually exhibit the following characteristics:

   - A clear, elevating goal
   - A results-driven structure
   - Competent team members
   - Unified commitment
   - A collaborative climate
   - Standards of excellence
   - External support and recognition
   - Principled leadership (Larson and LaFasto 1989, as cited in French and Bell 1995)

   Teams contribute to organizational learning by fostering cooperative behaviors, improving communication, becoming better able to perform work tasks together, and building strong relationships. Creative solutions are developed through the synergy of individuals working together for a common goal. Learning takes place that would otherwise not have happened through individual efforts. (It is important to note here that the use of teams in certain cultures must be handled with care, taking into consideration existing cultural norms such as power distance. For example, cultures high in power distance *expect* leaders to tell subordinates what to do, and the notion of participating in any sort of joint decision-making process is totally alien.)
4. **Shared vision.** The shared vision of an organization must be built on the individual visions of its members. What this means for the leader in the learning organization is that the organizational vision must not be created solely by the leader; rather, the vision must be created through interaction with the individuals in the organization. This is particularly difficult in large organizations that may prefer to operate in a top-down fashion with executives and/or boards of directors creating the vision for the mining company. Effectively communicating the vision in a logical, meaningful fashion, however, and permitting various locations to adopt their own version of the corporate vision can be very effective.
5. **Systems thinking.** In order to understand the source of and the solutions to modern problems, linear and mechanistic thinking must give way to nonlinear and organic thinking, more commonly referred to as systems thinking—a way of thinking where the primacy of the whole is acknowledged. The defining characteristic of a system is that it cannot be understood as a function of its isolated components. First, the behavior of the system does not depend on what each part is doing but on how each part is interacting with the rest. Second, to understand a system, one must understand how it fits into the larger system of which it is a part. Third, and most important, the "parts" need not be taken as primary. In fact, how the parts are defined is fundamentally a matter of perspective and purpose; it is not intrinsic in the nature of the "real thing" being examined (Koffman and Senge 1993).

   On a practical level, systems thinking is perhaps more critical in mining than in most industries where maintaining a balance between safety and productivity is paramount. The interaction of people, equipment, and natural resources—often in potentially hazardous environments—can have disastrous unintended consequences if not fully understood from a systems perspective by all employees. Organizations that understand how systems thinking can improve individual learning by inducing people to focus on the whole system are able to gain a competitive advantage.

Global Transfer of Knowledge

In an age when multinational organizations dominate many sectors of the mining industry, it is essential to develop organizations that foster an environment conducive to learning and are able to effectively share actionable, practical knowledge among far-flung operations all over the world. Most mining engineers are aware of practices that are successful in other parts of the world, or at a nearby mining operation within their own company, that could be an asset to their existing operation. Having the ability to effectively share a new process or technological advancement developed in, for example, North America with mining operations in Asia can provide a tremendous competitive advantage.

Knowledge Management

The transfer of knowledge from one part of the organization to another is easier said than done. It is an aspect of knowledge management that seeks to organize, codify, create, capture, and distribute knowledge to ensure its availability for other users. It is more than just a communication process. If it were merely a question of communication, then a memorandum, an e-mail, or a meeting would accomplish the knowledge transfer. Knowledge transfer is more complex because (1) knowledge resides in organizational members, tools, tasks, and their subnetworks (Argote et al. 2000), and (2) much knowledge in mining organizations is often tacit, localized, and hard to articulate (Nonaka and Takeuchi 1995). Obstacles to organizational learning include the following:

- **Organizational culture.** A culture that is control-oriented and formal can inhibit learning. For knowledge to be shared effectively, employees at all levels of the organization must be free to communicate formally and informally.
- **Organizational structure.** Hierarchical structures, as opposed to those with fewer layers, can impede knowledge transfer much in the same way that formal cultures can. If information must be shared only formally, learning can be impeded. Often, informal

communication can lead to serendipitous learning, which is more beneficial than learning shared through formal means. "Organic" organization structures not hampered by rigid chains of command are better at facilitating knowledge transfer.

- **Trust.** Organizations that lack trust have difficulty in effectively sharing knowledge. Lack of trust is often based on fear and insecurity, which tends to make people hoard information, even in the same company; there is an unjustified acceptance that "knowledge is power."
- **Personal interaction.** Although various electronic media, including teleconferencing, e-mail, and the Internet, have made the transfer of large amounts of information easy and virtually instantaneous, no medium is better at facilitating learning than personal interaction between employees. Organizations that encourage extensive face-to-face interaction will always fare better in knowledge transfer than those that do not.
- **Being intentional.** Starting from the top, organizations must be intentional about learning and the transfer of knowledge. In mining, the day-to-day business of making a profit can often overshadow efforts to constantly learn, improve, and share information. Leaders must make sure that time and resources are available to learn, as well as to capture and share what has been learned with other parts of the organization.

Learning from the Outside

One of the greatest sources of new knowledge can come from outside the organization. Suppliers, vendors, and contractors can be a great source of information and learning opportunities. Often, efforts to keep trade secrets or unique processes from falling into the hands of competitors can backfire and cause organizations to lose opportunities to learn from, as well as teach, outside organizations that are essential to the company's success. Toyota is renowned for developing relationships that are mutually beneficial and result in tremendous cost savings and productivity improvements for itself and outside organizations. Rather than keep outside contractors at arm's length, Toyota strives to develop close, long-term working relationships and engage in mutual problem solving. They do not focus on price and detailed formal contracts as much as Ford, GM, or Chrysler, but rather on relationships grounded in trust that are beneficial to all partners. Such relationships result in tremendous costs savings and other improvements.

Alliances, partnerships, joint ventures, government partnerships and other forms of inter-organizational relationships are also tremendous sources of learning. They provide opportunities for knowledge sharing that mining companies must capitalize on to maintain a global competitive advantage. As in the case with contractor and supplier relationships, intentional efforts to develop ties based on mutual, long-term benefits, as opposed to short-term profits or cost savings, can create a competitive advantage for all parties.

Scope of Training

Each new miner should be provided thorough training in the areas discussed below. Safety and health training should never be viewed as a task to simply complete, regardless of the degree to which miners appear to understand what they are supposed to have learned. During the first week on the job, new miners should be assigned to a well-trained mentor, who will work closely with them to ensure that they are performing their tasks in a safe manner. Adequate time, effort, and resources should be devoted to the following subjects before a new miner begins work at the mine.

New Miner Training

An example of a training outline recommended for a new-miner training program is shown in Figure 5.3-2. This example is modeled on a U.S. program and as such is designed to meet the safety regulations of that nation. New-miner training must always begin with the safety and health training required by the regulatory agency that governs the locale where the miner will be working; most importantly, it must address the safety requirements specific to the new miner's job.

Initially new-miner training should include the following:

- Introduction to the work environments—information specific to the location(s) where the new miner will be working. Because training required by regulations will never cover the exact particulars of any miner's work environment, it is vital that this section of the training be extensively covered.
- General hazard observation—instruction on the recognition and avoidance of electrical hazards and other hazards present at the mine, such as traffic patterns and control, mobile equipment, and loose or unstable ground conditions.
- Emergency procedures—a review of the emergency medical procedures and the escape and emergency evacuation plans in effect at the mine, as well as instruction on fire warning signals and firefighting procedures.
- Health and safety—instruction on the health and safety aspects of the tasks to be assigned, including the safe work practices associated with such tasks, the physical and health hazards of chemicals, and the mandatory health and safety standards pertinent to such tasks.
- Legal rights—instruction on the statutory rights of miners and their representatives under the prevailing labor laws of the location. (Some nations may require that union representatives be present at such meetings or require that specific notice be given to representatives before training occurs. Managers must therefore be clear on what local laws require.)
- Line of authority—a review and description of the line of authority of supervisors and miners' representatives and the responsibilities of such supervisors and miners' representatives. Depending on the nature of prevailing labor laws, it is important that these lines be clearly established to avoid potential communication issues and/or the violation of miners' rights.
- Hazard reporting procedures—an introduction to the company rules and procedures associated with reporting hazards. Once again, these must be location and task specific. In all cases, the most critical health and safety information that all new employees need to know is that which is unique to their job and location.

Immediately after new miners begin work, they should be provided with training in the following subjects (as well as others that may be required by local regulations), if these subjects have not been taught earlier:

- Self-rescue devices—instruction and demonstration on the use, care, and maintenance of self-rescue and respiratory devices, if used at the mine.

Title: Blaster (blaster's helper)

Tasks: Responsible for safe blasting operations or assisting experienced contract blaster in proper loading procedures, shot guarding, and postblast examinations.

Materials needed: Company policy and operational procedures; location-specific regulations; company energy guide; loading poles; blasting equipment for type of initiation system used; information sheet for hazardous materials as applicable.

What instructor will do: Review company operating policy and rules; review material safety data sheet; demonstrate proper use of equipment; demonstrate proper preloading procedures; demonstrate borehole inspection prior to loading; demonstrate emergency procedures; demonstrate safe loading and handling procedures; demonstrate proper hookup and initiation procedures; demonstrate proper shot-guarding procedures; demonstrate proper warning signals; demonstrate postblast examination; review regulations for blasting; explain inherent job hazards; observe and give guidance to student during simulation and operation.

What student will do: Read company policy, company energy guide, and information sheet for hazardous materials; observe and listen to demonstrations and reviews; understand the proper use and care of equipment; simulate all steps of safe blasting—procedures for preblast operations, loading and handling explosives, firing of blast, and postblast procedures; simulate proper housekeeping procedures, as well as any other deemed necessary by instructor.

Evaluation procedures: Student will give correct response to questions and will successfully demonstrate knowledge of care and use of blasting equipment, safe handling and loading of explosives, hand and audible warning signals, emergency procedures, shot guarding, and any other demonstration deemed necessary by instructor. The student will be observed by an experienced blaster and supervised during light production; any retraining needed will be accomplished until the student can operate without significant error through three error-free cycles.

Title(s) of personnel conducting training: Experienced blaster, supervisor/foreman

Figure 5.3-2 Example outline of training procedures for new-miner training program

- First aid—a review of first-aid methods.
- Other health and safety training—any remaining training on any other subjects that promote occupational health and safety for miners at the mine.

Newly Hired Experienced Miner Training

Newly hired experienced miners should be provided essentially the same training as new miners. One of the biggest mistakes managers make when hiring employees from another mining company is assuming they have been properly trained. The issue here is that some have acquired bad habits, particularly if they have come from a culture that did not stress or enforce safe work habits or good work ethics. All course topics covered under "New Miner Training" should be reviewed with experienced miners.

Annual Refresher Training

Following is a list of topics that should be readdressed with all miners. This list is not exhaustive, and, as previously described, local regulations or mine conditions or processes should be carefully considered to ensure that no training requirement is overlooked.

- Mandatory safety and health standards—mandatory health and safety standards for assigned tasks, including information about the physical and health hazards of chemicals in the miner's work area, the protective measures a miner can take against such hazards, and the contents of the mine's hazardous chemicals awareness program.
- Transportation and communication—specific information about transportation controls, as well as communication systems, with particular reference to anything that may have changed since the miners were last trained.
- Fire and emergency evacuation—escape and emergency evacuation plans, fire prevention, warnings, and fire fighting.
- First aid—first aid methods and location of first aid stations and equipment.
- Ground conditions—ground conditions and control and other critical issues, since such conditions often change.
- Traffic patterns—traffic patterns and controls going to and from the mine site.
- Highwall practices—working in highwall areas.
- Water hazards—water hazards, pits, and spoil banks.
- Illumination—procedures for ensuring that areas are properly illuminated for night work.
- Electrical hazards—hazards directly and indirectly related to electrical equipment, wiring, and controls.
- Accident prevention—prevention of accidents, giving particular attention to the processes or pieces of equipment that are associated with most injuries (and fatalities).
- Health—personal health awareness with specific reference to the avoidance/prevention of health issues associated with the miners' environment.
- Explosives—awareness of safe work practices when explosives are in use, with primary focus on those not in blasting positions or others who will receive detailed, job-specific instruction about blasting.
- Respiratory devices—proper selection and use of respirators for certain positions and annual fit testing.
- Maintenance and repair—proper procedures for reporting damaged or malfunctioning equipment, with particular reference to issues that may cause unsafe working conditions.

- The use of hand tools and other equipment—instruction on the use of hand tools or other equipment not previously covered.
- Hazard communication—familiarization with the specific chemicals in use in the workplace, and how to read and follow the safety procedures in the case of accidental exposure from contact, inhalation, or ingestion.
- Other—additional job- or site-specific instruction not covered in any of the programs previously mentioned.

Training the Trainer

Although trainers must have some knowledge about mining operations, having the necessary skills to conduct a good training program is critical. Obviously, good trainers can make or break any training program. Regarding on-the-job training (OJT), many mining operations managers make the mistake of assuming the best miners make the best trainers, or "mentors" as they are sometimes called. Mentors or trainers should always be trained in basic training skills. Mechanisms must be in place to monitor the quality and effectiveness of OJT and to make sure that trainers are thorough, that they always emphasize safe work habits, and that they do not encourage new employees to shortcut processes or procedures. In some cases, experienced miners have, over time, learned how to simplify steps in a job, sometimes at the expense of safe work practices. Assigning a skilled trainer to periodically observe and assess OJT is a good investment. Training employees properly from the start is much more cost-effective than retraining, particularly if mistakes resulting in property damage or injuries have triggered the retraining needs.

Methods of Training

Various methods of training are available today. Each has strengths and limitations and should be evaluated based on needs, audience, and subject matter. Methods available include the following:

- Self-instruction, which may involve a workbook, DVD series, computer-based training, or an audiotape series.
- Mentoring, which usually involves assigning an experienced employee to work with and "coach" an inexperienced worker. It is "hands-on" training and permits the person to learn on the job a task that's been assigned. It can enable a transfer of tacit knowledge that could not be accomplished through other methods.
- OJT, or hands-on training, conducted in the actual work situation. This may or may not be done in conjunction with the mentoring process previously described. The advantage of this method is that, because the person learns by doing, the training is very practical. The disadvantage is that, if permitted to observe and train only on the assigned position, the person may not know how the job fits into the rest of the mining operation. It is therefore critical to provide employees with an overview of the entire mining process at this particular location.
- Seminars presented in formal training settings and covering a wide range of topics. These generally address material relevant to management, engineering, and other staff, although some seminars are appropriate for mining operations employees. It is important to carefully evaluate the subject matter, prior to delivery of a seminar, to make sure it is pertinent relative to the training needs assessment.
- Distance learning, which may include Web-based seminars, on-line degree programs, and teleconferences. These alternatives can afford some of the same benefits as formal or off-site seminars and avoid the time and expense of travel.
- Work-site visitation, which involves permitting individuals at all levels of the organization to visit other mining operations to observe and learn. This allows visitors to personally observe the operation, interact with those working in the process, ask questions, and/or hear presentations. This is an extremely effective method for sharing knowledge as well as for developing relationships. It is particularly helpful in the transfer of tacit knowledge that cannot be codified and stored.

Training, Motivation, and Retention

Behavior modeling first appeared as a method of instruction in the 1970s. It is based on the work of psychologist Albert Bandura. He concluded that people learn best by imitating the behavior of others (modeling), and then receiving an immediate, observable reward. B.F. Skinner expanded this into reinforcement theory. He observed that positive consequences produce positive results, and negative consequences produce negative results. The concept has been adopted by many corporations today and is the theory behind a great deal of current training material (Zemke 1978). Behavior modeling is based on the principle that if a person observes a task being done correctly, and then is given an explanation as to why it is being done that way, this knowledge and practice will be transferred and will transfer to the workplace more effectively. Employees will know not only how to do it but also why it must be done in a particular way.

Behavior modeling works in the following way: The trainer performs the task while the participant observes, and then explains to the person what has been done and why it was done that way. Then the trainer asks the employee to perform the task and explain how to do it and why. The trainer gives feedback as to the employee's performance. This process may be repeated several times until the employee has the skill. The method is highly effective in mining operations where training is done under the direction of a mentor or in an OJT setting (Figure 5.3-3).

Four tools are helpful in supporting behavior modeling: coaching, positive reinforcement, counseling, and feedback. Models for each of these tools appear in Figures 5.3-4 through 5.3-7 respectively.

Training Evaluation and Record Keeping

Evaluation and record keeping are essential to an effective training process. Records must be maintained for topics required by law. Records are also essential for knowing who is qualified to perform what jobs so that the company can avoid placing untrained, unqualified employees in certain positions. Evidence as to how well training has been conducted and retained by those being trained can be obtained through written or oral feedback. Either way, the results should be documented for future reference.

Simulation Training

Simulation permits training to be conducted in a realistic setting without the safety risks and property damage that are associated with OJT in an underground or a surface mining

Step 1	Explain the skill	The trainer explains how to do the job and why it is done this way.
Step 2	Show how to do it	The trainer shows how to do the job.
Step 3	Have employee show how to do it	The trainer asks the employee to demonstrate how to do the job.
Step 4	Have employee explain the skill	The employee explains how the job was done and why it was done that way.

Figure 5.3-3 Steps in behavior modeling

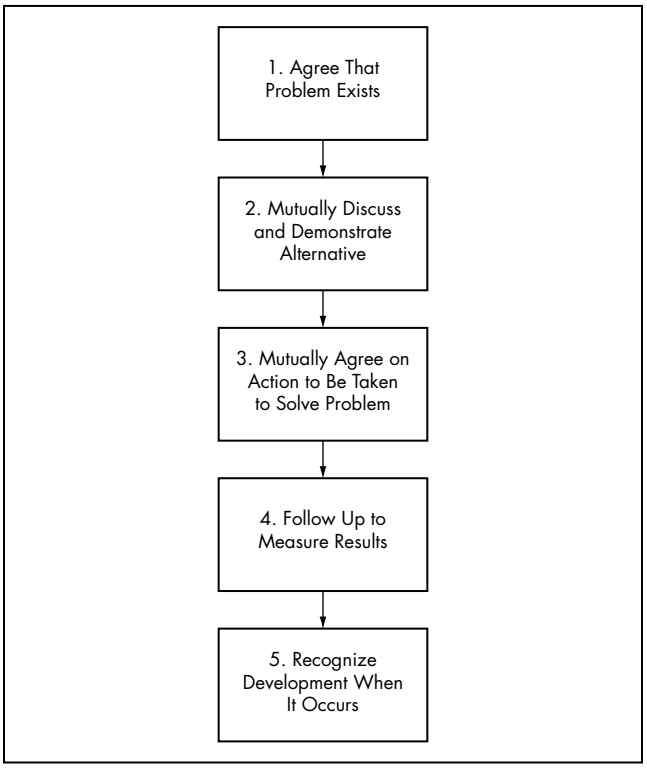


Figure 5.3-4 Steps to coaching technique

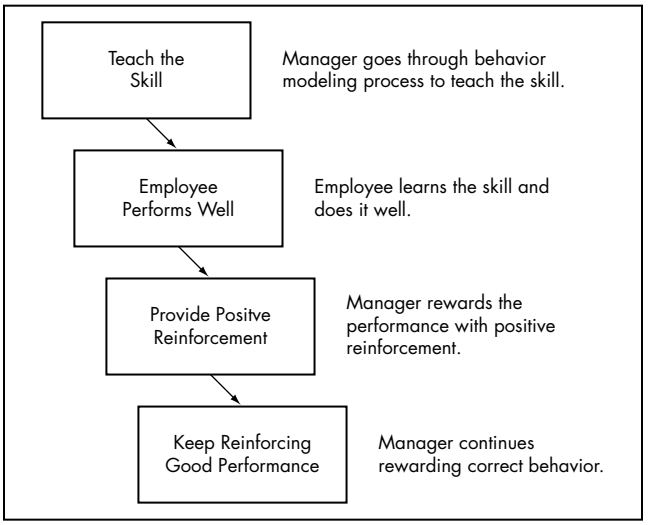


Figure 5.3-5 Positive reinforcement as a support to behavior modeling

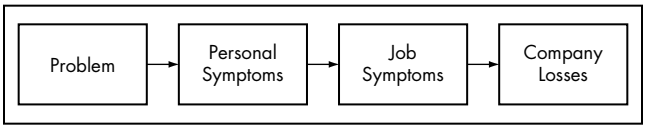


Figure 5.3-6 Counseling as an essential element in supporting new skills learned and maintaining productivity

1. Give frequent feedback	Do it all of the time.
2. Remark about the behavior	Let the person know when behavior is correct.
3. Reflect the feelings of the worker	Be sensitive to problems with the new skill.
4. Check perceptions	Ask how the person feels about the progress being made.
5. Give timely feedback	Provide feedback when the situation occurs. Do not wait several days.

Figure 5.3-7 Feedback as a means of reinforcing training

environment. In addition, simulators can be an asset to concept and capability development, to system design refinement and validation, and to process and systems modeling. Mining managers are therefore encouraged to evaluate the use of simulators and consider using them when practical and available.

SUMMARY AND CONCLUSIONS

The management of mining operations has become increasingly more complex in the last decade because of the rapidly growing global demand for mined resources. The need to maintain effective employee relations in far-flung global mining operations requires special leadership skills in order to effectively recruit, train, and motivate employees, particularly in light of today's mining labor shortages.

Mining leaders need to understand that efforts to maintain good employee relations must be ongoing and incorporated into the organization's strategic plan. Efforts should be tailored to fit the culture of the employees, and managers should understand that leadership styles and approaches must sometimes be adapted and modified to be effective in global operations.

A well-trained work force and a commitment to organizational learning can be a tremendous competitive advantage that must be nurtured and encouraged in today's world of rapidly changing mining technologies. Leaders must be innovative in seeking ways to develop, capture, and disseminate knowledge throughout the organization.

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CHAPTER 5.4

A Global Perspective on Mining Legislation

Diana Dalton

The laws relating to the exploration for and mining of minerals, although often different in approach from one country to the next, do have similarities. The purpose of this chapter is to advise the reader of what approaches may be found in the international context and what they mean for the practitioner.

No mining law can be read in isolation. Many other laws impact mining operations and can affect the way a mining law is implemented administratively. In developed countries, the law has progressed over the last three centuries, changing in approach as new issues, technologies, and practices have evolved. In this environment, good practice exists, and change is addressed in the normal course of events by highly trained experts. However, in developing countries, such experience is usually lacking.

This uncertainty in dealing with sophisticated mining entities and expertise can be frustrating for companies but is understandable. It is important to recognize this and be willing to take time to transparently explain what is required. It is also important to understand the social, economic, and political environment in the country where mining operations are to occur.

COMMON LAW AND CIVIL LAW

Two systems of law exist in various forms: common law and civil law. Most of the world's large mining companies are headquartered in countries that practice common law but often run operations in countries that practice civil law. It is important to know what type of legal system applies in the country where the applicant is planning to work in order to understand the laws and regulatory system.

Common law is generally called English or British law and is the system one finds in British Commonwealth countries as well as the United States. A common law system is based primarily on custom (before there were written laws). Although much of common law has been codified, these customary laws continue to be applied by the courts. For instance, the United Kingdom does not have a written constitution but has a series of laws, agreements, and judicial decisions that set out the fundamental rights of citizens. Most other common law countries have written constitutions and statutes that encompass these common law principles and legal approaches. In a common law system, although the statutes are a primary source of the law, judicial interpretation and *stare decisis* (precedent) in case law co-exist as important elements of legal interpretation. In civil law systems, judges may look to other legal decisions but are not bound by them.

A *civil law* system is based on statutory law made by a legislature with regulatory powers given to the executive. Treaties, orders, decrees, and other such documents are also a part of the written law. Everything is written, and judges apply the law; they do not interpret it. Civil law is the system found in South America, Europe (except for the United Kingdom and those parts of Africa that are not part of the Commonwealth), and Asia. The civil code systems are French (Napoleonic), German, Scandinavian, Portuguese and Brazilian, Dutch, and Chinese. Each has a different approach to lawmaking. Although Brazil has transitioned to a German type of civil law, Quebec and Louisiana have French civil code systems (except for criminal law which is national codified law). Both Scotland and South Africa employ a mixture of civil law and common law. Japan, South Korea, and Taiwan are based on the German civil law system.

The difference in the two legal systems is important to understand, especially when it comes to disputes regarding agreements, which will be discussed later in this chapter.

WHAT IS A MINING LAW?

The words *mining law* and *mining code* are often used. They can be one and the same thing or totally different. Mining code most often refers to all the laws, regulations and guidelines, and agreements pertaining to exploitation of minerals. Thus the term encapsulates all the relevant legal documents that oversee that activity.

Mining law stems from property law. It is similar to land laws in which an owner of a property leases it to another who uses it for his or her purposes in return for payment for that use. So a mining law should determine

- Who owns the mineral resources;
- Who issues leases;
- How the rights will be allocated and what terms and conditions apply to those rights;
- How the resource is to be treated;

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- How the land from which those minerals are extracted is to be treated;
- Who pays what and to whom; and
- Whom the law applies to, whether the public, private investors, domestic companies, or foreign organizations.

OWNERSHIP OF MINERALS

In most civil and common law countries, the ownership of minerals is vested in the state. In unitary countries this would mean the national government. In federal countries, such as Canada and Australia, the constitutions have vested the provinces and states with natural resource jurisdiction.

The two major countries where this is not the case are the United States and the United Kingdom, where minerals form part of the land and thus are owned by whomever holds title to the land in which they exist. However, in the United Kingdom, there have been significant changes from private ownership. In 1946, coal was nationalized, and in 1994, all private rights disappeared and ownership was vested in the Coal Authority. Oil and gas are also vested in the state, while precious metals such as gold and silver are owned by the Crown. To mine these precious metals, it is necessary to obtain the right to mine from the Crown Estates Commission. However, all other nonenergy minerals are privately owned (though some are owned by the Crown or a department of government, just to complicate matters), and thus there is no national licensing system. To exploit these minerals requires regional planning approval.

Other than these two countries, either the constitution or a mining law will outline who owns the minerals and by what authority the law is administered. It sounds simple: the state owns the minerals and has jurisdiction to license the exploration and exploitation of all minerals. However, while the state may own the minerals, in many countries, it does not own the land in which those minerals exist. This is problematic, because one must deal with the private landowner in order to have use of the land.

LAND ISSUES

Most mining laws will address a number of land-related issues:

- What areas are reserved or can be declared to be reserved, such as national parks and nature reserves? These reserved lands may be permanent or temporary and are not usually declared pursuant to a mining law. Other legislation affecting land may determine when and how land is reserved for certain uses and who has the power to make these decisions.
- What is the process by which the applicant must get permission from the landowner to enter and/or use the land? Some laws are silent on this, but usually some section gives a right of entry and sets out how the applicant must deal with local landowners. This could be in the mining law but normally is in land-use laws, especially if there is a customary land title, as there is in most countries where tribal ownership is the way land is held. It is important to check these laws to determine the process for gaining entry to customary land.

In many developing countries, especially those with tribal ownership, there is no title registration system, thus making the determination of land ownership much more difficult. In developed countries, a land registration system has already been established, making it easy to determine the titleholder.

In other countries, such as Canada or Australia, large tracts of land are owned by the government (Crown land). Many countries have laws that allow the government to expropriate privately owned land according to a formula that ensures fairness to the landowner. This is used only in cases where the mining license applicant has not been able to reach a deal with the landowner and the discussions have been protracted and difficult. In Canada and Australia, it is also necessary to establish whether the lands are under a treaty agreement.

Because every country deals with land in a different way, no one model can be put forward as the norm. Numerous laws deal with land, such as forestry, agriculture, water, air, health, safety, transport, public security, zoning, parks, and reserves. All of these will impact land access and use.

The issue of entry and the use of customary land is a difficult one for mining companies everywhere as it is not a topic usually covered in the mining law. One country that has addressed this issue is Papua New Guinea. The Mining Act 1992 (established by Parliament of Papua New Guinea) sets out the principle of compensation, which is defined as "all loss or damage suffered or foreseen to be suffered by them [landowners] from exploration or mineral" operations. It also sets out what compensation is payable and in what circumstances. The act allows compensation for

- Damage to the natural surface,
- Severance of land from other land owned by the landholder,
- Loss or restriction of rights-of-way or easements,
- Loss or damage to improvements,
- Loss of earnings in land under cultivation,
- Disruption of agricultural activities, and
- Social disruption (usually meant to include the necessity of moving people from the area).

The act clearly states that no compensation is payable for allowing entry, for the value of any mineral on the land, or for rents or royalty assessed regarding the mining of minerals. It also states that any dispute over customary land is heard under the Land Disputes Settlement Act. In addition, because trees are an important resource for customary landholders, the value of trees is set by the Valuer-General under other legislation. (The Valuer-General is a public servant appointed by the minister under the Valuation Act, 1967, a law relating to the valuation of property.) In many countries these principles are not contained in legislation so these matters remain to be negotiated with the private or customary landowners. Despite the attempt by Papua New Guinea to address the issue, a mining company must determine land boundaries in tribal and clan-owned areas—a long and difficult process because there is no land registry system.

Land entry and access are complicated issues. They are time-consuming and often difficult, especially in customary land-owned areas. Therefore, before planning to start exploration or mining operations in such a country, it is important to solve any land and ownership interests to ensure that land access is available for both the mining and land-use activities connected with mining.

CONTENTS OF MINING LAW

A mining law is essentially a property law. It is important to understand how to get a legal right to explore for or mine the minerals—normally, a license or lease; how it is retained; and

what rights and responsibilities are part of the license. In some laws, the word *license* is used, while in others it is *lease*. For purposes of this chapter, license is used.

Most mining laws do not deal with the financial aspects of mining, such as taxation, although some laws may have a section dealing with royalties and so forth. Those provisions are usually defined by the tax laws (see "Financial" section later in this chapter).

Definition of *Mineral*

The definition of a mineral is different in most laws. For instance, in many countries one will see separate laws for coal (India) and precious minerals (Australia, Botswana), geothermal water, spring water, construction minerals, and alluvials. As such, the definition in the mining act may vary, making it necessary to determine which law establishes rights for the particular mineral of interest.

Who Can Apply

The law should state who can apply for mineral rights. In many laws, only local citizens have rights regarding alluvial or nonmechanized mining of minerals. Differences with regard to construction, industrial, and radioactive minerals also exist. Usually foreign companies can apply for rights, but it is necessary to register a local company at some stage under the company law of that jurisdiction.

Anyone who obtains a license is subject to certain rights established in the law:

- The right to enter the land (subject to access provisions)
- The right to explore for or mine the land
- The right to construct facilities and use water (subject to water laws)
- The right to ownership of minerals that are mined

This latter provision may not be specifically stated in the law, and, if it is not, there should be a condition in the mining license that transfers the ownership of the mined minerals from the state to the titleholder. This is necessary to ensure that the mining company has the right to market the extracted minerals.

Security of Tenure

One of the key elements of a law is security of tenure (or title). This allows the holder of the license to have the exclusive right to explore and mine minerals in the license area. In most laws, a license gives the exclusive right to explore (or mine) minerals in the area covered by the license, ensuring that no one else has access to that particular area. However, this right will not be given for a reconnaissance license, which is generally available for anyone at any time and is nonexclusive. It allows for access to an area for initial inspection, aerial surveys, and so forth but gives no right to proceed to exploration.

In many countries, an exploration license gives the exclusive right to explore within the area comprising the license for all minerals. In other jurisdictions, a license will only be given for a mineral or group of minerals specified in the license. This means that within the area of the license, two or more licensees could be exploring for different minerals—that is, exclusivity of the mineral but not the land area. The result of such an approach can lead to difficulties with other explorers when land access becomes an issue or if each finds the same or a different mineral in the same area, both of which are capable of being developed.

Security of tenure and continuity of such title are necessary. In most legislation, provisions give an explorer the right to a mining license (provided certain conditions are met) and the right to exploit any mineral that is found that can be economically mined. It is important to ensure that the company that has taken the risk and spent large sums of exploration dollars is able to proceed to the mining license.

Licenses

In most countries, there is a two-license system: exploration and mining. However, some countries have a claim system (e.g., all Canadian provinces and Namibia), which can replace the exploration license or be for smaller areas while an exploration license is granted for a larger area. Exploration licenses are usually granted for short periods: 1 to 2 years with renewals up to a specified total. Other laws have short terms, which are renewable for an indefinite period.

In many countries, it is mandatory to relinquish a specific percentage of the land (usually 50%) on each renewal. This mechanism is used by governments to ensure that licensees concentrate on the most prospective land in their license areas. In addition, it ensures that more land is available to new players who can apply for licenses over relinquished areas. The theory is that the more land being explored, the greater the possibility of finding an economically viable resource, which is in the interest of both the licensee and the government. Other conditions for exploration are minimum work programs and expenditures that must be carried out during the license term. Failure to comply means the loss of the license, unless an amendment is obtained.

A mining license usually runs for a specific period that allows for long-term mining activity. Many laws will specify a term (e.g., 25 years) with longer-term renewals. Others put no term on the license but allow for mining until abandonment or the resource is depleted in accordance with the mine plan.

Many countries have a license that allows a licensee to retain an exploration area for a specified period of time (usually 5 years) for a number of reasons, including the deposit cannot be developed at current market prices, the technology is not currently available, or the resources are required to sustain future operations of an existing or proposed mine. Each year, reports must be submitted demonstrating whether the circumstances have changed so mining can commence.

As stated previously, in most countries mineral resources are owned by the state. This allows governments to require reports be made to the government with regard to the work conducted, resources found, and geoscientific data (including cores and interpretations, etc.) collected. Submission of these reports is required, and failure to file may well result in the cancellation of the license. This information is then usually available for new explorers once a license terminates. Such information is also available to those who apply to explore areas that have become available due to relinquishments by another company.

Registration

Because mining laws deal with property, they include a system of registration for mining titles, which is overseen by an official, usually called a registrar. The registrar accepts applications for mining titles and registers the granting of those titles. The title is not valid unless it is registered, much the same as a deed for a home. During the term of a title, any legal matter that is undertaken, such as a transfer of title, joint

venture, mortgage, or other encumbrance on that license, must be registered in order to make any such document operative for the purposes of that title. Transfers of title usually require approval of a minister or another designated official before they can be registered. In many jurisdictions, agreements between joint venturers or partners also need to be registered.

One issue that arises has to do with multiple exploration applications for the same land. In some laws, it will be stated that the first application made to the registrar is the one that will be considered. If that application is found to be in good form with all the information required, it will be approved and other applications will not be considered. This is called a *first-in-time* provision and is considered important by the investment community. If two or more applications arrive at the same time, the law or regulations will outline how the registrar decides which application is considered first.

In other jurisdictions, multiple applications are considered on the basis of the work programs and expenditures. In other words, all applications are reviewed and a recommendation is made to the minister or official granting the license as to which program is most acceptable. This system is appropriate when the administration is open; however, the absence of such transparency is often problematic.

Cancellation, Termination, Suspension, and Surrender

Most mining laws have provisions that deal with the circumstances of cancellation, termination, suspension, and surrender of mining titles. All of these have different meanings and require different processes in order to be effected.

Cancellation is the termination of a mining title and is triggered by the government when the licensee does not conform in a material way to the terms and conditions of a license or is in contravention of the law. The government must usually follow a process in order to cancel a title. In common law systems, administrative law requires that a licensee be given adequate time to remedy the situation before a cancellation is made, and most laws will outline a process. If such is not the case, then an agreement should be made between the government and the licensee to address the issue.

The law should clearly specify the conditions under which cancellation occurs. This is particularly important for companies that require financing by banks or other lending institutions. Lenders use the mining title as a security for lending, and, because titles have a specified term, they will want to ensure that cancellation is a remedy in only the most serious of situations. Most lenders will insist on an agreement if no provisions are available in the mining law.

Termination is often referred to as the end of the title's term when the resource is depleted in the case of the mining license or when exploration is at an end. At this stage, abandonment works become operative, and such procedures are usually spelled out in the law or regulations. However, in some countries, provisions for site reclamation and facilities abandonment are covered by other laws, such as environmental, industrial, or land laws. For each project, such provisions are usually part of the environmental and/or mine plan and clearly outline the specifics for that project. These provisions could also be part of an agreement between the government and investor.

Suspension can be initiated by either the licensee or the government and is for a limited period of time. If done by the government, it is usually because of issues that arise if the licensee is not in compliance with the conditions of the license. In many instances, suspension is because of safety or health issues at an operation. An inspector often has the power to order a suspension of operations until safety issues are addressed.

If initiated by a company, suspension is usually due to market conditions, equipment failures, resource problems, force majeure, or other issues that require operations to cease for a period of time. Most laws will state that the licensee must inform the government and continue to report on any change in these conditions. The law should address both of these instances.

Surrender is the relinquishment of an area or license and is usually initiated by the licensee. Mandatory relinquishment of exploration land is a type of surrender.

Rentals and Fees

Some countries impose land rentals on the areas covered by licenses. These are usually set at a certain cost per hectare and may rise as land is relinquished. It is a mechanism used by governments to ensure that exploration proceeds on a timely basis. Rentals and fees for applications and other administrative procedures are usually published in regulations or guidelines.

Powers of Officials

It is always prudent to determine who is responsible for the administration of the mining law, and this determination will normally establish to whom applications must be sent, who grants such titles, and who administers those titles during the license term. It is extremely important to review inspection powers, because inspectors usually have the power to suspend operations and give orders to compel rectification of issues. In more developed countries, these inspection powers are seriously enforced, whereas in developing countries, enforcement can be more problematic.

Small-Scale Mining

In most developed countries, small-scale mining is treated similarly to large-scale and is implemented under the same law. In developing countries, small-scale, alluvial, and gemstone mining are more problematic because much of the small-scale mining is illegal. Despite provisions in mining laws to license such activities, the legal systems have not been able to address these activities due to the lack of enforcement and other social issues, such as poverty.

Alluvial mining is often addressed in the same law as larger-scale mining. A number of countries issue a small-scale license that is available only to its citizens, is of a shorter term, and can cover the same land as an exploration or large-scale mining license. Gemstone mining is treated similarly. Quarries are often managed under the mining law that applies to all mining activities, but some jurisdictions have a separate quarry law (for instance, the Aggregate Resources Act of Ontario, Canada), while some mining laws give a landowner the right to mine construction minerals on its own land without a license.

What is important to know is whether foreigners can mine on a small-scale basis and what minerals can be mined. It is also important to understand that these small-scale activities may well take place on larger areas that are already the subject of an exploration or mining license.

The presence of illegal mining poses real problems for a serious investor. Many governments are not equipped to deal

with these activities. Environmental degradation, as well as corrupt practices and criminal activity, are inherently connected to these illegal small-scale operations. For years many governments have tried through legislative means to address this illegal activity, yet it continues to be a serious problem. For governments it represents a loss of revenue, whereas for investors it can create an environment in which proper mining practices are absent and consequently the potential risks are much higher.

STATE PARTICIPATION

Many developing countries take a participating interest in mining projects. There are as many different approaches to this issue as there are countries. Because of pressures from international development agencies, most countries have what is termed a *participating interest*. This means that the country, at a point usually outlined in an agreement or the law, will buy into the project (up to a certain percentage) as a partner. The cost to the government would be the same as if a new joint venture partner were buying in—that is, past costs are paid for up to the time the participating interest is operative.

Some countries have what is termed a *carried interest*. This is extremely unpopular with industry as the company must *carry* the government and absorb its costs. Rather than buying in, the country gets a percentage of the project and does not contribute to any funds upfront. In most instances, the carry is for the exploration period through to feasibility. At the development stage, the country would commence paying its share.

Laws usually state the amount of participating interest. An agreement, similar to a joint venture agreement but which also includes financing provisions, outlines the details of how that interest will operate. The country usually takes its participation through a state-owned mining company or a holding company with which the agreement will be made.

AGREEMENTS

In general, agreements in common-law countries tend to be more detailed because of the focus on specific language. This is because law courts may have interpreted the meaning of certain words, and thus one must be careful as to how terms are stated. In countries that practice the civil code, this is not as much of a consideration. It is important to seek the advice of local counsel before commencing agreement negotiations, as they are aware of the various nuances and differences in the specific system of law.

The law should state who has the right to enter into the agreement on the part of the country. Usually other laws (or the constitution) establish who has power to make an agreement on behalf of the country, and there may be constitutional prerequisites. In some countries, agreements, once negotiated, are put before the parliament for final approval before they are signed. Companies usually have board directives as to who can sign agreements on their behalf. The most important thing to remember is that an agreement cannot change the law; only the legislature can do so. Therefore, it is extremely important to have the company legal counsel, as well as local counsel, involved in the process of negotiations.

It is now a practice in many places to sign letters of intent or memoranda of understanding that outline the broad issues on which the parties concur. These are not legally enforceable and are merely used to establish the general principle on which the parties will subsequently base a written agreement. Agreements are often necessary because investors want certainty in politically risky countries, especially to secure financing from lenders who want to ensure a predictable environment for the operation. They also provide for clarification of issues that may be unclear in the law or address issues that are not part of the law. Socioeconomic benefits accruing to local populations are also important consideration for governments, and the agreement will often address obligations on these issues.

Agreements can supplement or implement law. Most often investment or development agreements with a country are supplemental to legislation. In order words, they address issues that are either unclear or absent in the law. Because there are as many agreements as there are countries, it is impossible to outline what a standard agreement might entail. In general, however, such agreements address the following:

- Clarification of mineral rights, if there are no legal specifics on things such as duration of rights, relinquishments, selling or transferring, clarification of cancellation or suspension of right, and work obligations.
- Rights to sell and market minerals, as well as export them.
- Obligations of both parties. These should be agreed upon, especially with regard to who will build the infrastructure (roads, transmission lines, buildings, etc.) to support mining operations, the terms for employment and training for nationals, local procurement obligations, and reporting. This may include safety and health obligations and/or environmental provisions if no law addresses these matters. These can also cover what will happen to facilities and equipment on abandonment and could deal with reclamation issues in the absence of legislation.
- Fiscal and financial provisions: royalty and process of payment, exemptions, stability (ensuring that taxation will remain stable over the life of a mining operation), foreign exchange arrangements, repatriation of proceeds, debt–equity ratios, management fee charges, and so forth.
- Legal provisions, including force majeure, dispute settlement (arbitration) process, and expropriation of an operation. Many countries have arbitration acts, which are used to settle disputes. However, in developing countries, this is often not the case, and so provisions in the agreement usually appoint an international tribunal to hear the arbitration. Agencies most commonly used for arbitrations are the United Nations Commission on International Trade Law, the International Centre for Settlement of Investment Disputes, or the International Chamber of Commerce.

These agreements may or may not include state participation. In most instances, a separate agreement for such participation is necessary because the state's interest will be held by a state-owned company or holding company, which is legally separate from the government.

HEALTH AND SAFETY

In most developing countries, the same strict culture of safety or good health in the workplace found in developed countries does not normally exist. Therefore, it is incumbent on foreign companies working in these countries to teach and instill that culture in the workplace.

Most mining laws address the inspection powers required to enforce proper safety and health practices at mine sites but do not necessarily provide the detailed provisions a company

must follow. In fact, these provisions can usually be found in regulations made under the mining law, in separate mine safety laws, or in generally applicable occupational health and safety (OHS) laws. In addition, environmental, water, and/or health laws have provisions with respect to clean air and water, and requirements for the use of chemicals and toxic substances. Regulations addressing the conditions of premises as well as provisions for safety clothing, gear and equipment, safe work practices, hazard assessment, food storage and preparation, health hazards, disease, radiation, and noise can normally be found in a variety of these laws. There will also be provisions for accident investigations and reporting, health outbreaks, and other issues that may arise.

Many OHS laws allow for a mine safety plan to be submitted and approved by the authorized department or agency that oversees mining activities. The licensee would have to ensure that its plan sets standards that meet or are better than those in the various laws addressing OHS. Senior managers are obligated to implement and maintain safety programs in the workplace and could be legally liable if any accidents do occur. They must provide a continuously compliant safe and healthy environment.

ENVIRONMENT

Because mining affects air, land, and water, numerous laws and regulations address these three areas that exist outside mining law. Many countries have clean air and water laws that apply to exploration and mining operations. Environmental standards for toxic substances and chemicals can also exist in health laws, as noted previously. It is important that all these laws and regulations be made a part of any exploration or mining work plan.

Environment laws per se usually address the need for pre-assessment of the areas where operations will take place (baseline studies and impact assessment), resulting in an environmental plan that addresses the impacts on land, water, and air. This plan may be subject to a public review process, which involves local populations affected by the operations. These can be controversial (e.g., tailings and reclamation issues that are of paramount importance with respect to safety and health and the need to minimize an operation's physical impacts). The environmental plan will usually be approved by another ministry, separate from the mines department, so it will require coordination to ensure that the environmental approvals do not cause a delay in the project. It is often wise to employ an agent knowledgeable in local practices, as it is notoriously difficult to deal with multiple layers of bureaucracy, even in the most developed countries.

Mining laws or agreements may address abandonment procedures, such as what can be removed and the time lines for plant and equipment removal. These may be required at the outset of a project or may be developed toward the end of mine life. If the law does not address this subject, it would be wise to ensure that the agreement has provisions that clarify this in detail so the costs of abandonment can be adequately forecast and there is certainty as to what will remain after mine life. For example, if a company has built and maintained a road to a mine site, it should be considered whether the government will take over the responsibility of this road after the mine is gone. Local populations may use such a road during the exploration period or mine life, and, if so, the government may agree to assist in maintenance costs.

In general, a reclamation plan is required at the beginning of an exploration program or a mining operation to ensure that reclamation proceeds on land that is no longer required. These plans are usually mandated by environmental laws but can also be required as part of the application for a mining right. How the reclamation is paid for over the life of the mine is also found in environmental and/or tax laws.

Penalties are still the usual way in which countries deal with breaches of environmental standards—that is, the polluter pays. However, tradable or marketable permits are available in some countries for certain chemical compounds, such as carbon and sulfur dioxide, and this practice will become more widespread as climate change provisions are implemented around the world.

A proper approach to environmental matters impacts a company's international reputation, so it is important to take this seriously.

FINANCIAL

Taxation of minerals is a complex issue. It must be recognized that the mining industry is extractive and nonrenewable by nature, and at the same time a mining tax must take account of the huge risk that is connected to extraction and is not found in other industries. The risks—geology, market price, large capital investment connected with production, and the fact that it is a single-purpose industry—mean that it is a unique investment. In addition, extractive industries are subject to the effects of supply and demand; therefore, the project economics are always at risk because of world market prices.

Another unique feature is that it is necessary to find an economically viable deposit. The exploration part of the mining scenario is extremely high risk; it is estimated that only 1 in 10,000 exploration projects result in the discovery of an economically viable, exploitable resource. Companies may spend millions of dollars before finding a mineable deposit. Development requires large capital outlays before production can commence. Because it is so different from any other industrial enterprise, tax laws generally have special provisions to address these unique factors.

In order to attract and retain the industry, governments know they must be competitive with other mining jurisdictions. In addition, a stable fiscal regime is an attractive part of the investment portfolio. Many countries have enacted a number of special tax treatments, which are usually found in the tax laws.

Accelerated Depreciation

Most income tax laws allow for the depreciation of assets for all industries. In mining, accelerated rates are allowed because of the huge capital investment that is incurred before production. This accelerated depreciation usually applies to all tangible assets that have a useful life of more than 1 year and include structures, mine equipment, and mine processing plants as well as roads, schools, housing, power lines, and so forth, built to service the mine. The rate of this accelerated depreciation varies from country to country.

Amortization

This has wide variances in tax laws; some countries allow reconnaissance and exploration expenses to be amortized if development results from those activities. Others allow amortization on a declining basis on all tangible assets under

generally acceptable accounting principles as well as all survey, exploration, feasibility, and development expenses. Other approaches allow research and development costs either to be expensed in the year they are paid or to be amortized over a period of years.

Loss Carryforward

Most income tax laws allow extractive industries to carry their losses forward against income in subsequent years. These allowances vary from country to country.

Dividend Withholding Tax

This is applicable to any shareholder who receives dividends as a result of his or her investment in a project. The tax can vary according to whether the recipient is a foreigner or a resident of the country and from 5% to 10%. It is generally seen as a deterrent to mining investment.

Import/Export Duties

Because mining projects are capital intensive and most imports are for heavy equipment and machinery, it is usual to allow relief from duties on items required specifically for mining during the early stages of construction, development, and production while profitability is low or even nonexistent. After the mine is constructed and operating, these special provisions may no longer be applied, except for specific mine plant replacement or mine expansion activities. Mining operations seldom have customs duties relief on anything that is readily available in the country and which is of the same quality and standard.

Export duties should not be incurred on ore or product, except in the case of gemstones, in which an export price is often the only way governments receive revenue.

Reclamation

Because it is now standard in most countries to require mining companies to reclaim disturbed lands, it is necessary for governments to define how the costs of reclamation will be dealt with in their tax regimes. Many jurisdictions now require trusts to be set up, into which the companies contribute on a regular basis. As these trusts build up, they are used to pay reclamation costs both during the life of the mine and at its closure.

Although different countries use different mechanisms, there is often an allowance for contributions for reclamation. In trust situations, it is generally acceptable that the company can deduct contributions in the year they are made. Most often, funds received from the trust are considered income for tax purposes in the year they are taken. Some jurisdictions also tax the interest earned at the usual corporate tax rate, which results in double taxation in many instances and is considered unfair by many mining companies.

Other mechanisms, such as performance bonds, are also used to address reclamation. The cost of these bonds may or may not be deductible.

Royalty

The royalty on the mined resource is applied in different ways in different countries. It can be based on production, net smelter return, or sales. The royalty on production may give certainty to the government, with respect to royalty return on a project, especially when there is a downturn in prices. However, such a royalty is not simple to determine, because it is difficult for the government to monitor the actual production of the mined product and to differentiate between what is product and what is waste.

For these reasons, many governments usually base their royalty on an ad valorem basis—that is, sales price using arm's-length contracts. Bills of sale and invoices are easily audited by the government, and, therefore, a royalty based on this approach is easier to collect. A royalty based upon sales price means the government revenue will fluctuate according to the market. For companies, the sales price basis for royalty allows them to address price fluctuations and maintain economic viability.

Royalty payments may also be allowed as deductions for tax calculations.

Allowable Expenses

The corporate income tax law should also define the exemptions allowed against the payable income tax:

- **Capital cost:** The buildings, machinery, and plant required for extraction and processing; assets providing services to the mine or community (where a substantial proportion of employees of the mine reside at the mine); loading facilities; equipment; mining vehicles; and so forth.
- **Exploration expenses:** The full amount of all exploration and predevelopment costs.
- **Development expenses:** Acquisition, surface development, pre- and postproduction underground development.
- **Operating costs:** The costs incurred during normal operations, excluding depreciation, noncash costs, and financing charges.

Auditing Capacity

Governments are always concerned that they receive fair income from the mining resource. It is therefore necessary to address transfer pricing issues, which are extremely complicated. Many countries have special auditors who verify all expenses claimed by companies to ensure that they are reasonable and necessary. Usually, income tax legislation defines arm's-length transactions and has provisions for the power to audit and disallow expenditures that are not considered valid.

CONCLUSION

The mining law of a country is merely a small part of what must be addressed in order to obtain a license to explore or mine. Other laws impact heavily on the operations and must be followed if the activities are to be approved. Even administrative practices, which implement the provisions of a law, can affect whether approvals will be timely. It is wise to employ a local agent and legal counsel to assist in dealing with the processes and procedures that must be followed to commence mining activities. Contact with a local chamber of mines can also be extremely helpful.

When entering a new country, a company will be judged on the reputation it builds, not just on its reputation elsewhere. Following the rules, providing information throughout the process, working with local populations, and running safe and healthy operations are paramount to establishing a viable mining operation and maintaining a positive reputation.

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PART 6

Mining Method Selection

CHAPTER 6.1

Evaluation of Mining Methods and Systems

Michael G. Nelson

INTRODUCTION

The relative merits of surface and underground mining are widely discussed and frequently debated. Some deposits can be mined entirely with surface methods, while others can only be worked underground. With all other conditions equal, surface mining is normally regarded as preferable, because of lower development costs, quicker start-up time, and lower accident rates generally associated with surface mining.

When choosing between surface and underground methods, some of the factors that must be considered include

- Size, shape, and depth of the deposit;
- Geologic structure and geomechanical conditions;
- Productivities and machinery capacities;
- Availability of experienced work force;
- Capital requirements and operating costs;
- Ore recoveries and revenues;
- Safety and injuries;
- Environmental impacts, during and after mining;
- Reclamation and restoration requirements and costs; and
- Societal and cultural expectations.

Some deposits may reasonably be mined entirely by surface methods. In general, such deposits are close to the surface and have a relatively uniform geology. Similarly, some deposits can only be mined economically by underground methods. These deposits are usually deeper, with geological and mineralogical characteristics that require more selective ore extraction. Finally, other deposits are best mined initially as open pits, with production shifting to an underground method as deeper portions of the ore body are extracted. An example of each type of deposit follows.

In suitable deposits, surface mining is more productive, more economic, and safer for workers. However, changes in environmental regulations and societal expectations may lead to fewer large open-pit mines, particularly if operators are required to backfill open pits and recontour waste dumps. These conditions may result in the development of small, high-grade deposits by very shallow open pits or in the development of high-grade underground mines in place of large open-pit mines. Where applicable, large, low-grade deposits may be mined by in-situ methods (Hitzman 2005). In some cases, especially in built-up areas, it has become almost impossible to obtain permits for new surface mines. This is the case for producers of crushed stone and dimension stone in large metropolitan areas in many developed countries. For this reason, several underground quarries have recently begun operating in the United States, and many more are in the planning stages.

Table 6.1-1 Major equipment in use at the North Antelope Rochelle coal mine

Description	No. in Use	Description	No. in Use	Description	No. in Use
Heavy haulers	2	Scrapers	7	Graders	13
Draglines	3	Water trucks	9	Mining shovels	14
Loaders	3	Drills	9	Dozers	27
Backhoes	6	Wheel dozers	13	Haul trucks	73

SURFACE AND UNDERGROUND MINE EXAMPLES

In some cases, the choice of surface or underground is obvious. One such example is the North Antelope Rochelle mine in Wyoming, United States, owned by Peabody Energy.

The North Antelope Rochelle mine shipped 88.7 Mt of compliance coal in 2008 and has produced more than 1,000 Mt since the mine began in 1983. It is the largest coal mining operation in the United States. Remaining coal reserves dedicated to the mine cover nearly 8,800 hectares with about 1,200 Mt of recoverable coal. The coal seam ranges from 18 to 25 m thick and lies from 15 to 105 m below the surface.

The complex employs a large fleet of big equipment, listed in Table 6.1-1. The three draglines have bucket capacities of 84, 76.5, and 65 m^3, respectively. The key to success is high volume and low unit costs.

Similarly, in the case of the Henderson mine, underground mining by panel caving was the most logical choice. The Henderson mine, located in Colorado, United States, is owned by Freeport-McMoran Copper and Gold, Inc. A cross section of the Henderson ore body is shown in Figure 6.1-1. Although

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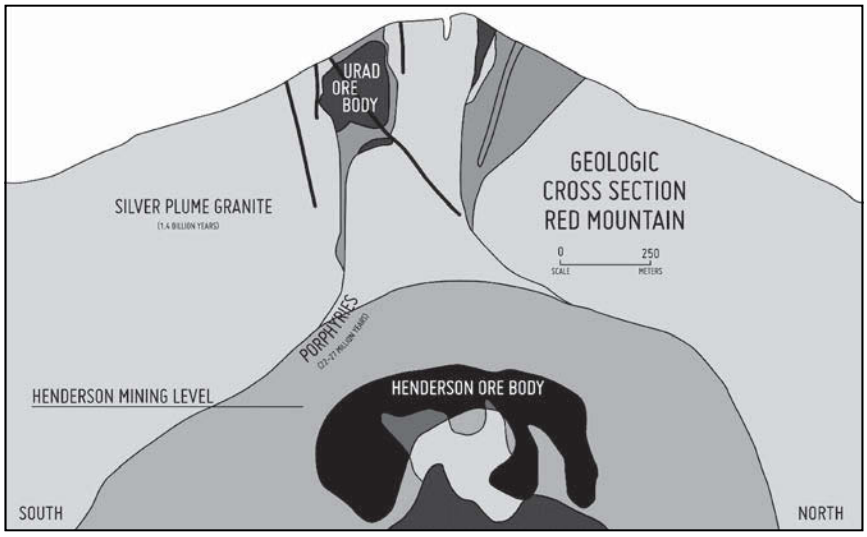
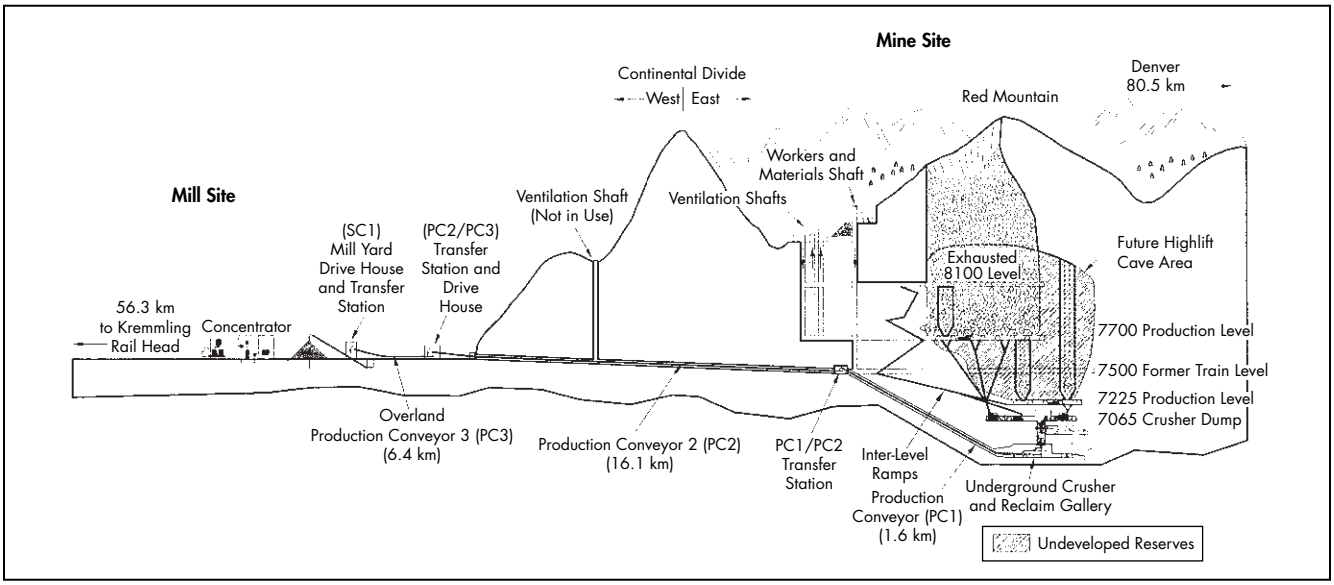


Figure 6.1-1 Cross section of the Henderson ore body



Source: Rech 2001.

Figure 6.1-2 Cross section of the Henderson mine

the ore body is relatively large, it is also quite deep—about 1,040 m below the top of Red Mountain. Development of an open-pit mine would have required removal of a large amount of overburden before the ore body was exposed. This would have required construction of roads, power lines, and other infrastructure to the top of Red Mountain, at 3,751 m above sea level. Thus development would have been extremely expensive, with no initial production to support development costs. The Henderson mine was developed as shown in Figure 6.1-2.

Finally, the Northparkes mine in Queensland, Australia, provides an example of a mine that began as an open pit and is now an underground operation. As shown in Figure 6.1-3, the Northparkes ore body is a narrow porphyry, 200–300 m across and about 900 m in height.

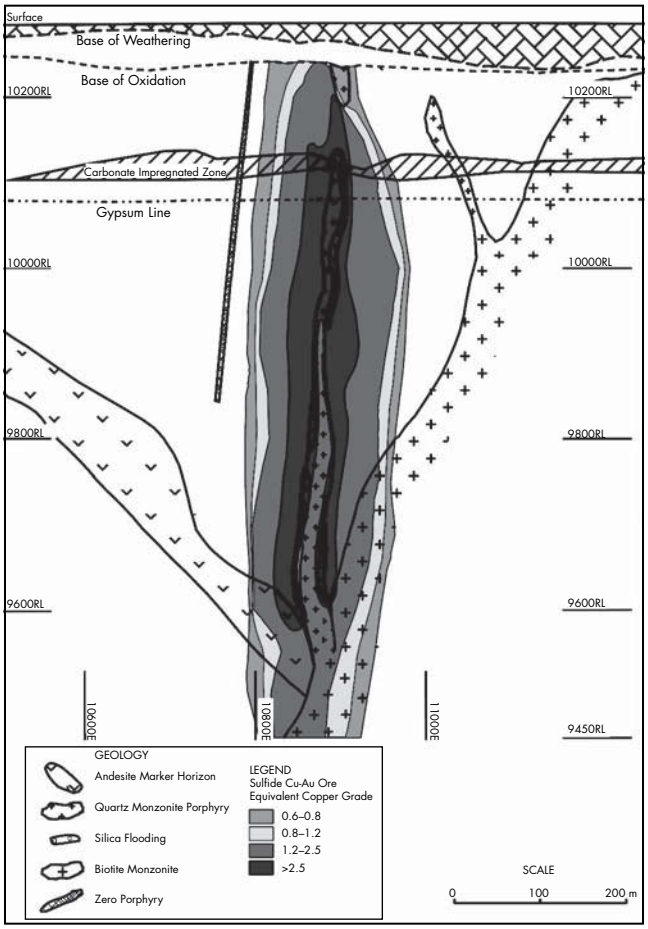
Beginning in the late 1970s, Northparkes was mined with two open pits, each about 150 m deep. Because the ore body has such a small cross section, the stripping ratio increased rapidly, and in 1993 development was begun for underground mining by block caving, as shown in Figure 6.1-4.

With these examples in mind, it is worthwhile to consider some specific differences between surface and underground mining.

PRODUCTION

Much more material is produced by surface than by underground mining. This is shown in Table 6.1-2, which gives recent data for the United States. It is apparent from Table 6.1-3 that sand, gravel, and stone products represent more than 90% of the material produced in surface mines each year.

The fraction of mined material produced by underground methods in the United States has decreased in recent years, as shown in Table 6.1-3. This results from the decrease in the fraction of coal produced underground; the fractions for metal and nonmetal minerals vary over the period.



Source: House et al. 2001.

Figure 6.1-3 Northparkes ore body

MINE SIZE

In terms of daily production tonnage, surface mines are almost always larger than underground mines producing the same commodity. This is partially true because open-pit mines must mine much more waste rock (and therefore have much more dilution of the in-situ mineral), whereas many of the underground methods can mine the same mineral much more selectively, with less dilution and therefore fewer metric tons.

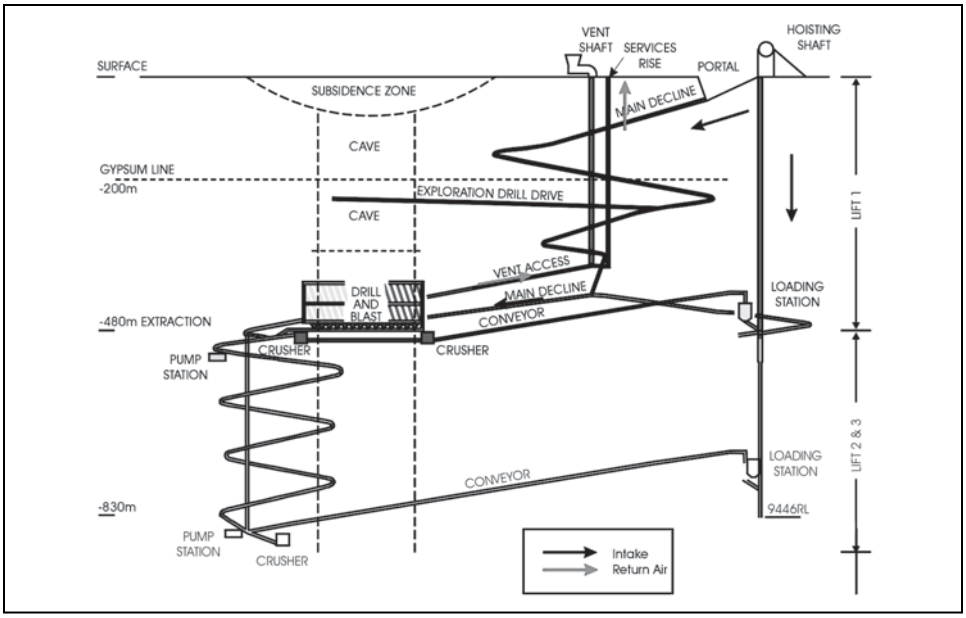
Table 6.1-4 shows approximate daily production rates for selected large surface mines; Table 6.1-5 shows similar data for large underground mines. These tables show the predominance of surface methods for large, high-tonnage operations worldwide. (Tables 6.1-4 and 6.1-5 are not intended to be complete but are included to provide an indication of the respective numbers and sizes of larger surface and underground operations.)

PRODUCTIVITY

When productivity is measured in metric tons mined per worker-hour, surface mines are almost always more productive. Table 6.1-6 shows data for coal mining in the United States. During 2006 and 2007, productivity in surface mines was more than three times that in underground mines. However, when choosing a mining method, it is important to go beyond a simple consideration of metric tons per worker-hour. For example, in a gold deposit, it may be more meaningful to examine grams or ounces of gold produced per worker-hour. In many gold mining districts, comparing the productivity of the surface mines and underground mines in this way shows much more comparable results.

SAFETY

The mining industry throughout the world continues to reduce the incidence of accidents and fatalities. The underground mining environment is recognized as being more hazardous than the surface. Table 6.1-7 shows the incidence rates per 200,000



Source: House et al. 2001.

Figure 6.1-4 Northparkes mine layout

Table 6.1-2 Ore and coal produced in the United States (in megatons)*

Material	2003	2004	2005	2006	2007
Metals					
Surface†	1,090	1,190	1,270	1,310	1,330
Underground‡	14	14	20	16	19
All mines	1,100	1,200	1,290	1,330	1,350
Nonmetals					
Surface†	2,850	3,000	3,090	3,130	3,040
Underground‡	107	139	137	129	129
All mines	2,960	3,140	3,230	3,260	3,170
Coal					
Surface†	719	745	763	804	795
Underground‡	353	368	369	359	352
All mines	1,072	1,112	1,131	1,163	1,147
Total					
Surface†	4,659	4,935	5,123	5,244	5,165
Underground‡	474	521	526	504	500
All mines	5,132	5,452	5,651	5,753	5,667

Source: U.S. Geological Survey 2009; U.S. Energy Information Administration 2009b.
*Data are rounded and may not add to totals shown.
†Includes materials from wells, ponds, and pumping operations.
‡Includes solution mining.

Table 6.1-3 U.S. underground ore and coal production (% of total mine production)

	Underground Production, % of total			
Year	Metals	Nonmetals	Coal	Total
2003	1.27	3.61	32.92	9.23
2004	1.17	4.43	33.05	9.55
2005	1.55	4.24	32.58	9.30
2006	1.20	3.96	30.88	8.76
2007	1.41	4.07	30.68	8.82

Courtesy of the Mine Safety and Health Administration.

hours for all accidents in the United States during the years 2003–2007; Table 6.1-8 shows the incidence rates for fatal injuries. These rates are higher in all cases for underground mining, and notably higher for underground coal mining.

DEVELOPMENT

Development for surface mining of coal and other bedded minerals involves the removing of cover layers of soil and rock to expose the coal. Surface mining is used when the coal seam is relatively close to the surface, usually within 60 m. The time between overburden removal and the mining of the product mineral should be as short as possible to optimize overall cash flow. However, for larger deposits covered by large amounts of overburden and waste, the amount of pre-stripping will also be large, leading to high preproduction development costs. The time required for pre-stripping can range from 2 to 6 years. Thus, interest costs during development will be high and will represent a significant portion of the pre-mining capital requirement before mining can start. When an ore body is steeply dipping and at or near the surface, open-pit mining can start with a small amount of stripping. However, as mining of such a deposit progresses, increasing amounts of waste rock must be removed. This must often be done many years before mining of the corresponding amount of ore at deeper levels can take place. Thus, the ultimate pit limits must be projected early in the mine planning process, and the investment cost for waste rock removal in advance of mining must be included in the economic evaluation. Waste rock stripping should be delayed as long as possible to avoid high interest cost for all the money spent in waste stripping activities. The increasing cost of stripping at greater depths is one of the major factors in deciding when to transition from surface to underground mining of a given deposit.

Table 6.1-4 Daily production tonnages for selected large surface mines

Mine	t/d	Product	Country
CBG Bauxite	32,000	Al	Guinea
Bata Hijau	182,000	Au	Indonesia
Zarafshan Newmont	38,000	Au	Uzbekistan
Goldstrike	32,000	Au	United States
Cripple Creek and Victor	30,000	Au	United States
Rhineland Lignite*	274,000	Coal	Germany
North Antelope Rochelle	251,000	Coal	United States
Black Thunder	250,000	Coal	United States
Cordero Rojo	180,000	Coal	United States
Kaltim Prime	100,000	Coal	Indonesia
Chuquicamata	375,000	Cu	Chile
Escondida	240,000	Cu	Chile
Grasberg	240,000	Cu	Indonesia
Collahuasi	170,000	Cu	Chile
Bingham	150,000	Cu	United States
El Abra	120,000	Cu	Chile
Hamersley Yandacoogina	143,000	Fe	Australia
Carajas	100,000	Fe	Brazil
Alegria	65,000	Fe	Brazil
Samarco	65,000	Fe	Brazil
Mount Wright	62,000	Fe	Canada
Iron Ore Company of Canada	60,000	Fe	Canada
Mt. Keith	32,000	Ni	Australia
Syncrude Oil Sands†	500,000	Oil	Canada

Source: Data from InfoMine USA 2009 and Mining-technology.com 2009.
*Operation includes three pits.
†Operation includes five pits.

Table 6.1-5 Daily production tonnages for selected large underground mines

Mine	t/d	Product	Country
El Teniente	100,000	Cu	Chile
Grasberg Underground	50,000	Cu	Indonesia
Olympic Dam	25,000	Cu, U	Australia
Palabora	20,000	Cu	South Africa
Kiruna	40,000	Fe	Sweden
Henderson	32,000	Mo	United States
Norilsk	30,000	Ni	Russia

In an underground mine, a significant amount of infrastructure must be installed before mining begins. This will include shafts, hoists, ventilation fans, underground shops, travel ways for workers and machinery, ore storage bins, underground crushers, and so forth. This requires detailed long-range planning from the very beginning so that the requirements of future workings at deeper levels can be accommodated. A large capital investment is often necessary before production can start.

Underground mining methods require a more careful design and planning process, because it is difficult to make changes in a design after the infrastructure has been installed and the equipment purchased. This condition is often exacerbated when variables such as ore grade, mine water make, and ground control conditions change or are different than expected. It is very important that the underground mine design and the machinery capacities are properly chosen from the beginning. For all of these reasons, it is often prudent to develop a small test mine to accurately determine many of the unknown mine characteristics. A test mine and a properly conducted feasibility study will minimize these risks.

The development of a large underground mine can take as many as 5 to 10 years. Interest costs during development will therefore be high and may comprise 30% to 40% of the premining capital requirement before mining can start.

Table 6.1-6 Coal mine productivity in the United States

Productivity Data	2006	2007
Underground		
Number of mines	666	610
Production, kt	430,374	423,296
Number of employees	47,475	46,723
Productivity, t/worker-hour	3.07	3.04
Surface		
Number of mines	875	855
Production, kt	1,199,194	1,196,915
Number of employees	35,398	34,450
Productivity, t/worker-hour	9.27	9.33
Total		
Number of mines	1,541	1,465
Production, kt	1,629,568	1,620,210
Number of employees	82,873	81,173
Productivity, t/worker-hour	6.04	6.05

Source: U.S. Energy Information Administration 2009a.

COST COMPARISONS

Estimates of capital and operating costs for surface and underground mines of various sizes and configurations are compiled regularly and in considerable detail by InfoMine, Inc. Those estimates are provided to customers as the *Mining Cost Service* and can be purchased in printed or electronic form, or accessed on-line. The cost estimates do not include permitting, environmental analysis, reclamation, or closure costs. Figures 6.1-5 and 6.1-6 summarize the cost estimates for surface mines.

The *Mining Cost Service* also provides estimates of capital and operating costs for underground mining. The data are more extensive, with estimates for eight mining methods, and shaft and adit access for each. Figures 6.1-7 and 6.1-8 summarize selected data.

While surface mining methods are relatively simple and uniformly applied, there are many underground mining methods, and application of any given method will vary from mine to mine. Thus it is much more difficult to accurately summarize costs for underground mining methods. Nonetheless, Figures 6.1-5 through 6.1-8 show the following trends:

- For small mines, capital and operating costs per metric ton of ore produced are lower for surface methods. Of course, dilution and ore grade must also be considered in a full economic analysis. For large tonnage production, capital and operating costs may be higher for surface

Table 6.1-7 Mining accident incidence rates per 200,000 hours in the United States

	Metal/Nonmetal		Coal		Total	
Year	Underground	Surface	Underground	Surface	Underground	Surface
2003	5.41	3.49	8.62	2.88	7.98	3.31
2004	5.30	3.50	8.16	2.54	7.58	3.22
2005	5.54	3.44	7.43	2.48	7.05	3.15
2006	4.64	3.18	7.13	2.38	6.62	2.94
2007	4.84	2.94	6.78	2.20	6.34	2.72

Courtesy of the Mine Safety and Health Administration.

Table 6.1-8 Mining fatal injury incidence rates per 200,000 hours in the United States

	Metal/Nonmetal		Coal		Total	
Year	Underground	Surface	Underground	Surface	Underground	Surface
2003	0.02	0.02	0.04	0.02	0.03	0.01
2004	0.03	0.02	0.04	0.02	0.03	0.01
2005	0.01	0.02	0.06	—	0.03	0.01
2006	0.01	0.02	0.08	0.01	0.06	0.01
2007	0.05	0.02	0.05	0.02	0.04	0.01

Courtesy of the Mine Safety and Health Administration.

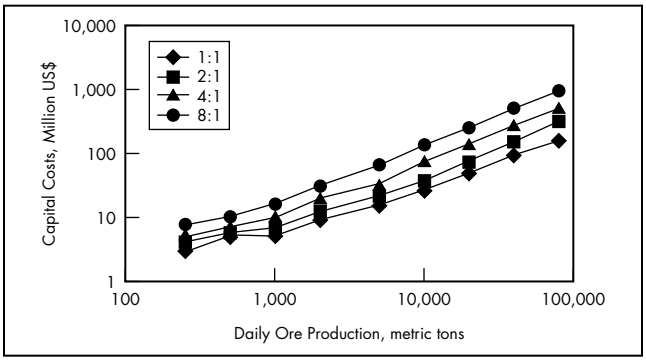


Figure 6.1-5 Estimated capital costs for surface mines at four stripping ratios

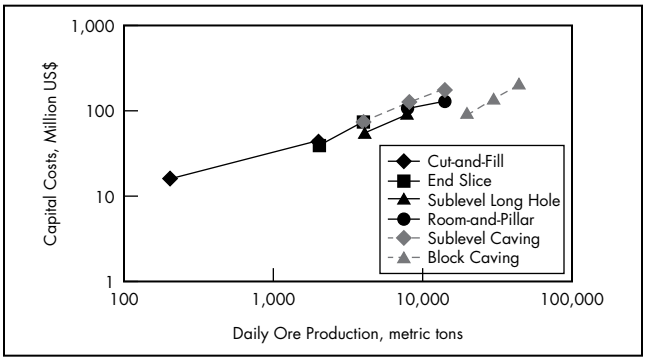


Figure 6.1-7 Estimated capital costs for six types of underground mines, all with shaft access

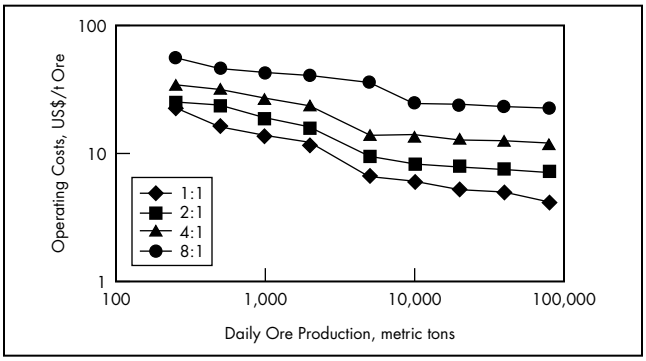


Figure 6.1-6 Estimated operating costs for surface mines at four stripping ratios

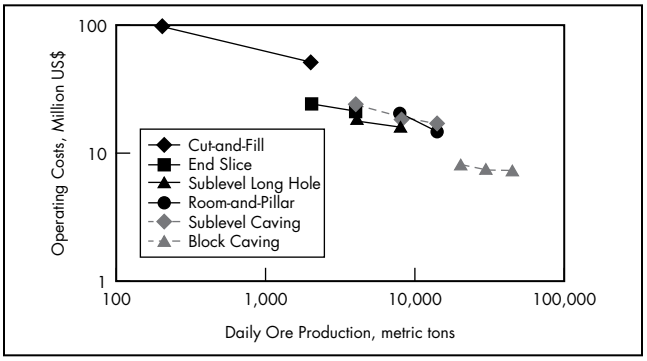


Figure 6.1-8 Estimated operating costs for six types of underground mines, all with shaft access

mines, depending on stripping ratio. In these cases, a dual feasibility study must be performed comparing the open-pit option to the best underground mining option.

- In all cases, capital costs increase and operating costs decrease with increasing production tonnage.

ENVIRONMENTAL AND CLOSURE REQUIREMENTS

Surface mines create a much larger footprint than underground mines. In the United States, surface coal mines are required to backfill mine excavations, and recontour and revegetate waste piles. This is not the case for metal, but there are strong indications that the situation is changing. Indeed, in most countries, society no longer looks favorably on large, abandoned excavations, and the mandated costs of reclamation and closure for large surface mines are likely to increase. A permit for construction of a new surface mine or expansion of an existing surface mine cannot be obtained in some areas. In these cases, underground mining should be examined.

Costs of environmental compliance, reclamation, and closure are seldom published. However in 1999, Mudder and Harvey reported that closure costs for U.S. surface mines ranged from US$1,236 to US$3,707 per disturbed hectare, with coal mine costs on the higher end of the range at US$2,471 to US$3,707 per hectare. Costs for metal mine sites were lower, yet they were much higher in cases where extensive water management and acid rock encapsulation were required. In 1996, Homestake Mining Company reported average company-wide reclamation costs of US$3,361 per hectare. Between 1980 and 1992, 136 abandoned coal mine sites in Pennsylvania were reclaimed, at a cost of about US$2,348 per hectare (Bogovich 1992).

In 2004, Wilson and Dyhr estimated environmental and closure costs as a percentage of total operating costs for medium-sized mines with on-site processing and tailings disposal. Those estimates are summarized in Figure 6.1-9, where the higher costs associated with surface mining are clearly shown.

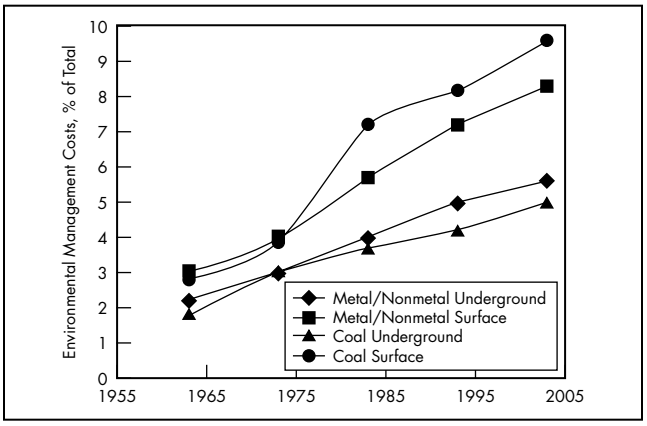
SELECTION OF A MINING METHOD

Based on this brief introduction, it may appear that surface mining is preferable to underground methods, particularly in regard to productivity and worker safety. However, as has been pointed out, selection of the best mining method for any deposit requires analysis of many factors besides the simple productivity in metric tons of ore per worker-hour.

The following subsections discuss in detail the procedures for selecting a mining method and include factors that influence the choice between surface and underground mining.

Location of the Deposit

In some cases, a mineral deposit may be located in a place where a large surface mine is simply unacceptable. The case of stone and other construction materials, already mentioned briefly, is an excellent example. These materials have a relatively low value and are used in large quantities, so it is important that they be mined as close as possible to the locations where they will be used. Those places are almost always heavily built-up areas. For example, the amount of concrete used in



Source: Data from Wilson and Dyhr 2004.

Figure 6.1-9 Environmental costs as percentage of total operating costs

Manhattan every 18 months—3.33 million m^3—is about the same amount as was used in the construction of the Hoover Dam on the border between Arizona and Nevada in the United States (Owen 2003). In such areas it is difficult, if not impossible, to expand an existing stone quarry, let alone open a new one, and the production of stone increasingly comes from underground quarries.

Of course, underground mining methods can also have adverse effects when operated under built-up areas. Surface subsidence and mine water release must both be monitored and controlled. Other factors also enter into choosing the location for a mineral deposit, including processing requirements, political and social conditions, and work-force availability, which are discussed in the following sections. Another factor, environmental and permitting requirements, is not discussed in this chapter.

Geology of the Deposit

Three aspects of a deposit's geology relate to the choice of surface or underground mining: the intrinsic value or grade, the morphology, and the structure.

A material with a higher intrinsic value will support a more expensive mining method. For example, Jim Walter Resources mines high-quality metallurgical coal at its Blue Creek mine in Alabama (United States) under very difficult conditions that include spontaneous combustion, deep cover (450 to 730 m), and high methane levels (Howell et al. 1991). A lower-quality coal would not support the high costs of mining in this geological setting. Similarly, an unusual narrow-vein gold deposit, where the gold occurs in very high-grade but sporadic pockets, supports a labor-intensive underground mining method (Original Sixteen to One Mine 2009). In general, deposits with lower intrinsic value or grade are more amenable to surface mining methods when other conditions permit.

Deposit morphology, including shape, extent, and depth, is also important. The economics of most surface mining methods (and some underground methods) are based on high production volume and low unit costs, and use of equipment that has high capital costs. These require large deposits with relatively uniform grade and few irregularities in shape or extent. Deposits that meet these criteria can often be mined profitably, even when the ore grade or product value is relatively low. Good examples of such surface mines are large coal mines in northeastern Wyoming, as described previously; the large porphyry copper mines, such as Bingham Canyon in Utah (United States) and Chuquicamata in Chile; and the large, low-grade gold mines such as Round Mountain and Goldstrike in Nevada. Similarly, coal deposits that can be mined by the underground longwall method must have large areas of coal with relatively uniform thickness to allow the development and production of large panels that will support the costs of development and purchase of equipment. The depth of a deposit also influences the surface versus underground decision. The depth of the Blue Creek mine requires the use of barrier pillars between longwall panels, at a cost that could probably not be supported by a lower-value coal. In other cases, metal deposits are often mined initially by the open-pit method but switch to an underground method when the costs of removing overburden become too high. This has happened, for example, at Kiruna in Sweden, Northparkes in Australia, and Palabora in South Africa.

Finally, the geologic structure of a deposit must be considered. It is more difficult to generalize about this factor, but a good example is the Homestake deposit in South Dakota (United States). George Hearst, who consolidated the claims and put them into production, is reported to have said to his partners, "Here's to low-grade ore and plenty of it" (Smith 2003). During its 125 years of operation, the Homestake mine produced almost 1.2 million kg of gold and 0.3 million kg of silver. Of course, open-pit mining was unknown when operations began at Homestake, but lacking other information, one might conclude that this large, low-grade deposit was an ideal candidate for that method. However, the deposit was highly folded and faulted, and required selective mining to extract the ore in a manner that could only have been done by underground methods.

Processing Requirements

The processing required to produce an economic product also influences the choice of mining method. It may be possible to mine a low-grade ore at very low cost using a surface method, but the resulting dilution may make processing so expensive that the overall operation is not profitable. In such a case, more selective mining using an underground method may be used to produce a higher-grade ore, which is less expensive to concentrate. Such selective mining can also be used to leave in place portions of the deposit containing impurities or contaminants that can increase reclamation or remediation costs if they enter the process stream.

It is also important to consider the locations of available processing facilities and the ease with which new facilities can be permitted and built. The difficulty of obtaining permits for new operations in built-up areas for aggregate pits and stone quarries was described previously. The same challenge has been encountered in permitting new coal preparation plants in the eastern United States, and in some cases this has been the main factor in deciding how to mine new coal resources.

Political and Social Conditions

Political and social conditions can determine not only whether or not a mineral deposit can be mined, but also the method by which it is to be mined. There may be significant opposition to the large, highly visible disturbance that occurs in surface mining, making permitting too expensive or impossible. In other cases, the legal rights to the minerals in an area may be

separated from the rights to control the surface in the same area, so that accessing and removing the minerals by an underground method is preferable.

Work Force

In general, underground mining methods require a more specialized work force than surface methods. Workers with experience in operating heavy equipment in agriculture or construction can often transfer their skills for use in surface mining operations, but underground mining equipment and processes are significantly different. For a deposit in which there is no clear choice between surface and underground mining, based on other constraints, the presence or absence of a suitably skilled work force can be a deciding factor.

CONCLUSION

Selection of the best mining method for a given deposit, including the choice between surface and underground mining, is a complex process involving the analysis of many interrelated variables. These variables are not just technical; they include consideration of environmental, social, and political conditions and constraints, and of the time and expense required to obtain the required government permits.

The process is usually iterative in nature, looking at many possible approaches and determining how all the variables interact in each. Mining companies and consultants now use detailed and sophisticated models that incorporate all the technical and financial data, and provide detailed output showing mine and mill production, direct and indirect costs, taxes and royalties, cash flows, internal rate of return, and net present value for each alternative considered. These models often incorporate probabilistic routines for sensitivity analysis so that decision makers can look at how the predicted outcomes for each alternative are affected by changes in the values of key variables such as ore grade, labor and material costs, and commodity prices.

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CHAPTER 6.2

Mining Methods Classification System

L. Adler and S.D. Thompson

INTRODUCTION

The purpose of a classification system for mining methods is to provide an initial guideline for the preliminary selection of a suitable method or methods. Its significance is great as this choice impinges on all future mine design decisions and, in turn, on safety, economy, and the environment.

The choice of a mining method assumes a previous but cursory knowledge of the methods themselves. It also assumes a brief understanding of ground control and of excavating and bulk handling equipment. In the formal mine design procedure, the choice of mining methods immediately follows geological and geotechnical studies, and feeds directly into the crucial milestone diagram where regions of the property are delineated as to prospective mining methods (Lineberry and Adler 1987). This step in turn just precedes the subjective, complex, and critical layout and sequencing study.

To develop the proposed classification system adopted here, many existing ones (both domestic United States and foreign) were examined and incorporated to varying degrees. The result is deemed more systematic, inclusive, and understandable than its predecessors (i.e., Stoces 1966).

Subsequent parts of this handbook elaborate on the selection and comparison of mining methods.

INPUT STATEMENT

A comprehensive statement has been developed to provide a rapid checklist of the many important input parameters (Adler and Thompson 1987). The three major areas are (1) natural conditions, (2) company capabilities, and (3) public policy (Table 6.2-1). Those parameters appearing early are generally the most important. Natural conditions require that a dual thrust be maintained concerning resource potentials and engineering capabilities. An additional basic distinction occurs between geography and geology. For company capabilities, fiscal, engineering, and management resources must be recognized. This includes the scale of investment, profitability, and personnel skills and experience. Public policy must be considered, particularly as to governmental regulations (especially safety, health, and environmental), tax laws, and contract status. Some of the latter input factors are held in abeyance until near the end of the investigation, and then considered as modifying factors. This organization duplicates but tightens others (Hartman 1987).

SPATIAL DESCRIPTION

Most mineral deposits have been geometrically characterized as to an idealized shape, inclination, size, and depth. Complex or composite bodies are then composed of these elements.

Ideal shapes are either tabular or massive, with chimneys (or pipes) being subordinated. Tabular deposits extend at least hundreds of meters (feet) along two dimensions, and substantially less along a minor dimension. Massive bodies are approximately unidimensional (cubic or spherical), being at least hundreds of meters (feet) in three dimensions. A modification is recommended later to achieve closure with tabular deposits. For tabular deposits, the inclination (attitude or dip) and thickness are crucial. Inclinations range from flat to steep (Table 6.2-2) (Hamrin 1980; Popov 1971).

Table 6.2-1 Input statement categories

Primary Categories (Dependency)	Secondary Categories
Natural conditions (invariant)	Geography
	Geology
	Economic engineering
Company capabilities (variant)	Business administration
	Monetary aspects
	Management aspects
Public policy (semivariant)	Regulations
	Taxes
	Contracts
	Incentives
State of the art (mining engineering)	Salient distinctions
	Total systems (design/control)
	Encumbered (and regulated) space
	Full-spectrum practice (manage/evaluate)
	Professionalism

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S.D. Thompson, Assistant Professor, University of Illinois at Urbana–Champaign, Illinois, USA

Table 6.2-2 Tabular deposits classified by attitude and related to bulk handling and rock strength

Class	Attitude or Dip	Bulk Handling Mode	Rock Strength
Flat	≤20°	Use mobile equipment (and conveyors)	Weak rock (surficial)
Inclined	20–45°	Use slashers (metal plate can also vibrate—as gravity slides)	Average rock
Steep	≥45°	Gravity flow of bulk solids	Strong rock (at depth)

Table 6.2-3 Surface pit slopes related to rock strength and time

	Maximum Pit Slope	
Rock	Short Term	Long Term
Strong	41°–45°(–70°)*	18°–20°
Average	30°–40°	15°–18°
Weak (soils also)	15°–30°	10°–15°

*Infrequently up to 70°.

Table 6.2-4 Underground deposits classified by thickness

	Deposit Thickness		
Class	Coal	Ore	Comments
Tabular			
Thin	0.9–1.2 m (3–4 ft)	0.9–1.8 m (3–6 ft)	Low profile or narrow mine equipment
Medium	1.2–2.4 m (4–8 ft)	1.8–4.6 m (6–15 ft)	Post and stulls ≤3.1 m (10 ft)
Thick	2.4–4.6 m (8–15 ft) pillar problems	4.6–15.3 m (15–50 ft) can cave (steep dip)	Small surface equipment; crib problems
Massive	≥4.6 m (15 ft)	≥5.3 m (50 ft)	Pillar problems or poor recovery; benching necessary; caving considered

Table 6.2-5 Deposits classified by depth

	Deposit Depth		
	Underground (a measure of overburden pressure)		
Class	Coal	Ore	Surface
Shallow	≤61 m (200 ft) slope entries possible	≤305 m (1,000 ft)	≤61 m (200 ft)
Moderate	122–244 m (400–800 ft) pillar problems	305–457 m (1,000–1,500 ft)	61–305 m (200–1,000 ft)
Deep	≥915 m (3,000 ft) bumps, burst, closure	≥1,830 m (6,000 ft)	≥305–915 m (1,000–3,000 ft) open pit

Table 6.2-6 Deposit classified by geometry and type

Geometric Class	Deposit Type	Comments
Tabular	Alluvium (placer)	Near surface—weak
Flat and inclined	Coal (folded too)	Weak country rock—an erosion surface
	Evaporites (domes too)	
	Sedimentary	Good country rock, thicker
	Metamorphic (folded too)	
Steep	Veins	Can be weakened or rehealed (gouge and alteration)
Massive	Igneous (magmatic)	Strong
	Disseminated ores	Can be weakened

In surface mining, the inclination limits the advantageous possibility of being able to cast waste material nearby, as opposed to hauling it a distance and then storing it. For flat deposits, especially when fairly shallow, an area can be successively opened up and the waste can then be cast into the previously mined-out strips, a substantial economic advantage. Casting, in its normal sense, is not restricted to the use of rotating excavators; broadly, it means relatively short-distance hauling of waste, which can also be done with mobile loaders and/or trucks or with mobile bridge conveyors. For steeper (and deeper) deposits, stable pit slopes become important (Table 6.2-3) (Hartman 1987; Popov 1971). Where the deposit inclination exceeds that of the stable slope, both the hanging wall and footwall must be excavated and the increased waste then handled and placed.

For both surface and underground mining methods, the inclination cutoff values nearly coincide (one for pit slopes, the other for face bulk handling mechanisms, whether mechanical or by gravity). While not identical, they are close enough to use similar values (20° and 45°; see Table 6.2-2).

The thickness of a tabular deposit is also important (Table 6.2-4), with reference primarily to underground work (Popov 1971). When three or more benches are required, the deposit tends to be treated as massive. Primarily in flat underground deposits, thickness governs the possible equipment height (low profile), and in steep ones its narrowness. Also, in underground mining, the deposit thickness becomes a support problem, especially if effective pillars become so massive that recovery is compromised. When the upper limit of any of these concerns is reached (e.g., benching, equipment size, and pillar bulk), closure with massive deposits occurs for all practical purposes. Pillar size vs. recovery can dictate caving except where pillar sizes may be decreased because backfilling is used, such as in postpillar cut-and-fill.

Finally, the depth below the ground surface is important (Table 6.2-5) (Popov 1971; Stefanko 1983). For surface deposits, even flat ones, this can obviate casting and require increased waste haulage and expanded dump sites. For underground mining, earth pressures usually increase with depth, consequently raising the support needs. The ground surface location above a deposit must be clearly identified to evaluate other parameters (see "Input Statement" section previously).

CORRELATING DEPOSIT TYPES

The inclination (dip) can be roughly related to the deposit type (Table 6.2-6). Rocks can also be related to strength (Table 6.2-7) (Hartman 1987). The strength of the deposit and its envelope of country rock can then be related to its type (Table 6.2-8). For determining pit slopes, (surface mining) and support requirements (underground mining), these relationships become important. Some variations are noted, especially for veins and disseminated deposits.

CLASSIFYING SURFACE MINING METHODS

Depth Related to Inclination

The surface mining classification, although based on the crucial ability to cast waste material rather than to haul it, has other features. These are primarily based on the depth of the deposit being a function of its inclination. Flat seams tend to be shallow, and casting is possible; steep and massive deposits trend to depth. From this, a number of relationships result.

Table 6.2-7 Rocks classified by strength

Class	Compressive Strength	Examples
Weak	≤41.3 MPa (6,000 psi)	Coal, weathered rock, alluvium
Moderate	41.3–137.9 MPa (6,000–20,000 psi)	Shale, sandstone, limestone, schist
		Evaporites, disseminated deposit
Strong	137.9–206.8 MPa (20,000–30,000 psi)	Metamorphic, igneous, veins, marble, slate
Very strong	≥206.8 MPa (30,000 psi)	Quartzite, basalt, diabase

Depth Related to Excavating Technique and Stripping Ratio

Because of the effects of weathering and stress release, excavating becomes more difficult and expensive with depth, following a continuum from hydraulic action and scooping through to blasting (Hartman 1987).

As a matter of definition, the stripping ratio (ratio of waste to mineral) usually increases with depth. However, the relatively inexpensive handling of waste near the surface by casting tends to mitigate this increase, permitting higher ratios. The use of mobile, cross-pit, high-angle conveying allows greater pit depths and, along with the mineral value, also influences this ratio.

Surface Mining Classification System

Based on the foregoing factors, a surface mining classification has been developed (Table 6.2-9). The classification incorporates information dependent on the intrinsic characteristics of the geometry of the deposit. Quarrying appears to be anomalous because of (1) relatively steeper pit slopes, (2) specialized means of excavating and handling, and (3) less critical amount of overburden. "Glory hole" mining or its equivalent is making a comeback in very deep open pits using inclined

Table 6.2-8 Deposits related to geometry, genesis, and strength (in order of induration)

Deposits Type	Geometry	Genesis	Strength and Stiffness, Deposit/Country Rock	Examples
Alluvium (placers)	Tabular-flat	Surface-stream action deposition (fans, deltas, meanders, braids)	Poor/poor	Sand and gravel; precious metals and stones (tin)
Erosion surface (swamps)	Tabular-flat and thin (possible folding)	Swamps (possible dynamic metamorphism)	Poor/poor to good	Coal
Disseminated	Massive	Underground channels and multifaceted advance	Poor/poor	Hydrothermal ores (porphyry coppers and sulfides)
Vein (can be rehealed)	Tabular-inclined (pipes, chimney shoots)	Major underground channels (fissures), gouge, alteration (reheal)	Poor to good/good	Hydrothermal ores (porphyry coppers and sulfides)
Evaporites	Tabular-flat-thick	Interior drainage	Good/good	Salt, phosphates
Sedimentary (bedded)	Tabular-flat-thick	Shallow seas	Good/good	Limestone, sandstone
Metamorphic	Tabular-flat-thick	Dynamic and/or thermal	Good/good	Marble, slate
Igneous (magnetic)	Massive	Plutonic emplacement	Good/good	Granite, basalt, diabase

Table 6.2-9 Classification of surface mining methods

Shape, Attitude (dip)	Deposit Characteristics	Stripping Ratio	Excavation		Mining Method
			Waste Handling	Excavation	
Tabular					
Flat	Near surface	Low	Onsite	Hydraulic, scoop, dig	Placers—hydrosluicing, dredging, solution—at depth
	Shallow	Moderate	Cast	Scoop, dig, light blast	Open cast (strip)—area, contour, mountain top
Inclined	Moderate	Moderate (remove hanging wall)	Need highwall	Auger	Auger
			Haul (to waste dump)	Blast	Open pit
	Deep	High (remove both hanging wall and footwalls)	Haul (to waste dump)	—	Open pit
				Saw, jet pierce (joints)	Quarry
Massive	Full range	Depends on depth	Haul (to waste dump)	—	Open pit; glory hole

Note: In-situ mining is always possible.

Table 6.2-10 Structural components located and described for underground mining

Component (time dependent)	Location/(Material)	Loaded by	Supported by	Comments
Roof (can deteriorate, slough, slake—dry and crumble)	Back and hanging wall (envelope)	Main roof—all, especially overburden (cap rock)	Pillars and fill, also arched (1/5)	Spans ~3 m (10 ft) for coal to 30.5 m (100 ft) for rock
		Immediate roof—body	Artificial supports can remove	Spans ~3.1 m (10 ft) (stand-up time)
Pillars and walls (can deteriorate—slough, slake)	Sides, deposit and waste (horses mainly deposit)	All—especially overburden	Floor	Critical: 1. Stiffness: (slenderness ratio: approximately 10/1 [coal] to 1/3 [rock]) 2. Strength (material) 3. Percentage recovery
Floor (can settle and heave)	Footwall (envelope)	All—through pillar watch water	Country rock can be compacted, removed, drained	Critical: 1. Stiffness 2. Strength (bearing capacity especially if water) 3. Heave (deep-seated)
Fill (for permanent stability)	Crushed waste, sand, water	All—especially as pillars are removed	Footwall and floor	Good mainly to support hanging wall. Requires greater than angle of slide and confinement.
Artificial support (limited time)	External: Timber (props, sets, cribs, stulls, posts); concrete gunite (mesh)	Mainly immediate roof	Floor	Deterioration (chemical and stress)
	Internal: Bolts (headers), trusses, cables, grout, cementation	Mainly immediate roof	Anchorage in roof, etc.	Anchorage a concern

hoisting. Glory hole mining utilizes a single large-diameter raise located in the lowest point of the pit, down which all blasted material is dumped. The bottom of the hole feeds into crushers and a conveying system, which transports the material to the surface through a horizontal or inclined drift (Darling 1989).

In contrast to the underground classification, the surface one is not formed into a matrix. This is because depth and therefore the excavating technique, waste handling, and stripping ratio are all functionally related to the deposit geometry, particularly the seam inclination. No preceding classification recognizes this relationship (Hartman 1987; Lewis and Clark 1964; Morrison and Russell 1973; Stout 1980; Thomas 1973).

CLASSIFYING UNDERGROUND MINING METHODS

Normally, two major independent parameters will be considered that form a matrix, unlike for surface methods. These two parameters are (1) the basic deposit geometry, as for surface methods, and (2) the support requirement necessary to mine stable stopes, or to produce caving, a ground control problem (Boshkov and Wright 1973; Hamrin 1980; Hartman 1987; Lewis and Clark 1964; Thomas 1973).

Deposit Geometry

Deposit geometry employs the same cutoff points for tabular deposits as in the surface classification, but for different reasons. Flat deposits require machine handling of the bulk solid at or near the face; steep ones can exploit gravity (Table 6.2-2), with an intermediate inclination recognized. If stopes are developed on-strike in steep seams as "large tunnel sections" or "step rooms" (Hamrin 1980), machine handling can still be used. The resulting stepped configuration causes either dilution or decreased recovery, or both. Because this face can also be benched, stope mining simply reproduces tunneling.

Ground Control

Ground control requires knowledge of the structure (opening), material (rock), and loads (pressures). Structural components are detailed in Table 6.2-10. Earlier tables detailed the deposit by its depth and detailed rocks by strength (Tables 6.2-5 and 6.2-7, respectively). From the point of view of support, the roof, pillars, and fill are of primary concern.

Main Roof

The main roof (sometimes the hanging wall) is distinguished from the immediate roof by being the critical load transferring element between the overburden and pillars. The immediate roof can be removed (mined out) or supported artificially and lightly. The main roof is defined as the first close-in, competent (strong) seam. If it is only marginally competent, heavy artificial support may keep it stable; if not, then caving can be expected. For a flat seam, the vertical (perpendicular) loads on the main roof are largely due to the overburden and its own body load. Horizontal (tangential) loads or pressures will tend to be uniformly distributed, resulting in a low stress concentration. If bed separation occurs above the main roof, this stress uniformity is enhanced; but at depth, overburden loading tends to decrease separation. Body loads are invariant, whereas edge loads—particularly those due to the overburden—can be shifted (pressure arching). The main roof is often sufficiently thick so that it can be arched below 1/5 (i.e., at less than 1 horizontally and 5 vertically) to increase stability. A guideline for coal is that stable spans are usually less than 3 m (10 ft), whereas for hard rock they are generally less than 30 m (98 ft).

For an inclined seam, the main roof is the hanging wall, and the results are similar to a flat seam. Pressures perpendicular to it are more significant then tangential ones, and bed separation due to gravity is less likely.

Table 6.2-11 Deposit and structural components related to underground mining methods

Deposit Geometry	Structural Main Roof and Floor	Components Rated (pillars, walls)*	Underground Mining Methods	Type
Tabular				
Flat (and inclined)	Good	Good	Room-and-pillar (spans ≤6 m [20 ft]); stope-and-pillar (spans ≤31 m [100 ft])	Self-supported
	Good	Poor	Room-and-pillar; stope-and-pillar	Supported
	Poor (roof collapses about free-standing pillars)	Good	Longwall; pillaring	Caved
	Poor	Poor	Immediately above	Caved
Steep	Good	Good	Sublevel stoping (spans 6–31 m [20–100 ft]); large tunnel section	Self-supported then filled
	Good	Poor	Hydraulicking—coal (spans 6–21-m [20–70-ft] arch); shrinkage	Supported then filled
	Poor	Good	Cut-and-fill	
	Poor	Poor	Sublevel caving and top slice spans ≥6 m (20 ft) (for gravity flow)	Caved
Massive	Good	Good	Vertical slices†	Self-supported
	Good	Poor	Vertical slices	Supported then filled
	Poor (cap rock)	Poor	Block caving (spans ~34 m [110 ft] active—end stope used)	

*Rated as to strength (and stiffness of pillar).
†Horizontal slices can introduce the many problems associated with multiple-seam mining.

Pillars

Pillars serve to support the main roof and its loads, primarily the overburden acting over a tributary area. Pillar material consists mainly of the seam itself and sometimes waste incorporated within the seam. Pillars must not only be sufficiently strong but also must be sufficiently stiff, a frequently overlooked requirement. If pillars are not adequately stiff, but still adequately strong, the roof will collapse about the still free-standing pillars, especially when differential pillar (and floor) deflection occurs. The minimum slenderness ratio for pillars to avoid this crippling is inversely proportional to the recovery. The mining of flat, thick seams of coal dramatically reflects this relationship and is a factor in classifying seam thicknesses (Table 6.2-4). For massive deposits, even in strong rock, this makes freestanding pillars of doubtful value. Upper slenderness ratios range from about 10/1 for coal to 1/3 for rock. Continuous vertical pillars are used to separate vertical stopes in hard rock that employ steep, tabular stoping methods. Even with stable ground, these are usually filled soon after mining for long-term stability. When massive deposits along with their cap rock are weak, caving is necessitated, usually performed as horizontal lifts or as block caving. Caving always requires a sufficient span 9 m (30 ft), good draw control, and also risks dilution and/or poor recovery. Soft or nonuniform floors (footwalls) act the same as do soft and irregular pillars.

Fill

Fill, often a sandy slurry consisting of crushed waste, cement, and water, can be readily introduced into confined (plugged), inclined, and steep tabular stopes. When drained and dried, this hardened slurry provides permanent resistance to ground movement, especially for the walls or pillars. It is widely used in all but the caving methods. It is either run in progressively as a stope is mined out or done all at once at the end of stope mining. Because of settlement and shrinkage away from a flat back, it is marginally useful for flat deposits.

When timbering is densely placed, especially with square sets, it rivals pillars. It, too, is usually filled as stoping progresses (overhand mining). These relationships are summarized in Table 6.2-11 and lead into the formal classification.

Underground Mining Classification System

Based on an understanding of bulk handling and ground control, the underground classification system shown in Table 6.2-12 closely follows previous ones. The primary difference is that sometimes shrinkage stoping is considered self-supported rather than supported. However, although the broken mineral provides a working floor, it is still supporting the hanging wall (roof). On the other hand, when the stope is drawn empty, it remains substantially self-supported until fill is introduced. The disadvantages of the shrinkage method are unique: (1) an uncertain working floor, (2) dilution due to sloughing and falls of rock, (3) possibly adverse chemical effects, and (4) tying up about two-thirds of the mineral until the stope is drawn.

Vertical crater retreat mining is included in the classification between sublevel and shrinkage stoping (Hamrin 1980).

OTHER FACTORS

While subordinated, there are additional factors that must be closely evaluated. These deal with the broad impacts on the environment, health and safety, costs, output rate, and others. They are usually evaluated on a relative basis, although numbers may also be employed (Table 6.2-13) (Boshkov and Wright 1973; Hartman 1987). An example of where the environmental considerations on the surface are beginning to affect mining methods is in the use of high-density paste backfilling in order to return most of the tailings back

Table 6.2-12 Classification of underground mining methods based on deposit geometry and support

Deposit Shape, Attitude (dip)	Degree of Support: Unsupported (open stopes)	Supported	Caved
Tabular			
Flat (mobile bulk handling)	Room-and-pillar; stope-and-pillar	Some degree of artificial support for room-and-pillar and stope-and-pillar	Longwall (shortwall); pillaring (especially room-and-pillar)
Inclined (mixed bulk handling)	Above with scrapers	Above with scrapers	Longwall (difficult)
	Large tunnel section (on-strike)	Large tunnel section with artificial support	
Steep (gravity bulk handling)	Coal hydraulicking	Shrinkage stoping; cut-and-fill stoping	Sublevel caving
	Sublevel stoping	Timbered stoping (square sets, stulls, gravity)	Top slicing (control dilution-and-recovery)
	Vertical crater retreat	Fill as needed	
	Shrinkage stoping	Gravity fill as needed	
Massive	Immediately above mine in vertical slices. Fill—gravity placement. To remove pillars, can mine and then fill horizontal lifts.*		Immediately above in horizontal lifts block caving (bulk mining)

*For ground control problems, especially those associated with coal, treat as if they were to be extracted by thick-seam and/or multiple-seam mining. As pressure increases (especially with depth), or as rock strength decreases, shift right for suitable method (toward supported and caved).

Table 6.2-13 Secondary factors to be considered when selecting a mining method

Method	Relative Cost	Flexibility/ Selectivity	% Recovery/ % Dilution	Environment	Safety and Health	Output (t/h) and Productivity (t/employee)	Miscellaneous
Surface Mining							
Placers and dredging	0.05	Low/high	High/low	High impact, and water pollution	Fair	Moderate	Need water; impact of weather
Open-cast	0.10	Moderate/ moderate	High/low	Blasting can lead to frequent claims and water pollution	Fair	High	Flat topography and impact of weather
Open-pit	0.10	Moderate/ moderate	High/low	Ground disturbance, waste piles, and some water problems	Slope stability (slides)	High	Impact of weather
Quarry	1.00	Low/high	High/high	Ground disturbance and waste piles	Slope stability	Very low	Skilled workers and impact of weather
Underground Mining							
Room-and-pillar (coal)	0.30	High/high	50–80/20	Subsidence and water pollution	Ground control and ventilation	High	Pillaring common
Stope-and-pillar	0.30	High/high	75/15	Good	Ground control and ventilation	High	Benching common
Sublevel stope	0.40	Low/low	75/15	Fill to avoid subsidence	Less, blast from long holes	Moderate	Fill common
Shrinkage	0.50	Moderate/ moderate	80/10 plucking during draw	Fill to avoid subsidence	Poor floor (collapse) and stored broken mineral*	Low	Tie up ⅔ of ore
Cut-and-fill	0.60	Moderate/ high	100/0	Fill to avoid subsidence	Some	Low	Sort in stope
Timbered square set	1.00	Moderate/ high	100/0	Fill to avoid subsidence	Smolder, and fall (of personnel)	Very low	Sort in stope
Longwall	0.20	Low/low	80/10	Subsidence and water pollution	Good	Very high	High capital ≤12° dip ≤2.4 m (8 ft) thick
Sublevel caving (top slicing)	0.50	Low/low	90/20	Severe subsidence disruption	Fair and stored broken mineral*	High	Cave width ≥9.2 m (30 ft)
Block caving	0.20	Low/low	90/20	Severe subsidence disruption	Air blasts and stored broken mineral*	High	Tie up mineral

*Can pack (cement), oxidize, and smolder.

underground (in order to obtain mining permits from environmental agencies).

In addition, innovation is always occurring and some is currently of proven value. These include rapid excavation, methane drainage, underground gasification, and retorting (Hartman 1987). Many methods are now automated and robotized.

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CHAPTER 6.3

Selection Process for Hard-Rock Mining

Peter G. Carter

INTRODUCTION

The purpose of this chapter is to describe the process for determining which mining methods should be considered by a mine planner when evaluating a hard-rock mineral deposit. (For the purpose of this chapter, hard rock is defined as ores that cannot be mined by mechanical mining machines.) Mining methods applicable to the generic groupings of *open pit* and *underground* are discussed. Quarrying, dredging, methane drainage, and solution mining are covered, along with other mining methods, in more detail in Chapters 6.6 and 10.3 of this handbook, although several of these methods are mentioned here for completeness, because they involve hard rock. The term *mining method* is used to describe a unique combination of variables that describe the process of excavating rock to recover valuable minerals, together with the essential characteristics of the excavation—an open-pit or an underground mine.

Mining in the 21st century is steadily gravitating toward large-scale excavations in either open-pit or underground mining schemes. This is driven by ever decreasing grades and product values (measured in real terms), all of which place pressure on profit margins and drive the requirement for improved economies of scale.

The single most important variable influencing the selection of a mining method will always be the style and geological characteristics of the mineral deposit. Mineralization styles may include, for example, thin, steeply dipping, tabular vein gold deposits or large, disseminated porphyry copper deposits. Typically, more than one mining method or variation, such as open stoping systems with and without fill, will need to be evaluated. Other variables likely to have a material bearing on the selection process include

- Engineering properties of the mineral deposit and host rock mass;
- Required rate of mineral production from the mine;
- Forecasts of the mineral products' value;
- Comparative capital and operating costs of the various mining activities and mineral processing activities necessary to implement the mining method;
- Availability, cost, and skill levels of labor needed to operate the mine;
- Prevailing regulatory environment; and
- Environmental impacts together with the costs of mitigation and, ultimately, mine closure costs.

The optimum mining method will always be the one that maximizes the economic returns while keeping the environmental impact within acceptable levels, maintaining acceptable work conditions (especially in regard to levels of safety risk) for employees, and satisfying statutory obligations (including resource recovery stipulations). Collectively, these goals will also satisfy the objective of efficient use of the mineral resource.

The following sections examine in more detail the key influences briefly referred to here, followed by discussion on the selection and evaluation methodologies that should be considered when planning a mine. Quantitative and qualitative methodologies are examined.

KEY INFLUENCES ON SELECTION OF MINING METHOD

Key influences include the style of the mineralization and the strength and character of the rock mass.

Style of Mineralization

The term *style* refers to the range of geometric attributes and the mechanisms that have controlled and in many cases also determined the distribution of valuable minerals within the deposit. Typically, this information is coded within a geological model for the mineral deposit, based on the quantity of factual data. During the early phases of exploration, the geological model may be substantially conceptual, perhaps based on mapping and remote sensing technologies such as gravimetric or seismic methods and limited drill-hole sampling. The limitations of knowledge about the style of the mineralization must be carefully considered in the selection process.

In some circumstances, dependent on the geological setting, these conceptual models may provide a reliable foundation for considering the applicable mining methods.

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Improvements in geological knowledge for a coal seam within sedimentary rocks that have not been materially disturbed by faulting are unlikely to cause the mining method selection process to be revisited, whereas additional geological data (drill-hole sampling) of a steeply dipping, tabular gold deposit substantially affected by faulting, folding, or shearing and displaying pronounced grade trends may require a wider range of possible mining methods to be considered.

The method of preparing the geological model will also be a significant consideration for the mine planner. Most of the general mine planning software packages currently available provide powerful three-dimensional (3-D) visualization, wireframe triangulation facilities, and block modeling systems to facilitate the development of sophisticated and often complex geological models. These tools commonly have the facility for preparing long sections illustrating grade isopachs, thickness isopachs, and structurally controlled grade trends, which can be useful in mining method selection processes.

Interpolation of grade distributions applying simple polygonal methods through to complex geostatistical methods incorporating uncertainty are now common. Mineral resource models that reflect the inherent uncertainties provide enhanced assistance for the optimum selection of a mining method.

Strength and Character of the Rock Mass

This section has been adapted from Hoek (2007). Any process intended to aid the selection of an excavation method must consider the strength and character of the host rock mass. One of the more complex tasks for the mine planner is the determination of representative mechanical properties of the host rock mass. Although tests have been devised to quantify many of the properties of laboratory rock specimens, it is a considerably more difficult task to predict the expected behavior of a rock mass. Numerous empirical rock mass classification methods (derived from actual case studies) have been devised to assist mine planners. It is important to understand the limitations of rock mass classification schemes (Palmstrom and Broch 2006) and that their use does not (and cannot) replace some of the more elaborate design procedures or decisions made from economic analyses. However, the use of these design procedures requires access to relatively detailed information on in-situ stresses, rock mass properties, and planned excavation sequence—none of which may be available at an early stage in the project. As this information becomes available, the rock mass classification scheme adopted should be updated and used in conjunction with site-specific analyses.

Open-Pit Slopes

The stability of rock slopes has traditionally been evaluated by limiting equilibrium methods (Hoek and Bray 1981; Wyllie et al. 2004), although probabilistic-based approaches are increasingly more commonly applied, because they acknowledge the implicit uncertainties of limit equilibrium methods. Limit equilibrium models fall into two main categories: (1) models that deal with structurally controlled planar or wedge slides and (2) those that deal with circular or near-circular failure surfaces in homogenous materials. Many of these models have been available for more than 25 years and can be considered reliable slope design tools. Wyllie et al. provide a methodology for assembling basic geological data, rock strength information, and groundwater observations, and integrating this with engineering rules in the form of design charts and graphical methods to permit a nonspecialist engineer to obtain approximate answers suitable for assessing open-pit alternatives.

Several rock mass classification systems have been specifically adapted for rock slope engineering (Haines and Terbrugge 1991; Romana 1995; Chen 1995). These methodologies have been adapted from classification systems for the highly confined rock mass conditions associated with underground mining as distinct from the low-confining stress conditions characteristic of open-pit slopes. These systems, if used with appropriate caution, are useful in specifying a range of slope conditions to assist in mining method selection practices but can never replace the requirement for more rigorous processes such as limit equilibrium and numerical modeling of slopes (Figure 6.3-1).

Numerical modeling of slope deformation behavior is now a routine activity on many large open-pit mines. Software programs such as FLAC and UDEC are typically used for such modeling, although a significant amount of expertise is required to ensure realistic input information and reliable interpretation of outputs. In best practice, a combination of limit equilibrium and numerical modeling approaches are applied to generate an array of solutions for the range of inputs that typically exist at a site, because it is far more reliable to look at the array of results from a parametric study than a single deterministic study.

With the greater depths characteristic of modern open-pit mines, the role of the in-situ stress field in slope stability is becoming an increasingly important consideration. In these cases, mine planners must seek advice from specialists about the applied assumptions when comparing deep open-pit alternatives with underground methods.

In terms of arriving at a suitable set of slope parameters for assessing the applicability of any open-pit method, a process that recognizes the implicit uncertainties and considers a range of slopes as inputs to the evaluation should always be adopted.

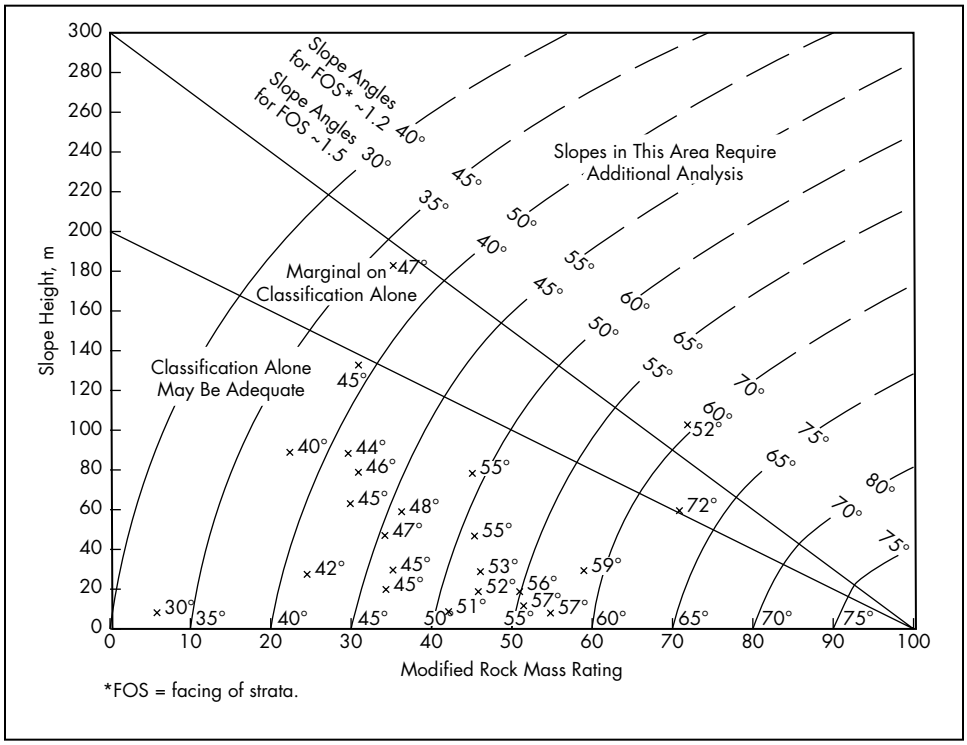
Underground Excavations

Rock mass classification systems applicable to underground excavations have been evolving for more than 100 years since Ritter (1879) attempted to formalize an empirical approach to tunnel design for the purposes of determining support requirements.

Terzaghi's rock mass classification. The earliest reference to the use of rock mass classification for the design of tunnel support is in a paper by Terzaghi (1946) in which the rock loads, carried by steel sets, are estimated on the basis of a descriptive classification. It is useful to examine the rock mass descriptions included in his original paper, because he draws attention to those characteristics that dominate rock mass behavior, particularly in situations where gravity constitutes the dominant driving force. The clear and concise definitions and the practical comments included in these descriptions are good examples of the type of engineering geology information that is most useful for engineering design.

Terzaghi's descriptions (quoted directly from his paper) are as follows:

- Intact rock contains neither joints nor hair cracks. Consequently, if it breaks, it breaks across sound rock. Because of injury to the rock due to blasting, spalls may drop off the roof several hours or days after blasting—known as a spalling condition. Hard, intact rock may also be encountered in the popping condition involving the



Source: Haines and Terbrugge 1991.

Figure 6.3-1 Preliminary slope angle chart

spontaneous and violent detachment of rock slabs from the sides or roof.

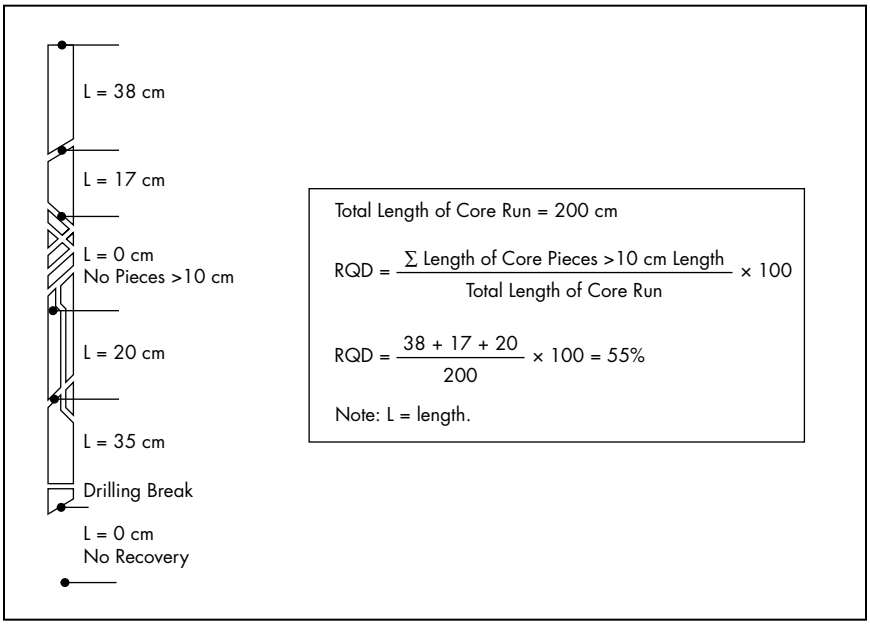
- Stratified rock consists of individual stratum with little or no resistance against separation along the boundaries between the strata. The strata may or may not be weakened by transverse joints. In such rock the spalling condition is quite common.
- Moderately jointed rock contains joints and hair cracks, but the blocks between joints are locally grown together or so intimately interlocked that vertical walls do not require lateral support. In rocks of this type, both spalling and popping conditions may be encountered.
- Blocky and seamy rock consists of chemically intact or almost intact rock fragments which are entirely separated from each other and imperfectly interlocked. In such rock, vertical walls may require lateral support.
- Crushed but chemically intact rock has the character of crusher run. If most or all of the fragments are as small as fine sand grains and no re-cementation has taken place, crushed rock below the water table exhibits the properties of a water-bearing sand.
- Squeezing rock slowly advances into the tunnel without perceptible volume increase. A prerequisite for squeeze is a high percentage of microscopic and submicroscopic particles of micaceous minerals or clay minerals with a low swelling capacity.
- Swelling rock advances into the tunnel chiefly because of expansion. The capacity to swell seems to be limited to those rocks that contain clay minerals such as montmorillonite, with a high swelling capacity.

Lauffer (1958) proposed that the stand-up time for an unsupported span is related to the quality of the rock mass in which the span is excavated. In a tunnel, the unsupported span is defined as the span of the tunnel or the distance between the face and the nearest support, if greater than the tunnel span. Lauffer's original classification has since been modified by a number of authors, notably Pacher et al. (1974), and now forms part of the general tunneling approach known as the New Austrian Tunnelling Method.

This method includes a number of techniques for safe tunneling in rock conditions in which the stand-up time is limited before failure occurs. These techniques include the use of smaller headings and benching or the use of multiple drifts to form a reinforced ring inside which the bulk of the tunnel can be excavated. These techniques are applicable in soft rocks such as shales and phyllites, and in which the squeezing and swelling problems, described by Terzaghi, are likely to occur. The techniques are also applicable when tunneling in excessively broken rock, but great care should be taken in attempting to apply these techniques to excavations in hard rocks in which different failure mechanisms occur.

Rock quality designation (RQD). The RQD index was developed by Deere et al. (1967) to provide a quantitative estimate of rock mass quality from drill core logs. RQD is defined as the percentage of intact core pieces longer than 100 mm in the total length of core. The core should be at a minimum size (54.7 mm in diameter) and should be drilled with a double-tube core barrel. The procedures for measurement of the length of core pieces and the calculation of RQD are illustrated in Figure 6.3-2.

RQD is a directionally dependent parameter, and its value may change significantly, depending upon the borehole orientation. The use of the volumetric joint count can be quite useful in reducing this directional dependence. RQD is also intended to represent the rock mass quality in situ. When



Source: Deere 1989.

Figure 6.3-2 Procedure for measurement and calculation of RQD

using diamond drill core, care must be taken to ensure that fractures, which have been caused by handling or the drilling process, are identified and ignored when determining the value of RQD.

Deere's RQD was widely used, particularly in North America, after its introduction. Although various investigators have sought to relate RQD to Terzaghi's rock load factors and to rock bolt requirements in tunnels, the most important use of RQD is as a component of the rock mass rating and Q rock mass classifications covered later in this chapter.

Rock structure rating (RSR). Wickham and Tiedemann (1974) described a quantitative method for describing the quality of a rock mass and for selecting appropriate support on the basis of their RSR classification. Most of the case histories used in the development of this system were for relatively small tunnels supported by means of steel sets, although historically this system was the first to make reference to shotcrete support. Although the RSR classification system is not widely used today, Wickham et al.'s work, in which they devised a basis for rating the geological, geometrical, and joints condition, played a significant role in the development of the classification schemes discussed in the remaining sections of this chapter.

Rock mass rating (RMR). Bieniawski (1973, 1976) published the details of a rock mass classification, called the geomechanics classification or the RMR system. Over the years, this system has been successively refined as more case records have been examined, and the reader should be aware that Bieniawski has made significant changes in the ratings assigned to different parameters (Bieniawski 1989). Bieniawski's RMR system was originally based on case histories drawn from civil engineering. Consequently, the mining industry tended to regard the classification as somewhat conservative, and several modifications have been proposed in order to make the classification more relevant to mining applications. A comprehensive summary of these modifications was compiled by Bieniawski (1989).

Both this and the 1976 version deal with estimating the strength of rock masses. The following six parameters are used to classify a rock mass using the RMR system:

1. Uniaxial compressive strength of rock material
2. RQD
3. Spacing of discontinuities
4. Condition of discontinuities
5. Groundwater conditions
6. Orientation of discontinuities

In applying this classification system, the rock mass is divided into a number of structural regions, and each region is classified separately. The boundaries of the structural regions usually coincide with a major structural feature such as a fault or with a change in rock type. In some cases, significant changes in discontinuity spacing or characteristics within the same rock type may necessitate the division of the rock mass into a number of small structural regions.

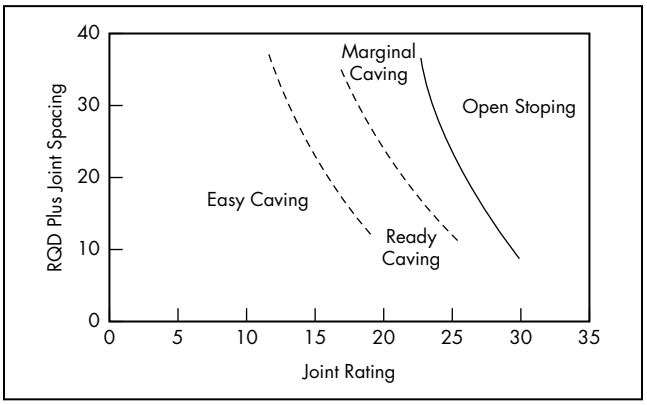
Modified rock mass rating (MRMR). Laubscher (1977, 1984), Laubscher and Taylor (1976), and Laubscher and Page (1990) have described an MRMR system for mining. This system takes the basic RMR value, as defined by Bieniawski, and adjusts it to account for in-situ and induced stresses, stress changes, and the effects of blasting and weathering. A set of support recommendations is associated with the resulting MRMR value. In using Laubscher's MRMR system, it should be borne in mind that many of the case histories upon which it is based are derived from caving operations. Originally, block caving in asbestos mines in Africa formed the basis for the modifications but, subsequently, other case histories from around the world have been added to the database. The selection of an appropriate mass underground mining method has been presented by Laubscher (1981) (Figure 6.3-3). The selection process is based on his rock mass classification system, which adjusts for expected mining effects on the rock mass strength.

Laubscher's scheme is aimed at the mass mining methods, primarily block caving and open stoping methods, although

his main emphasis is on cavability. The two parameters that determine whether a caving system is used over a stoping system are the degree of fracturing, RQD (Figure 6.3-3), joint spacing, and the joint rating, which is a description of the character of the joint—that is, waviness, filling, and water conditions. This scheme puts emphasis on the jointing as the only control for determining cavability.

Laubscher (1990) has subsequently modified the MRMR classification to relate the MRMR rating to the hydraulic radius (HR) (Figure 6.3-4). By including the hydraulic radius, cavability becomes feasible for more competent rock if the area available for undercutting is large.

Cummings et al. (1982) and Kendorski et al. (1983) have also modified Bieniawski's RMR classification to produce the modified basic RMR (MBRMR) system for mining. Developed for block caving operations in the United States, this system involves the use of different ratings for the original parameters used to determine the value of RMR and the subsequent adjustment of the resulting MBRMR value to allow for blast damage, induced stresses, structural features, distance from the cave front, and size of the caving block. Support recommendations are presented for isolated or development drifts, as well as for the final support of intersections and drifts.



Source: Nicholas 1992.

Figure 6.3-3 Laubscher's 1981 classification for cavability

Rock tunneling quality index, *Q*. On the basis of an evaluation of a large number of case histories of underground excavations, Barton et al. (1974), of the Norwegian Geotechnical Institute, proposed a rock tunneling quality index (Q) for the determination of rock mass characteristics and tunnel support requirements. The numerical value of Q varies on a logarithmic scale from 0.001 to a maximum of 1,000 and is defined by

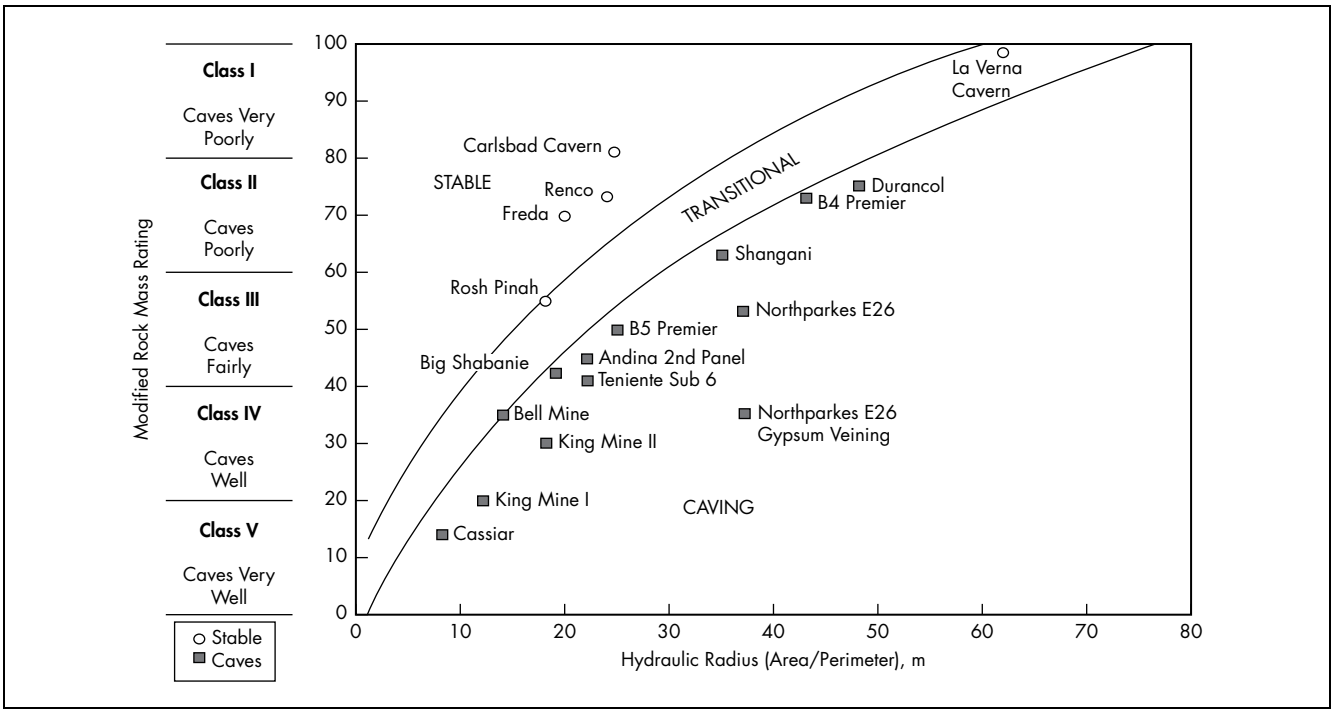
$$Q = \frac{\text{RQD}}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{\text{SRF}}$$

where

RQD = rock quality designation
J_n = joint set number
J_r = joint roughness number
J_a = joint alteration number
J_w = joint water reduction factor
SRF = stress reduction factor

In explaining the meaning of the parameters used to determine the value of Q, Barton et al. (1974) offer the following comments:

- The first quotient (RQD/J_n), representing the structure of the rock mass, is a crude measure of the block or particle size, with the two extreme values (100/0.5 and 10/20) differing by a factor of 400. If the quotient is interpreted in units of centimeters, the extreme particle sizes of 200 to



Source: Laubscher 1990.

Figure 6.3-4 Laubscher's cavability related to hydraulic radius and MRMR

0.5 cm are seen to be crude but fairly realistic approximations. Probably the largest blocks should be several times this size and the smallest fragments less than half the size. (Clay particles are, of course, excluded).

- The second quotient (J_r/J_a) represents the roughness and frictional characteristics of the joint walls or filling materials. This quotient is weighted in favor of rough, unaltered joints in direct contact. It is expected that such surfaces will be close to peak strength, will dilate strongly when sheared, and therefore will be especially favorable to tunnel stability. When rock joints have thin clay mineral coatings and fillings, the strength is reduced significantly. Nevertheless, rock wall contact after small shear displacements have occurred may be an important factor for preserving the excavation from ultimate failure.
- The third quotient (J_w/SRF) consists of two stress parameters. SRF is a measure of (1) loosening load in the case of an excavation through shear zones and clay bearing rock, (2) rock stress in competent rock, and (3) squeezing loads in plastic incompetent rocks. It can be regarded as a total stress parameter. The parameter J_w is a measure of water pressure, which has an adverse effect on the shear strength of joints due to a reduction in effective normal stress. In addition, water may cause softening and possible outwash in the case of clay-filled joints. It has proved impossible to combine these two parameters in terms of inter-block effective stress, because paradoxically a high value of effective normal stress may sometimes signify less stable conditions than a low value, despite the higher shear strength. The quotient (J_w/SRF) is a complicated empirical factor describing the active stress.

In relating the value of the index Q to the stability and support requirements of underground excavations, Barton et al. defined an additional parameter, which they called the equivalent dimension, D_e, of the excavation. This dimension is obtained by dividing the span, diameter, or wall height of the excavation by a quantity called the excavation support ratio (ESR). Hence:

$$D_e = \frac{\text{excavation span, diameter or height (m)}}{ESR}$$

The value of ESR is related to the intended use of the excavation and to the degree of security that is demanded of the support system installed to maintain the stability of the excavation.

Modified rock quality index (Q). Barton's Q has been used with a great deal of success in the design of tunnels in rock. However, the SRF parameter is redundant when the classification system is used for the estimation of rock mass properties for the purpose of analytical or numerical modeling for design, because the influence of stress is taken into account within the model. Thus the SRF is set to 1.0, which is equivalent to a moderately clamped but not overstressed rock mass. In addition, in most underground hard-rock environments, the excavations are relatively dry (not considering transient mine water inflows from drilling or backfilling), in which case the J_w parameter can also be set to 1.0.

Along with several other factors (accounting for jointing, stope geometry, and mining-induced stress), Q can then be used to determine the modified stability N′ which is then used with the modified stability graph method (Mathews et al. 1981; Potvin 1988; Bawden 1993; and Hoek et al. 1995) for the dimensioning of open stopes in underground mines.

Modified stability number (N'). Mathews et al.'s empirical method for dimensioning stopes is based on the first and second quotients to dimension each face of a stope together with the HR, where this parameter accounts for the shape and size of the face (Figure 6.3-5). Potvin modified this method and calibrated it using 175 case histories. Nickson (1992) added further case histories, which considered hanging walls, footwalls, endwalls, and backs from a wide variety of mining environments. Other case histories can be found throughout recent literature—Bawden et al. (1989) and Greer (1989). The classification of the rock mass and excavation problem is accomplished in the modified stability graph method, which relates N' to HR where

$$N' = Q' \times A \times B \times C$$

where

A = measure of the ratio of intact rock strength to induced stress in the range 0.1–1.0
B = relative orientation of the dominant jointing relative to the excavation surface in the range 0.2–1.0
C = measure of the influence of gravity on the stability of the face being considered in the range 2–8
N' = values in the range 0.0005–8000 with typical values for hard-rock mines of 0.1–1000

Regardless of whether an open-pit or underground mining method is being contemplated, rock mass classification systems can be of considerable benefit, even when little detailed information is available to characterize the rock mass, its strength, and the hydrogeological conditions. At a minimum, these systems can provide valuable insight into the data, which must necessarily be collected following a decision to move the evaluation of a mining method through concept, preliminary, or final feasibility studies.

MINING METHOD ALTERNATIVES

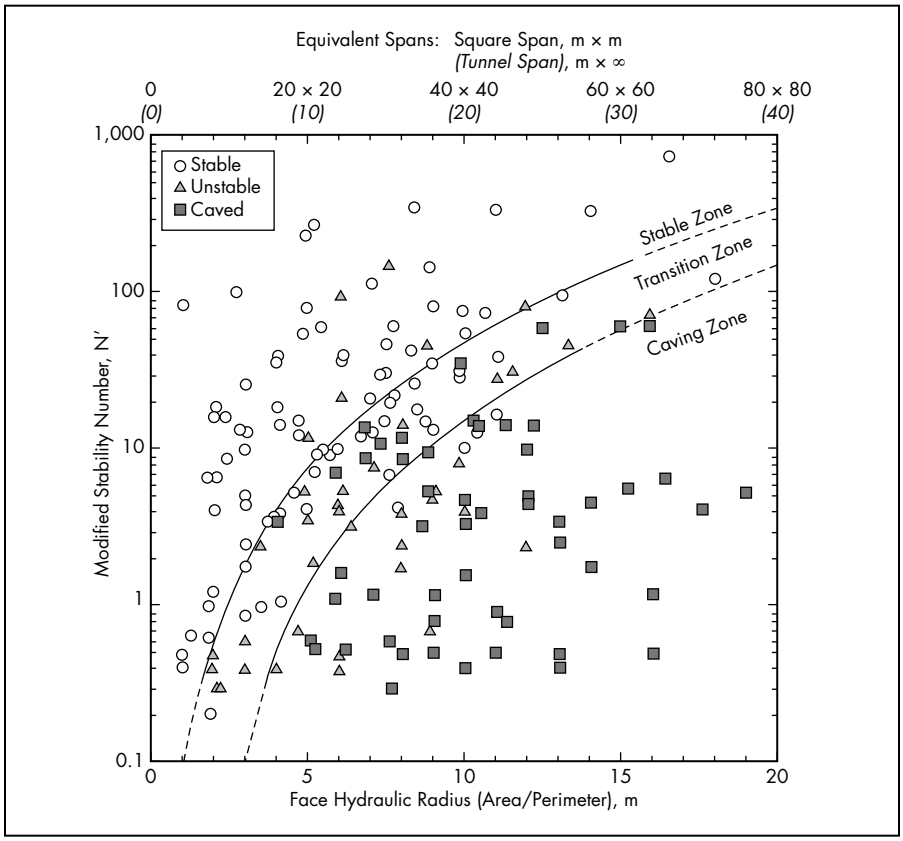
To facilitate the following discussion on mining method selection and evaluation methodologies, a brief discussion to characterize the range of surface (i.e., open-pit) and underground mining methods follows. Additional information in relation to open-pit mining methods can be found in Hustrulid and Kuchta (2006) and Kennedy (1990), while further information on underground mining methods can be found in Hustrulid and Bullock (2001).

Surface Mining Methods

Surface mining methods are defined here as any excavation that commences from the natural surface and does not entail the construction of a tunnel or shaft. Most often, the style of mineralization will significantly impact the features of a surface mining method, particularly the character and thickness of overburden. The type of equipment deployed also commonly affects the classification of a surface mining method.

Open-Cut Mining

Open-cut mining refers to a particular kind of surface mining that most commonly deploys large rope shovels, hydraulic shovels, or excavators together with suitably sized rear-dump trucks and progresses the excavation in a series of slices. For



Source: Adapted from Hutchinson and Diederichs 1996.

Figure 6.3-5 Mathews et al.'s stability graph



Figure 6.3-6 Iron ore open-cut mine in Australia



Courtesy of BHP Billiton.

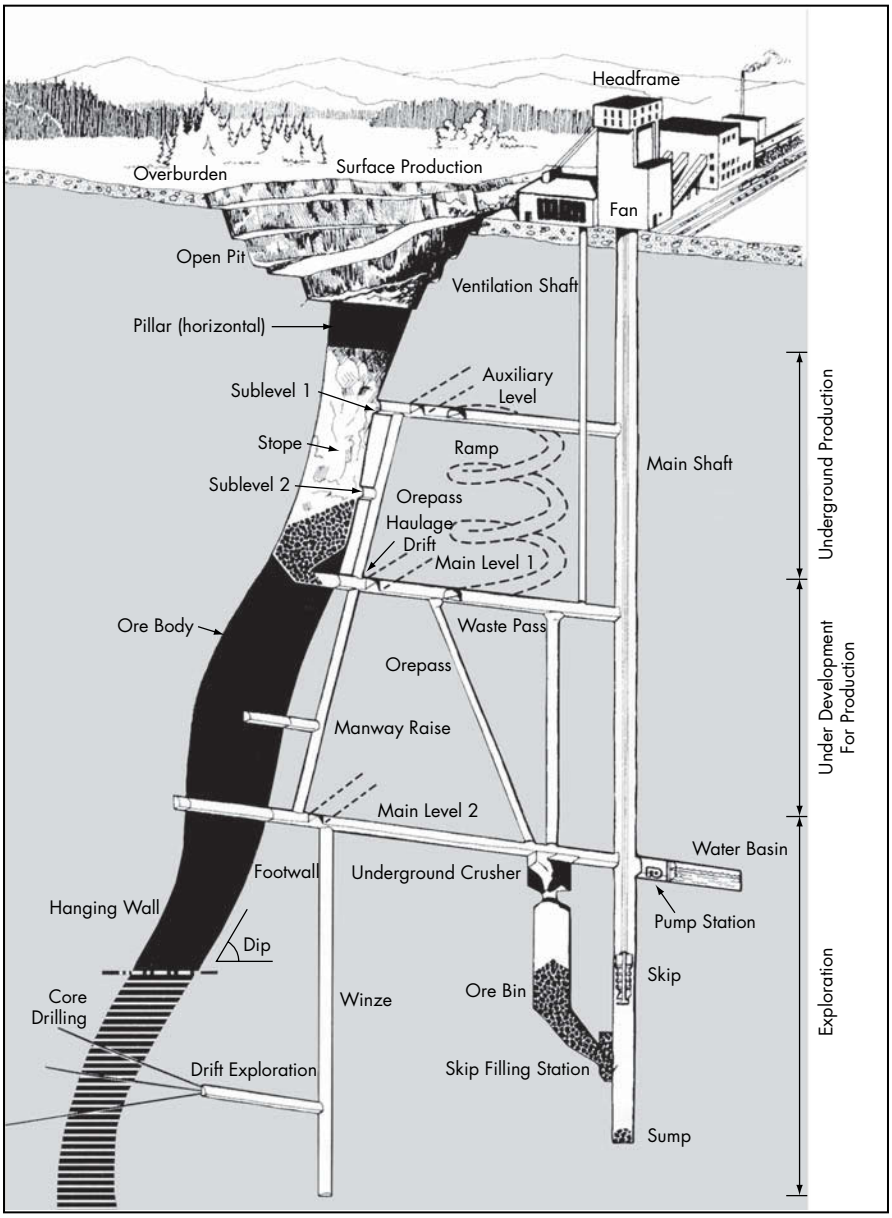
Figure 6.3-7 Strip mine in Australia

hard-rock open-cut mines, drilling and blasting practices are often an integral part of the excavation system. The mining slice height may or may not be consistent with the vertical interval applied to construct berms on a slope. For this method, pit slopes commonly emerge as a sequential series of cutbacks, designed to manage the progressive strip ratios and maximize cash flows. Many of these features are illustrated in Figure 6.3-6.

The term *open-cut mining* is also commonly used to describe excavation in soft rock in which drilling and blasting systems may not be required and continuous excavation technologies such as bucket-wheel excavators are utilized. Truck and shovel systems may well still have a role in these environments.

Strip Mining

Strip mining describes a particular type of surface mining method that relies heavily on the progressive and sequential disposal of overburden spoil into a previously mined void (Figure 6.3-7). Coal mining often falls in this category, although bauxite miners often adopt a variant of strip mining. Dragline equipment, supplemented by truck and shovel



Source: Hamrin 1998. © Atlas Copco.

Figure 6.3-8 Underground mining terminology

systems, are often observed in strip mines. Strip ratios can be relatively high, and slope angles can be relatively steep, largely due to the relatively low overall height of these slopes.

Underground Mining Methods

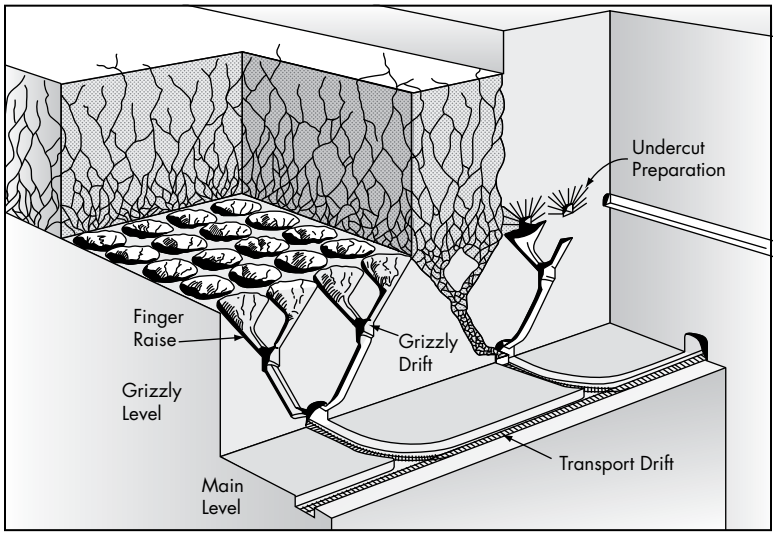
Underground mining methods invariably rely on tunneling networks to gain access to the zones of valuable minerals (Figure 6.3-8). These tunnel networks can be linked to vertical shafts equipped with rock-hoisting facilities or inclined ramps, also known as declines, through which rubber-tired equipment can pass, both to facilitate movement of this equipment and also for the transport of rock products to the surface.

Underground mining methods can be divided into three broad classes; caving, stoping, and other methods. The term *caving* implies the controlled collapse of the rock mass under the force of gravity, whereas the term *stoping* implies the excavation of a stable opening of small or large dimensions.

Caving Methods

Three generic methods of removing the valuable minerals and triggering caving processes can be described:

1. **Block caving technologies** are suitable for large low-grade ore bodies, either vertical or inclined, which are undercut over a large area, thereby inducing collapse of the entire rock mass, with the broken rock being extracted via a purpose constructed system of extraction points (Figure 6.3-9). The collapsing rock mass usually propagates to the natural surface and requires careful draw management to contain dilution from unmineralized material. Large scales of operation are possible with



Source: Hamrin 1998.

Figure 6.3-9 Schematic block cave

this technology, which is successfully being applied at increasing depths and on increasingly stronger rocks, compared to reference points in the 20th century.

2. **Sublevel caving technologies** are also suitable for large ore bodies of a generally tabular geometry requiring a more selective mining system and extraction by conventional drilling and blasting technologies (Figure 6.3-10). This method differs from block caving in that all of the ore is drilled and blasted, and only the overburden waste rock caves by gravity. Depending on the ore body geometry, this mining method is amenable to high rates of ore production. Sublevel caving is extensively used in the Swedish iron ore mines at Kiruna (26 Mt/a).
3. **Longwall mining systems** are extensively applied to deposits of coal and rely on rock-cutting technologies to excavate the coal using shearers and conveyor transportation systems to deliver the coal to surface. Coal seams are removed in a single slice, which may be 300 to 400 m across and several kilometers long with working heights of between 1.5 and 6 m. The overlying rock collapses into the mining void as the shearing system is advanced. The shearing system is protected by a series of heavy-duty shields operated by means of hydraulic jacks, which provide a movable canopy, thereby preventing the immediate roof over the workings from collapsing onto the shearer. Recent innovations also include longwall top coal caving, which emphasizes coal extraction efficiency in thicker seams.

Stoping Methods

The excavation of a stable void of small or large dimensions may be stoping's first defining feature—that is, the shape of the mineralization and/or the nature of the rock mass—in which the excavation is to be constructed. The term *stoping* infers the excavation of a stable void of small or large dimensions and is a defining feature of these methods (Table 6.3-1). The character of this void is substantially influenced by the shape of the mineralization and/or the nature of the rock mass in which the stope is being constructed.

A second defining feature of the range of possible stoping methods comprises the use of a fill material, which typically falls into two main groups: (1) waste rock or tailings, which may either be unconsolidated and therefore have negligible strength; or (2) consolidated, typically with a pozzolanic material such as cement or fly ash. In this context the term *strength* implies a capacity of the fill material to stand without collapse when otherwise virgin rock confining the fill material to the original void is removed in a second phase of mining.

A third defining feature of stoping methods is the nature of the drilling and blasting technology deployed, which may be characterized as either short-hole or long-hole systems. In this case, short holes are typically less than 4 m in length and consistent with a single-pass tunneling jumbo, while long holes are drilled with purpose designed long-hole drilling machines applying segmented drill strings.

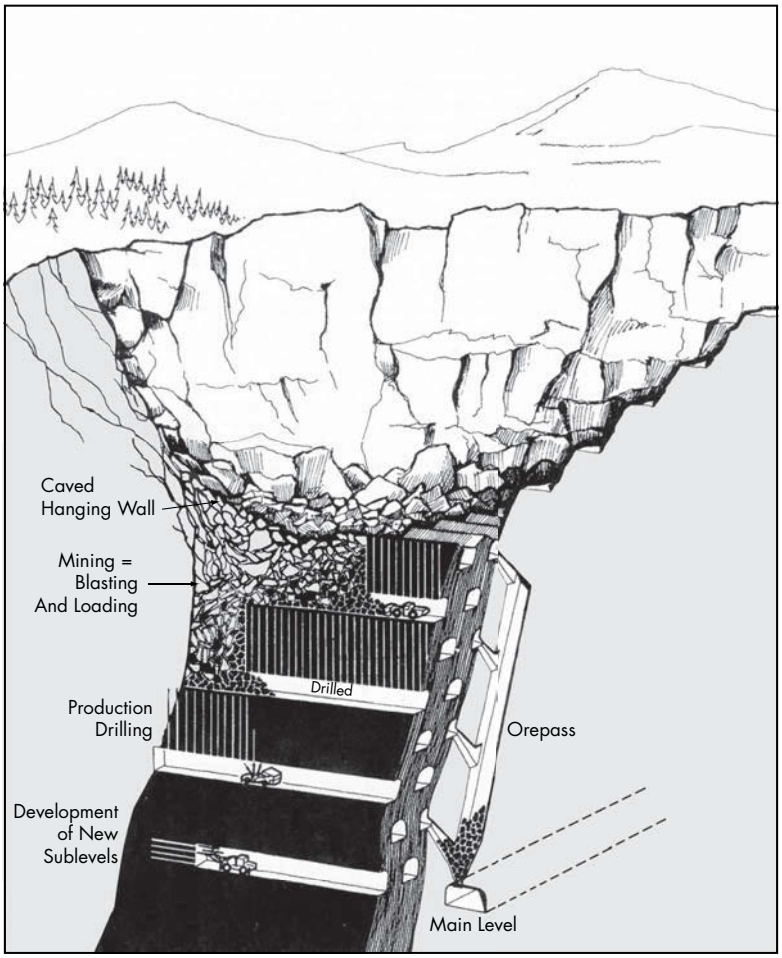
A fourth defining feature occurs where numerous independent stopes and the sequence of creation (and possibly filling) of these voids are contemplated. Collectively, these features give rise to a large number of stoping methods, as in Table 6.3-1, including shrink stopes (Figure 6.3-11) and overhand cut-and-fill stoping (Figure 6.3-12).

A fifth defining feature is the degree to which mineralization that has economic value is not mined so as to maintain the required stability of the stope—thus the principle of pillars such as in the room-and-pillar method, commonly applied to mineralization with low dips and modest heights (4–6 m) (see Figure 6.4-1 in Chapter 6.4). However, it can be applied to heights of more than 30 m.

Vertical crater retreat (VCR) is a term that encapsulates open stopes being developed by applying a particular drill-and-blast methodology but is otherwise similar to conventional open stoping (Figure 6.3-13).

Other Methods

The extraordinary diversity of mineral deposits inevitably leads to innovative mining methods, including combining methods. One such method is postpillar cut-and-fill, where the



Source: Hamrin 1998. © Atlas Copco.
Figure 6.3-10 Sublevel caving

pillars are mined small but immediately backfilled. Another example is the Avoca method, which combines sublevel longhole drilling with immediate backfilling. Other approaches are described in the following paragraphs.

Horodiam. The Horodiam method utilizes a large diameter raise-bored shaft as the access for a drill jumbo to drill horizontal radial blastholes over the entire height of the stope. The method is amenable to remote control technologies and has been patented as a remotely operated excavation system (ROES) by Australia's Commonwealth Scientific and Industrial Research Organisation.

Coal seam methane drainage. The success of this method is a function of the "physical properties of the coal seam (diffusivity, reservoir pressure, permeability, and gas content), mining method (if in progress), and drainage method" (Hartman 1987). The method is used in Europe but only now is gaining widespread acceptance in the United States to produce coal seam methane. This method is a type of borehole mining in which the wells are used to recover the methane.

Coal seam gasification. This method is applied to coal and is also related to borehole mining, in which the coal is burned at one end and the gases given off are recovered at another borehole. Use of this method is based on whether the cost of burning the coal and recovering the gases is cheaper than traditional mining. The key parameters that impact the method are the fracturing and the chemical composition of the coal. This method may become more feasible in cases where the coal seam is too narrow for traditional methods, in the recovery of multiple seams, where the second seam is too close to the first to be recovered in a traditional fashion, or where seam depth or quality precludes the economic application of conventional surface or underground methods. Subsidence considerations apply as for caving methods of mining.

Underground retorting. This method is being tried with oil shales and tar sands. After the area is mined to some extent using traditional drifting techniques and pillar designs, the rock in the stope (retort) is blasted in place. Oil is released from the rock and recovered under the stope. This method is chosen based on the retorting characteristics rather than on the mining parameters. From a mining perspective, the critical factor would be the cost and methodology of fracturing the ground. The degree of fragmentation will impact the percentage recovery of the oil, which is probably the most critical concern.

Surface to Underground Transition Methods

Occasionally, newly discovered mineral deposits are amenable to both a surface mining and an underground method, which presents a particularly interesting challenge to mine planners. Typically, a surface mining method would be applied to

Table 6.3-1 Summary of common stoping methods

Stoping Method	Style of Mineralization	Type of Fill	Drill-and-Blast Solution	Stope Sequencing Method	Use of Pillars	Other Comments
Room-and-pillar	Typically flat tabular	Unusual if applied—unconsolidated	Short holes with tunneling jumbos	Typically unconstrained	Essential feature of method	Typically highly mechanized trackless diesel drilling and ore transportation machinery applied with medium to high productivity outcomes
Square-set stoping	Typically steeply dipping tabular deposits	Timber sets substitute for fill	Short holes drilled with handheld drilling machines	Typically unconstrained	Avoided by substitution of intensive timber works	Poor productivity and expensive mining method rarely used in favor of other modern methods
Sublevel open stopes	Massive and tabular steeply dipping deposits	Can apply either unconsolidated or engineered fills	Long-hole systems with purposed design long-hole drilling machines	Considerable variety of transverse and longitudinal sequences	Secondary stopes may effectively act as temporary pillars if engineered fills are applied; otherwise can be permanent	Typically one of the most productive and lower-cost mining methods applied across many different styles of mineralization
Shrink stopes (Figure 6.3-11)	Typically steeply dipping tabular deposits	Broken ore used as work platform and temporary support of stope walls	Short holes drilled with handheld drilling machines	Usually independent of other stopes	Permanent pillars to separate stopes	Poor productivity, rarely used
Overhand cut-and-fill (Figure 6.3-12)	Tabular, moderately or steeply dipping	Unconsolidated fills	Short holes with tunneling jumbos	Usually independent of other stopes	Permanent pillars to separate stopes	Typically, heavily mechanized method applying tunneling jumbos and load-haul-dump machines accessing the stope from ramp system
Underhand cut-and-fill	Tabular, moderately or steeply dipping	Engineered fills	Short holes with tunneling jumbos	Usually independent of other stopes	Not typically needed as artificial pillars made of engineered fill materials used in lieu	Highly mechanized method applied where minimum stress is anticipated to cause mining to cease, thereby avoiding extensive ramp systems extending into mineralization that may not be able to be mined
Avoca	Tabular, moderately or steeply dipping	Unconsolidated fills	Long-hole systems with purposed design long-hole drilling machines	Typically very constrained, longitudinal advance or retreat systems	Pillars of very low-grade material below economic cutoffs, may be left but usually do not override stope sequence solution	Moderately high productivity solution able to use modern mechanized long-hole drills and load-haul-dump machines. Fill dilutions can be significant factor.
Sublevel retreat open stopes	Tabular, moderately or steeply dipping	Unconsolidated or engineered fills	Long-hole systems with purposed design long-hole drilling machines	Highly constrained by longitudinal retreat sequences in underhand or overhand configurations	Typically avoided in higher stress environments but may be used on regional scale	Highly mechanized mining method used in deposit with lower tons per vertical meter to avoid extensive access development in waste external to mineralization

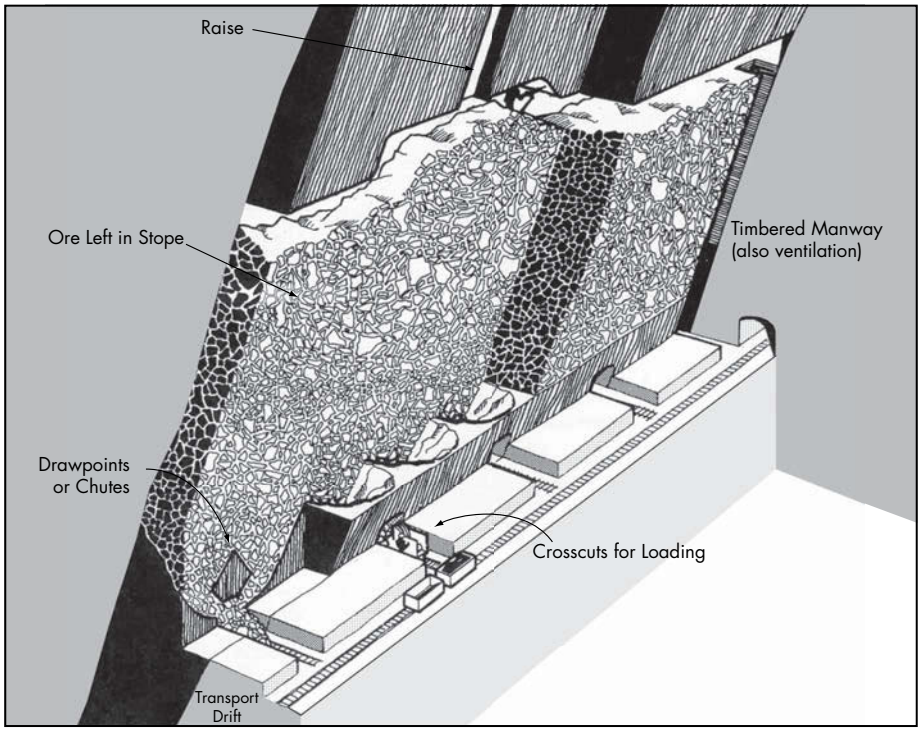
initially develop the deposit, although instances of an underground development preceding an open-pit development have occurred. The answer to critical questions regarding the depth at which a transition should occur is dependent on many factors, including the relative scale of the surface mine and the underground mine, the lead time required to develop the underground mine, and the optimum underground mining method.

MINING METHOD SELECTION AND EVALUATION METHODOLOGIES

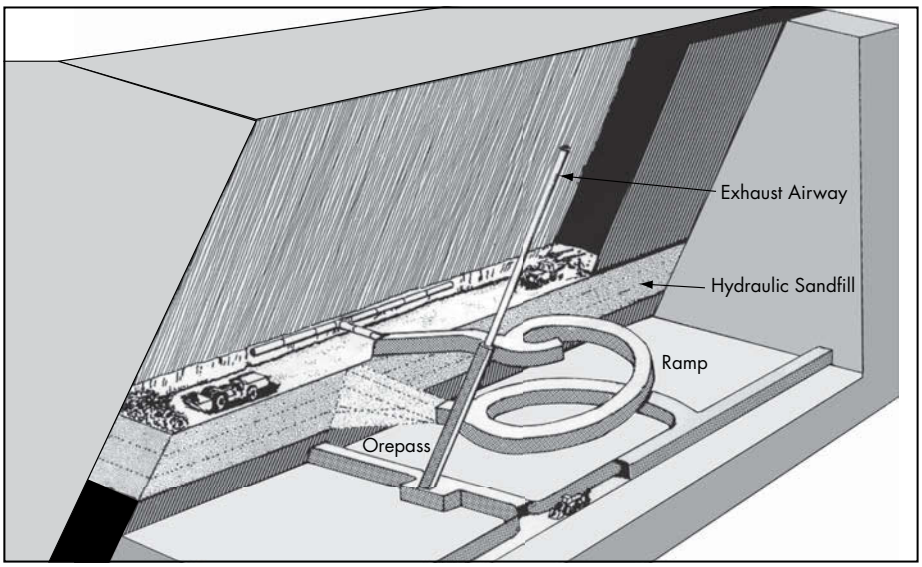
In many cases, the style and geometry of the mineralized system will be the dominant factor in identifying the most appropriate mining method for evaluating the potential economic value of the deposit. It is uncommon, however, to encounter a mineral deposit that is amenable only to a single mining method. Consider, for example, a flat tabular potash deposit within a halite sequence occurring at a depth that unequivocally precludes open-pit methods. In this instance, solution mining and conventional room-and-pillar mining systems should both be considered during the initial appraisal (and carried forward in subsequent appraisals) until such time as a clear economic benefit from a preferred method, after accounting for risk and uncertainty, can be demonstrated.

A second factor that will always have substantial influence on the possible mining methods is the characteristics of the host rock mass and the mineralized rock mass. The quantity and quality of this information will almost certainly be influenced by the status of exploration over the deposit. An all too common problem is the failure in early-stage exploration activities to allocate sufficient funds for the collection of critical rock mechanics data by which the characteristics of the rock mass can be adequately ascertained.

The techniques for evaluating mining methods are only attempts at defining and quantifying in a written format what miners in years past determined through discussion, previous experience, and intuition. Therefore, each of the method selection schemes presented here is similar and yet different, reflecting personal preferences in their emphases. The purpose



Source: Hamrin 1998. © Atlas Copco.
Figure 6.3-11 Shrink stoping

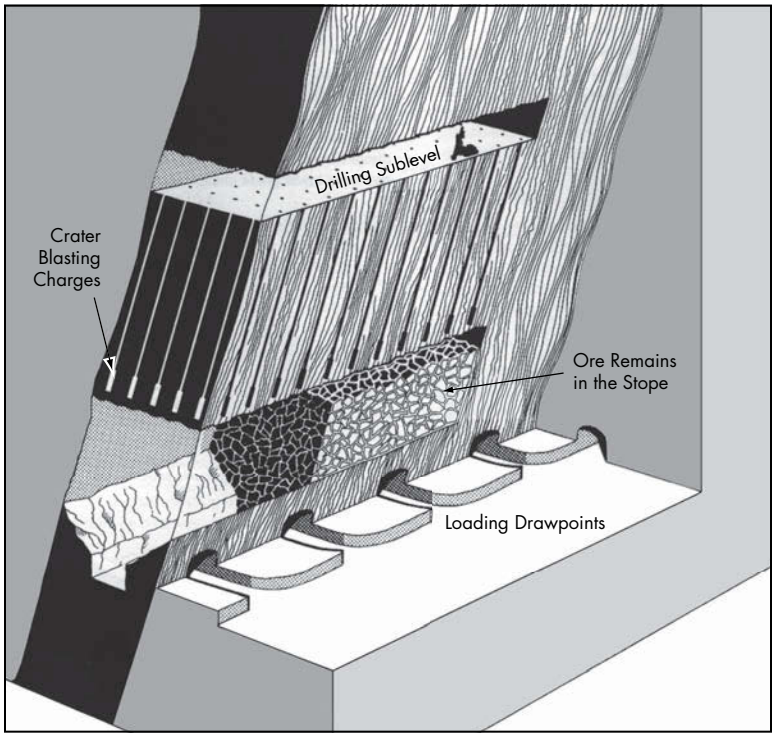


Source: Hamrin 1980. © Atlas Copco.
Figure 6.3-12 Overhand cut-and-fill stoping (mechanized)

of discussing these techniques is not to critique them but simply to present the alternatives available to aid in selecting the most appropriate.

Most of the schemes are aimed at determining the appropriate underground method, as there are many possible choices. However, the purpose of this chapter is to discuss the selection of the best mining methods, including surface, hydraulic, and more novel methods. The method selection process should first determine whether the deposit should be mined using a more traditional surface, underground, or in-situ leach mining method. A novel method should only be considered if traditional methods are not economically or technically feasible. To start a mine with a novel mining method requires adequate funding and an enormous commitment from the board of directors to technical development; the board must also have the patience to work out the technical problems.

If the deposit cannot be mined using a surface method, then an underground method should be considered. The



Source: Hamrin 1980. © Atlas Copco.
Figure 6.3-13 Vertical crater retreat open stoping

mining method selection techniques are limited, because selection is based solely on the known physical parameters and rock strength characteristics. Sometimes several mining methods may appear to be equally feasible. In order to further determine which method(s) is the most suitable, the input variables of mining costs, mining rate, labor availability, and environmental regulations should be considered in more detail.

(Note: None of the mtethod selection systems deal with in-situ stress. Although the techniques account for the vertical stress via depth, none of the methods discuss how a high horizontal stress impacts the choice of the mining method.)

Qualitative and Quantitative Ranking Systems

Boshkov and Wright

The classification system proposed by Boshkov and Wright (1973) was one of the first qualitative classification schemes developed for underground method selection (Table 6.3-2). Their system, which assumes that the possibility of surface mining has already been eliminated, uses general descriptions of the ore thickness, ore dip, and strength of the ore and walls to identify common methods that have been applied in similar conditions. The results of this classification provide up to four methods that may be applicable.

Hartman

Hartman (1987) has developed a flow-chart selection process for defining the mining method, based on the geometry of the deposit and the ground conditions of the ore zone (Figure 6.3-14). This system is similar to that proposed by Boshkov and Wright but is aimed at more specific mining methods. Hartman admits the method is qualitative and should be used as a first-pass approach. This classification includes surface and underground methods, coal, and hard rock.

Morrison

The classification system proposed by Morrison (1976) divides underground mining into three basic groups: (A) rigid pillar support, (B) controlled subsidence, and (C) caving (Figure 6.3-15). General definitions of ore width, support type, and strain energy accumulation are used as the criteria for determining a mining method. This classification helps to demonstrate the selection continuum, choosing one method over another based on the various combinations of ground conditions. In this system, the ground conditions have already been evaluated to determine the type of support required.

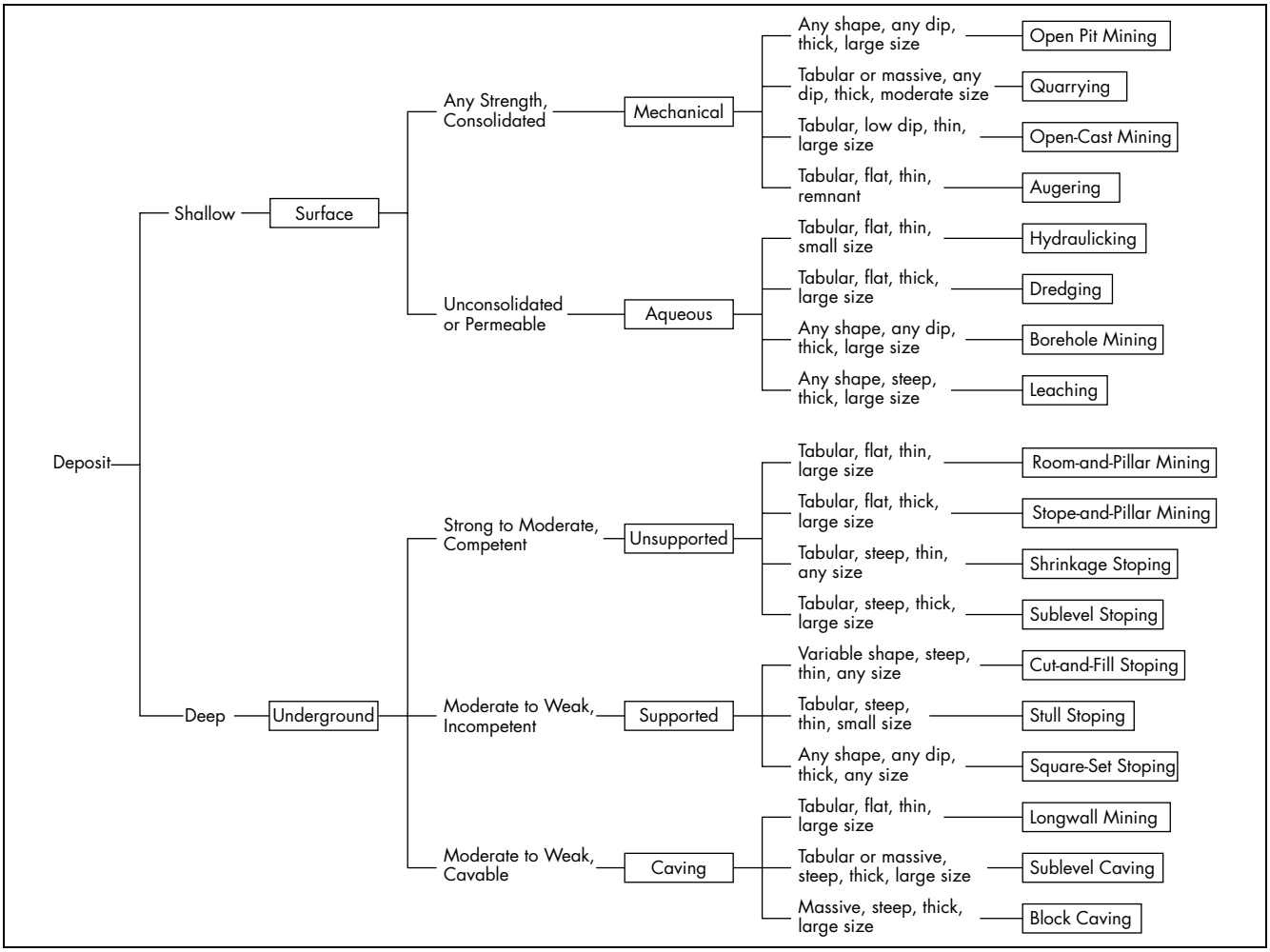
Laubscher

A process for the selection of an appropriate mass underground mining method has been presented by Laubscher (1981). The selection process is based on his rock mass classification system, which adjusts for expected mining effects on the rock mass strength. Laubscher's scheme is aimed at the mass mining methods, primarily block caving versus stoping, and his main emphasis is on cavability. The two parameters that determine whether a caving system is used over a stoping system are the degree of fracturing, RQD, joint spacing, and the joint rating, which is a description of the character of the joint—that is, waviness, filling, and water conditions (Figure 6.3-3). This scheme puts emphasis on the jointing as the only control for determining cavability. More recently, Laubscher (1990) has modified the classification to relate his rock mass rating to the hydraulic radius (Figure 6.3-16). By including the hydraulic radius, cavability becomes feasible for more competent rock if the area available for undercutting is large.

Table 6.3-2 Boshkov and Wright classification system

Type of Ore Body	Dip	Strength of Ore	Strength of Walls	Commonly Applied Methods of Mining
Thin beds	Flat	Strong	Strong	Open stopes with casual pillars
				Room-and-pillar
				Longwall
		Weak or strong	Weak	Longwall
Thick beds	Flat	Strong	Strong	Open stopes with casual pillars
				Room-and-pillar
		Weak or strong	Weak	Top slicing
				Sublevel caving
		Weak or strong	Strong	Underground glory hole
Very thick beds	NA*	NA	NA	Same as for "Masses" below
Very narrow veins	Steep	Strong or weak	Strong or weak	Resuing
Narrow veins (widths up to economic length of stull)	Flat	NA	NA	Same as for thin beds
	Steep	Strong	Strong	Open stopes
				Shrinkage stopes
				Cut-and-fill stopes
			Weak	Cut-and-fill stopes
				Square-set stopes
		Weak	Strong	Open underhand stopes
				Square-set stopes
			Weak	Top slicing
				Square-set stopes
Wide veins	Flat	NA	NA	Same as for thick beds or masses
	Steep	Strong	Strong	Open underhand stopes
				Underground glory hole
				Shrinkage stopes
				Sublevel stoping
				Cut-and-fill stopes
				Combined methods
			Weak	Cut-and-fill stopes
				Top slicing
				Sublevel caving
				Square-set stopes
				Combined methods
		Weak	Strong	Open underhand stopes
				Top slicing
				Sublevel caving
				Block caving
				Square-set stopes
				Combined methods
			Weak	Top slicing
				Sublevel caving
				Square-set stopes
				Combined methods
Masses	NA	Strong	Strong	Underground glory hole
				Shrinkage stopes
				Sublevel stoping
				Cut-and-fill
				Combined methods
	NA	Weak	Weak or strong	Top slicing
				Sublevel caving
				Block caving
				Square-set stopes
				Combined methods

Source: Boshkov and Wright 1973.
*NA = not applicable.



Source: Hartman 1987, reproduced with permission of John Wiley and Sons, Inc.

Figure 6.3-14 Hartman chart for the selection of mining method

Nicholas

The classification proposed by Nicholas (1981) determines feasible mining methods by numerical ranking and thus is truly quantitative. The first step is to classify the ore geometry and grade distribution using Table 6.3-3.

The rock mechanics characteristics of the ore zone, hanging wall, and footwall are similarly classified using Table 6.3-4.

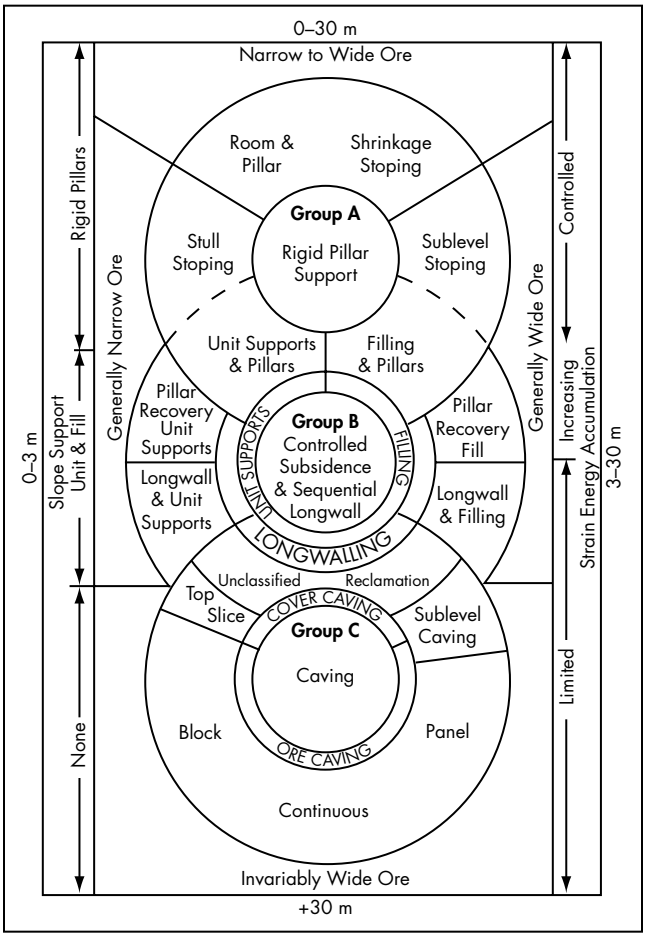
A numerical ranking is then performed by adding up the values of each mining method, using Tables 6.3-5 and 6.3-6. The values of the tables represent the suitability of a given characteristic for a particular mining method. A value of 3 or 4 indicates that the characteristic is preferred for the mining method. A value of 1 or 2 indicates that a characteristic is probably suited to that mining method, while a value of 0 indicates that a characteristic will probably not promote the use of that mining method, although it does not rule it out entirely. A value of –49 would indicate that a characteristic will completely eliminate consideration of that method. A recent modification to the system is the weighting of the categories for the ore geometry, ore zone, hanging wall, and footwall. To give each of these categories equal weight, the ore zone, hanging wall, and footwall need to be multiplied by 1.33. However, the importance of each category is not equal; the ore geometry is more important than the ore zone, which is more important than the hanging wall, which is more important than the footwall.

The proposed weighting for each category, summarized in Table 6.3-7, can be changed based on personal experience. The net weighting is then multiplied by each of the categories. Those two or three (as in the case of Table 6.3-7) mining methods that have the highest overall numerical ranking should be economically analyzed.

The proposed values for the characteristics can be changed as the technical expertise with mining equipment and mining processes improves. In addition, each individual has a different point of view as to the relative importance of the various characteristics for each method.

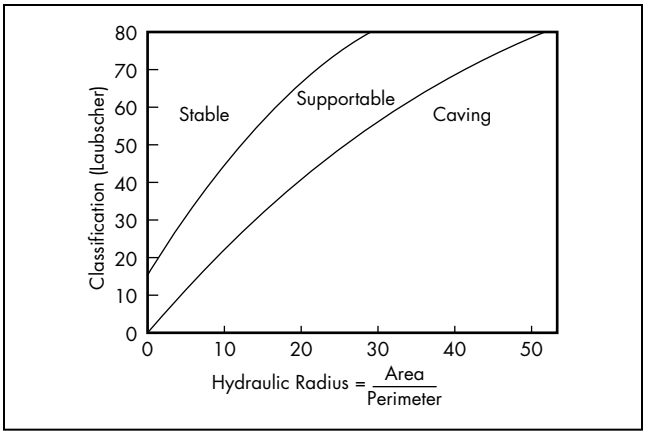
Numerical Evaluation Methodologies

The continuing development of mineral resource modeling and mine planning technologies in the 21st century is providing a range of sophisticated computer-based tools largely unknown in the latter part of the 20th century. Without exception, these mine planning technologies rely on numerical resource models, most commonly some sort of a block model, whereby the mineral deposit and the surrounding host rock is discretized into uniform orthogonal blocks with grade



Source: Morrison 1976.

Figure 6.3-15 Morrison's classification system



Source: Nicholas 1992.

Figure 6.3-16 Laubscher's 1990 classification system

Table 6.3-3 Definition of deposit geometry and grade distribution

General shape/width	
Equi-dimensional	All dimensions are on same order of magnitude.
Platy–tabular	Two dimensions are many times the thickness, which does not usually exceed 100 m.
Irregular	Dimensions vary over short distances.
Ore thickness	
Narrow	<10 m
Intermediate	10–30 m
Thick	30–100 m
Very thick	>100 m
Plunge	
Flat	<20°
Intermediate	20°–55°
Steep	>55°
Depth below surface	Provide actual depth.
Grade distribution	
Uniform	Grade at any point in deposit does not vary significantly from mean grade for that deposit.
Gradational	Grade values have zonal characteristics, and the grades change gradually from one to another.
Erratic	Grade values change radically over short distances and do not exhibit any discernible pattern in their changes.

Source: Nicholas 1981.

Table 6.3-4 Rock mechanics characteristics

Rock Substance Strength (uniaxial strength/overburden pressure)		
Weak	<8	
Moderate	8–15	
Strong	>15	
Fracture Frequency	**No. of Fractures/m**	**% RQD**
Very close	>16	0–20
Close	10–16	20–40
Wide	3–10	40–70
Very wide	<3	70–100
Fracture Shear Strength		
Weak	Clean joint with smooth surface or fill with material with strength less than rock substance strength	
Moderate	Clean joint with rough surface	
Strong	Joint filled with material that is equal to or stronger than rock substance strength	

Source: Nicholas 1992.

distributions determined by using sophisticated geostatistical analysis and interpolation algorithms.

With rare exception, determination of the mining limits for mineral deposits amenable to open mining methods is evaluated by applying a profit maximization algorithm with the most commercially available packages based on developments of the Lerchs–Grossmann graph theory. The technology relies implicitly on block model representations of the mineral resource and, when combined with linear programming technology to evaluate the optimum mining sequence for the proposed open-pit excavation, represents the minimum acceptable approach to the evaluation of open-pit prospects at any point in the evaluation cycle (concept, prefeasibility, or final feasibility study). Similar technologies are becoming

Table 6.3-5 Ranking process for grade and geometry (values)

	General Shape*			Ore Thickness†				Ore Plunge‡			Grade Distribution§		
Mining Method	M	T/P	I	N	I	T	VT	F	I	S	U	G	E
Open-pit mining	3	2	3	2	3	4	4	3	3	4	3	3	3
Block caving	4	2	0	-49	0	2	4	3	2	4	4	2	0
Sublevel stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sublevel caving	3	4	1	-49	0	4	4	1	1	4	4	2	0
Longwall mining	–49	4	–49	4	0	–49	–49	4	0	–49	4	2	0
Room-and-pillar mining	0	4	2	4	2	–49	–49	4	1	0	3	3	3
Shrinkage stoping	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut-and-fill stoping	0	4	2	4	4	0	0	0	3	4	3	3	3
Top slicing	3	3	0	–49	0	3	4	4	1	2	4	2	0
Square-set stoping	0	2	4	4	4	1	1	2	3	3	3	3	3

Source: Nicholas 1992.
*M = massive, T/P = tabular or platy, I = irregular.
†N = narrow, I = intermediate, T = thick, VT = very thick.
‡F = flat, I = intermediate, S = steep.
§U = uniform, G = gradational, E = erratic.

Table 6.3-6 Ranking process for rock mechanics

	Rock Substance Strength*			Fracture Spacing†				Fracture Strength*		
Mining Method	W	M	S	VC	C	W	VW	W	M	S
Ore Zone										
Open-pit mining	3	4	4	2	3	4	4	2	3	4
Block caving	4	1	1	4	4	3	0	4	3	0
Sublevel stoping	–49	3	4	0	0	1	4	0	2	4
Sublevel caving	0	3	3	0	2	4	4	0	2	2
Longwall mining	4	1	0	4	4	0	0	4	3	0
Room-and-pillar mining	0	3	4	0	1	2	4	0	2	4
Shrinkage stoping	1	3	4	0	1	3	4	0	2	4
Cut-and-fill stoping	3	2	2v	3	3	2	2	3	3	2
Top slicing	2	3	3	1	1	2	4	1	2	4
Square-set stoping	4	1	1	4	4	2	1	4	3	2
Hanging Wall										
Open-pit mining	3	4	4	2	3	4	4	2	3	4
Block caving	4	2	1	3	4	3	0	4	2	0
Sublevel stoping	–49	3	4	–49	0	1	4	0	2	4
Sublevel caving	3	2	1	3	4	3	1	4	2	0
Longwall mining	4	2	0	4	4	3	0	4	2	0
Room-and-pillar mining	0	3	4	0	1	2	4	0	2	4
Shrinkage stoping	4	2	1	4	4	3	0	4	2	0
Cut-and-fill stoping	3	2	2	3	3	2	2	4	3	2
Top slicing	4	2	1	3	3	3	0	4	2	0
Square-set stoping	3	2	2	3	3	2	2	4	3	2
Footwall										
Open-pit mining	3	4	4	2	3	4	4	2	3	4
Block caving	2	3	3	1	3	3	3	1	3	3
Sublevel stoping	0	2	4	0	0	2	4	0	1	4
Sublevel caving	0	2	4	0	1	3	4	0	2	4
Longwall mining	2	3	3	1	2	4	3	1	3	3
Room-and-pillar mining	0	2	4	0	1	3	3	0	3	3
Shrinkage stoping	2	3	3	2	3	3	2	2	2	3
Cut-and-fill stoping	4	2	2	4	4	2	2	4	4	2
Top slicing	2	3	3	1	3	3	3	1	2	3
Square-set stoping	4	2	2	4	4	2	2	4	4	2

Source: Nicholas 1992.
*W = weak, M = moderate, S = strong.
†VC = very close, C = close, W = wide, VW = very wide.

Table 6.3-7 Weighting factors

Mining Method	Weighting		
Ore geometry	1.0	1.0	1.0
Ore zone ground conditions	1.33	0.75	1.0
Hanging wall ground conditions	1.33	0.6	0.8
Footwall ground conditions	1.33	0.38	0.5

Source: Nicholas 1992.

available for the evaluation of optimized mine plans for underground deposits.

The type of tool selected will depend on the required confidence in the mine planning outcome, which is also typically related to the position in the deposit evaluation cycle. Importantly, all of these technologies facilitate rapid deposit evaluations (compared to classic uncomputerized mine planning methodologies). This speed and facility frequently allows for the analysis of subtle variations within a mining method, which could well mean the difference between a successful or an unsuccessful development for large low-margin deposits.

Risk

Increasingly, business investment decision making requires quantification of the risks and uncertainties associated with a mine plan and particularly with the value of the proposed investment, in which an investment opportunity is competing against many other investment opportunities. Ample evidence within the global mining industry demonstrates that the highest-ranked risk to investment value is the quantity and quality of the mineral resource (the key revenue drivers) on which deterministic mine plans are based.

Consequently, the trend is toward the preparation of probabilistic resource models in mine planning evaluations, both to describe the probability-weighted estimate of the value of a mineral deposit constrained by deterministic limits and to describe the value of a mineral deposit reflecting the probabilities of different mineral limits and grade distributions. These latter techniques, sometimes known as resource range analyses, are particularly useful to investment decision-making processes in the early stages of exploration of a mineral deposit and are supplanted by the probabilistic resource models as the exploration process matures.

In its simplest form, a resource range analysis for a mineral deposit comprises the development of five equally plausible mineral resource models, sometimes characterized as the minimum, low, most likely, high, and maximum cases, with probabilities assigned to each case such that the cumulative probability equals unity. In some instances, three models reflecting the low, most likely, and high cases may suffice. Similarly, a mine plan for each model of the mineral resource is prepared and evaluated with a probability weighting to derive an expected value. The different resource models may well dictate that materially different mining methods are selected for the evaluation. The significant increase in mine planning effort arising from these approaches manifestly relies on the emergence of numerical evaluation methodologies previously described.

Economic Analysis

Every corporation has a range of metrics in which it is interested when evaluating a mineral deposit. These could include issues such as scale and the level of participation in the market for the product, the likely life of the mining project, the size of the investment required to bring the deposit into production, and similar physical metrics. Invariably, this must also include financial metrics. With little exception, the value of prospective future cash flows, discounted at an appropriate rate as a *present value* and netted against the investment required so as to describe a net present value (NPV), will also be required, particularly where competing investment alternatives must be assessed.

In conducting any economic analysis of a mineral project, definition of the future cash flows should consider the four basic cash streams.

1. **Product income:** the incoming cash stream that describes income from the sale of product, perhaps net of royalties, as may be considered appropriate.
2. **Capital cost expenses:** the cash stream representing the outgoing expenditures required to develop the mining project and create an income stream. Sustaining capital expenditures should also be regarded.
3. **Operating cost expenses:** the outgoing cash stream representing the expenditure on inputs to the production process incurred to produce units of output and generate an income stream.
4. **Tax expenses:** the outgoing cash stream representing the payment of income taxes and any offsetting depreciation allowances.

An accumulation of these four cash streams in any accounting period, typically years, provides the basis for describing discounted cash flows and NPV. Probabilistic evaluations should also be completed applying stochastic simulations for the income stream and mine plans that examine optimistic, most likely, and pessimistic scenarios for the capital cost and operating cost streams.

Stages in Application of Mining Method Selection Process

Application of any mining method selection process is necessarily iterative and, as knowledge about the deposit under consideration evolves, may have different conclusions.

Deposit Discovery

The first occasion on which evaluation practitioners may be called upon to provide a business case to justify further exploration expenditure is shortly after the emergence of promising exploration results. Enthusiasm for further drilling is inevitably high and, depending on the prevailing economic environment and the quality of the exploration results, may overshadow a more rational assessment of the potential economic value. This is an ideal time to apply a resource range analysis methodology to establish a range of resource models (and conceptual mine plans for each model) and, more importantly, to establish the business case for further investment in exploration relative to the corporation's business goals and required deposit size.

In this circumstance, the mineral resource models are likely to be simple polygonal models that give some sense of the shape and geometry. The conceptual mine plans are unlikely to be any more than high-level estimates of output that have been extended by relevant unit costs for the expected activities, perhaps applying many rules of thumb for mining and processing activities.

Advanced Exploration Concept Studies

The term *advanced exploration* here implies that preliminary mineral resource modeling has been completed and that the

mineral resource estimates emerging from the process are able to be classified by one of the internationally recognized systems, typically at least as an *inferred mineral resource*. The presumption is that the mineral resource model is based at least on block modeling technology with interpolated mineral concentrations, with the model extended to describe the surrounding host rock (for the purpose of modeling slopes) in the case of deposits requiring open-pit evaluations.

In this case, one of the readily available open-pit optimization software systems should be applied to evaluate the deposit. For underground deposits, similar systems are starting to become available, and these will evolve over time with industry research and development programs.

Many unvalidated critical assumptions may well need to be made about a number of mining method alternatives, derived by applying the methodologies previously outlined. Because of the relatively high levels of uncertainty surrounding many of the key drivers, resource range analysis methodologies are extremely useful in helping to establish the impacts of uncertainty on project value and identifying the priorities for resolving these uncertainties to more acceptable levels. In most circumstances, a risk-weighted NPV greater than zero should be sufficient to encourage further investigation by way of preliminary feasibility studies.

Preliminary Feasibility Studies

In a mineral project development, the advance to a preliminary feasibility study usually signifies a stronger commitment to drilling and sampling the mineralization, complemented by further rounds of mineral resource modeling. Hydrogeological, geotechnical, geometallurgical, and environmental studies are also likely to proceed, welded together under a strategic mine planning umbrella. Improved resource estimate confidence is likely. Mining method studies are likely to be of less significance, as the focus is on variations within the selection of methods identified at the concept study stage. In most cases, the study effort is now focused on ensuring that all reasonable alternatives are investigated with the intent of identifying the best alternative to move forward to a final feasibility study.

Final Feasibility Studies

For final feasibility studies, the focus usually shifts from mineral resource definition to detailed mine and infrastructure planning at a level that is adequate for the capital expenditure approvals necessary to take the project into construction. Evaluation of alternative mining methods should never figure in this stage of project evolution.

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CHAPTER 6.4

Selection Process for Underground Soft-Rock Mining

Antonio Nieto

INTRODUCTION

The soft rocks usually are part of the sedimentary minerals classification, which is subdivided into clastic, organic, and chemical. Examples of the soft-rock ores include coal, metalliferous shales, oil shales, potash, salt, trona, and possibly kimberlites. Where coal, metalliferous shales, potash, and trona occur as economic ores, they are typically laterally extensive beds in a nearly horizontal inclination but with, at most, a shallow dip angle. This differentiation is key, because it enables the application of large-scale mechanization to the mining process. The economy of scale that results from mechanization is often the determinant factor for economic success. As the capabilities of mechanical cutting expand into more demanding applications, the possibility exists that ores previously considered hard-rock deposits, such as the platinum-bearing reefs in southern Africa, may be cut instead of blasted.

Coal has its origins in the accumulation of plant debris that becomes buried by sediments. Through a process dependent on time, burial depth, and chemical transformation, the plant debris becomes coal. Therefore, coal is classified as an organic sedimentary mineral. Coal varies in quality from lignite to anthracite, with sub-bituminous and bituminous ranked as intermediate in the progression and the most commonly mined types worldwide.

Coal is primarily used for electricity generation and steelmaking and is commonly referred to, respectively, as *steam coal* and *coking coal*. Coal with attributes such as an appreciable free-swelling index (FSI) is used to make coke, which is used in primary production of steel from iron ore. The scarcity of such coals elevates their value compared to steam coal.

Trona is a carbonate mineral of sodium used to form soda ash, used in glassmaking and other industrial processes, including baking soda. It occurs naturally in a few locations worldwide as laterally extensive evaporite deposits, suitable for underground mining methods. It is a moderate value ore and competes with an alternative process that synthesizes the same product from chemical feedstocks.

Potash is the name loosely applied to a variety of potassium salts, particularly potassium chloride, which are encountered in laterally extensive evaporite deposits found worldwide. These are often associated with intermixed or stratigraphically adjacent halite (sodium chloride). Potassium is key to plant growth, and potash is mainly used as fertilizer, although it is also used to produce soaps, ceramics, and drugs, among others.

Coal, potash, trona, and salt are the principal soft-rock ores and, within limits, share similar production methods that focus on economies of scale. The most common mining techniques for soft-rock ores are longwall, room-and-pillar (R&P), and stope-and-pillar. For water-soluble minerals, solution mining is an alternative. The process of properly selecting an underground mining method for a particular ore deposit is critical to the ultimate success of the operation. An improperly selected method will increase costs, lower productivity, create unnecessary hazards, and reduce resource recovery. Due to the complex nature of ore bodies, no two mines are completely alike, and all operations must adapt to the particular conditions of their deposits.

ORE DEPOSIT CHARACTERISTICS

Numerous considerations must to be recognized when selecting the best method to mine a soft-rock ore deposit. Some of the considerations are based on ore deposit characteristics favorable to the mining method being considered:

- Ore strength
- Host rock strength
- Deposit shape
- Deposit dip
- Deposit size
- Deposit thickness
- Deposit grade
- Ore uniformity
- Deposit depth

Other characteristics are a function of mining method:

- Operating cost
- Capital cost and development timing
- Production rate
- Mechanization
- Selectivity and flexibility

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- Health and safety
- Environmental effects

Ore Strength

The material properties of the ore often drive mine design decisions. Although there are many mechanical properties, compressive strength is often discussed as an indicative characteristic inferring structural performance and suitability for mechanical cutting. Mining methods such as R&P and stope-and-pillar depend on the strength of the ore rock to support the roof and overburden in order to create a structurally stable excavation. In soft-rock applications, the relative strength of the ore is often weak, with a compressive strength less than 6,000 psi. This low strength is generally associated with a low to moderate specific energy of cutting (kilowatt-hour/ton). This allows the application of mechanized cutting and loading, which is elemental to the success of many modern mines. As the ore strength increases, the options for mechanical cutting are reduced, and the application productivity declines while costs increase. There is a marked difference between the cost and productivity performance of mechanized cutting versus drilling and blasting in the majority of soft-rock applications, with mechanized methods decidedly preferred.

Table 6.4-1 gives the strength designations and ranges of values based on the compressive strength of the material.

It is important to note that the strength and mechanical properties of a rock are significantly affected by fracturing and planes of weakness in the deposit. Fracturing is characterized by small discontinuities in the rock mass and may be caused by heat, vapor expansion (as in porphyry deposits), depositional conditions (i.e., slickensides), or tectonic movement (faults). Cleat is a fracture system ordinarily observed in coal. Two different fracture directions are typically present: face cleat (primary direction) and butt cleat (secondary direction). During exploration, the degree of fracturing should be quantified and utilized to reduce ore structural properties, potentially leading to smaller openings, larger pillars, and increased ground control costs. Limited fracturing may be a positive factor for some mining methods, because it promotes caving, lowers blasting requirements, and aids mechanical cutting. However, excessive fracturing can have a negative influence on ground control, water, and gas inflows.

Host Rock Strength

The strength of the rock enclosing the ore is also an important driver in mining method selection. Temporary and permanent openings must be developed either in the host rock, in order to access the ore, or with the host rock as roof (back or hanging wall) or floor (footwall) for the ore openings (entries or crosscuts). To execute an appropriate design, the material properties must be understood. The behavior of the roof and floor can be pivotal in the success of mechanized mining systems. Floors that become muddy and easily rutted can disable production and send maintenance costs skyrocketing.

It is inaccurate to assume that the ore and host material will have the same characteristics, so each must be independently characterized by geomechanical testing.

Deposit Shape

Ore deposits are classified into two broad categories: tabular and massive. A tabular deposit is flat and thin, and has a broad horizontal extent. This classification typically refers to materials formed by sedimentation. Similar in shape to tabular ore bodies, lenticular deposits are shaped like lenses and lack the large areal extent of most tabular deposits. Most methods designed to exploit tabular deposits may be adapted to mine lenticular ones. The ore materials must often be of higher value than applications such as coal, because production costs are generally higher but reserve tonnages are lower.

A massive deposit may possess any shape. The ore is often distributed in low concentrations over a wide area with varying horizontal and vertical extents. Frequently, the difference between ore and waste may be a function of grade rather than rock type. Massive deposits may be unpredictable and require a considerable exploration investment in order to document and fully understand the resource. For the purposes of mining method selection, massive deposits are often accompanied by a more specific clause like "massive with large vertical extent." These additions are necessary because the shape of a massive deposit is variable and may be unsuitable for certain mining methods. The deposit shape definitions are summarized in Table 6.4-2.

Deposit Dip

Dip is defined as the angle of inclination of a plane measured downward, perpendicular to the strike direction. The deposit dip is more relevant to tabular ore bodies than massive ones, although it may sometimes be a consideration for the latter. Deposit dips are categorized and defined in Table 6.4-3.

Both flat-lying soft-rock ore beds and near-vertical ore veins may be classified as tabular, but the mining methods used to exploit them are dramatically different. Several methods are highly dependent on gravity for material flow and cannot function in flat-lying deposits. Alternatively, low working slopes are a key factor in the application of mechanization for cutting and loading as well as material haulage by rubber-tired, rail, or conveyor-belt methods.

Table 6.4-1 Ore strength definitions

Relative Strength	Example Material	Compressive Strength, psi
Very weak	Coal	<6,000
Weak	Shale, siltstone, sandstone	6,000–14,500
Moderate	Limestone and sandstone	14,500–20,000
Strong	Granite	20,000–32,000
Very strong	Quartzite	>32,000

Table 6.4-2 Deposit shape definitions

Deposit Type	Shape	Width	Extent
Tabular	Flat	Thin to moderate	Horizontal
Lenticular	Flat, elliptical	Thin to moderate	Horizontal
Massive	Any	Thin to thick	Horizontal and vertical

Table 6.4-3 Deposit orientation definitions

Inclination Category	Dip Angle, degrees
Low	0–5
Moderate	5–25
Fairly steep	25–45
Steep	45–90

Table 6.4-4 Tabular deposit thickness definitions

Deposit Thickness (T)	
Thin (small)	T < 5 ft
Moderate	5ft < T < 12 ft
Fairly thick	12 ft < T < 20 ft
Thick (large)	T > 20 ft

Table 6.4-5 Ore grade definitions

Grade	$/ton
Low	10–50
Moderate	50–250
High	>250

Deposit Size

The volumetric size of an ore body must also be considered. Several of the methods discussed in this chapter rely on large deposits with long mine lives to justify their high initial capital costs and promote economies of scale. Other methods simply do not work efficiently in ore bodies, which are either too large or too small. Deposit size is characterized subjectively by the terms *small*, *medium*, and *large*. As a generalization, large ore deposits have tens to hundreds of million cubic yards of ore and suggest mine lives in the 10- to 50-year range.

Deposit Thickness

Deposit thickness refers to the ore thickness of tabular deposits. Thickness plays an important role in opening stability and may prevent certain equipment from functioning efficiently or mining methods from being effective. The deposit thickness (nominally the mining extraction height) definitions are listed in Table 6.4-4. These definitions are most relevant to mechanical cutting and loading applications, such as longwall or continuous miner R&P. The thickness ranges roughly correlate with the types of equipment available to implement a mining system and the cost/productivity that might be expected.

Deposit Grade

Grade is discussed in terms of the amount/value of recoverable/salable material in a unit weight or volume of in-place mineral resource. Where it becomes economically viable to produce the mineral resource, the in-place resource becomes ore. As such, the end-outcome economics of different mining methods may vary the amount of ore that an in-place mineral resource may yield. A gold ore may contain as low as 0.1 oz/ton and still be economic, whereas iron ore grades may approach 60% by weight. Coal is generally characterized by its attributes—that is, energy content (Btu/lb); percentage of ash, moisture, and sulfur; FSI; and so forth. Some mining methods with high operating costs necessitate high-grade ores in order to be economic. Large-scale methods may be suitable for large, low-grade deposits, such as bituminous coals. Ore grades are categorized subjectively and must be investigated on an individual site basis. Ore grade definitions are provided in Table 6.4-5. Value estimates associated with the classifications give some relative sense of the range involved.

Ore Uniformity

The uniformity of the ore in the mineral deposit must be considered, as poor uniformity may render some mining methods unviable. It is undesirable to excavate subeconomic material, unless it is necessary to reach ore or create necessary infrastructure, such as belt-conveyor galleries. A mineral deposit may be segmented by faults, subeconomic mineral occurrence, or legal/environmental issues. Some mining methods are well suited to flexibility because they can selectively extract specific sections of a deposit without disrupting the overall operation. An example of this is the case where an R&P coal mine adapts the panel geometry while in panel to reflect new findings about unsatisfactory coal quality, adverse roof conditions, or insufficient coal thickness.

Other methods, such as longwall mining, limit selectivity and must produce at least some amount of material leading to equipment advance in order to continue to the panel's intended end. Faults with significant displacements compared to the bed thickness can seriously disrupt longwall or R&P operations. In some areas, igneous or sedimentary materials may be injected into tabular deposits, such as dolerite dykes in coal seams, and create impediments to mechanized cutting and loading. An inconsistent feed of material may disrupt processing plant performance or require blending, rehandling, or disposal of mined material. These situations can be anticipated and minimized with a thorough knowledge of the ore body's uniformity. Ore uniformity designations are

- Variable,
- Moderate,
- Fairly uniform, and
- Uniform.

Deposit Depth

Another deposit-related consideration that impacts mining method selection is ore deposit depth relative to the surface. Shallow deposits are generally more suited for surface mining. Deeper deposits may require progressively greater ground control measures (increased costs), larger pillar sizes (lower recovery), or decreased applicability of some mining methods in order to ensure safety and sustainability. Commonly applied variations of R&P or longwall mining occur over deposit depth ranges from 250 to 3,500 ft. The definition of shallow/moderate/deep is relative depending on the value of the ore and the strength of the material. A deep coal mine might have workings to a depth of 3,500–4,500 ft. Alternatively, a deep gold mine producing from a meta-quartzite reef might have workings to nearly triple that depth. Classification for deposit depths are shallow, moderate, and deep.

MINING METHOD CHARACTERISTICS

Every mining method has characteristics that will produce different outcomes based on the ore deposit to be mined. As such, prior to selecting the best mining method, the methods to be applied and their expected outcomes must be clearly understood.

Operating Cost

The operating cost of a mine is the cost associated with the production of ore from the primary mining method. The total cost is higher and incorporates items such as depreciation, depletion, taxes, and royalties. The operating cost divided by the number of salable units of production mined creates a metric used to compare efficiency between competing production alternatives—that is, $/ton. When the total cost is the basis of the metric, it can indicate the potential viability of the project

in total. In mining, the operating cost is composed of fixed and variable expenses. Variable expense totals change in proportion with activity, such as roof-control cost ($/ft) that typically accumulates with the amount of entry development. In comparison, fixed costs, such as labor and ventilation, stay relatively constant over a moderate range of activity variation. Some methods are labor intensive or may require a large quantity of materials in order to operate, thereby necessitating valuable ores to compensate for the greater price of extracting them. Other methods cost little once implemented but have high initial capital costs. These methods, such as longwall mining, may be able to excavate large low-grade deposits economically.

Capital Cost and Development Timing

Initial capital cost is defined as the amount of investment needed before the mine begins to generate revenue. A small quarry excavating an outcropping limestone bed has little capital cost because it can start extracting ore almost immediately with little investment in equipment. Alternatively, a deep potash mine might have to sink one or more shafts beyond a depth of 3,000 ft, build a surface plant, and implement a mechanized mining equipment fleet to produce the first salable ton of product. Thus, first production may come after several years and tens to hundreds of million dollars have been committed.

Higher capital costs are frequently associated with long development or start-up times. Equipment manufacturers often have wait times of months or even years before assembly and delivery of new equipment. Typically, this equipment is customized for the mine-specific application.

Production Rate

The production rate of a mine is highly dependent on the mining method. A high production rate can accommodate a large market and may overcome low-value ore if operating costs are low. The ability to stockpile and blend ores of varying grade in order to maintain a consistent feed to the mill is typically advantageous. Higher production is generally more desirable because mines are rarely opened in areas where selling more product is disadvantageous. The economics of mines that can sell product up to the limit of their production capacity are drastically different than mines that can produce at levels above what their markets can consume. In the latter case, production enhancement proposals readily embraced by the former case, intended to distribute fixed costs over a larger total production, are rejected, and the focus sharpens on costs contributing to the fixed component of operating cost.

Mechanization

Mechanization is a critical element of a modern mine. Utilizing machines to perform production tasks is much safer and more efficient, in cost or production performance, than using manual labor. To justify a large capital investment in equipment, it is common to need a longer mine life and thus a larger ore body. Highly mechanized mining is safer than less mechanized methods because fewer workers will be needed and thus the overall hazard exposure will be lower. Several methods lend themselves to a high degree of mechanization, including longwall and continuous miner R&P methods.

Selectivity and Flexibility

Selectivity and flexibility can significantly contribute to the success of a mining method. It is generally valid to assume that mining conditions, market prices, and technology will change over the course of a mine's life, so the mining method must be adaptable to these fluctuations. Sacrificing optional alternatives in any mining method is not desirable unless there is compelling reason to do so. If commodity prices were to drop substantially, a portion of the ore in a massive deposit may become uneconomic to mine. If the mining method is able to bypass the low-grade sections and continue mining economic material, the mine will continue to be successful.

Health and Safety

The safety and health of a mine's workers should be the top priority of every operator. Several methods are inherently safer than others, because the openings are more stable or personnel are less likely to be subjected to hazardous conditions. Although no modern methods are considered to be unsafe, it bears mentioning that specific health and safety concerns are often mitigated by the mining method selection. Longwall mining is recognized as the safest method of mining applied to soft-rock deposits.

Environmental Effects

The largest environmental impacts of an underground mine typically fall into three categories: subsidence, groundwater, and atmospheric emissions. Subsidence is defined as the sinking of the surface above mine workings as a result of material settling into the voids created by mineral extraction. It is contentious in urban or suburban areas where it can affect homes, schools, and roads. The surface subsidence created by modern longwall mines is largely predictable in its timing and magnitude, in contrast to the unpredictable outcomes associated with some R&P mines. In this way, longwall subsidence is less hazardous to human-made surface structures, because impacts occur soon after mining and rarely change much after initial stabilization. This allows remediation of surface damage in a time contemporary with mining.

Most areas with a history of mining also have developed legal processes to address damage from mining-induced subsidence. High-extraction mining methods will foreseeably induce surface subsidence. If selected, provisions must exist to mitigate or remedy damages.

Water impacts may arise by accidental causes. Acid-generating rock of multiple types in excavated ore, waste, or overlying strata may produce acid mine drainage. Water produced by the rock mass and mining process must be afforded appropriate controls, as it will be necessary to keep the mine drained. In all cases, strict controls must be effectively applied to mitigate groundwater or surface-water impacts by mining-related water discharges.

Air quality in underground mines is typically affected by the natural liberation of mine gases (i.e., methane [CH_4], hydrogen sulfide [H_2S], and carbon dioxide [CO_2]), blasting by-products and equipment emissions (i.e., nitrogen oxides [NO_x], sulfur oxides [SO_x], and diesel particulate matter), and mineral dust from ventilation fans. Generally, exposure and emission thresholds exist for these emissions and are strictly applied. In the case of coal dust and methane, special precautions are followed to avoid the hazards of fire and explosion.

Zero harm is a sustainability principle applied by the foremost mining enterprises in the context of health, safety, environment, and communities where mines in their portfolios actively operate. It is an acknowledged goal that communities will be forever improved because of the global and local activities of these mining enterprises.

MINE PLANNING

Mine planning has three well-defined stages in order to have a successful implementation of the project and operation of the mine: identification, selection, and definition.

Identification

The initial assessment is a review of information about the potential mining site and involves the analysis of geographic, geologic, environmental, technical, and economic data. This assessment helps the mining company to evaluate the advantages and disadvantages of the potential site. In this phase, resource that has potential to become ore is characterized and limited mining methods are considered to aid in a coarse valuation of the prospect. At the conclusion of this phase, a limited number of feasible alternatives for exploiting the opportunity should be identified and adequately framed for further evaluation in the selection phase.

Selection

The reserve determination from the identification phase is the basis for semiquantitative mine plan comparisons. Competing mine design alternatives are compared in pro forma economic evaluations and investment performance measures such as net present value, along with scored risk assessments. Uncertainty that leads to variability of outcomes (risk) will be characterized, and mitigating strategies or controls will be developed should the decision to move into the implementation phase be approved. The preferred mine design (in terms of financial value and technical feasibility) results from this stage of planning. At the conclusion of this phase, a single preferred alternative for the mine plan should be selected for optimization in the definition phase.

Definition

In this phase, all of the detailed planning and resource estimation of the prior phases are refined and optimized to deliver a final plan prior to implementation. The success of this phase will define the success or failure of the venture. Gaps in information, inaccurate planning, or even human resource failures can lead to loss of investment, environmental damage, human injury, and negative community impacts. The key to success in execution is to invest in front-end planning and design prior to implementation, which should follow a rigorous plan that includes sufficient contingency and flexibility to manage the variability that is inevitably encountered.

ROOM-AND-PILLAR MINING METHOD

The R&P mining method is a popular mining method for underground mining in tabular and lenticular deposits, as shown in Figure 6.4-1. It is the dominant choice for noncoal underground mining and is frequently applied in coal mines. The concept is to sink a shaft or construct a slope or drift, depending on depth of ore, to the elevation of the mining horizon and begin excavating the ore laterally within the deposit. Where drilling and blasting are not required, the focus of the operation is the continuous miner (Figure 6.4-2), which utilizes a large rotating drum to break the material in front of it. An internal gathering system then loads the broken ore onto an onboard conveyor.

The onboard conveyor feeds onto a shuttle car or articulated hauler, which takes the product to an optional mobile belt feeder. If present, the feeder meters the product onto a conveyor belt, which in turn carries the ore to the surface. Alternatives to shuttle cars and rubber-tired haulers (battery- or diesel-powered) are generally termed *continuous haulage* systems and include bridge conveyors, composed of multiple independent *bridge carriers*, and flexible conveyor trains using a single continuous belt mounted on a mobile base frame, with bends to follow a producing machine.

Roof support is an integral part of the mining process and is usually done with roof bolts and their relative, the roof truss. In *place-change* continuous mining, the continuous miner makes a cut, and roof bolts are installed with a mobile machine called a roof bolter. Of course, ventilation and face drainage are requirements of any mining method. Alternatively, in very weak roof conditions, continuous miners with roof-bolting equipment onboard are common. These machines, called bolter miners, cut and load the ore simultaneously with the installation of roof bolts. This results in *in-place* mining in contrast to the place-changing method. Another variation of continuous miner has cutting

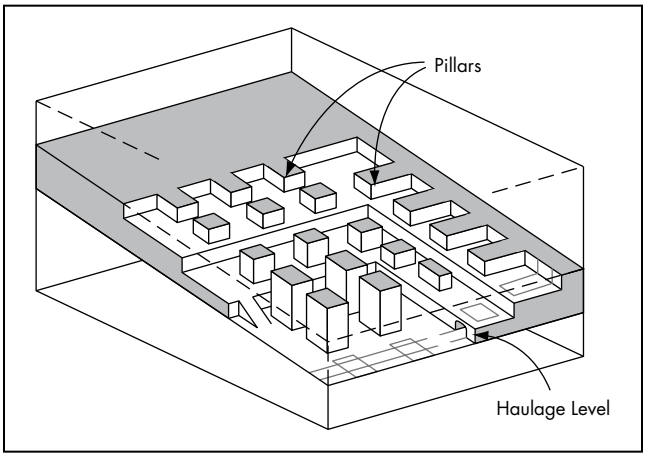
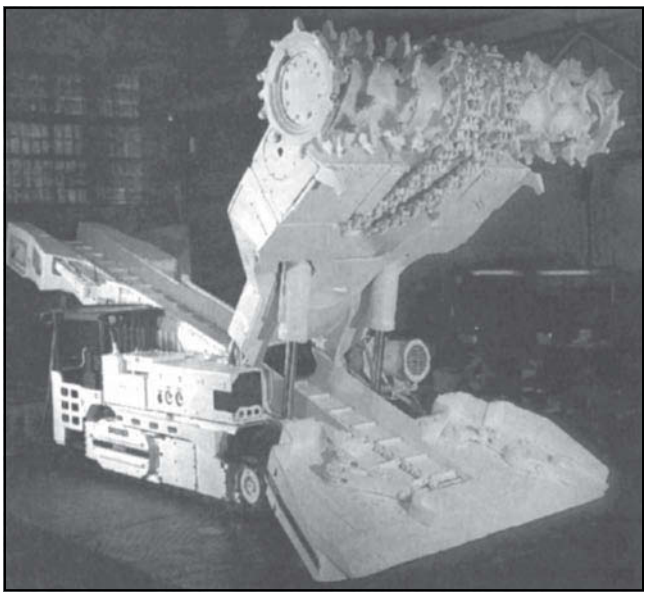


Figure 6.4-1 Room-and-pillar method



Courtesy of Dresser Industries.

Figure 6.4-2 Continuous miner

rotors that rotate parallel to the working face. These *borer miners* are popular in potash and trona mining because of their weight and power. The inability to adequately ventilate methane in the face with borer miners has forced their decline in coal mining. Broken ore haulage can be the same for any of the continuous mining methods, although the combination of in-place mining with continuous haulage methods has some advantages.

In the R&P method, a continuous miner excavates the deposit in a grid-like pattern, driving entries (rooms) approximately 15–20 ft wide at the intended mining height. These openings run parallel to each other along the long axis of the workings. Crosscuts, driven in the same manner at an acute angle to the entries, connect the entries to complete the grid-like pattern. Pillars are left behind to support the roof, hence the term *room-and-pillar*, alternately know as *bord-and-pillar*. The optimal or favorable characteristics for R&P mining are shown in Table 6.4-6.

This mining method is optimal for minerals with lower ore strength such as coal, potash, salt, or trona. R&P mining can be practiced with partial extraction to leave behind larger pillars (lower resource recovery and higher cost) where concerns exist over ground stability or surface subsidence. Alternatively, high-extraction mining can be executed where pillar recovery is done after initial panel development. This method is productive and cost-effective but has rising concerns associated with ground control during the final phases of pillar recovery. Mobile roof supports have been introduced to help mitigate roof-control concerns and reduce logistics related to roof support. Although an improvement, mobile roof supports do not fully address the concern.

Another variation of the R&P method is rib-pillar extraction, where long, narrow pillars are developed and recovered in a progressive process intended to improve safety and productivity. This technique is effective where some attribute of the deposit does not lend itself to efficient longwall operation. Yet another variation of R&P mining, arguably hybridized with longwall mining, is shortwall mining, in which a continuous miner works with shuttle cars and specially designed roof supports similar to longwall shields or chocks. Again, this method has only found successful application in a few cases. For the most part, underground soft-rock operators are migrating to longwall mining, which has left high-extraction R&P mining on the decline where alternatives are available.

The R&P mining method has distinct advantages: the foremost that, with a continuous miner, operations are nearly continuous in nature. Most mining sequences require drilling, blasting, loading, hauling, dumping, and roof support, as well as continuous ventilation and drainage. The invention of the continuous miner eliminated the independent steps of drilling, blasting, and loading, which substantially increases the overall efficiency of the method and improves general productivity. Low operating costs and high production rates are typically associated with modern mechanized R&P mining. Continuous miners can cut through soft-rock deposits, particularly coal, with ease, resulting in rapid development rates. R&P mining is generally more flexible than other methods, because continuous miners can move to other working places within a panel or possibly across a mine with limited difficulty. Also, the grid layout of the mine allows for straightforward ventilation with consistent airflow to all working faces.

The major disadvantage of continuous mining is that it can only be applied to a limited variety of applications. A continuous miner cannot operate efficiently, if at all, in harder rocks like limestone or granite, and, thus, its principal advantages cannot be shared.

Table 6.4-6 Room-and-pillar favorable characteristics

Key Deposit Indicators	Characteristics
Ore strength	Weak to moderate
Host rock strength	Moderate to strong
Deposit shape	Tabular
Deposit orientation	Flat to shallow
Deposit size/thickness	Large, thin
Ore grade	Moderate
Uniformity	Fairly uniform
Deposit depth	Shallow to moderate

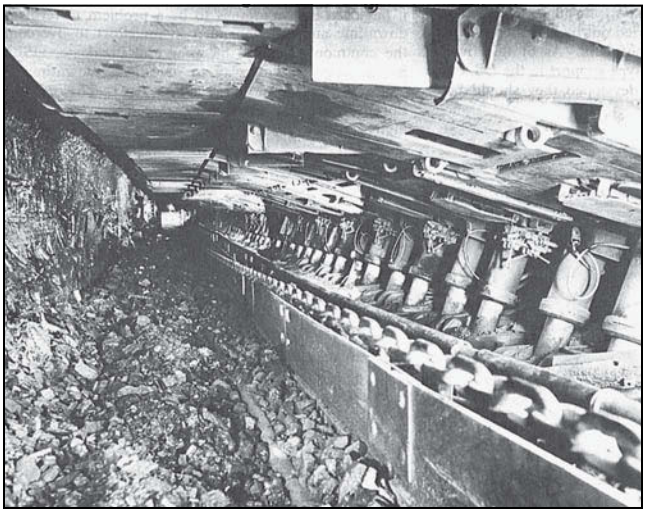
Table 6.4-7 Room-and-pillar advantages and disadvantages

Advantages	Disadvantages
Continuous production	Moderate capital costs
Rapid development rate	Limitation on depth
Excellent ventilation	Moderate selectivity and flexibility
High productivity	Variable subsidence
Moderate operating cost	Higher cost with partial extraction
Good recovery (with pillar extraction)	Moderate recovery (without pillar extraction)

R&P has been used in a variety of soft-rock applications, as well as a few hard-rock mining applications, but on a small scale when compared to coal. To purchase equipment and perform development excavations, R&P requires a moderate capital investment. The method is also limited by depth. The pillar size is dictated by the weight of the overburden above the deposit, so conceptually the deeper the ore body, the larger the pillars must be. Larger pillars result in lower recoveries and overall mining efficiencies. Pillars can be recovered after initial development by utilizing retreat mining (high extraction). The primary advantages and disadvantages of continuous miner–based R&P mining are summarized in Table 6.4-7.

The difference between coal and noncoal production methods are four main factors:

1. **Strength.** Higher strength generally correlates with higher specific energy of cutting and lower productivity in cutting applications.
2. **Scale.** Coal mines are usually larger in throughput than other soft-rock mines because of the need to economically produce a lower value product.
3. **Methane (CH_4).** Where most coal mines are "gassy," many noncoal mines are free of that hazard. In most countries, coal mines and their related equipment are governed by strict regulations designed to prevent methane or coal dust explosions. Large mine explosions are often the product of methane explosions, which entrain coal dust in the air, leading to subsequent and more energetic coal dust explosions in rapid succession. Worldwide, systems involving water or incombustible dust (rock dust) are implemented to prevent coal dust explosions. Spontaneous combustion is also a hazard in many coal mines worldwide and is an attribute of some coal seams, but not all.
4. **Coal workers pneumoconiosis, more commonly known as black lung.** This chronic, debilitating disease is related



Source: Hustrulid 1982.

Figure 6.4-3 Klockner Ferromatik shield roof supports on U.S. longwall faces

to excessive exposure to respirable coal dust, usually during employment in coal mining. The management of dust in coal mining is subject to strict regulations but continues to be an area of industry and regulatory focus.

LONGWALL MINING METHOD

Longwall mining is combined with R&P mining to create some of the most efficient and highest-producing underground mines in the world. First, the main entries are driven with the conventional R&P techniques using continuous miners. A series of panels branching perpendicular from the mains or submains are outlined by a 2–3 entry R&P border, leaving a very large solid block (panel) of ore within its confines. Typical panel dimensions in contemporary coal mines are 800 to 1,400 ft of face length (width) with 6,000 to 15,000 ft of panel length. In coal, panel tonnages of almost 12.1 million metric tons are possible. In modern longwall faces, a shearer, armored face conveyor (AFC), stage loader, and line of powered roof supports (shields) are assembled in a setup room at the beginning of the panel before longwall mining commences (Figure 6.4-3). Utilities used by a longwall include emulsion pumps with capacities of 300–500 gpm at 4,000 to 4,800 psi. Electrical controls powered by 3,300 to 14,400 V provide power to the longwall face equipment at 1,000–4,160 V.

The shearer moves back and forth across the coal block, excavating 100% of the ore within its height capability, causing the material to fall onto the AFC and be transported to the main belt conveyor system via the stage loader, which normally has an integral crusher to provide suitably sized material for conveyor belts. The shields advance sequentially following the shearer to hold up the roof directly above the face equipment and advance the AFC to repeat the cutting cycle. The excavated area behind the shields is allowed to collapse. Retreat of the longwall progresses, as continuous miners develop additional adjacent longwall panels. When the longwall reaches the end of the panel, specialized activities are executed by a carefully choreographed plan to withdraw the longwall equipment from the completed panel and reinstall it in the next panel. During this process, key elements of the equipment are refurbished or exchanged with machine

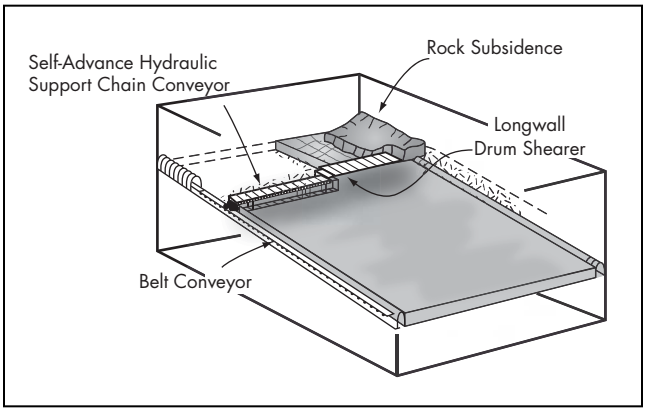


Figure 6.4-4 Longwall method

manufacturers for an already refurbished machine or component. Commonly, shearers, AFC and stage loader components, pumps, and selected electrical equipment are refurbished as necessary to allow high availability in service during the new panel. For world-class longwalls, production can range from 6.6 to 13.2 million tons per year, with unplanned production outages resulting in lost opportunity costs estimated to range from $200 to $1,000/min. A sketch of the longwall mining method and the components used are shown in Figure 6.4-4.

In coal seams lower than 60 in., thick plow-type longwalls are sometimes applied. These systems do not have the productivity of higher-height shearer-based systems and are more vulnerable to abnormal geologic conditions or roof falls on the face or in the tailgate. However, they are a viable alternative if mining heights below the limits of shearers are required.

Alternatively, interest is emerging in mining very thick seams, more than 18 ft in height, by longwall methods. This has led to development of some very large single-pass longwall systems; multilift longwalls, with limited success; and top-coal caving longwalls, which seem to offer good potential.

Although both retreating and advancing style longwall systems have been used in the past, most installations worldwide are now retreating faces. This choice causes higher initial development but minimizes the huge task of maintaining gate roads in the caved area behind the face (gob/goaf).

The largest number of longwalls is composed of dated equipment styles, including low-capacity chock-type roof supports, or even the earliest style, prop-and-bar or timber roof support. These are notable only because many such installations still exist worldwide but are clearly inferior to modern technology from productivity and safety perspectives.

It is of passing interest that a variation of the longwall method is also applied to hard-rock gold and platinum reef deposits in southern Africa. There, drilling and blasting break the rock, and low-production conveying systems and slushers clear broken ore from the face. Nonexplosive rock breaking or cutting is being evaluated but is not yet commercialized.

Not all soft-rock ores are suited for longwall mining, which works best in deposits that are laterally extensive, flat lying, of fairly uniform thickness, and primarily free of discontinuities such as faults. Coal beds deeper than 1,000 ft usually must be extracted by way of longwall mining, because using R&P methods would require the use of much larger pillars to support the roof and thus reduce the amount of coal that can essentially be recovered.

Table 6.4-8 Longwall favorable characteristics

Key Deposit Indicator	Characteristics
Ore strength	Low
Host rock strength	Weak to moderate
Deposit shape	Tabular
Deposit dip	Flat to shallow
Deposit size/thickness	Large, thin
Ore grade	Low and above
Uniformity	Uniform
Deposit depth	Shallow to deep

Rock bursts, mountain bumps, and outbursts are all manifestations of stored energy release where R&P or longwall mining has been conducted with some combination of the following conditions present:

- Depth greater than 1,500 ft
- Strong ore or stiff/strong rock members in the nearby underlying or overlying strata
- Unexpectedly high in-situ horizontal stresses or stress increases from interaction between workings
- Substantial reservoir or pore pressures of pressurized fluids, particularly CO_2 or CH_4

These events can range from mild thumps of little significance to catastrophic events capable of serious equipment damage and fatal injury to personnel. Expert assistance should be enlisted to assist the mine planning process where such events may occur.

The optimal characteristics for longwall mining are shown in Table 6.4-8.

Even more so than R&P mining, the longwall method is exceptionally efficient and has outstanding production rates and low operating costs. The operation is almost completely mechanized and recovers an extremely high percentage of the ore body. The working face is also safe since the roof is directly supported at all times by heavy-duty shields. Electronic controls and automation allow personnel to position themselves away from most of the recognized hazards. If conditions allow, longwall mining is the most effective way to excavate a thin tabular deposit with lower ore strength.

Over the years, significant improvements to longwall mining equipment have been made to help yield higher production rates. Shields with high-yield capacities and electro-hydraulic controls have replaced manually operated frames and chocks. AFCs have become more robust and powerful with increased size and speed of the chains, allowing higher conveyor capacities. The shearers have also become much more powerful and reliable, which enables more production, less downtime, and greater equipment longevity.

There are, however, a few disadvantages to longwall mining. First, it requires a substantial capital investment to purchase the highly specialized equipment to create a longwall section. The development time is significant because the continuous miners have to progress the main entries and develop gate roads for the longwall panel before the longwall can be installed. Finally, there is little selectivity or flexibility after mining commences. Longwall mines should be large, long-lived operations, with high production rates, in order to make certain of an adequate return on the mine operator's investment. The primary advantages and disadvantages of longwall mining are summarized in Table 6.4-9.

Table 6.4-9 Longwall advantages and disadvantages

Advantages	Disadvantages
Low operating cost	High capital investment
High productivity	Significant advance development
High recovery	Low selectivity
Safest method	Predictable subsidence
High production rate	Low flexibility
High mechanization	
Continuous method	

SMALL-SCALE MINING METHODS

Typically, in small-scale mining operations, a more traditional mining method is favored, where pneumatic drills are used to drill holes to be charged with explosives, and the ore is then blasted and hauled away. Small-scale coal mines may use this method because access to capital is difficult and the cost of equipment for a continuous miner section is prohibitive. Where conditions and capital availability permit, some operators employ continuous miners in small R&P mines. The evolving regulatory and socioeconomic climate is likely to systematically diminish such small operators in preference for larger-scale operations.

CONCLUSIONS

The process of selecting the optimum mining method for a given deposit is complex and requires extensive collection of geological, metallurgical, and mining-related data. In addition to the analysis of multiple alternatives, a thorough understanding of the sociopolitical setting, pertinent environmental concerns, and applicable regulations is critically important. This chapter has discussed the primary deposit characteristics and the mining method performance variables that are involved when selecting a mining method for soft-rock extraction. Because mineable ore deposits exist in all shapes and sizes and no two are alike, the best method selection process is not always evident. However, several key tasks should always be undertaken during method selection for any ore deposit. The first step is to identify the mineral resource available. The second step is to match the most suitable mining method to the ore body. As part of this step, it is important to identify pertinent economic or environmental factors that may constrain methods selection.

Above all else, it cannot be overemphasized that mine planners must value the principle of zero harm, which encompasses health, safety, environment, and community impacts. Failure in any of these areas can affect the sustainability of a mining operation just as seriously as a planning or execution failure. A review of numerous case histories of success and failure highlights the fact that failed projects are usually due to inadequate deposit characterization, inadequate risk assessment and consequent acceptance of elevated risk, inadequate planning or overestimation of operating performance, or inadequate capital to correctly implement plans. There are many paths to failure, and the paths to success are few and normally difficult.

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CHAPTER 6.5

Comparison of Underground Mining Methods

Richard L. Bullock

An overview of the planning required for an underground mine is necessarily complicated by the availability of many different types of underground mining methods. For a detailed description of these mining and development methods and associated case studies, see Hustrulid and Bullock 2001.

The major objective when deciding upon and planning a mining method should be to maximize value, which is achieved after consideration of the following (R.L. Grayson, personal communication):

- Safety of all personnel
- Lowest production cost per metric ton
- Maximum productivity
- Required quality and quantity of the final product
- Maximum recovery of reserves
- Optimal environmental considerations

There are obviously trade-offs in trying to reach these non-aligned objectives.

ROOM-AND-PILLAR MINING METHOD

Room-and-pillar (R&P) mining is a system where a series of rooms (horizontal openings) are extracted, leaving ore, rock, or coal—called pillars—in place between the rooms. These rock/ore/coal pillars can be smaller horizontally than the rooms typical for hard-rock mining (see Figure 13.1-5 in Chapter 13.1) or larger than the rooms typical for soft-rock or coal mining. Normally rooms and pillars are of consistent size and shape and are laid out in a uniform pattern. However, for some metal mining, because the ore is not uniform, pillars can be varied in size and location to enable placement in low-grade areas of the stope. This is sometimes called stope-and-pillar mining.

The term *room-and-pillar* mining has been applied to metal mining in the United States for more than 150 years in the vast mining districts of the Missouri Lead Belts, the Tennessee Zinc District, and the Tri-State Zinc District of southwestern Missouri, southeastern Kansas, and northeastern Oklahoma. Considering the number of underground mines of coal, dolomite, gypsum, limestone, potash, salt, and trona, as well as all of the Mississippi Valley–type lead and zinc mines, it should not be surprising to realize that approximately 60% to 70% of all underground mining in the United States is some form of R&P mining. This amounts to nearly 340 Mt/a (370 million tons/yr) (Zipf 2001). For the aggregate industry alone, according to NIOSH there are 90 to 100 underground mines, all of them R&P (Iannacchione 1999). In today's permitting environment, at any given time there are probably between 20 and 40 R&P underground aggregate mines being planned.

Access to the R&P Mine: Hard-Rock and Coal Mining Methods

Although the access to a mine is not always influenced by the mining method, some discussion is warranted on the various approaches to the initial mine and production opening where R&P mining is to be applied:

- If it is possible to develop the resource from a hillside adit, doing so obviously provides the least expensive and most complete method of entry. In coal mining, this is called a drift entry.
- If a shaft is sunk (Bullock 1973), it should be
  - Sunk somewhere close to the center of gravity of the ore body, unless the ground is going to be allowed to cave, in which case it should be placed well outside the cone of subsidence;
  - Sunk to a depth that allows most of the ore that is be hauled downgrade to reach the shaft storage pockets;
  - Sunk deep enough to accommodate adequate storage pockets, skip loading, and a crusher station (if needed); and
  - Located, for aesthetic reasons, such that the headframe is out of sight of the public.
- If a decline is to be driven, the maximum grade of the decline depends on the equipment that will be driven on the decline. For truck haulage, the decline gradient should be matched to the gearing and optional power train of the trucks. Here are a few planning guidelines, depending on how a decline will be used for truck haulage:
  - For trackless haulage, 12.5% is the maximum grade recommended for normal mine trucks. If

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super-powered trucks are planned, then grades up to 17% may be acceptable.
– For conveyor-belt haulage where rubber-tired trackless equipment must negotiate on a regular basis, 15% is the maximum grade recommended. For conveyor-belt haulage only, the theoretical maximum grade is approximately 0.17 to 0.31 radians (10° to 18°), depending on the type of material (CEMA 1994). Beyond these recommended angles of incline, material will slide down the belt en masse and internally on top of itself, and lumps will roll down the belt and over the top of the fines. However, equipment must be able to access the belt to occasionally clean up spill rock unless hand shoveling for cleanup is planned.

Room-and-Pillar Stoping for Hard Rock

The differences in strength, hardness, and abrasiveness of rocks such as limestone, dolomite, or sandstone as compared to those of the soft materials coal, potash, salt, or trona necessitate different extraction methods.

Primary Extraction Methods

One advantage of modern R&P mining systems is that every task can be mechanized to some degree, provided that it is economically sound to do so. Mechanization minimizes the operating labor force and simplifies staffing. The high-capacity equipment for modern R&P operations is relatively simple to operate.

Although most R&P mining is done by drilling and blasting, particularly for aggregates and metals, some mining is done by mechanical excavation, usually with roadheaders (Bullock 1994). With the power of today's mechanical excavating machines and with improvements that are being made in tools such as disk and pick cutters, the possibility of mechanical excavation should be at least considered during the feasibility study for any rock under 100 MPa (15,000 psi) or even up to 136 MPa (20,000 psi) if it contains fractures and is low in silica content. Where mechanical excavation is truly viable, its use equates to higher production rates and reduced operating cost. The following are advantages of mechanical excavation where it is viable (Ozdemir 1990):

- Improved personal safety
- Minimal ground disturbance
- Reduced ground support needed
- Continuous, noncyclic operations
- Low ground vibrations and no air blast
- Uniform muck size
- Less crushing and grinding in the mill
- Reduced ventilation requirements
- Conducive to automation

Room Width

For productivity reasons, rooms should be as wide as is practical and safe. The wider the rooms, the more efficient the drilling and blasting, and the larger and more efficient the loading and hauling equipment. However, room width is always limited by the rock mass strength of the ore body, back, and floor compared to the stress levels induced into the rock. It is inappropriate to design room widths simply from elastic theory, without taking into account rock-mass strength. However, since rock and pillars can be reinforced to increase the effective rock-mass strength, final room width may be a matter of economics.

Many discussions have been written on how to design a roof span. For further insight into the rock mechanics of R&P roof spans, see Hustrulid and Bullock 2001. What is important at this point is to determine what information is needed for the design and how much of the needed information is already at hand. Chapter 12.1 contains a general summary of geological and structural information that should have been determined during exploration of an ore body. Unfortunately, most exploration groups spend little time or money in determining the information that is needed to construct a rock-mass classification of the mineralized areas and rock surrounding the mineralization. A best-guess rock-mass analysis may have to be done with nothing but the exploration information. In any case it is hoped that mapping of underground structures from core logs, surface mapping, possibly mapping of surface outcrops of the same underground structures, and geophysical information, along with the rock quality designation (RQD) of the rock core, are sufficient for a crude rock-mass classification to be constructed.

Pillar Width

The overall strength of a pillar is related to its height; that is, the ratio of pillar width w to pillar height h is important. The amount of load that a pillar can safely carry is proportional to the ratio w:h. Thus a pillar of ratio 4:1 has a much larger safety factor than does a pillar of ratio 1:1 or 1:5. Ratios of 1:3 to 1:4 are not uncommon in some competent hard-rock metal mines.

The theoretical load as calculated by the overburden load distributed to the pillars may or may not be the load that is actually carried. There is a good chance that the load may arch over some of the interior pillars of the stope and transfer load to barrier pillars or waste areas. In some such cases, interior pillars can be made smaller as yielding pillars. If a stopes mine is very wide, a row of large, rectangular barrier pillars should be retained at regular intervals. In areas of very large lateral extent, this prevents cascading pillar failure of the entire area in a domino affect (Zipf and Mark 1997).

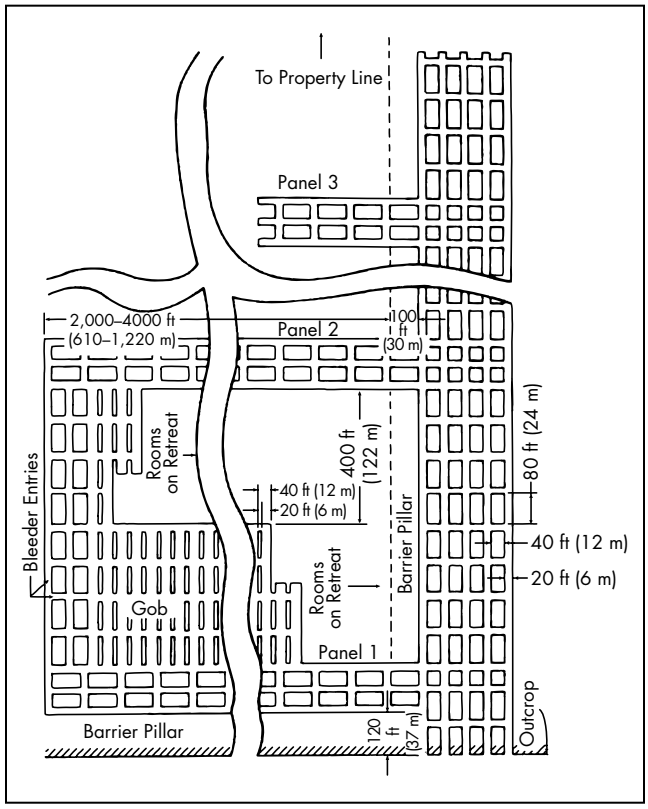
For more information, the reader is referred to case studies on R&P stoping in Hustrulid and Bullock 2001. These studies describe how different mines approach these design problems as well as catastrophic failures that have occurred when proper precautions were not taken.

Secondary Extraction Methods

Pillar removal should be planned as part of the overall mining of areas where the economic value of what remains warrants the extraction of some or all pillars. For example, it is not uncommon for some very-high-grade pillars in the lead/zinc/copper mines of the Viburnum Trend in Missouri to have a value of more than $1 million per pillar.

For optimum recovery, the initial pillar design must be correct and include barrier pillars to prevent catastrophic failures. There are five basic approaches to removing pillar ore after completion of primary mining:

1. Partially extract, by slabbing, the highest-grade part of each pillar.
2. Remove a certain number of the high-grade pillars completely, but leave enough to support the back.
3. Encapsulate pillars with fenced cemented rock fill or paste backfill, then come in underneath the pillar and drop it into the sublevel below.



Source: Stefanko and Bise 1983.

Figure 6.5-1 Typical soft-rock or coal R&P mining system with five entries

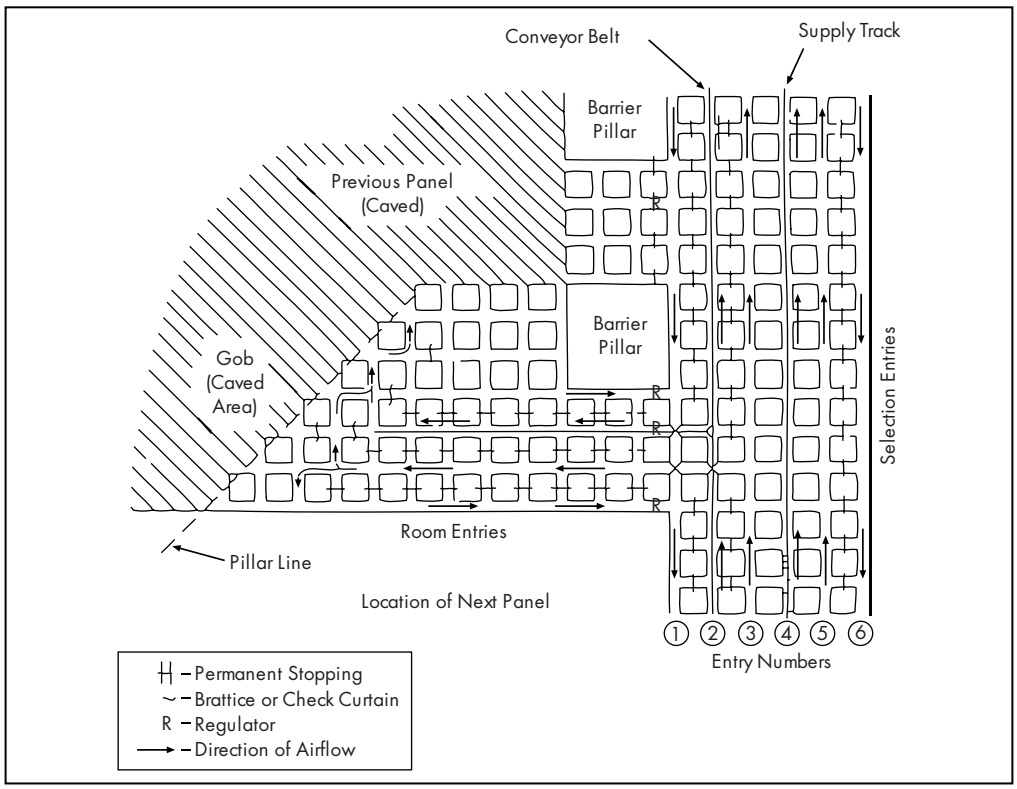
4. Encapsulate low-grade pillars with fenced cemented rock fill or paste backfill to form a barrier pillar, then remove the surrounding pillars (Lane et al. 1999, 2001).
5. For narrow areas and strong/reinforced back, remove all pillars.

Room-and-Pillar Mining for Coal and Soft Rock

For the typical R&P mine layout for mining of bituminous coal shown in Figure 6.5-1, five main entries allow access to the production panel through panel entries. Coal-mine dimensions are generally as follows:

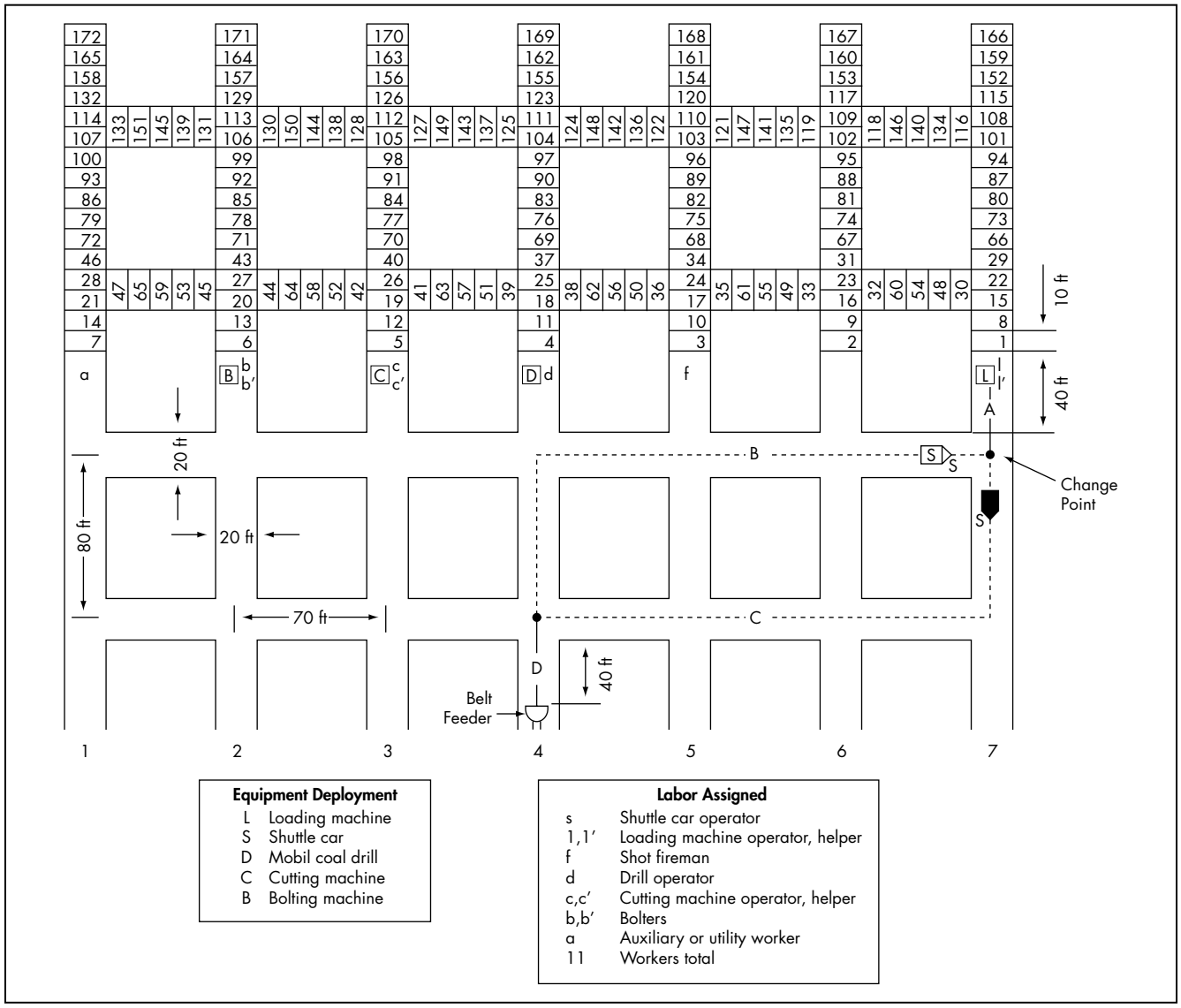
- Entry widths (in the United States) can be up to 6 m (20 ft) and are generally driven 18.3 to 30.5 m (60 to 100 ft) apart, center to center.
- Panel widths can be 122 to 183 m (400 to 600 ft), limited primarily by the cable reach of the electric shuttle cars that are usually used to move the coal. Panel lengths generally vary from 610 to 1,220 m (2,000 to 4,000 ft) but can be longer.

In Figures 6.5-1 and 6.5-2, note that the panel pillars are being mined. In the United States this is called pillaring or caving; elsewhere it is called stoping. Normal practice in pillaring is to drive rooms and crosscuts upon advance (mining into the virgin coal seam in a panel) and to pillar upon retreat (mining while moving back out of the panel). In the United States, the caved area is called the gob; elsewhere it is called the goaf. Pillars are not removed if the surface must be supported; pillaring is often omitted for other reasons as well (Hartman and Mutmansky 2002).



Source: Hartman and Mutmansky 2002.

Figure 6.5-2 Typical soft-rock or coal R&P mining system showing retreat minivng with six entries



Source: Stefanko and Bise 1983.

Figure 6.5-3 Conventional R&P section with seven entries showing auxiliary operations

Mining Methods

Two methods of R&P coal mining exist: conventional and continuous. Conventional operation involving drilling and blasting of the undercut coal seam is an antiquated method practiced in <5% of coal mines in the United States. Conventional operation using conventional equipment is cyclic, with the following sequence:

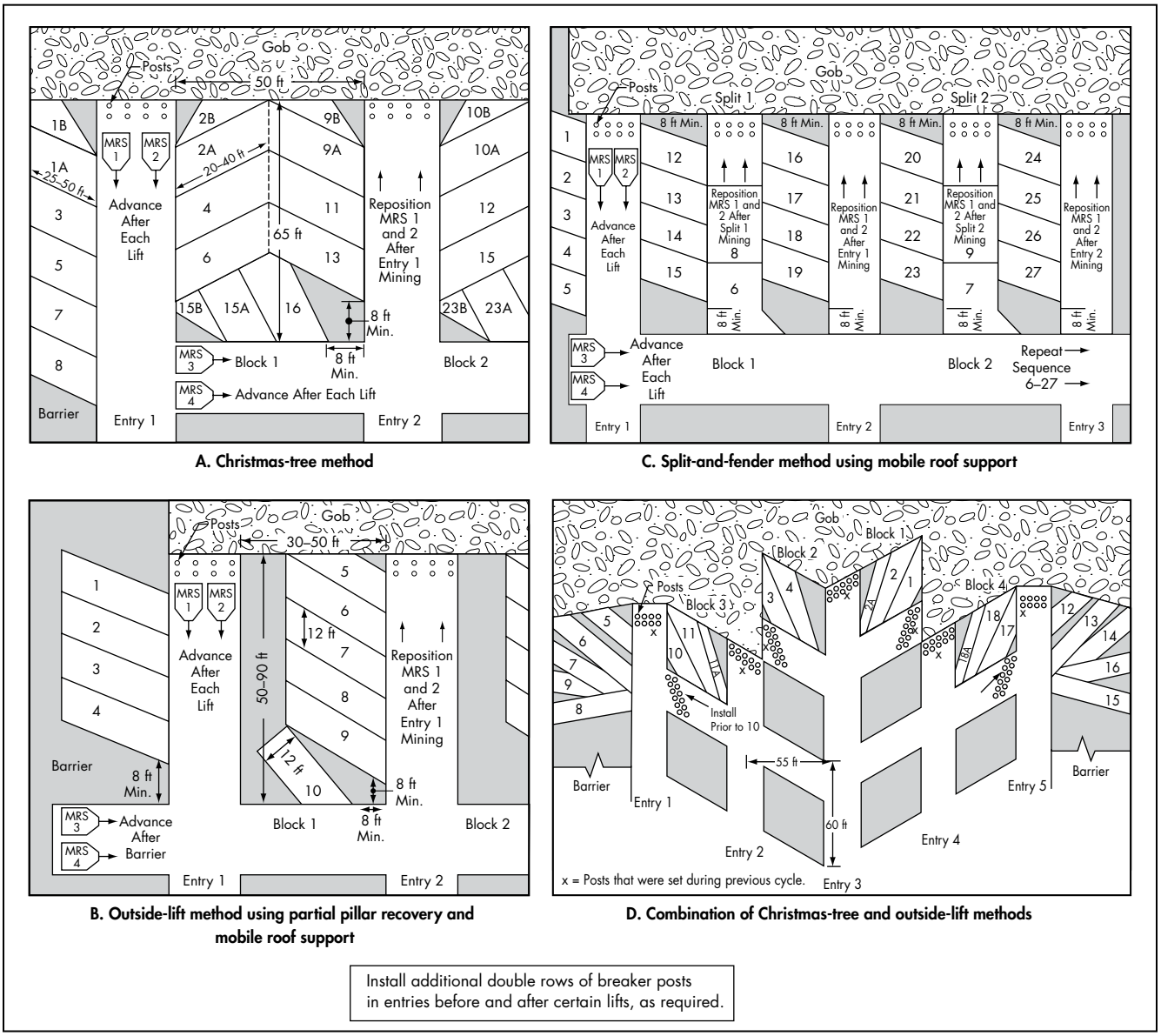
1. Undercut a kerf of 127 to 178 mm (5 to 7 in.) to improve coal breakage during light blasting.
2. Drill blastholes with an electric rotary drag-bit auger drill.
3. Charge the holes lightly with permissible explosive and then blast.
4. Load the coal with an electric gathering-arm loader into a shuttle car for transport away from the face.

Auxiliary operations of roof control, ventilation installation, and cleanup must also be performed. A typical mining section that uses this system is shown in Figure 6.5-3, which shows a seven-entry development. The numbers represent the sequence of faces where each crew member performs their function of the cycle.

Continuous operation is so named because a mechanical excavating drum miner continuously extracts the coal. Coal is mined continuously and then hauled from the face, usually by electric shuttle car but sometimes by conveyor belt or diesel scoop machine. There is usually some waiting between the loading of each shuttle car or scoop. As with conventional operation, auxiliary operations of roof control, ventilation installation, and cleanup must also be performed. The plan view in Figure 6.5-2 shows section entries, room entries, and rooms associated with a production panel within a mine. Room entries, rooms, and associated crosscuts are mined upon advance, and pillars are mined upon retreat. In the figure, note the numbering on the entries in the section entry set.

Retreat Methods

Retreat mining methods vary greatly in the coal industry. The method used can depend on seam height as well as local mining



Source: Feddock and Ma 2006.

Figure 6.5-4 Pillar-extraction methods

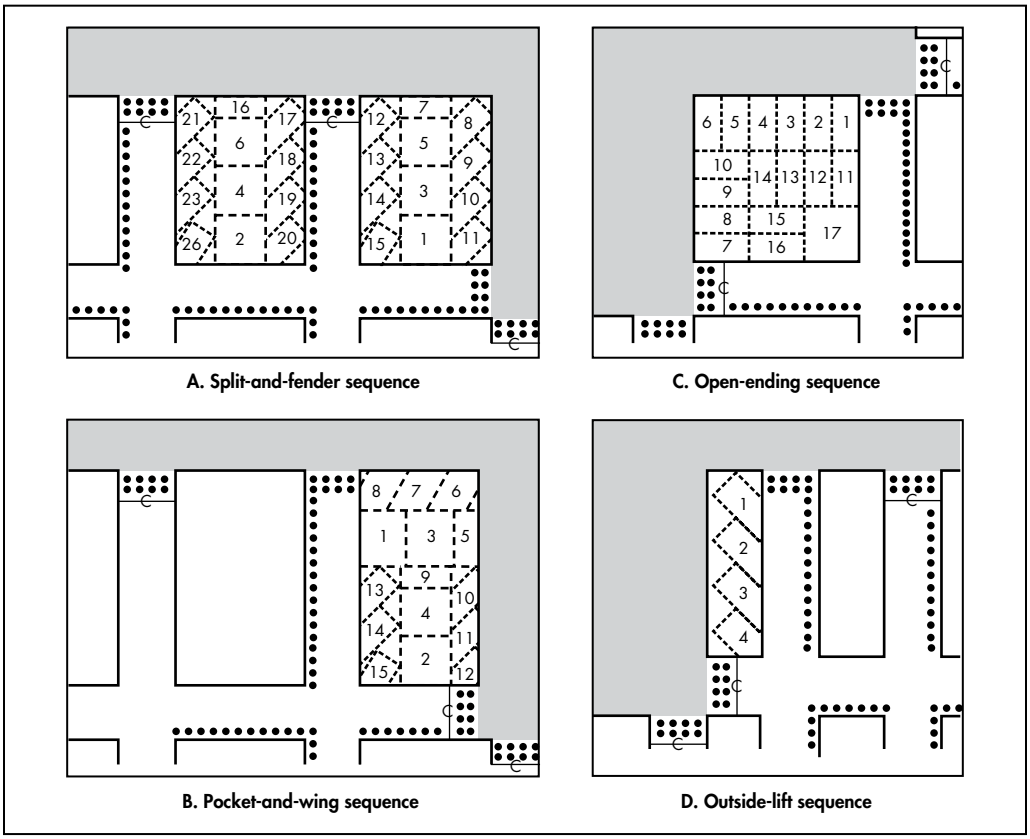
conditions, availability of mining equipment, pillar size, success of similar methods in adjacent mines, opinions of mine engineers, and state and federal roof-control specialists. Pillar-extraction methods widely practiced in the industry include Christmas tree, outside lift, split and fender, and combinations thereof, as shown in Figure 6.5-4.

In a study by the U.S. National Institute for Occupational Safety and Health (NIOSH; Mark et al. 2003), it was estimated that, nationally, about 60% of retreat mining used the Christmas-tree method, 35% used the outside-lift method, and only 5% used the split-and-fender method. The Christmas-tree method is usually favored because it does not require placing changes and bolting.

Using 150 case histories, NIOSH has developed a methodology that has proved successful in generating and validating the Analysis of Retreat Mining Pillar Stability (ARMPS) method and program developed by Mark and Chase (1997). The method aids in the correct design of retreat mining and helps show potential problems such as bump- (seismic-) prone areas (Chase et al. 2002). An excellent review of all current practices in the United States has been written by Mark (2009).

The Christmas-tree method (Figure 6.5-4A) is also called left–right or twinning. Cuts are taken both left and right on both sides of the entry. A continuous miner removes most of the coal on each side until a chevron-type pillar—typically a corner wedge-shaped remnant pillar—is left. The figure shows a common sequence in which lifts are extracted during barrier and production pillar extraction using mobile roof support (MRS) units.

The outside-lift method (Figure 6.5-4B) is the original method developed for use with MRS units. This method is suitable for narrow pillars when combined with extended cut mining. There are many variations, depending on conditions, pillar dimensions, and coal-haulage equipment. Generally the



Source: Kauffman et al. 1981.

Figure 6.5-5 Pillar-extraction cut sequences

pillar is sized so that lifts taken from one side of the pillar are sufficient to extract the pillar without going beyond the supported roof. Cuts are taken starting from the pillar near the gob (goaf) and moving toward solid coal. The sequence of cuts shown in the figure is typical, and MRS units are moved in a retreat mining process or adjacent entry with supplemental support generally provided by posts, mechanical chocks, or MRS units. Multiple pillars are usually extracted simultaneously to provide an adequate number of working places and so avoid production delays.

The split-and-fender method (Figure 6.5-4C) involves a specific cut sequence for continuous mining equipment, as shown by the numbers in the pillars. Again, supplemental support in the form of MRS units is used.

The disadvantages of MRS units are twofold: costs for their initial purchase and for recovery if they are trapped by a rock fall are high, and their operating range is usually limited to seams thicker than approximately 1.1 m (42 in.).

Typical Retreat Mining Methods Using Breaker Posts or Mechanized Chocks

The vast majority of thin-seam mines (seams ≤1.3 m or 52 in.) in the United States use wooden breaker posts (Mark et al. 2003). In a 2002 survey of mines in southern West Virginia (United States) by NIOSH (Mark 2002), the vast majority of thicker-seam mines were already using MRS units. But of the 54 thin-seam mines, only 7 (13%) were using them. In thin-seam mines, a timber plan that requires an adequate number of posts installed at the proper times and locations is essential.

More than 100 roadway, turn, and breaker posts can be required to extract a single pillar. Where hardwood timber is not abundant, mechanical chocks can be used, as is often the case in Europe, Australia, and South Africa. Typical examples of this application are sited in the following paragraphs.

In very low (i.e., thin) seams where continuous haulage is used, the Christmas-tree and outside-lift methods have been used in combination for pillar systems developed with continuous haulage (Figure 6.5-4D). When using mobile bridge conveyors—the most common type of continuous haulage—crosscuts are driven at ~60° angles to facilitate movement of bridges and carriers. The parallelogram-shaped pillars create a panel configuration that is usually referred to as a herringbone or turkey-foot design. Each mining cycle starts with recovery of the two central pillars (blocks 1 and 2) left standing out in the gob by the previous cycle. Each pillar is extracted by the outside-lift method. After cutting lifts 1 and 2 in block 1, the continuous miner is maneuvered to cut lifts 3 and 4 in block 2. A variation is to cut the two central pillars using the Christmas-tree method. The extraction sequence removes the left barrier and block 3 followed by the right barrier and block 4, and then the sequence repeats (Feddock and Ma 2006).

In the low-seam coal mines of Appalachia (United States), many mines still use the four basic pillar-extraction cut sequences described by Kauffman et al. (1981) and shown in Figure 6.5-5: split and fender, pocket and wing, open ending, and outside lift. The numbers in the figure represent the sequence of cuts made by a continuous miner. Temporary prop supports are shown as rows of black dots.

There are several disadvantages to using posts rather than MRS units (Mark et al. 2003):

- Setting posts exposes miners to ground falls.
- Posts have a limited load-bearing capacity; a typical hardwood post 152 mm (6 in.) in diameter can carry about 45 t (50 st).
- Posts have limited roof-convergence range—from 25 to 51 mm (1 to 2 in.)—before they break.
- Because of their weight and bulk, posts can cause material-handling injuries, particularly in high coal.

SUBLEVEL OPEN-STOPING MINING METHOD

R&P methods are often used for subhorizontal ore bodies of relatively uniform thickness. As strata dip and/or ore-body thickness increases, other extraction methods are usually used. For example, a moderately thick, flat-dipping deposit would normally be mined by R&P methods. But for a dip of 90°, loading on the pillars is from the horizontal direction and blasted ore falls down for collection at the bottom of the stope. Although the general geometry is the same as that for R&P methods, the generic name given to this system is sublevel open stoping. Blasthole stoping, vertical crater retreat mining, and vein mining fall under this general heading.

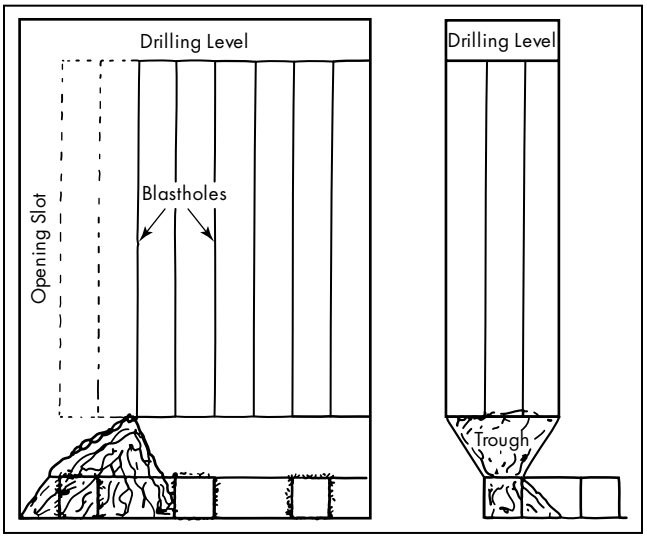
A special form of sublevel open stoping is called shrinkage stoping. In general, the method is applied to ore bodies having dips greater than the angle of repose of broken material (~50°) so that material is transported to the collection points by gravity. For massive deposits, stopes with vertical walls are created and the overall dip of the deposit is immaterial. The criterion for applying the method is that, during extraction, the openings created must remain open. After extraction, the openings may be filled or left open, and the pillars left between stopes may be extracted or left in place. This section presents some typical layouts used for extracting ore. It is assumed that mobile equipment is used with ramp access.

Throughout planning, it is important to be aware of the potential for ore dilution or ore loss when drilling long holes and when mining undulating veins. In feasibility studies, the value of dilution for sublevel long-hole stoping should be ≥15% and ore loss ≥5%. However, for short-blasthole methods such as shrinkage methods, dilution and ore loss might be <5%.

Extraction Principles

One can consider an ore block of width w = 1 to 10 m (3.3 to 33 ft), length l = 10 to 40 m (33 to 131 ft), and height h = 20 to 30 m (66 to 98 ft). For simplicity, it can be assumed that the block is vertical, although for this method the dip of the block is immaterial since, if the block is thick enough, a stope can be developed that will flow by gravity and can be mined by a number of sublevel stoping techniques. Blasted ore falls to the bottom of the block and is removed with load-haul-dump (LHD) equipment. There are various designs for the extraction level. It can also be assumed that a trough is created using fans blasted toward an opening slot. The LHD units travel in a footwall haulage drift running parallel to the trough. Access to the trough is from the side. The location and number of access drawpoints are such as to provide full extraction coverage.

With the availability of remote- or teleremote-operated LHD units, it is now common to design stopes with flat bottoms rather than trough undercuts. Final cleanup of up to 20% of the stope ore is done by remote control, which simplifies stope layout and makes drilling and blasting more efficient. With remote LHD units, there may be no requirement for a footwall haulage and crosscuts, because the stope is accessed along the strike within the ore, reducing waste-development costs.



Source: Bullock and Hustrulid 2001.

Figure 6.5-6 Blasthole stoping method, starting with a slot opening

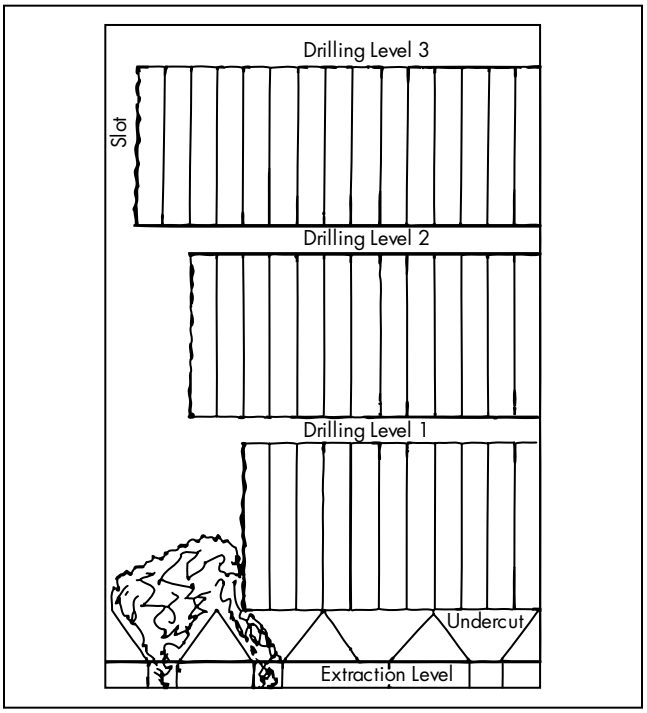
Blasthole Stoping (End Slicing)

Because it is the simplest and usually lowest in cost, blasthole stoping (also called end slicing) is the first method considered for mining a block (Figure 6.5-6). From the drilling level at the top of the block, rows of parallel blastholes are drilled down to the top of the extraction trough. A raise is driven at one end of the block and is slashed to full stoping width to form a slot. Holes are blasted one or several rows at a time toward the open slot. The blasting design and layout is similar to that used in bench blasting. Hole diameters vary widely but are typically in the range 76 to 165 mm (3 to 6.5 in.). For wide blocks, diameters of 165 mm (6.5 in.) are often used.

Hole straightness is an important design consideration that affects fragmentation, ore loss, and dilution. In general, the largest hole diameter possible for the stope geometry is selected, because straight hole length is strongly dependent on hole diameter. The amount of development required to exploit a certain volume of ore is inversely proportional to block height, and, because the cost of development is significantly higher than that of stoping, the tallest extraction blocks possible should be planned for the stope.

Sublevel Stoping

If geomechanic studies indicate that very tall blocks (those whose height exceeds the straight drilling length from one drill location) can be extracted using the same extraction level, then several drilling levels at various heights within the block can be created by a method called sublevel stoping (Figure 6.5-7). The layout is similar to that for blasthole stoping with an extraction level and an opening slot, but now there are multiple drilling levels. Mining is done either overhand (lower drilling blocks are extracted first) or underhand



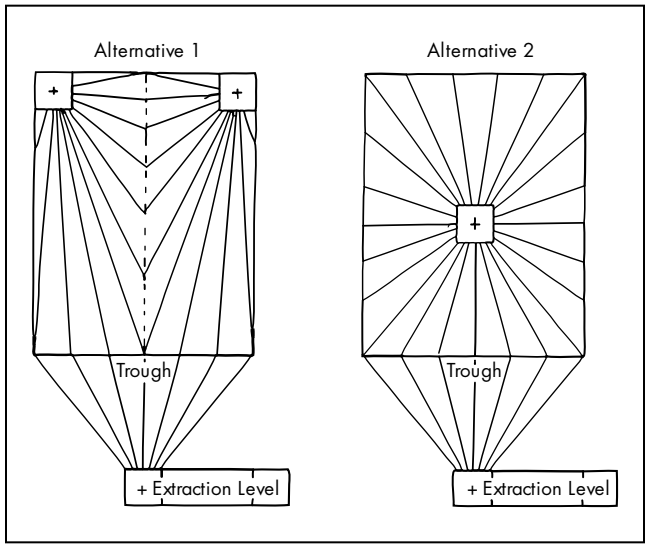
Source: Bullock and Hustrulid 2001.

Figure 6.5-7 Sublevel stoping method, showing a typical parallel-holes drill pattern

(upper drilling blocks are extracted first); overhand stoping is usually assumed. The simplest approach is to repeat the drilling layout for one-level blasthole stoping. Ore-body thickness is assumed to be such that the full width is undercut and becomes available for drilling access, in which case parallel holes can be drilled.

An alternative approach is to drill fans of holes rather than parallel holes from the sublevels (Figure 6.5-8). There can be one or multiple drill drifts on each sublevel and the rings can be drilled downward, upward, or in full rings. There are many variations of drill pattern, based on a number of factors. For example, the distance between sublevels needs to be maximized so as to minimize the amount of development per ton of ore broken. But the accuracy of the drilling equipment in that particular rock determines how long holes can be without excessive drill-hole deviation, which causes boulders, ore loss, ore dilution, or so-called misbreaks (rock that breaks incorrectly or not at all) in blasting. Likewise, the width of the ore block to be taken can determine whether sublevel drifts are placed on the hanging wall and the footwall, or, if the ore is narrow, whether a single drift in the ore might approach the limits of the ore. Another variable is hanging-wall strength. If a hanging wall needs support, then some of the sublevels need to be on the hanging-wall side so that sublevel drifts can also be used in place of cable bolts for support as well as for drilling long blastholes. Yet another variable is whether a primary stope or a pillar between two previously mined filled stopes is being mined. In the latter case, the blastholes break more easily to the free face of the rock–fill interface. For a guide to these planning decisions, see Chapter 13.4.

Use of sublevel stoping assumes good stability of the openings created. Stability surprises can mean partial or even full collapse of partially extracted stopes. Production can be



Source: Bullock and Hustrulid 2001.

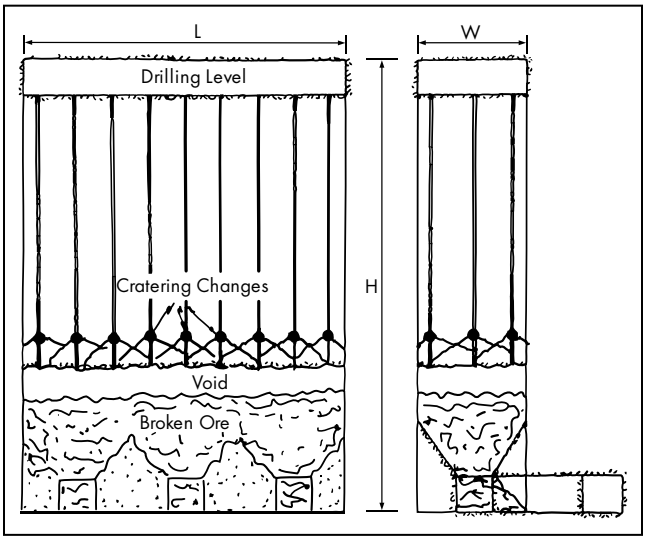
Figure 6.5-8 Multilevel sublevel stoping method, showing typical fans-of-holes drill patterns

stopped fully because of the presence of large blocks in the drawpoints. Even in the best-case scenario, there is likely to be some ore loss and dilution. The footwall, hanging wall, and roof can be reinforced before or during mining. Extraction blocks (stopes) can be oriented parallel or perpendicular to the ore body, depending primarily on the width of the ore body and how the mine planners want to drive the access drifts to support extraction of the stope block.

Vertical Crater Retreat Stoping

In the cases discussed previously, rings of holes are blasted toward a vertical slot. In vertical crater retreat (VCR) or vertical retreat mining (VRM) systems, the need for a slot connecting the drilling and extraction level has been eliminated, simplifying development (Figure 6.5-9). Rather, the slot is replaced by a horizontal slot (undercut) created at the bottom of the block on the extraction level. A real trough can be created but is not necessary. From the drilling level, large-diameter (~165 mm or 6.5 in.) parallel holes are drilled downward to the undercut level. Short explosive charges (length = 6× the hole diameter) are lowered to positions slightly above the top of the undercut. These "spherical" charges are detonated, dislodging a crater or cone-shaped volume of rock into the underlying void. As each layer of charges is placed and detonated, mining of the stope retreats vertically upward in a VCR.

The design of the blasting pattern is based on full coverage of the block cross-section by the adjacent craters. Normally the blasting pattern is tighter (holes are spaced closer) than for large-hole blasthole stoping, and hence the powder factor is larger. During blasting under these confined conditions, fragmentation is generally finer than it is for blasthole stoping. Prior to charge placement, care must be taken in determining the location of the free surface. In some cases, ore blocks can overbreak vertically along fractures or faults and leave a vertical opening for the next round to break to, in which case the next round in a nearby hole will break sideways to any nearby free face. Thus observations and measurements are essential prior to planning every blast. In this system, the level of broken rock remaining in the stope can be controlled to provide



Source: Bullock and Hustrulid 2001.

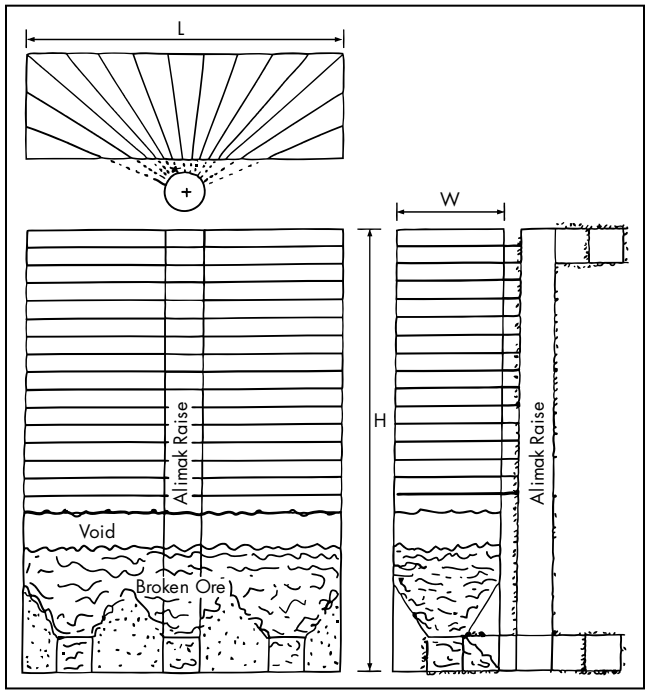
Figure 6.5-9 Vertical crater retreat stoping method

varying levels of support to the stope walls. If the stope is kept full except for a small slot to provide a free surface and swell volume for the blasted rock in the slice, the method used is classified as a shrinkage method and the remaining ore is drawn out at the completion of mining.

Vein Mining

Another approach to extracting the ore block is called vein mining (Figure 6.5-10). At the highest level of the block to be extracted, a connection is made to the ore body. Access is assumed to be via the footwall side, and connection is made in the middle of the extraction block. On the extraction level, an undercut or extraction trough is prepared. A raise is driven between the extraction level and the upper access point from which long blastholes can be drilled. Using a Boliden-type cage or the Alimak technique (Ovanic 2001), a raise is driven upward in the footwall a small distance from the ore-footwall contact. The next step in the process is the drilling of subhorizontal fans of blastholes using the Alimak platform or Boliden-type cage in such a way that the plan area of the extraction block is fully covered. The hole diameter is determined by the capacity of the drilling machine but should be as large as possible since the toe spacing and the burden (distance between fans) is determined by the hole diameter and the explosive used. When drilling of the entire extraction block is complete, the fans are charged and blasted one or more at a time, working off the raise platform. Access to the block is now only from the upper level as the stope is retreated upward. The ore in the stope can be removed after each blast or can be left in place and only enough removed to provide swell volume for the next one or more slices. Rock reinforcement can be installed in the hanging wall if required from the raise platform during the drilling of the production holes.

An advantage of this method is that it enables extraction of very high ore blocks with a minimum of development (upper access point, extraction level, and connecting raise). The overall length of the extraction block is determined by the straight-hole drilling length of the available drilling equipment. However, if larger blocks are more economically mined, then multiple raises can be used and this possibility should



Source: Bullock and Hustrulid 2001.

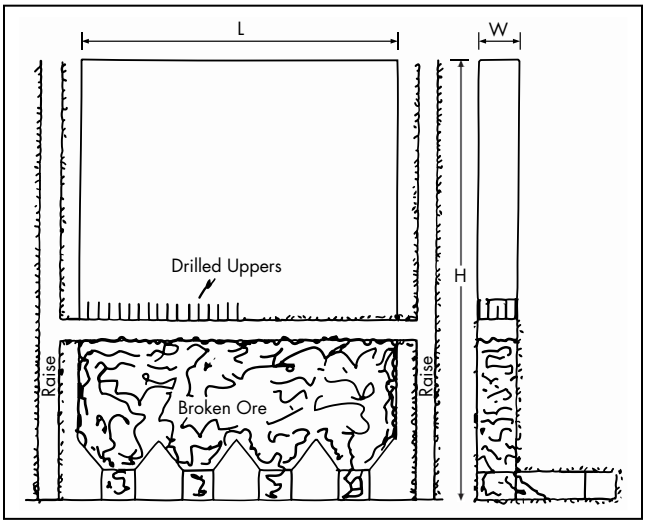
Figure 6.5-10 Vein mining method using long holes

be considered in the feasibility study. A disadvantage is that drilling and charging must be done from a raise environment, which can be hazardous unless well managed. However, major advances have been made in the mechanization and automation of rigs used for drilling.

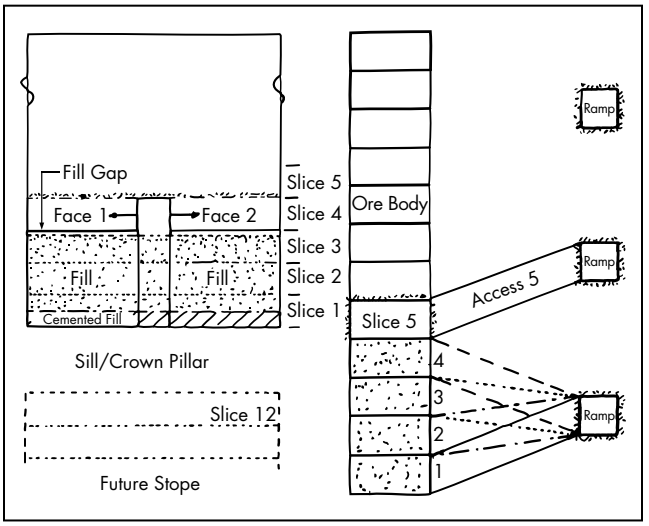
Shrinkage Stoping

Although normally considered as a separate method, it is logical to also discuss shrinkage stoping at this point, since it is an open-stope method (Figure 6.5-11). The method is generally applied to very narrow extraction blocks that have traditionally not lent themselves to a high degree of mechanization. It has been applied successfully in high-grade precious metal mining because of its low dilution and low ore loss. The extraction block is laid out longitudinally due to the very narrow nature of the ore body recovered. An extraction drift is located in the footwall with loading crosscuts positioned at regular intervals. Raises are driven at each end of the extraction block connecting to the above-lying level. An initial horizontal extraction slice is driven across the block from raise to raise. Extraction troughs are created by drilling and blasting the rock between this level and the underlying extraction points. When the extraction system has been created, short vertical holes are drilled into the roof of the first extraction slice using the raise access. The miners stand on the broken ore that forms the working floor. Jackleg or stoper drills are used to drill small-diameter holes. The holes are charged and then ore is extracted from the stope to provide room for the blasted material. The blast is initiated and the miners reenter the newly created void to bar down loose material before drilling out for the next slice. The process continues, working upward one slice at a time. When the upper end of the extraction block is reached, the ore is drawn out. Until that time, the stope is filled with broken ore.

A severe disadvantage of this method is that it is among the most hazardous of mining methods. The miner is exposed to



Source: Bullock and Hustrulid 2001.
Figure 6.5-11 Shrinkage stoping method



Source: Bullock and Hustrulid 2001.
Figure 6.5-12 Overhand cut-and-fill mining sequence

freshly blasted unsupported ore while working on rough broken ore. In addition, voids can be created in the ore being drawn, which can suddenly collapse in areas where miners are working. For these reasons as well as the relative high operating cost, its use is limited in the United States. However, it is still used fairly extensively in developing countries and in Canada.

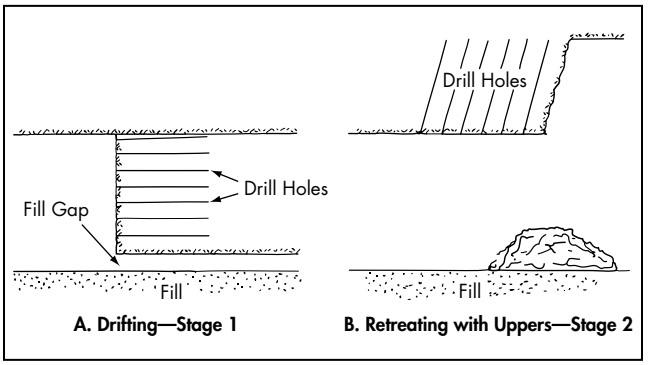
Summary of Open-Stoping Sublevel Mining

Depending on the geometry of the ore body, several varieties of sublevel stoping can be used. However, the ore bodies must all have strong wall rocks and competent ore, either naturally or by means of reinforcement, since in the process of ore removal large openings are created.

The extraction block used to illustrate the layouts for the different mining systems can now be duplicated and translated laterally and vertically in the ore body, leaving pillars to separate adjacent blocks. The size and shape of the extraction block can be adjusted to fit the ore-body geometry and the mine infrastructure. Openings created during this primary mining can be filled with various materials or left unfilled. The filling materials can be cemented or left uncemented, depending on the next stage of recovery envisioned. Various methods can be used to recover remaining reserves tied up in the pillars. During the feasibility studies, these secondary recovery methods should be examined at the same time as the primary system is designed. Although for simplicity the basic extraction block was considered to be vertical, the process can obviously be repeated for ore bodies having various dip conditions.

CUT-AND-FILL MINING METHOD

Where ore and/or wall rocks are weak, and hence both opening size and allowable time between ore removal and filling of the excavation is strictly limited, a number of extraction designs can be applied, all of which fall under the general category of cut-and-fill mining. This versatile method can be adapted to the extraction of any ore-body shape. With some exceptions, all of the ore is removed via drifts that are then filled. As a result, mining costs are higher than for other methods. But when the method is applied correctly, recovery is high and dilution is generally low. Thus it is appropriate for the extraction of high-grade ore bodies.

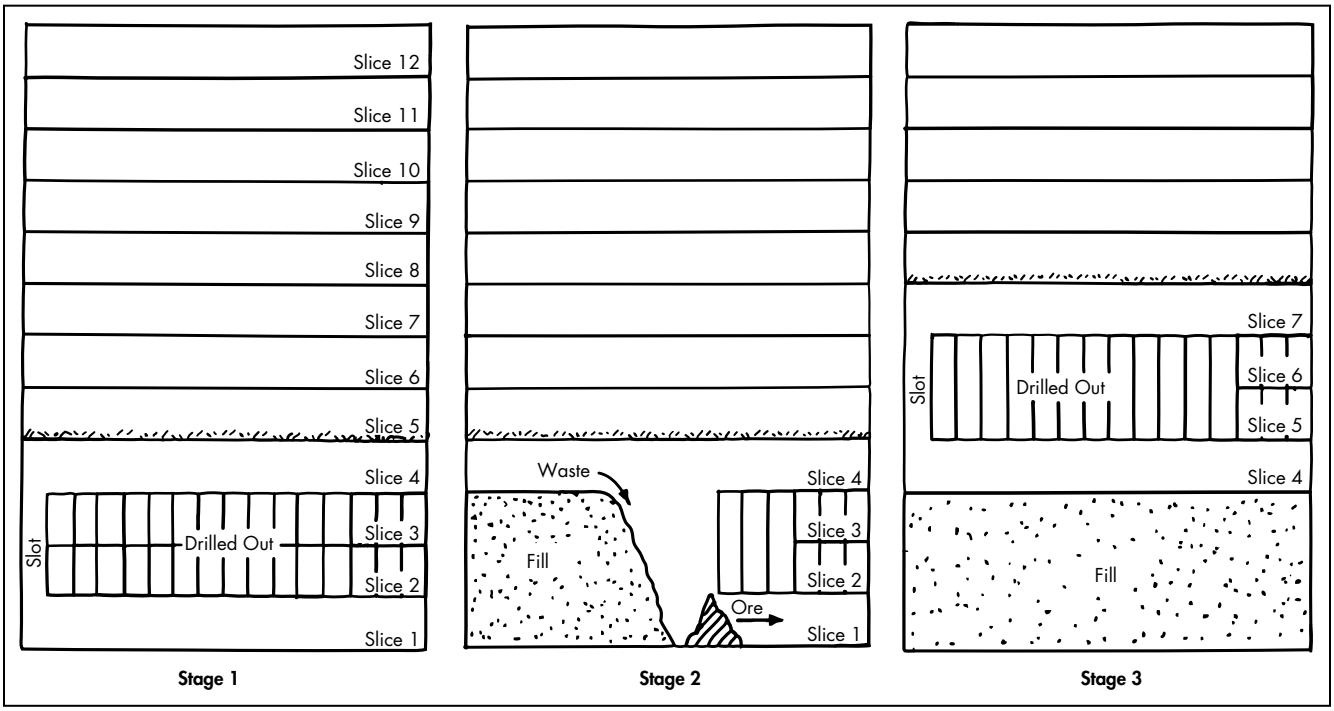


Source: Bullock and Hustrulid 2001.
Figure 6.5-13 Two-pass cut-and-fill mining sequence

Extraction Principles

For simplicity, an extraction block of the same type used in the previous section is assumed. Access is via a ramp driven in the footwall, and mobile equipment is used. Typically, drifts used in mechanized cut-and fill mining are about 5 m (16.4 ft) high. The ore block to be extracted is assumed to be vertical and of a width that can be removed by means of normal drifting. When ore-body strength is fairly good, overhand cut-and-fill mining is normally applied (Figure 6.5-12).

Ideally, access to each level is via crosscuts originating at the midlength position of the block such that two headings can be operated at one time. Typical drift rounds consisting of drilling, blasting, loading, scaling, and installation of rock reinforcement are used. This progression of operations can lead to delays unless carefully planned. Drilling of heading 2 is carried out while other operations are being done at heading 1. When the extraction slice is complete, backfilling takes place. Fill is placed so as to leave a small gap to the overlying ore (Figure 6.5-13A) such that a heading 5 m (16 ft) high recovers about 4 m (13 ft) of ore. On the next slice, this gap forms the free surface for blasting. The process continues upward, slice by slice, to the top of the block.



Source: Bullock and Hustrulid 2001.

Figure 6.5-14 Avoca mining sequence

Several extraction blocks can be operated simultaneously to meet production requirements. The horizontal pillar created between two such stacked extraction blocks is called the crown pillar for the underlying stope and the sill pillar for the stope above. Normally the first cut of the extraction block above the sill is filled with cemented fill to facilitate later extraction of the pillar.

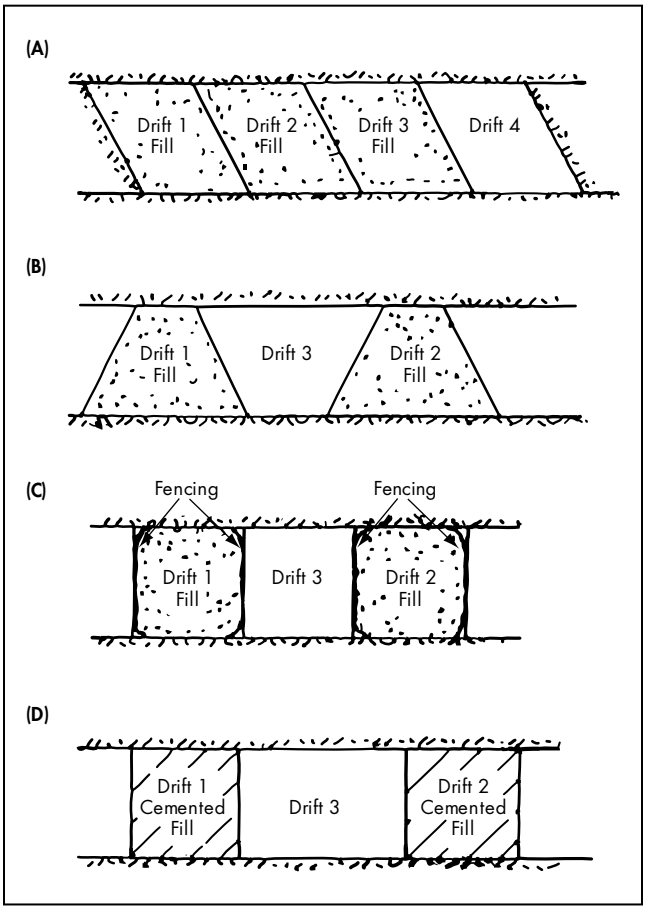
In some cases, the wall rock is strong enough to allow a double slice to be open at any given time. The first slice is mined by drifting and then, rather than filling directly, upward-oriented drill holes, called uppers, are drilled the length of the slice (Figure 6.5-13B). When drilling is complete, several rows of holes are charged and blasted, beginning at the ends of the extraction block and retreating toward the access. Ore is extracted by an LHD unit after each blast and transported to the orepass. Efficiency can thus be improved by changing the typical cycle to one in which all drilling is done first, followed by charging and loading. Then either one lift can be backfilled followed by the drilling of uppers, or both lifts can be backfilled, followed by drifting and then drilling of uppers.

Access to this one-drift-width cut-and-fill stoping is via an access ramp in the footwall. Often, where the wall rock is fairly competent, four slices are accessed from a given point on the ramp (Figure 6.5-12). In overhand cut-and-fill stoping, crosscut 1 is made first. When the slice is complete, the roof of the crosscut is slashed down to form crosscut 2. This continues for the four slices, at which time a higher point on the ramp is selected as the origin of the crosscuts. Generally the maximum crosscut inclination is limited to about 20°, although 12° to 15° is more normal. This sets the position of the ramp with respect to the ore body.

There are several variations of overhand cut-and-fill stoping using so-called ramp-in-stope development. This method has the advantage that development is in ore, although some development must still be done in the footwall. It can be applied to lower-grade deposits for which the development cost of conventional mechanized cut-and-fill mining cannot be justified, although overall operations are usually less efficient. One such mine development is the Bruce copper mine in Bagdad (Arizona, United States), described by Johnson et al. (1998). Another general description of ramp-in-stope cut-and-fill mining is in Pugh and Rasmussen (1998).

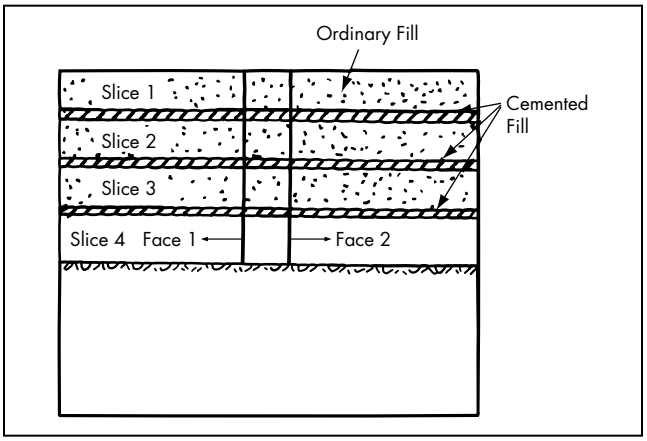
If the strength of the wall rock and ore is quite good, spans of more then two lifts can be created by means of the Avoca (rill mining or benching) mining method (Figure 6.5-14). In the figure, slices 1 and 4 are extracted by drifting (stage 1). Rows of vertical blastholes are then drilled from the floor of slice 4 to the roof of slice 1. A vertical slot is created and rows of holes are blasted one or more at a time toward the slot. Ore is extracted by an LHD unit operating in slice 1. At the same time that retreat extraction is under way, filling is being conducted from the opposite end of the stope (stage 2). A gap is maintained between the extraction and filling fronts to minimize dilution. When completed, slice 7 is removed by drifting (stage 3). Slices 5 and 6 are now removed using slice 7 as the drilling level and slice 4 as the extraction level.

If the extraction block is quite wide, the cut-and-fill method can still be used but now several drifts are driven side by side (Figure 6.5-15). This is similar to R&P (rib pillar) mining with the rooms being filled and the pillars then being extracted. Various techniques can be used to shape the drifts; the most common is shown in Figure 6.5-15C. Here, straight walls are used and every other drift is removed in a primary mining phase. Cemented fill is used to avoid dilution during removal of the interlying drifts. Another variation is to make the primary drifts narrow and the secondary drifts wide to minimize the use of cemented fill.



Source: Bullock and Hustrulid 2001.

Figure 6.5-15 Drift-and-fill mining techniques



Source: Bullock and Hustrulid 2001.

Figure 6.5-16 Underhand cut-and-fill mining method

If the strength of the ore or hanging wall is very poor (e.g., in the range of rock mass rating [RMR] = 20 to 40, which might be the equivalent of a standup time [depending on the roof span] of 1 to 100 hours), then the underhand cut-and-fill mining method—often shortened to *undercut and fill*—can be used (Figure 6.5-16). The first slice is taken and then various techniques are used to prepare a layer that becomes the roof when the slice below is extracted. In the past, the most common mining method involved a timber floor pinned into the walls. It is now more common to pour a layer of cemented fill with or without reinforcement. The remainder (upper portion) of the drift can be left open or filled with uncemented fill. The next slice is then extracted under the constructed roof. A more common practice in North America is to use engineered cement or paste fills. When they are used for mining wide ore bodies, the material must be jammed tight to the back or previous floor.

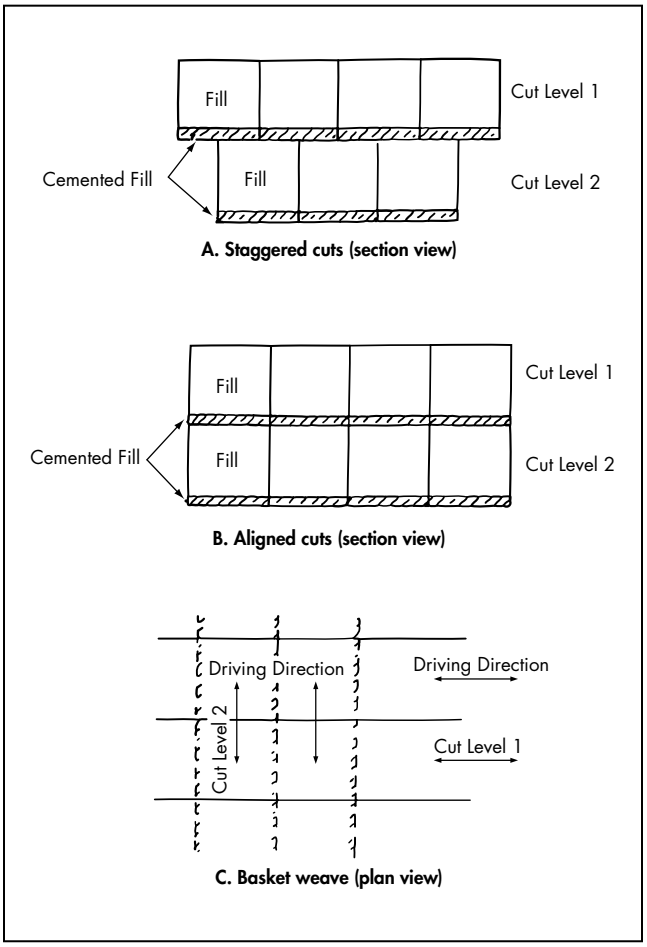
From the same development access level, some mines use the overhand cut-and-fill method to work upward from this level and the undercut-and-fill method to work downward. This doubles the number of working faces in operation at any one time from a given level. Whether or not this is advisable depends on the ore/rock strength. Wide ore bodies can also be mined using the underhand cut-and-fill method (Figure 6.5-17A and B). The process is the same as described earlier but now cold joints occur between the individual drift floor pours. Generally, whether it be engineered fills, cemented fill, or a paste fill, it is better to avoid positioning drifts of the underlying layer directly under those above. It is acceptable to shift the drifts sideways or drive them at an angle to those above; the latter case results in a basket-weave pattern (Figure 6.5-17C). Undercut drifts are developed usually not on 90º angles but more commonly on 45º or 60º angles (Figure 6.5-18).

In thick, inclined, or very wide ore bodies that are appropriate for overhand cut-and-fill mining, vertical pillars are sometimes left to provide additional support between the hanging walls and footwalls in a method called postpillar mining (Figure 6.5-19). If possible, the pillars are located in internal waste or low-grade areas. On the lowest slice, an R&P mine is created and the rooms are then filled. A second slice is taken continuing along the vertical upward extension of the pillars and this level is then filled. The process continues thusly. This method is so named because the pillars appear as vertical posts surrounded by fill. Some consider it to be a subset of R&P mining, others to be a subset of cut-and-fill mining.

There are no set dimensions for the pillars and rooms in this mining method. In Figure 6.5-19, the small post pillars can vary from only 1.8 to 6 m (6 to 20 ft) square, and the rooms from 7 to 15 m (24 to 50 ft), depending on the rock-mass strength of the back. The idea is to maximize the extraction ratio while maintaining a safe working area. Because of the presence of the surrounding fill, even very tall and slender pillars can be quite strong. The method is often used to maximize initial recovery from R&P mining, and therefore no additional recovery is expected by means of retreat mining or pillaring. Of course, in all such cases, measures must be taken to ensure that the pillars are correctly aligned and vertical.

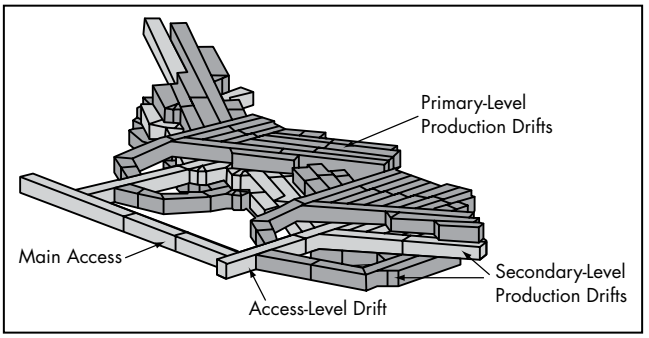
SUBLEVEL CAVING MINING METHOD

Sublevel caving was used initially in the United States for extracting soft iron ores in the iron ranges of Minnesota and Michigan. The method as practiced today differs significantly from that used earlier and should probably be given another name such as sublevel retreat stoping, continuous underhand sublevel stoping, or something similar that better reflects the process. Sublevels are created at intervals of 20 to 30 m (66 to 98 ft) beginning at the top of the ore body and working downward. On each sublevel, a series of parallel drifts is driven



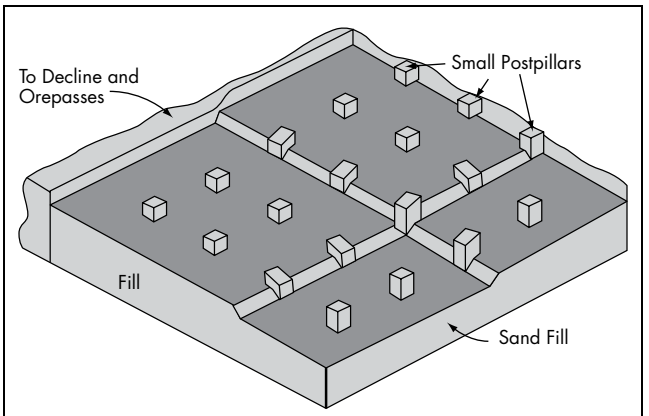
Source: Bullock and Hustrulid 2001.

Figure 6.5-17 Underhand cut-and-fill stope development patterns



Source: Brechtel et al. 2001.

Figure 6.5-18 Undercut drift-and-fill mining method, showing a primary level and underlying secondary levels to be mined under backfill (Murray mine, Jerritt Canyon, Nevada, United States)



Note: Pillar and room dimensions vary according to the rock-mass strength of the back.

Figure 6.5-19 Postpillar cut-and-fill mining

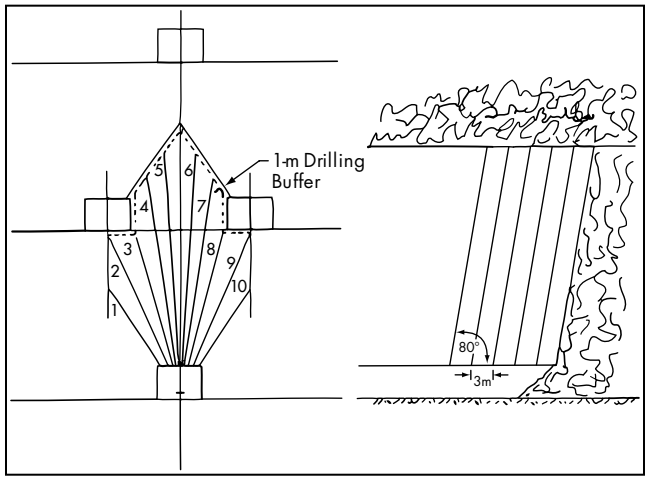
at a center-to-center spacing on the same order as the level spacing. From each sublevel drift, vertical or near-vertical blasthole fans are drilled upward to the immediately overlying sublevels. The distance between fans (the burden) is on the order of 2 to 3 m (6.5 to 10 ft). Beginning typically at the hanging wall, fans are blasted one by one against the frontlying material consisting of a mixture of ore from overlying slices as well as the waste making up the hanging walls and/or footwalls. Extraction of ore from the blasted slice continues until the total dilution reaches a prescribed mineral cutoff level. The next slice is then blasted and the process continued. Depending on ore-body geometry, the method can be used with transverse or longitudinal retreats.

Today, sublevel caving is used in hard, strong ore materials from which hanging-wall rocks readily cave. The key layout and design consideration is to achieve high recovery with an acceptable amount of dilution and ore loss. The uncertainties of fragmentation and ore cavability present in panel caving (discussed in the following section) are removed since each ton of ore is drilled and blasted from the sublevels. The method has been used most for mining magnetic iron ores, which can be easily and inexpensively separated from waste. However, it has been and can be applied to a wide variety of other ore types.

Sublevel Caving Layout

Ore is recovered by means of both drifting and stoping. Because the cost per ton for drifting is several times that for stoping, it is desirable to maximize stoping and minimize drifting. Thus, through the years, sublevel heights have steadily increased until today stopes are up to 30 m (98 ft) high. Whereas, in early designs, approximately 25% of the total volume was removed by drifting, today, in the largest-scale sublevel cave designs, that value has dropped to about 6%. Similarly, sublevel intervals have increased from 9 to nearly 30 m (30 to 98 ft). Key to this development is the capability to drill longer, straighter, and larger-diameter holes. Sublevel caving is an underhand method with all blastholes drilled upward. The ore moves down to the extraction/drilling drift by gravity.

A number of factors determine sublevel cave design. Sublevel drifts typically have dimensions (width × height) of 5 × 4 m (16 × 13 ft), 6 × 5 m (20 × 16 ft), or 7 × 5 m (23 × 16 ft) to accommodate LHD equipment. In the example used to illustrate the layout principles, the drift size is assumed to be 7 × 5 m (23 × 16 ft). The largest possible blasthole diameter, from the viewpoint of drilling capacity and explosive charging, is normally chosen. The maximum hole size in use today is 115 mm (4.5 in.), based largely on the capability to

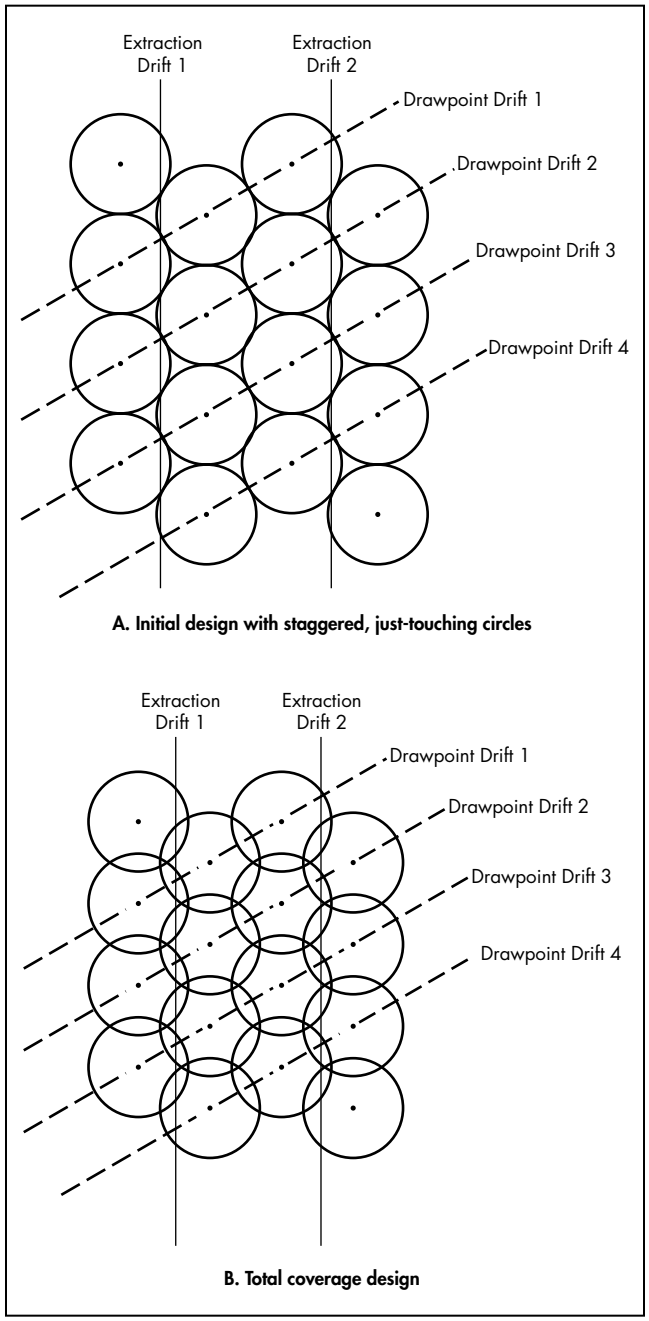


Source: Bullock and Hustrulid 2001.
Figure 6.5-20 Sublevel caving ring design

charge and retain explosive in the hole. These large holes can be drilled using either in-the-hole or top-hammer drills. The large diameters and large drift sizes permit the use of tubular drill steel of relatively long lengths, which minimizes the number of joints and maximizes joint stiffness, so that the required long, straight holes can be produced. The largest ring designs incorporate holes with lengths of up to 50 m (164 ft).

A sample sublevel caving ring design is shown in Figure 6.5-20. The sublevel drift interval is decided largely based on the capability to drill straight holes. In this example, it is assumed that the sublevel interval based on drilling accuracy is 25 m (82 ft). When the sublevel interval has been decided, it is necessary to position the sublevel drifts. In this example, drifts are placed so that the angle from the upper corner of the extraction drift to the bottom center of the drifts on the overlying sublevel is 70°. This is about the minimum angle at which the material in the ring moves to the drawpoint. The resulting center-to-center spacing is 22 m (72 ft). A single-boom drill is assumed to drill all of the holes in the ring. The inclination of the side holes is chosen as 55° although holes somewhat flatter than this can be drilled and charged. The function of holes drilled flatter than 70° is largely (1) to crack the ore, which is then removed from the sublevel below; and (2) to reduce the maximum drill-hole length. Holes flatter than 45° are difficult to charge because of the angle of repose of the ore at the extraction front. In the final configuration of drill holes shown in the figure, a buffer zone 1 m (3.3 ft) wide has been left between the ends of the blastholes and the boundary to the overlying drifts and outer fan holes. The layout is similar to that obtained using the theory of bulk flow described by Kvapil (1982, 1992). The fans can be drilled vertically or inclined from the horizontal at an angle, typically 70° to 80°. Inclining the fans improves brow stability and access for charging the holes. In the example, the inclination of the fans is 80° and the burden is 3 m (10 ft).

To initiate mining of a new sublevel, an opening slot must be made toward which the fans can be blasted. Several techniques can be used, including blind hole boring and slashing, fan drilling with increasing inclination angles until the production fan inclination is reached, and creating an opening slot longitudinally along the hanging wall. For transverse sublevel caving, upon reaching the footwall, the inclination of the



Source: Bullock and Hustrulid 2001.
Figure 6.5-21 Panel caving method

fans is sometimes steepened to permit recovery of additional ore and to minimize waste extraction. Figure 6.5-20 shows the importance of good drilling precision. If the forward or backward angular position due to incorrect initial alignment or in-hole deviation exceeds 2°, the ends of the longest holes find themselves in the wrong ring. Side-to-side angular deviations can mean that fragmentation is poor due to too little explosive concentration, dead-pressing of explosive, and so on. Thus, careful drilling is of the utmost importance for successful sublevel caving. (*Dead-pressing of explosive* means that the explosive density has increased, by detonation of another nearby blasthole, to the point that the explosive can no longer

detonate. This is most likely to happen with fairly insensitive explosives. More sensitive explosives are more likely to detonate sympathetically rather than become dead-pressed.)

Recovery and Dilution

Sublevel caving lends itself to a very high degree of mechanization and automation. Each of the different unit operations of drifting, production drilling, blasting, and extraction can be done largely without disturbance from the others. Specialized equipment and techniques have been developed leading to a near-factory-like mining environment. As indicated earlier, because every ton of ore is drilled and blasted, there are no longer the uncertainties regarding cavability and degree of fragmentation as occur with block caving. However, a very narrow slice of blasted ore surrounded by a mixture of waste and ore must be extracted with high recovery and minimal dilution. As is easy to visualize, the ore at the top of the ring in the example is more than 40 m (130 ft) away from the extraction point, whereas the waste–ore mixture lies only the distance of the burden in front of the drill ring pattern, on the order of 3 m (10 ft). With care, recoveries on the order of 80% with dilution held below 25% can be achieved.

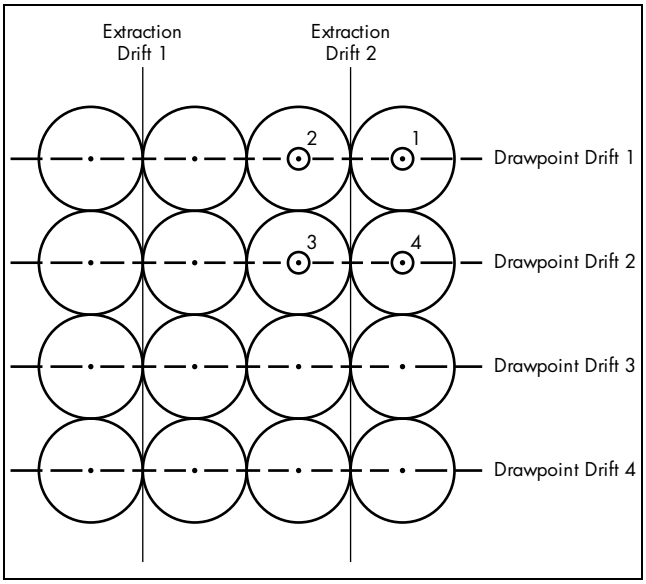
PANEL CAVING MINING METHOD

The term *panel caving* here represents block caving (suggesting the mining of individual blocks) as well as panel caving (used today mostly to indicate a laterally expanding extraction). As one would expect, the system has a great number of variants.

For panel caving, the three most important elements of the extraction system are (1) the undercut level that removes the support from the overlying rock column, (2) the funnel through which rock is transported downward to the extraction level, and (3) the extraction level itself. The basis for system design and performance is the degree of fragmentation present as the rock blocks enter the top of the funnel. In the early days of block caving, it was always considered essential that the materials be soft and cave readily. Today, the trend is to use cave mining on ever harder and tougher ores. As a result, the engineer must thoroughly evaluate the ore body and tailor the design so that successful extraction will result. This is the least expensive mining system, as measured by extracted tons. However, it requires a large amount of testing during the feasibility study, and thus only an engineer who is experienced in panel caving design should attempt to design a panel caving system.

Extraction Principles

Assuming the use of LHD equipment, the major development on the extraction level consists of extraction drifts, drawpoints, and extraction troughs/bells. To simplify discussion, it is assumed here that all drifts have the same cross section. Design is iterative and it is not always obvious where to begin. In this case, one begins with knowing or assuming the size of the material that must be handled. The physical size of the loading equipment is related to the required scoop capacity, which, in turn, is related to the size of the material to be handled. If fragmentation is expected to be coarse, a larger bucket and larger machine are required than if the extracted product is expected to be fine. Knowing the size of the machine, one arrives at a drift size. The orepass diameter should be 3 to 5 times the anticipated largest block size in order to avoid hang-ups. By this rule, the extraction opening should be about 5 to 7 m (16 to 23 ft) for block sizes up to 1.5 m (5 ft). Depending on density and shape, such a block would weigh 5 to 10 t (5.5 to 11 st). A large piece of equipment is required to handle such blocks. It is typical for extraction drifts to be sized according to width/height ratios $w{:}h$ = 4:3, 5:4, and 6:5. For the machine in the example in this section, the drift size would be about 5 × 4 m (16 × 13 ft) or larger.



Source: Bullock and Hustrulid 2001.

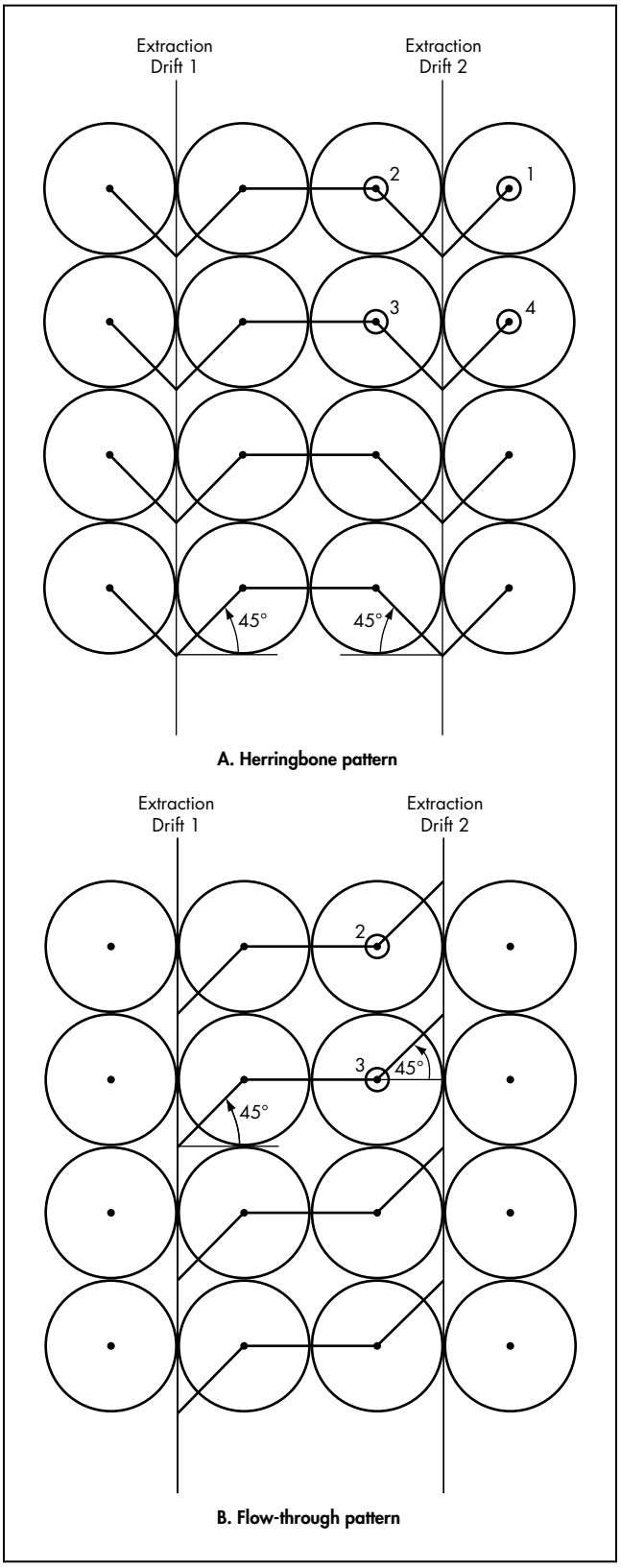
Figure 6.5-22 Panel caving method, showing square layout of extraction and drawpoint drifts

To begin the design of the extraction level, one creates a grid of extraction drifts that are to be traversed by LHD units and the lines of the associated drawpoints. In practice, a series of circles of radius R corresponding to the draw radius of influence on the undercut level are first drawn. Figure 6.5-21 shows two such patterns for staggered coverage with the locations of the extraction drifts superimposed. The value of R depends on the degree of fragmentation, and is larger for large fragments and smaller for finer fragments. This presents a design problem, since fragmentation is generally larger in the initial stages than at the later stages of draw.

The degree of desired coverage is an important design factor. The case of "just-touching" (Figure 6.5-21A) shows that triangles between the circles are not covered. Moving the circles to achieve greater overlap gives, finally, the case of total coverage (Figure 6.5-21B). In this example, it is assumed that R = 7.5 m (25 ft) and a square just-touching drawpoint pattern is used.

Figure 6.5-22 shows the locations of the extraction drifts, drawpoint drifts, and drawpoints on the extraction level. For extraction drift 2, drawpoints 1 and 2 are associated with drawpoint drift 1 and drawpoints 3 and 4 are associated with drawpoint drift 2. The figure shows that two draw circles are associated with each drawpoint in this design. For drawpoint 1, the draw circles are 1 and 4; for drawpoint 2, the draw circles are 2 and 3.

In continuing the design example, one must decide the orientation of the drawpoint drift with respect to the extraction drift. Figures 6.5-23 shows two possibilities involving the use of a 45° angle. Careful examination reveals that the choice affects both loading direction and ease with which openings



Source: Bullock and Hustrulid 2001.

Figure 6.5-23 Panel caving method, showing drawpoint drifts

can be driven. A drawpoint entrance made at 60° to the axis of the extraction drift is at a very convenient angle from the loader operator's point of view. Some designs use 90° angles (square pattern), enabling loading to be done from either direction; however, although 90° pillars provide good corner stability, loading is more difficult. When considering different drawpoint design possibilities, loading machine construction must be taken into account. It is important for the two parts of the LHD unit to be aligned when loading to avoid high maintenance costs and low machine availability.

As mentioned previously, the extraction level must be designed according to the degree of fragmentation expected, so that the openings are large enough to permit extraction. However, large openings are prone to stability problems and, because openings must last for the time required to extract the overlying column of ore, it is important that design, creation, and reinforcement of the openings be done with care. Fortunately, for harder rocks yielding coarser fragmentation, although one expects the ore to be coarse, the host rock is also strong and provides good construction material. For softer rocks yielding finer fragmentation, the openings should be smaller. Protecting the integrity of the openings is of the highest importance and is discussed in more detail in the next section.

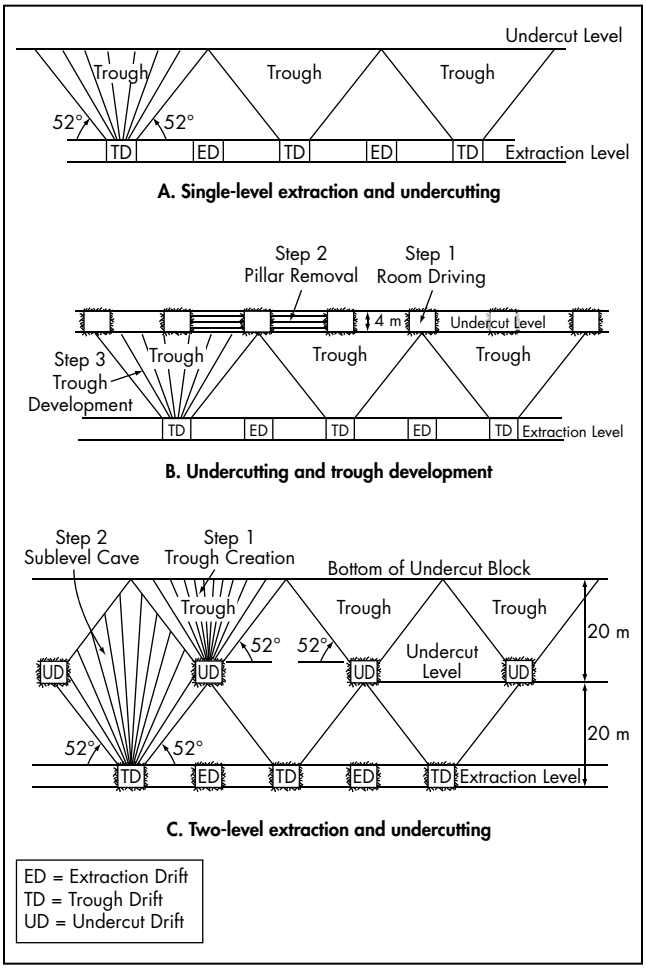
There are a large number of design possibilities for the extraction level. All involve the basic components of fragmentation, radius of influence, draw coverage, machine size, and drift size, which should be examined in roughly that order.

Undercutting and Formation of the Extraction Trough

In the undercutting process, a slice of ore forming the lower portion of the extraction column is mined. As this drilled and blasted material is removed, a horizontal cavity forms beneath the overlying intact rock. Because of the presence of this free surface, subhorizontal side stresses, and the action of gravity, the intact rock undergoes a complex process involving loosening, crushing, and caving. The ease with which intact rock transforms into a mass of fragments is called its characteristic *cavability*. One approach to addressing a material's cavability is to describe the size and shape of the area that must be undercut to promote caving. The other, and more important, approach is to describe the fragment size distribution. The latter is much more difficult to predict, but ultimately more important from a design viewpoint. In this section, both the undercutting process and the design of the trough required to deliver the resulting fragments to the extraction level are described.

The simplest design is to combine undercutting and trough formation into a single step. As described in the previous section, a series of parallel extraction drifts are driven. The center-to-center spacing of the drifts is determined by the size of the influence circles. This example uses the plan layout of Figure 6.5-22. The center-to-center spacing of the extraction drifts is 30 m (100 ft) (4*R*). A series of parallel trough drifts are now driven between the extraction drifts. Starting at the far end of the extraction block, fans of holes are drilled and then blasted toward opening slots.

Figure 6.5-24A shows an example in which the side angles of the fans are 52° and the resulting vertical distance between the extraction level and the top of the undercut is 20 m (66 ft). The trough drifts and troughs can be created either before or after the extraction drifts are driven. The latter case is called



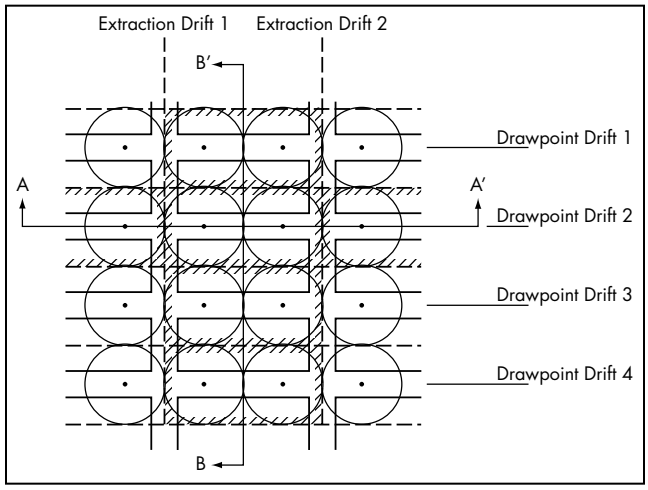
Source: Bullock and Hustrulid 2001.

Figure 6.5-24 Panel caving methods

advance or pre-undercutting. An advantage of this design is that all development is done from one level. An example of the use of this design has been presented by Weiss (1981).

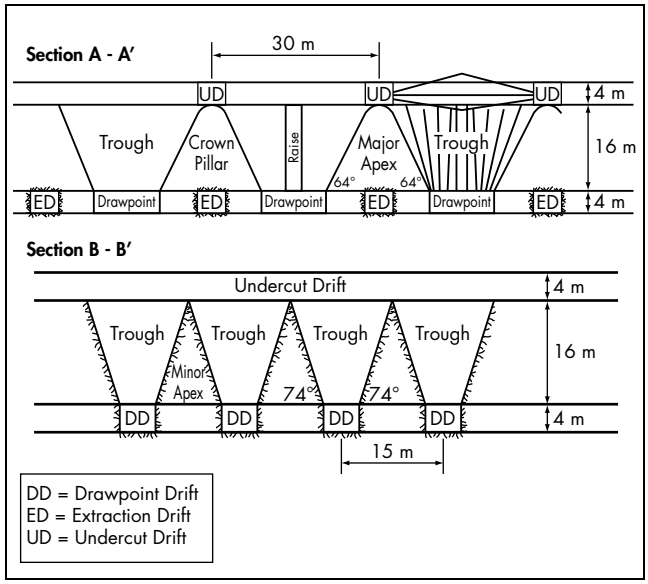
Most mining companies using panel caving have separate undercut and extraction levels. Figure 6.5-24B shows the undercut level designed as a rib pillar mine. The rooms are 5 × 4 m (16 × 13 ft) and the room center-to-center spacing is 15 m (50 ft). In step 2, the interlying pillars are drilled and blasted. In step 3, extraction troughs are created to complete undercut/trough development. It is possible and often desirable to develop the undercut level first, followed by the extraction level.

Figure 6.5-24C shows an alternative design for the same basic extraction-level layout. A separate undercut level has been used with the undercut drifts spaced 30 m (100 ft) center to center. From these drifts, fans of holes are drilled to form a trough. The angle of the side holes has been chosen as 52°. The undercut drifts are positioned directly above the underlying extraction drifts. When the undercut has been created, a sublevel-caving-type fan pattern is drilled from the trough drifts on the extraction level. This completes development. The total height of the undercutting in this case is 40 m (132 ft), which has some advantages in the caving of harder rock types because it greatly increases the undercut volume, thus inducing caving, and the greater volume improves secondary breakage of the harder rock.



Source: Bullock and Hustrulid 2001.

Figure 6.5-25 Panel caving showing plan view of traditional bell development



Source: Bullock and Hustrulid 2001.

Figure 6.5-26 Panel caving showing sectional views of traditional bell development

Figures 6.5-25 and 6.5-26 show plan and section views of a more traditional undercutting and bell layout for panel caving. The previous examples use primarily extraction troughs to demonstrate the principles involved. A trough has the advantage of simplicity of construction but the disadvantage that additional rock is extracted during development. This rock, if left in place, can provide extra stability to both the extraction drifts and the drawpoints. Drawbells are created rather than troughs. The first step in drawbell construction is to drive a drawpoint drift connecting adjacent extraction drifts. A raise is driven from this drift up to the undercut level. Fans of drill holes are then drilled from the drawpoint drift around the opening raise to form the bottom of the drawbell.

Fans of holes are also drilled from the undercut drifts to complete bell formation. A disadvantage of this method is that the amount of development and level of workmanship required is higher than for trough design. As a result, it is more difficult to automate.

For all designs, it is important that a complete undercut be made. If this is not done, very high stresses can be transmitted from the extraction block to the extraction level, causing major damage. Traditionally, the extraction level has been prepared first, followed by creation of the undercut and completion of the drawbells. This procedure has a number of advantages.

Unfortunately, however, very high near-vertical stresses are created just ahead of the leading edge of the undercut. These stresses are transmitted through the pillars to the extraction level and can induce heavy damage to the newly completed level. The result is that repairs must be made before production can begin. The concrete used for making the repairs is generally many times weaker that the rock that has been broken, and the structural strength can never be completely restored.

An alternative to this procedure is to create the undercut first (advance undercutting), thereby cutting off the vertical stress. The extraction level is then created under this stress umbrella. Where this has been done, conditions on the extraction level are markedly improved over those where undercutting was done afterward. Both techniques have advantages and disadvantages, but advance undercutting is preferred for most future mines.

Block Size

The size of a block refers both to the height of the extracted column and its plan area. In the early days of block caving, block height was on the order of 30 to 50 m (98 to 164 ft). With time, this has increased to the point that extraction heights of several hundred meters are now being used or planned. Obviously, because specific development is inversely proportional to the height of the block, there are incentives to make the extraction units as high as possible, within the restrictions imposed by factors such as ore-body geometry and mineral type. Limits are also imposed by the life of the extraction points. If the anticipated life of the extraction point is, for example, 100,000 t (110,000 st), there is no point in selecting a block height yielding 200,000 t (220,000 st) per drawpoint.

Drawpoints can be rebuilt but it is best if they can last the life of the draw. As mentioned earlier, most caving today is done in the form of panel caving rather than the caving of individual blocks. After the initial cave is started, the lateral dimensions are expanded. Cavability affects the size of the undercut that must be created to get a sustainable cave. Relationships have been developed among rock-mass characteristic, hydraulic radius (area/perimeter), and ease of caving. It is possible, unfortunately, to begin initial caving and then for a stable arch to form. The undercut area must then be expanded and/or other techniques such as boundary weakening must be used to get the cave started again. With a sufficiently large undercut area, caving can be induced in any rock mass, which is necessary but not sufficient for successful block caving. Another factor that affects the size of the undercut is the degree of fragmentation that results. Because panel caving is being considered for application to ever stronger rock types, both of these factors—cavability and degree of fragmentation—must be satisfactorily addressed before any method is selected. Unfortunately, the database upon which such decisions must be taken is very limited.

Cave Management

Cave management refers to maintaining control over how much is extracted from each drawpoint each day. It involves a number of different factors. The rate of draw is an important parameter in planning the required area under exploitation. Loosening of fragments appears to be a time-dependent process and must be recognized in planning the draw. The rate must not be so rapid that a large gap results between the top of the cave and the bottom of the block. Sudden collapse of the rock above can result in disastrous air blasts. In high-stress fields, too-rapid draw can cause rock-bursting conditions. In the drawpoint area, there is a zone in which the height of the column under draw increases from near zero (where extraction is just beginning) to the full column height. This is followed by a zone in which the ore-column height decreases to near zero where extraction is complete. It is important to maintain the proper ratio of draw height to slope distance to avoid early introduction of waste from above. Poor cave management can also mean buildup of high loads in various areas and subsequent stability problems. Typical rates of draw, taken from the available literature, are on the order of 0.3 to 0.6 m/d (1 to 2 ft/d).

The proper sequencing of undercut and extraction is a very important aspect of cave management. Unfortunately, relevant design guidelines are difficult to obtain from the literature.

An important design consideration for the extraction level is the means by which oversize is handled. There are a number of different problems to address. The first concern is the management of true hang-ups at the extraction points. Sometimes these can simply be blasted down by careful placement of explosives. At other times the boulders must be drilled first. This is not a simple procedure and involves danger to both machines and humans. The second concern is where and how to handle the movable oversize. These blocks can be (1) handled at the extraction points, (2) moved to a special gallery for blasting, (3) moved to an orepass equipped with a grizzly and handled there, or (4) dumped directly into an orepass for later handling. All variations are used and each company has its own procedures.

Initially, the sizes of blocks arriving at the drawpoints are defined by natural jointing, bedding, and other weakness planes. As blocks separate from the parent rock mass, they displace and rotate with the loose volume, occupying a larger volume than does the intact mass. The swell volume is extracted from the extraction points, providing expansion room for the overlying intact rock. Loosening eventually encompasses the entire column. As the column is withdrawn, the individual blocks abrade and split, resulting in finer fragmentation than in the early part of the draw. The initial fragmentation due to initial fractures in the rock is called primary fragmentation. As the column moves downward and new breakage occurs, the resulting fragmentation is called secondary fragmentation. Data concerning this transition from primary to secondary fragmentation are very difficult to obtain.

ACKNOWLEDGMENTS

The majority of this chapter is taken from Bullock and Hustrulid 2001.

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CHAPTER 6.6

Comparison of Surface Mining Methods

Ernest Bohnet

INTRODUCTION

Each mining operation is unique in some aspect—ownership; ore-body location, size, geometry, grade, mineralogy, hydrology, or geotechnical parameters; or constraints such as environmental regulations. However, all mining operations are alike in one way—the prime objective of any mining project is to maximize the return on investment. This is the golden rule, or in the language of mining, the investor's "Law of Conservation."

A mineral resource, by definition, does not become a mineable reserve unless the mineral or minerals can be extracted economically (and legally). The choice of the most economic method is dictated primarily by the type of deposit to be mined, the mining rate, and the physical characteristics of the deposit.

A comparison of surface methods must take into consideration two prime objectives of mine planning: to develop the most economic plan for the overall project that will maximize the return on the monies invested and to maximize the recovery of the resource. Surface mines can be subdivided into various classes and subclasses, which can be illustrated as follows:

Class	Subclass	Method
Mechanical	—	Open-pit mining
		Quarrying
		Strip mining
		Auger mining
Aqueous	Placer	Dredging
		Hydraulic mining
	Solution	Surface techniques
		In-situ leaching
		Evaporite processing

The purpose of this chapter is to discuss the relative advantages and applications of one method versus another, as well as some of the more important evaluation parameters that must be considered in any method comparison.

FACTORS AND CONDITIONS

Open-pit mining and strip mining are the two most dominant types of surface mining methods in the world, accounting for approximately 90% of the surface mineral tonnage. Strip (open-cast) mining is used for large, tabular, flat-lying ore bodies or mineral seams, such as coal, that are relatively close to the surface. Backfilling of these mines is usually economically feasible and desirable as part of the concurrent reclamation requirements.

Open-pit mining is typically applied to disseminated ore bodies or steeply dipping veins or seams where the mining advance is toward increasing depths. Backfilling usually cannot take place until the pit is completed; even then, the prohibitive cost of filling these pits with all of the waste rock removed at the end of the mine life would seriously jeopardize the project's economics. Very few large open pits in the world could support such a costly handicap.

Aggregate mines produce a bulk commodity for local markets. These quarries employ the same mining methods as other open-pit mines.

Seam deposit mining, like coal, uses different planning methods and terminology as compared to open-pit mining for base and precious metals. However, all surface mining methods have one common element: mining proceeds until the economic or legal limits are reached. The method of determining the economic limit is different for the two types of deposits. Coal, for example, in most instances, is readily distinguished from waste rock and is of a near-uniform quality and value per ton mined in each seam. Therefore, mine design limits are determined by the break-even strip ratio, commonly defined as cubic yards (cubic meters) of waste to tons (metric tons) of coal that can be mined. Since 1 ton of coal is approximately equal to 1 yd^3 (1 t ≈ 0.84 m^3), the ratio is similar to a volume ratio.

The pit limits for a disseminated deposit, such as base or precious metals, is determined by a break-even economic analysis of the value of a finite incremental pit expansion. This economic analysis is a two-step process where first the net operating value is determined for each ton of material in a

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block model, above and below the ore cutoff grade. Second, the pit limits are determined by expanding the pit shape in increments to a break-even point where the net operating margin equals the cost of waste stripping. Given that each ton of ore has a different grade and net value, a break-even stripping ratio cannot be used in the determination of the pit limits.

As a final test, the increment between an existing pit wall to the economic limit must have sufficient width for the proposed mining method. Complicating this matter further is the potential for multiple ore processes, such as milling and leaching of the same ore type. Milling is a higher cost process method used to increase metal recoveries, whereas leaching is a slower process with correspondingly lesser recoveries and lower costs.

Therefore, the mining engineer must have sufficient knowledge of the overall project, from ore geometries to the process extraction methods, in order to maximize the project return on investment. This includes an awareness of all cost and recovery parameters, as well as the factors that affect the value of the salable product. For example, where process capital and operating costs are high, ore selectivity and the choice of appropriate mining method play important roles in mine design. Mining cost increases that may be required to optimize the project's return can be more than offset by reduced process capital and operating costs per unit of salable product.

The total amount and type of material to be moved per time period, coupled with ore-body geometries, are the prime criteria in selecting the most economic and applicable mining method.

Other factors that affect the type of mining method as well as type and size of equipment include

- Topography;
- Remoteness of operation;
- Support infrastructure;
- Availability and cost of skilled labor;
- Climate and altitude;
- Environmental, legal, or physical boundary restrictions;
- Groundwater quantities;
- Slope stability and ground conditions;
- Investment risk;
- Blasting restrictions;
- Life of operation; and
- Waste rock disposal.

Topography can play a major role in selecting the most appropriate mining method. Generally, the more rugged the topography, the more limited are the choices of mining method and level of overall production.

Remote operations require substantial infrastructure to maintain the standard of living needed to attract a well-qualified work force. Therefore, the mining and processing methods selected must result in high productivities per employee shift. Better ore selectivity in the mine will reduce process plant requirements and could, therefore, reduce overall staffing requirements. If the ore grade can be raised sufficiently, the ore may be transported off-site for processing at a less remote location. Large equipment can be used for the bulk of the waste stripping and small equipment employed to selectively mine the ore.

Where labor costs are high, the same philosophy can be applied. Lower labor costs and the level of labor skills available can also affect the choice of mining method. For example, front-end loaders and mechanical drive trucks will require less diverse maintenance skills as compared to the labor force needed to maintain electric/hydraulic shovels and electric haul trucks.

With adverse climatic conditions, mining operations may be seasonal. Seasonal and short-life operations may use contractors for all mining operations or for only the bulk waste stripping.

Environmental and social considerations play an increasingly important role in the type of mining operation permissible. Open pits in populous areas may require backfilling, whereas the same type of operation in a remote location may have minimal environmental constraints. For example, hydraulic mining, outlawed in California (United States) in 1884, was used to remove the unconsolidated overburden at the Bougainville Copper mine (Papua New Guinea) in 1969–1971. Environmental and aesthetic conditions may be so restrictive in some areas as to eliminate surface mining options altogether. In some cases, this forces the use of underground mining, which will result in reduced mineral resource recovery.

Groundwater conditions and slope stability can play an important role in mining method options, mine design, production scheduling, and equipment selection. A large dragline may not operate safely where ground conditions are wet and unstable and the risk of slope failure is high. If large water inflows are anticipated, mining may have to be accelerated to avoid high pumping costs from a lengthy operation. Ore can be stockpiled at the processing plant and treated months (or even years) after the mine is completed.

The risk of investment varies from country to country; however, if risks are high, then capital expenditures are usually minimized. Front-end loaders, with shorter useful lives and lesser capital costs, are preferred over large shovels and draglines. The larger the project and the longer the development period, the greater the investment risk.

Blasting restrictions due to the operation's proximity to populous areas or in foreign countries with explosive use constraints can also affect mining method selection. Continuous mining machinery similar to pavement cutters or ripping can be used in some circumstances as an alternative to drilling and blasting.

Available waste disposal areas may be located a significant (trucking) distance from the surface mining operation. A viable alternative in this situation is crushing and transporting of the waste via a line of conveyor belts, and this in turn can affect the mining methods. Where the ore has a high value per ton, in excess of 100 times the cost of mining, very high strip ratios can be economically justified, as well as underground mining methods. The surface mining method in these circumstances is designed to be primarily a waste removal operation. Ore mining is a small cost component of the overall operation, and great emphasis must be placed on efficient waste production and mining methods. Given that most open-pit and strip mines have waste-to-ore ratios higher than 1:1, the majority of surface mines are waste rock operations. It is, therefore, important to ensure that any effort to reduce ore mining costs does not increase waste mining costs. For example, in a coal stripping operation using shovels and trucks, usually a slot is left on one side of the backfilled pit for coal haulage to minimize coal mining costs. This slot prevents efficient waste haulage around both sides of the pit, thereby increasing waste haulage distances and costs significantly. As strip ratios increase, the mining method and haul road system will require modification to allow more effective waste transportation and reduced overall mining costs. Digging conditions including the size

of the broken material will impact the choice of hydraulic/cable shovel, dragline, bucket-wheel excavator, and front-end loader.

Quarrying is a special type of open-pit mining used to produce aggregates and dimension stone products. In dimension stone quarries, rock joint fractures are infrequent, and, therefore, the bench faces are vertical. The lack of fractures often permits near-vertical highwalls that can approach up to 1,000 ft (300 m).

Auger mining is primarily used to remove coal from under a final highwall. The final highwall position is determined by economic stripping limits, property boundaries, or physical constraints such as a body of water.

In the case of physical constraints, auger mining may be the only alternative to increase the amount of coal recovered. However, if the highwall position was determined using a break-even stripping ratio analysis, the cost of stripping the final 100 to 300 ft (30 to 100 m) break-even increment should be compared to auger mining costs and profitability.

Aqueous surface mining methods, uniquely involving the use of water for extraction, can be used in special circumstances. The mining of placer-type deposits that contain concentrations of heavy metals in unconsolidated overburden are particularly suitable to dredging and hydraulic mining if an adequate water supply is available and the mining operation can comply with the applicable environmental regulations.

Hydraulicking, now often termed *hydraulic mining*, ideally requires the presence of a natural gradient away from the deposit to facilitate hydraulic transport of the resultant ore or waste slurry to the process or disposal area.

Dredging can be used if a natural or human-made body of water overlies the placer deposit. Dredging machines can be classified as mechanical, bucket-line, bucket-wheel suction, and dipper; or hydraulic, suction, and cutterhead. The type of dredging equipment selected depends on the type of material to be excavated.

Solution mining using surface or in-situ techniques is applicable to deposits of minerals that can be recovered usually by dissolution, but also by melting, leaching, or slurrying. The two methods are similar and are differentiated primarily through the location and the type of minerals recovered. Surface leaching generally employs heap (or dump) leaching of mineral values; copper, gold, silver, and uranium are common examples.

In-situ mining uses barren solution, introduced down one set of wells, and the loaded solution returns to the surface through concentric or another set of wells. This mining method is primarily used with sulfur, evaporite, or water-soluble minerals. In-situ leaching uses chemical or bacteriological reagents, usually mixed with water, to selectively dissolve the valuable minerals. Drill holes are typically used to inject and recover the solution. Extraction may also be enhanced by rubblization of the ore zone prior to solution application. Solution mining will increase in the future as more effective reagents are developed and application methods are improved.

The preceding discussion has focused primarily on factors and conditions that can affect surface mining options. New technologies will result in a greater range of alternatives in the future. Therefore, the mining engineer needs to continually review these developments as applicable to each old and new situation encountered. Basic assumptions must be revisited to ensure that past assumptions of extraction are still the most economically viable in light of new developments. Mining is dynamic in nature; therefore, mining methods will continue to evolve to meet new economic, social, and environmental conditions.

DEVELOPMENT VERSUS THE PRODUCTION SEQUENCE IN OPEN-PIT AND STRIP MINING

Selection of the most suitable mining method and the most appropriate size and type of equipment are closely associated. The following discussion primarily addresses loading and hauling fleets, as applied to quarries, strip mines, and open-pit operations.

The size of the operation over time has a direct bearing on the type of equipment selected. Loading and hauling fleets are dependent fleets, and, therefore, the effectiveness and availability of each affects the fleet requirements of the other. For example, if a mining operation had only one large shovel, the truck production would only be effective when the shovel is operable and broken rock is available at the shovel face for loading. Therefore, to have an effective operation, more than one unit per equipment fleet is desirable. A mining operation should have at least one drill per shovel and at least one backup shovel/loader to load trucks when a shovel breaks down or has to undergo a scheduled maintenance shutdown.

Large mining operations require time to develop sufficient working areas for effective use of the equipment. Therefore, the equipment selected for full production may not always be suitable for the development period of each pushback. The development (or preproduction) period may require different mining methods because of lesser material movements and different types of material to shift. During the development stage, access haul roads need to be constructed, and in rugged terrain, bench widths may be much narrower than those available during production. Therefore, smaller equipment may be needed during this stage of mine development.

Sufficient time must be allowed in the production schedule to drill, blast, and remove the material from the mine. Narrow benches do not allow continuous mining unless access for drilling and blasting is provided in the plan.

During preproduction and during the stripping phase of each mining sequence, taller benches may be used. For example, stripping may be accomplished on 40-ft (12-m) benches and ore production on 20-ft (6-m) benches to minimize ore dilution. The advantages of higher benches include the use of larger equipment, minimal access construction, and overall cost reductions. However, taller benches will also increase the vertical lift for all material below the pit exit elevation.

Large open-pit and strip mines are subdivided into smaller phases that are sequenced to optimize project economics. The objectives of dividing a surface mine into a development phase and then production phases are to

- Minimize preproduction stripping requirements to obtain sufficient stripped ore reserves to sustain production;
- Shorten the time between stripping and ore production;
- Develop the highest net-value ore first and mine the lowest net-value ore later;
- Avoid high stripping peaks that result in excessive equipment purchases and labor requirements;
- Blend the ore (or coal) to produce a less variable product; and
- Optimize equipment fleets for each of the development and production phases.

The development and production sequencing requirements must be considered in selecting the most appropriate mining equipment types and mining method.

METHOD ADVANTAGES AND DISADVANTAGES

The advantages and disadvantages of one type of surface mining versus another are often related to the types of equipment used and the associated costs and benefits derived from their use. In the following discussion, emphasis is placed on the mechanical extraction methods generally used in open pits, quarries, and strip mining.

Strip mining has the greatest choice of equipment, particularly for the mining of relatively flat-lying coal seams in gentle topography. In the following discussion, the various types of mining methods and equipment applications for strip mining (using draglines, shovels, bucket-wheel excavators, conveying systems, and combinations of these types of systems) as well as those for open-pit mining (usually limited to shovel/truck or front-end-loader/truck-type operations) will be discussed. (Shovel/rail-type operations of the past have almost been phased out in favor of the greater flexibility of mining with truck haulage.)

Open-pit loading equipment is usually matched with haul trucks that can be loaded in four to six passes. The choice of loading unit is dependent on total production requirements, the minimum number of active work areas, ore selectivity, and blending requirements. Additional factors to be considered are the type of material to be loaded and the ground and digging conditions. In hard digging or with sharp rock in wet climates, tracked shovels are preferable over rubber-tired loaders. However, if the mine production requirements are relatively small or the mine life is short, the lower capital cost of front-end loaders must be considered, particularly in high-risk investment areas.

The rubber-tired loader is commonly used to load material that is described as easy to medium digging and for operations requiring less than 100,000 tons (90,000 t) of daily material movement. Loaders are highly mobile and, therefore, are ideally suited for situations requiring frequent movement from one mucking face to another for ore blending requirements. Loaders also can be used to load, haul, and dump material into a crusher from blending stockpiles dumped near the crusher. Some operations have a mismatch of large shovels and a small crusher. A mismatch occurs when the crusher rating of tons per hour is much less than what the shovel can produce through the truck fleet. If this situation occurs, there are advantages to stockpiling the material at the crusher to avoid truck dumping delays and thereby maximize shovel and truck productivities. The extra cost of a stockpile-rehandling loader is more than offset by the increase in shovel and truck productivities and their respective reduced fleet requirements.

Loader cycle time is approximately 25% to 50% greater than a shovel's cycle time. Therefore, a loader has to be 25% to 50% larger (bucket capacity) than a shovel to attain the same productivity. In addition, loader availabilities are typically less than a shovel's, and this factor also needs to be considered in sizing the loader fleet. The loader bucket-fill factor is usually less than a shovel's in medium to hard digging. Hydraulic shovels have similar limitations and advantages as a cable shovel. The hydraulic shovel is typically not used in very hard digging. Hydraulic shovels are available in a range of sizes comparable to front-end loaders. However, because the larger sizes over 30 yd^3 (23 m^3) are not common, large cable shovels are used as the prime loader at mines where production requirements are in excess of 200,000 tpd (180,000 t/d).

Hydraulic shovels can effectively load from smaller bank heights than can cable shovels; therefore, if ore selectivity is very important, these shovels can improve ore selectivity and minimize dilution. A hydraulic shovel has additional advantages over a cable shovel in the sense that the bucket can be vertically positioned anywhere in the mucking face and selectively load from either the top half or bottom half of the face. This advantage is particularly useful in mining seam deposits where waste separation from the seam is necessary, or in some precious metal (gold) deposits that have highly variable grades and/or oxide–sulfide mineralization contacts. Hydraulic shovels are also more mobile and require less infrastructure because they are diesel versus large electric cable shovels with trailing cables, cable crossings, and power lines in the pit. There are capital and operating cost trade-offs as mentioned.

Hydraulic shovels can also be equipped with a backhoe-type loading bucket. This permits the shovel to sit on top of the bench and load trucks that are positioned on the same bench or on the bench below. Backhoe loading is particularly useful for loading a dipping seam or wedge of material where loading with a cable shovel would be impossible. Backhoe loading is also very useful in wet conditions, permitting the trucks to travel on a drier bench above the water, with correspondingly increased tire life.

The useful life of loaders, hydraulic shovels, and cable shovels varies according to the application, maintenance standards, and the company's general replacement policy. However, life for a front-end loader is typically in the 3- to 4-year range, hydraulic shovels in the 8- to 12-year range, and cable shovels are usually replaced at 20-year intervals. The longer the useful life, the more obsolete the equipment becomes in comparison to new technological advances. In order to combat this technological aging factor, many equipment manufacturers are progressing toward complete modular part replacement designs. These designs allow the mine operator to exchange older components for new, more reliable modules that may have more advanced features.

Another factor that influences the choice of loading equipment is the trade-off between lower and higher capital costs versus higher and lower operating costs. Complicating this analysis is the difference in useful life. Given that the prime objective in mining is to maximize the return on investment, the time value of money must be taken into consideration in any trade-off analysis. Discount rates of 10% to 15% are commonly used to convert future costs into a net present value (NPV). The higher the discount rate, the greater the emphasis on reducing current costs (capital). Expenditures after 10 years have less significance on the total NPV. This method of ranking alternative fleets can be applied to all choices: dragline versus a shovel-and-truck fleet in a coal stripping operation, or cable shovels with 170-ton (150-t) trucks versus front-end loaders with 85-ton (76-t) trucks, trucking versus crushing and conveying. However, other tangible and intangible factors may be present that often are overlooked in this capital-versus-operating-costs analysis. An example of this would be greater ore selectivity, which means less ore dilution and less ore discarded as waste, thereby resulting in higher-grade ore to the processing plant. A small-grade change can significantly affect overall project economics.

As an example, project economics are very sensitive to changes in commodity price. However, grade changes and recovery changes have nearly the same effect as a commodity price change. Therefore, if ore selectivity can improve the ore grade by 10%, this is more significant than a 10% increase in both mine capital and operating costs. This factor must be taken into consideration in any equipment trade-off economic analysis.

Other factors that influence the choice of equipment (and therefore mining method) are less tangible. As previously discussed, these include mobility requirements, ore blending, hard or easy digging, maintenance skills of the available work force, size of the operation, working area, and advance rate required.

In respect to this last point, open-pit mines are subdivided into phases. Each phase is somewhat independent of the following phases; however, the object is to remove waste rock that exposes ore in a timely manner before the ore reserves (from the prior phase) are exhausted. Circumstances exist where the advance rate or number of benches to be mined each year in a phase requires more than one loading unit. If the working width of a bench is too narrow to permit more than one unit, then larger equipment may be required.

An example of this situation is a mine that was using 15-yd^3 (11-m^3) hydraulic shovels as the prime loading fleet. The mine plan indicated that after the tenth year of operation, the fourth mining phase required the movement of 15 million tons (13.5 Mt) of material per year for 5 years to expose ore. Given that one shovel could not move this material and since two shovels could not effectively work on the bench, the alternatives were to purchase a large 40-yd^3 (30-m^3) cable shovel or start mining the phase earlier. A discounted cash-flow analysis indicated the better choice was to purchase one 40-yd^3 (30-m^3) shovel for the bulk of the waste stripping and to use the smaller shovels for ore production.

The purpose of subdividing an open pit into phases is to improve the project economics. This can also mean that material movement requirements through time will change significantly. It does not mean that mining quantities will change dramatically each year. An effective mine production schedule must be developed to consider equipment and personnel requirements. A smooth schedule will give the mine operations and maintenance departments sufficient time to build up their organizations without significant annual variances. If total material movements are increased during the life of the mine, these increases should be smoothed to form a new production level that remains constant at a higher level for a number of years.

During preproduction, or at some point in the life of the mine, this may not always be feasible, and an alternative to this is contract stripping. Contract stripping can be justified if the mine life is short or the stripping peak is of fairly short duration (i.e., between 3 and 4 years). Again, the trade-off is lower capital costs versus higher operating costs. The intangible benefits include the avoidance of hiring and training a large work force for a relatively short time period, greater assurance that the stripping will be carried out on time and per the contract terms, as well as fewer management resources being required. An intangible disadvantage is less control over the operation, as it must be recognized that a contractor's goal is to move a volume of rock, not necessarily to separate ore and waste in the mine owner's best interest. Adequate ore control procedures must be conducted as part of the mine owner's responsibility to minimize ore dilution and loss. It is also the mine owner's responsibility to ensure that the mine plan is accurate and well understood, otherwise extra costs will be charged by the contractor for scope deviations.

The life of a mining project plays an important role in selecting the most appropriate mining method, equipment fleets, and whether to use a contract miner. Smaller companies may have more capital constraints and therefore favor contract mining.

The lack of available capital also may force the mine operator to look for used equipment for its initial mining. This equipment may be replaced within a few years with new equipment if mine life is sufficiently extended and positive cash flow has been demonstrated to the lending agency's satisfaction. Any economic analysis of used versus new equipment is extremely subjective. Equipment maintenance costs are composed of maintenance labor and repair parts. The amount of maintenance required is related to the condition of the equipment purchased and also to the operating hours on the equipment. If the equipment has been overhauled adequately, maintenance costs will be similar to that of a piece of equipment that has been in operation for a relatively short time. The maintenance requirements of rebuilt equipment will increase at an aging rate that is approximately double that of a new piece of equipment.

In order to adequately explain this factor for economic analysis, consider the following example of haul trucks that normally have a useful life of a maximum of 75,000 hours. If the average maintenance cost during the life of a truck has a value of 1.0, then the relationship of maintenance costs to life can be stated as follows:

Operating Hours	Maintenance Cost Factor	Mechanical Availability, %
0–5,000	0.30	85
5,000–20,000	0.70	80
20,000–40,000	1.00	75
40,000–60,000	1.20	70
60,000–75,000	1.35	65

If a truck with 60,000 to 70,000 hours is overhauled, maintenance costs should drop to the equivalent of a new truck with 5,000 operating hours. However, after operating only 30,000 hours, the overhauled used truck would have the same maintenance requirements as a truck with 65,000 hours.

Used equipment availabilities are also much less than that for new equipment. New equipment usually will have manufacturers' warranties and close field support.

The effect of purchasing used equipment with lower availabilities is that more units are needed to obtain the same production level, and the maintenance work force has to be significantly larger. Nonetheless, if capital funding is limited, used equipment or a contract miner may be the only viable options.

The comparison of one type of mining method versus another should also compare the various types of equipment that are dependent on another type, such as loading and hauling equipment, loading productivity versus crusher capacity, and drilling unit capacity versus scheduled shovel production.

For any analysis, it is better to have the first unit in the chain of production with greater capacity than the downstream

unit. Therefore, drill capacity should exceed shovel/loader capacity, shovel capacity should be slightly higher than crusher capacity, or the capacity of two shovels should be matched with the capabilities of one crusher.

A mismatch occurs when there are only a few units in production and the capacity of one unit dependent on the production of another is significantly different. For any comparison of different mining methods or equipment types, it is very important to conduct the economic analysis using accurate and correct assumptions.

Two of the major expense areas in any open-pit or stripping operation are the functions of loading and hauling. Typically, 30% to 50% of the total mining cost is related to hauling all material types out of the pit. A cost comparison of equipment sizes and alternatives must be based on an optimum selection of loading and haulage fleet sizes, otherwise the comparison may be unfairly skewed one way or the other.

The first step in any comparison is the determination of unit productivities per operating shift or per hour. Loader productivities can be estimated by using the anticipated load time per specified truck size and dividing this into the estimated effective minutes per shift. Productive minutes per 8-hour shift (with no downtime) will vary from 330 to 420 minutes, depending on the efficiency of the assignment systems used and the balance between loading and haulage units. Industry standards without a computerized truck dispatch system typically use 350 effective minutes per 8-hour shift as a basis for comparison.

Truck productivities can be determined from haul profiles for each specific time period. For initial studies comparing one system to another, average productivities per time period can be used to determine fleet requirements.

One method that can be used to more accurately analyze the loading and haulage fleet requirements is the binomial method. The best way to explain this method is by using a simple case to demonstrate its use in determining dependent fleet production.

Example 1. Consider a fleet consisting of

- One loader, 80% mechanical availability and an estimated productivity of 9,000 tons (8,200 t) per operating shift, and
- Three haul trucks, 70% mechanical availability and an estimated productivity of 4,000 tons (3,600 t) per operating shift.

Determine the fleet capacity.

Solution. Assume that the fleet is scheduled 100% of the time and will only be inoperable if either the loader or all the trucks are down for repairs.

One could incorrectly assume that the average loader production would be 80% of 9,000 tons (8,200 t) per shift, or 7,200 tons (6,500 t) per shift. However, because the loader production is dependent on available haul trucks, the truck downtime distribution must be considered.

Table 6.6-1, developed using the binomial formula, shows the fraction of time that x units are available out of a fleet of n units with a given availability of p where this fraction equals

$$\frac{n!}{x!(n-x!)} \times p^x \times (1-p)^{(n-x)} \qquad (6.6\text{-}1)$$

Table 6.6-1 indicates that 34% of the time, all three trucks are available (row 3, column 3); 44% of the time, two are available; 19% of the time, only one is available; and 3% of the time, the whole truck fleet is down. When less than three trucks are operating, the loader production will drop to the amount that the trucks can haul—in this case, 8,000 tons (7,300 t; i.e., 44% of the time) and 4,000 tons (3,600 t; i.e., 19% of the time).

Table 6.6-1 Fraction of time that (*x*) units are available out of a fleet of (*n*) units that have a probable availability of 70%

	Number of Units Available, *x*						
Fleet Size, *n*	0	1	2	3	4	5	6
1	0.30	0.70					
2	0.09	0.42	0.49				
3	0.03	0.19	0.44	0.34			
4	0.01	0.08	0.26	0.41	0.24		
5	0.00	0.08	0.13	0.31	0.36	0.17	
6	0.00	0.01	0.06	0.19	0.32	0.30	0.12

In summary, the fleet capacity can be stated as follows: The loader operates 80% of the time and during this time, 34% will be at 9,000 tons (8,200 t)/shift, 44% will be at 8,000 tons (7,300 t)/shift, and 19% will be at only 4,000 tons (3,600 t)/shift. Total fleet capacity would average

$$0.80 \times 0.34 \times 9{,}000 = 2{,}448 \text{ tons/shift}$$
$$0.80 \times 0.44 \times 8{,}000 = 2{,}816 \text{ tons/shift}$$
$$0.80 \times 0.19 \times 4{,}000 = \underline{\quad 608 \text{ tons/shift}}$$
$$\text{Total} = 5{,}872 \text{ tons/shift (5,340 t)}$$

From this simple example, one can see that production from the loader would be 18% short of the initial estimate of 7,200 tons (6,500 t) per shift that was determined without consideration of the haul fleet. The same analysis holds true for haulage truck requirement determinations. For example, if only the number of haul trucks needed was examined, the following could result.

- Annual target objective: 1,800,000 tons (1,636,000 t)
- Shifts scheduled: 250
- Tonnage requirements per shift: 7,200 tons (6,500 t)
- Average truck productivity: 4,000 tons (3,600 t)/shift
- Operating trucks needed per shift: 1.80
- Three trucks at 70% availability will average 2.1 shifts

Therefore, it could be incorrectly assumed that three trucks would be sufficient. However, from the previous discussion it is known that if the loading fleet contained only one loader, then 20% of the time the haul fleet would be idle waiting for the loader to be repaired. It is also known that the loader could not keep up with three trucks, and production would be limited to 9,000 tons (8,200 t) per shift, not the 12,000 tons (10,900 t) per shift indicated by the haulage capacity. Therefore, haulage capacity would be

$$250 \text{ shifts} \times 0.80 \times 0.34 \times 9{,}000 = \quad 612{,}000 \text{ tons}$$
$$250 \text{ shifts} \times 0.80 \times 0.44 \times 8{,}000 = \quad 704{,}000 \text{ tons}$$
$$250 \text{ shifts} \times 0.80 \times 0.19 \times 4{,}000 = \quad 152{,}000 \text{ tons}$$
$$= 1{,}468{,}000 \text{ tpy}$$
$$(1{,}335{,}000 \text{ t/a})$$

The solution in this case would be to purchase another loader or work more shifts. With larger fleets, the shortfalls caused by calculating fleets independently are not so easy to quantify without using a computer program that considers

Table 6.6-2 Equipment operating shifts attainable per shift with a probable availability of 70%

	Number of Operators, *x*					
Fleet Size, *n*	**1**	**2**	**3**	**4**	**5**	**6**
1	0.70					
2	0.91	1.40				
3	0.97	1.76	2.10			
4	0.99	1.91	2.56	2.80		
5	1.00	1.97	2.80	3.33	3.50	
6	1.00	1.99	2.92	3.66	4.08	4.20

the availability interaction of the two fleets. Haulage truck productivities can be averaged for these calculations; however, in reality, haul distances will vary, and this can be used to counter some of the production shortfalls that would otherwise occur when there is a shortage of trucks or loading units. For example, if the mine were short of trucks, then the shortest haul distances would be used; and conversely, if the mine were short of loading units, longer hauls would be used to counterbalance the shortfalls that would otherwise occur. This may not always be possible, but haulage distance flexibility should be one objective of a good mine plan.

The lack of a loading unit more severely impacts haulage capacity than the lack of one truck. For this reason, it is better to have slightly more loading standby capacity and use an under-trucking philosophy to minimize both capital and operating costs. Haulage costs per ton of material mined will typically be several times the loading cost.

It is not always in the miner's best interest to have operators for each piece of loading or haulage equipment. In fact, as the fleet sizes increase or availabilities decrease, the extra operators are more of an operating cost liability.

Binomial distribution can be used to determine the most likely number of operating shifts that will be obtained for a given fleet size based on the number of operators or equipment scheduled, where x is the number of assigned operators, n is the number of units in the fleet, and b is the binomial probability of n things taken one at a time (see Table 6.6-2).

Table 6.6-2 is derived from the following formula:

operating shifts/shift =

$$x\frac{\sum_{j=x}^{n} bj}{} + 1(b_{(x-1)}) + x - 2(b_{(x-2)}) + \ldots 1(b_1) \qquad (6.6\text{-}2)$$

For example, the anticipated operating shifts for five pieces of equipment with 70% availability and four operators is 3.33. Adding one more operator to the payroll would only increase loader operating time by 0.17 shifts. From another viewpoint, if a truck fleet was purchased to cover five operating loading units, then the trucks purchased for the fifth loading unit would only operate 17% of the time. During the remaining 83%, the trucks and their drivers would be wasted capacity unless the remaining loading units could handle the extra trucks effectively.

In order for the remaining loading units to handle the extra trucks, the operation must be initially designed to be under-trucked when all the scheduled loading units are operating.

The following example is taken from an actual case in China for which the initial fleet specified for review was severely over-trucked.

Example 2. Initial production plans indicated an overburden stripping requirement of 160,000 tons (145,500 t) per shift. Production rates for the shovels and trucks were estimated to be 20,000 tons (18,200 t) and 2,500 tons (2,300 t) per operating shift, respectively. It was proposed to purchase and schedule 10 shovels to operate at 80% availability to give a loading capacity of 10 × 80% × 20,000 tons (18,200 t) = 160,000 tons (145,500 t). In order to cover each shovel, it was proposed to purchase 10 trucks and schedule 8 of them to operate at 80% availability, giving an average capacity of 10 × 80% × 2,500 tons (2,300 t) = 20,000 tons (18,200 t) per shift, thereby matching each shovel's productive capacity.

Solution. The devised production plan assumed that trucks and their drivers, due to physical constraints, were not transferable between the various loading units and the haul distances were always the same length. Therefore, the total equipment requirements were estimated to be 10 shovels and 100 trucks, of which 10 shovels and 80 trucks would always be scheduled to operate.

By applying the binomial function to a group consisting of one shovel and 10 trucks, the anticipated output would be approximately 15,000 tons (13,640 t) per shift, or 150,000 tons (136,400 t) for 10 shovels. This is 10,000 tons (9,100 t) short of the objective and is caused by the fact that there will be times when less than eight trucks are available, resulting in lost shovel production. Situations will also occur when more than eight trucks are available, but given that the shovel can use only eight trucks, no increase in production can be gained to counterbalance the shortfall.

Referring to Table 6.6-3, if the trucks could be transferred from shovel to shovel in order to balance the truck shortfalls (Case 3), then production will be increased by approximately 10,000 tons (9,100 t) and the target attained. However, this is avoiding the real problem, over-trucking. The 160,000-tons (145,500-t)-per-shift target can be obtained with different shovel/truck fleet combinations with a lower capital expenditure and at a reduced operating cost. Table 6.6-3 illustrates various fleet sizes that would produce 160,000 tons (145,500 t) per shift. For comparison, operating and capital costs of each fleet are listed, and the basis of these estimates is provided in Table 6.6-4. Case 1 is the base as originally specified; Case 2 is the base case but with the equipment increases that would actually be needed to reach 160,000 tons (145,500 t) per shift with the same constraints as Case 1. Case 3 is the production obtained if trucks can be transferred from one shovel to another, Case 4 is an optimum fleet balanced to minimize costs with fixed hauls, and Case 5 is an ideal case where haul distances can be flexed to accommodate truck or loading equipment shortages and maintain peak production efficiencies. In this last case, production efficiency is increased by changing the mine plan to allow flexible haul distances. Instead of a fixed 2,500 tons (2,300 t) per truck shift, dump points can be varied to shorten or increase haul distances to give a truck productivity range of 2,100 tons (1,900 t) to 2,600 tons (2,400 t) per shift but still average 2,500 tons (2,300 t) per effective truck shift.

This example shows the magnitude of the savings that can be achieved by selecting the optimal fleet size and removing some of the high-cost constraints.

Table 6.6-3 Optional fleets vs. cost (Example 2)

	Case				
Equipment Characteristics	**1**	**2**	**3**	**4**	**5**
Shovels in fleet	10	11	10	11	11
Shovels scheduled	10	11	10	9	9
Trucks in fleet	100	107	100	85	81
Trucks scheduled	80	85	80	70	70
Truck productivity, tons					
Minimum	2,500	2,500	2,500	2,500	2,100
Maximum	2,500	2,500	2,500	2,500	2,600
Fleet production, tons	150,340	160,049	159,544	160,498	160,240
Shovel operating shifts	8.00	8.80	8.00	8.39	8.39
Production per shovel operating shift, tons	18,792	18,187	19,943	19,125	19,094
Truck operating shifts	75.20	80.05	78.41	67.34	64.71
Effective truck shifts	60.10	63.98	63.82	64.20	64.13
Production per truck effective shift, tons	2,500	2,500	2,500	2,500	2,499
Operating cost per ton*	$0.4438	$0.4447	$0.4364	$0.4118	$0.4085
Fleet purchase cost ($ × 10^6)	$230	$249	$230	$215	$210

*Loading and haulage cost only.

A computer program that models the dependent fleet interactions can generate and investigate a large number of loader/truck combinations, select those that meet the production goal, and finally determine which combination will give the lowest overall combined cost per ton of material mined.

This type of evaluation is also a very useful tool for sensitivity analyses to investigate how changes in parameters such as availabilities, depreciation, and equipment and labor operating costs change the optimum fleet size.

The methods described using the binomial distributions were developed to quickly and easily analyze various loading/haulage fleet combinations with respect to costs, capacities, and the optimal fleets for comparing one mining method and equipment choices to another alternative.

There are obvious limitations to the accuracy of this method in its application to a true mining situation. This evaluation, for instance, cannot recognize the difference between the expected output from an operation that has a well-functioning computerized truck dispatch system as compared to one that is run with batch assignments. This simple method also does not consider the total picture in the sense that there are multiple haul profiles and that the loading equipment may have varying productivities and/or targets according to the material being mined. Only a fully developed mine simulation program can analyze an operation accurately to reflect all of the numerous variables applicable to one point in time.

The desired accuracy level and the complexity of the operation to be studied dictate the depth of study and method that should be used. The numerical examples illustrate the importance of correctly selecting the best loading and truck fleet sizes.

This case study of equipment proposed in a large strip mine has been given as a good example of comparing nonoptimal equipment fleets with other types of mining. If the comparison were made between a dragline versus a shovel/truck combination and that comparison indicated $10 million in capital cost savings, the mine owners would have made the wrong choice of mining method and equipment. Therefore, it is of significance to correctly analyze the proposed equipment fleet to ensure that any comparison is valid and the proposed savings of one system over another are real.

Table 6.6-4 Basic assumption for all cases (Example 2)

Equipment Characteristics	**Shovels**	**Trucks**
Mechanical availability	80%	80%
Maximum productivity per operating shift	20,000 tons	As noted
Operating cost per effective shift	$1,500	$665
Operating cost for nonproductive shift (waiting time)	$300	$220
Operating labor cost per shift	$240	$120
Capital cost per unit ($ × 10^6)	$8.0	$1.5

SUMMARY

In all surface mining operations, where there is a choice of mining method and equipment, the optimal alternative is often difficult to choose because the differences may be within the degree of accuracy of the trade-off study.

The five most important criteria in selecting the best mining method and equipment are

1. Type of deposit, seam or disseminated;
2. Geometry of the deposit and terrain;
3. Total material movement per time period;
4. Projected project life; and
5. Geographic location.

Other factors of lesser importance include

- Climatic conditions and altitude;
- Availability and cost of a skilled labor force;
- Environmental and legal restrictions;
- Slope stability and groundwater;
- Investment risk and available capital funding;
- Blasting restrictions;
- Method of waste rock disposal, external or backfilling;
- Availability of support services and infrastructure; and
- Availability of electric power versus the local cost of diesel fuel.

Table 6.6-5 presents a comparison of ore-body conditions favorable to the different surface mining methods. A summary

Table 6.6-5 Comparison of deposit conditions favorable to surface methods

		Mechanical Extraction				Aqueous Extraction		
Open Pit	**Characteristic**	**Quarrying**	**Strip Mining**	**Augering**	**Hydraulic Mining**	**Dredging**	**In-situ Mining**	**Surface Leaching**
Ore strength	Any	Any (sound structure)	Any	Any	Unconsolidated, few boulders	Unconsolidated, some boulders	Consolidated	Rubblized or cavable, permeable
Rock strength	Any	Any	Any	Any	Unconsolidated	Unconsolidated	Competent, impervious	Competent, impervious
Deposit shape	Any (preferably tabular)	Thick-bedded or massive	Tabular, bedded	Tabular, bedded	Tabular	Tabular	Any	Massive or thick tabular
Deposit dip	Any (preferably low dip)	Any, if thick	Any (preferably low dip)	Low dip	Low dip	Low dip	Any (preferably low dip)	Steep
Deposit size	Large, thick	Large, thick	Large, moderate thickness	Limited extent, thin	Limited extent, thin	Moderate extent and thickness	Moderate to large	Any (preferably large)
Ore grade	Low	High (assay not critical)	Low	Low	Very low	Very low	Intermediate	Very low
Ore uniformity	Uniform (or sort or blend)	Uniform	Fairly uniform	Uniform	Fairly uniform	Fairly uniform	Variable	Variable
Depth	Shallow to moderate	Shallow to moderate	Shallow	Shallow	Very shallow	Very shallow	Moderate to deep	Shallow to moderate

Source: Hartman 1987.

Table 6.6-6 Comparison of advantages and disadvantages of surface methods

		Mechanical Extraction				Aqueous Extraction		
Characteristic	**Open Pit**	**Quarrying**	**Strip Mining**	**Augering**	**Hydraulic Mining**	**Dredging**	**In-situ Mining**	**Surface Leaching/ Hydraulic Mining**
Mining cost (relative)	10%	100% (highest)	10%	5%	5%	<5%	5%	5%
Production rate	Large-scale	Small-scale	Large-scale	Moderate	Moderate	Large-scale	Moderate	Moderate
Productivity	High	Very low	High	Very high	High	Highest	Very high	Very high
Capital investment	Large	Small	Large	Small	Small	Large	Large	Moderate
Development rate	Rapid	Moderate	Rapid	Rapid	Rapid	Moderate	Moderate	Moderate
Depth capacity	Limited	Limited	Limited	Limited	Limited	Limited	Unlimited	Unlimited
Selectivity	Low	High	Low	Low	Moderate	Low	Low	Low
Recovery	High	High	High	Moderate	Moderate	High	Low	Very low
Dilution	Moderate	Low	Low	Low	High	High	High	Very high
Flexibility	Moderate	Low	Moderate	Very low	Moderate	Low	Low	Low
Stability of openings	High	Highest	High	High	Moderate	Moderate	High	Moderate
Environmental risk	High	Moderate	Very high	Low	Severe	Severe	Moderate	Moderate
Waste disposal	Extensive	Moderate	Minor	None	Moderate	Extensive	Minor	Minor
Health and safety	Good	Good	Good	Good	Fair	Good	Good	Good
Other	Low breakerage cost; rainfall and weather problems; large-scale best	Waste-intensive; labor-intensive; high breakage cost	No waste haulage; low breakage cost; large-scale best	Restrictive; used for remnant coal	Unconsolidated deposit; water required; no breakage cost; beneficiates	Unconsolidated deposit; water required; no breakage cost	Unconsolidated deposit; water required; no breakage cost; beneficiates	Unconsolidated deposit; water required; no breakage cost

Source: Hartman 1987.

of the advantages and disadvantages of each surface mining method is presented in Table 6.6-6.

In conclusion, many factors need to be considered in selecting the best mining method coupled with the most economic fleet of equipment. When there is more than one option, it is best to select the alternative that gives the mining operation the greatest degree of flexibility. Mining is dynamic, and the operation that can adjust the easiest to changing conditions is the operation that will survive future crises.

Successful analysis of the options available ensures that the proposed project will operate at its competitive peak. The ideas presented in this chapter will hopefully help the mine operator meet the prime objective, that is, the investor's law of conservation.

REFERENCE

Hartman, H.L. 1987. Surface mining: Method comparison and summary. In *Introductory Mining Engineering*. New York: Wiley-Interscience.

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PART 7

Rock Breaking Methods

CHAPTER 7.1

Mechanical Rock Breaking

Jamal Rostami

INTRODUCTION

One of the main components of mining activities and unit operations is rock breakage, which has increasingly shifted toward mechanized systems in the past few decades. Mechanical excavation refers to methods of rock breakage where the rock is entirely removed from the mine face by the action of mechanical cutting tools. In recent years, mechanical excavation has become increasingly common, especially in underground mining and construction. Mechanical rock excavation is traditionally compared to drill-and-blast operations. The inherent advantages of mechanical excavation are increased safety, higher production rate, less labor-intensive operation, higher degree of automation, minimum damage to the walls, better control over the process, uniformity of product size, elimination of blast vibrations, and better ventilation in underground openings.

Many improvements have been made in the process of rock breakage, and several world records have been set at various operations. This includes drilling and excavation systems across the board and sizes ranging from 50 mm to >15 m (2 in. to >50 ft) in diameter. Mechanical excavators are well positioned in underground construction and mining but are less common in surface operations. However, excavators such as surface miners and road milling machines are becoming more common in surface mining operations, especially in soft to medium-hard rocks.

The most common types of mechanical excavators are tunnel boring machines, raise boring machines, roadheaders, continuous miners, and longwall drum shearers. However, mechanical excavation methods also include unconventional techniques such as laser drills, microwave rock-breakage systems, projectile impact, flame jets, and water jets. Of these techniques, water jetting is the only one that is used on a regular basis for precision rock cutting and occasionally in drilling operations. The mechanism of rock cutting by water jet can be considered erosional, somewhat different from the other methods listed. The use of electric shock to induce stresses, or "plasma explosion," has also been investigated but has not yet been used in mining applications or rock excavation.

Using the physical, chemical, and mechanical properties of rock formations can induce rock breakage. This is to say that the breakage occurs when bonds between the constituent grains and minerals in rock are broken using one of these forms of energies. This can be done by using chemical processes to weaken and break the bonds, or by inducing stresses between the particles to separate pieces or chunks of rock from the ground or from each other. Optical, thermal, and mechanical properties of rocks can also be used to generate stresses.

The most efficient form of breaking such bonds is by the use of force via a mechanical device or tool. This has been the most dominant method of rock breakage since the industrial revolution, when high-quality steel became available at reasonable prices. The rock-cutting tools, coupled with various types of machinery that could deliver concentrated forces, allowed indentation and penetration into the rock surface to drill the holes for blasting operation or for large-scale excavation. Other means of rock breakage are still being explored and experimented on. At this stage they are considered to be "novel techniques." This includes using a laser beam for melting rock or using oscillating laser beams for breakage of rock into pieces through resonance of the grains.

New methods and techniques, as well as new tools for rock fragmentation, are being developed every day. Some of these ideas will find acceptance in practical applications relatively fast, while others will not. In addition, manufacturers continue to develop new machinery, and as such, miners gradually adopt new systems into their design and operation. As one expert put it, this is "a subject that in general has as many exceptions as rules" (W. Hissem, personal communication). This means that the information provided in this chapter on the tools and the norms/rates indicated for different types of machinery are likely to change within a few years. Although this information is provided to offer a brief background and basic understanding of the techniques, it is imperative that users look up the most recent development to obtain quantitative values for their design and engineering estimates.

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ROCK/BIT INTERACTION AND ROCK INDENTATION PROCESS

Mechanical rock breakage involves using a tool to create a high-stress area within the rock to cause failure and disintegration. As such, the tool penetrates the formation when sufficient forces are applied. This process is called indentation and can be accomplished under static or dynamic loading. Static loading is used primarily in excavation, while dynamic or impact loading is mostly used for drilling applications. The distinction between the two is that impact loading primarily involves a reciprocating hammer that delivers impact energy to the tool, whereas with static loading, a bit is dragged or rolled across the surface, which causes rock chips to spall from the rock surface.

In mechanical rock excavation (static loading), the cutters or bits can be categorized into the following two main groups: drag-type tools and roller-type tools. Each group is associated with particular ground conditions, rock properties, and machine types. Thus, it is important to understand the relationship between the rock properties and the selection of proper or optimum tools. Intact rock properties have the most pronounced impact on bit or cutter selection. These properties include the following:

- Rock strength (compressive and tensile)
- Abrasiveness or hardness of the rock and its constituent minerals
- Rock texture, grain size, and shape
- Ductility or plastic behavior

Several rock mechanics tests are commonly used to characterize the rock type and potential behavior pertinent to excavation and breakage. The following is a list of the typical tests used for this purpose:

- Unconfined compressive strength
- Indirect ("Brazilian") tensile strength
- Point load index
- Abrasiveness measured by the Cerchar abrasivity index (CAI) or by the Taber abrasion test
- Siever's J-value (SJ), brittleness (S_{20}) index, and abrasiveness value, as measured by Norwegian drillability tests (Bruland 2000)
- Elastic moduli measurements (Young's modulus and Poisson's ratio)
- Thin-section analysis

To successfully select a cutter type, measurements of the values for some of these properties are required.

Mechanical breakage starts with rock indentation by a bit or part of a bit that is in contact with the rock surface. This process involves a hard and relatively stiff object applying concentrated load on the brittle medium or rock formation. The process starts with elastic deformation of the rock, followed by localized failure of the rock in the area of high tensile stresses, which forms the initial cracks in the directions of maximum tensile stresses. The process of fracturing causes irreversible deformations. A continued increase in the stress at the point of contact results in crushing of the material under the contact area forming a fine powder. The grain size of the crushed zone gradually increases as the distance from the point of contact increases until the crushed zone gives way to radial cracks. A detailed study of this phenomenon was performed by Lindqvist (1984). In Lindqvist's analysis, the approach offered by Lawn and Swain (1975) was adopted,

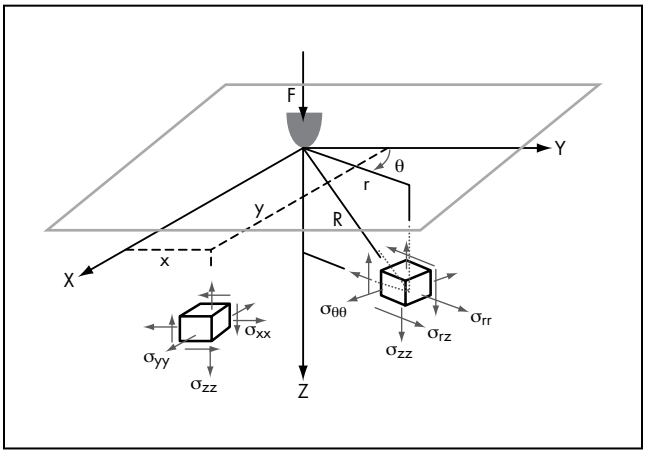


Figure 7.1-1 Coordinate system for indentation stress field

and stresses in the rock were estimated using the Boussinesq solution for the stresses in a plane from a concentrated load. These stresses can be expressed by the following equations (Fung 1965):

$$\sigma_{rr} = \frac{F}{2\pi R^2}\left[\frac{3r^2 z}{R^3} - \frac{R(1-2\nu)}{R+z}\right]$$

$$\sigma_{\theta\theta} = \frac{(1-2\nu)F}{2\pi R^2}\left[\frac{z}{R} - \frac{R}{R+z}\right]$$

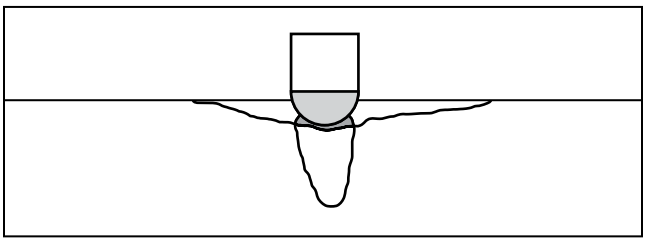
$$\sigma_{zz} = \frac{3Fz^3}{2\pi R^5}$$

$$\sigma_{rz} = \frac{3Frz^2}{2\pi R^5}$$

where

σ_{rr}, $\sigma_{\theta\theta}$, σ_{zz}, and σ_{rz} = stresses in various directions, including radial, tangential, and shear (see Figure 7.1-1)
F = applied load
R = radius vector from the stress point to the force application point, which equals $(r^2 + z^2)^{1/2}$

As can be seen in these equations and in Figure 7.1-1, the stress in the rock is proportional to the applied force and inversely proportional to the square of the distance from the point of contact. These equations, and all similar sets of equations for estimation of stresses using other coordinate systems (i.e., the Cartesian system), indicate that the stress reaches a singularity at $R = 0$; however, the indenter actually has a finite width and the load is distributed over a small area. This setting creates an area of concentrated compressive stresses in the radial direction developed directly under the indenter, and the resulting deformation creates tensile stress in tangential directions. The stress field quickly loses its magnitude as the points get further away from the load. The area of concentrated compressive stresses is best represented by the powdered rock that forms under the cutting tool. This powder is often referred to as pressure bubble. The process of indentation is followed by formation of the chips as radial cracks extend from the pressure bubble in all directions, especially toward the adjacent cut or toward an existing free surface or surfaces.



Source: Gertsch 2000.

Figure 7.1-2 Schematic of a sample after an indentation test

Gertsch (2000) defines two parameters—specific penetration (SP) and specific energy (SE)—to express rock cuttability. SP is defined as the amount of indentation obtained for a given applied force, and SE is defined as the energy required to excavate a unit volume of rock. Figure 7.1-2 shows a schematic drawing of a rock specimen after an indentation test. Thus, the response of a rock to indentation can be represented by a force–penetration curve. A sawtooth shape, resulting from rapid loading and unloading, indicates brittle behavior, whereas a steadily increasing curve with no unloading or a drop in the force indicates a rather plastic or ductile rock. If an indenter is dragged across a face at a certain depth, simulating the action of a drag bit, it will create a sawtooth curve; however, the magnitude of forces will not increase over time, as it does in an indentation test.

BIT MATERIALS

Bits or cutting tools are the most important components of any system of mechanical rock breakage. Tools must be very strong to resist high-stress concentrations without failure, but they must also be able to resist abrasion and rapid wear by hard minerals in the rock formation. A tool is subject to impact loading; however, most materials that are hard and abrasion resistant are also brittle and fracture under impact loading. The best cutting tools are strong and ductile on the inside with a hard surface, which is typically obtained by chemical or heat treatment of steel.

The most common materials used in rock-cutting tools are special types of steel and tungsten carbide. Tungsten carbide is stronger, harder, and more impact resistant than steel, and it is commonly used as an insert at the cutting end of a steel bit (bit tip). Rock-cutting tools are manufactured from high-grade steel that is machined and heat treated to increase the surface hardness to a range of 54 to 56 on the Rockwell hardness scale. As for the carbide, various grades of carbide can be used for manufacturing of the bit tips. This includes the size of the grains used in the mix and the amount of cobalt used in the mixture to prevent carbide from becoming brittle at higher temperatures.

Through extensive testing and field trials, the optimum percentage of cobalt in carbide mix has been considered to be around 10%. This gives the best combination of hardness, strength, and toughness to the carbide inserts for application in rock excavation. In certain applications, polycrystalline diamond is used to coat the cutting tips of tungsten carbide bits. Polycrystalline diamond tools have shown excellent resilience in various applications, but they are still rather expensive and are only used for special applications such as oil-well drilling bits or diamond drilling for deep rock exploration, where the extended life and higher tool costs are justified due to operational considerations and typically very high rig/operational costs.

BIT TYPES

Mechanical rock excavation tools are classified into the following two main categories: drag tools and roller tools. Drag tools scrape rock from a face by dragging the cutting edge of the tool across the rock surface. These tools have rather sharp edges to reduce the forces for penetrating, thereby cutting the rock in tensile and shear mode. The amount of force (and thus energy) that can be transferred to the rock by these tools is limited by the strength of the material used for the tip of the tools. The use of stronger, harder materials such as tungsten carbide at the tips of drag-type tools has resulted in increased bit life and an ability to cut harder rocks. Although using carbide can increase the range of application of the drag-type tools, it cannot withstand the extreme forces and heat generated while cutting harder rocks (especially those with high quartz content). Drag bits are generally limited to cutting rocks with strengths less than 100 MPa (15,000 psi). For higher-strength rocks or rocks with higher hardness and abrasivity, roller tools are often used.

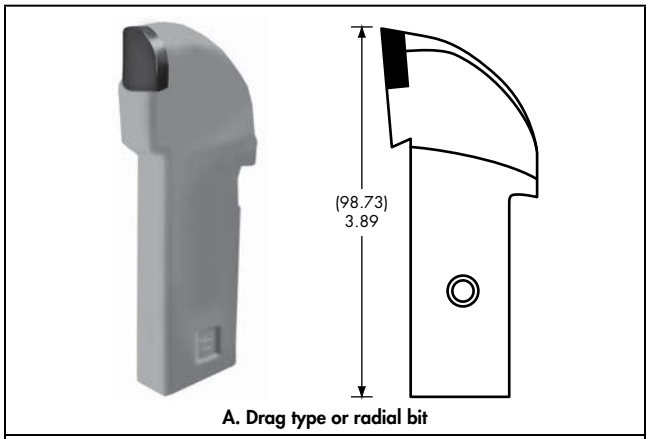
Figure 7.1-3 shows some of the typical drag bits used in the industry. These tools are very common on partial-face machines used to mine soft to medium-strength rocks. There are two types of tools in this category—drag bits and conical picks. Drag (or radial) bits are used in very soft and nonabrasive rocks and are very efficient. However, they lose their efficiency very rapidly as the sharp edge of the tip erodes.

Drag bits are gradually being replaced by conical picks, which are used for higher-strength and more abrasive rocks. Rotation of the bit, which allows for uniform wear on the tip and the sustaining of its profile during the excavation, is an important function of the system. The new sleeve system allows for better performance and longer bit life due to improved rotation. Sleeves provide additional surfaces for rotation and protection of the bit block. When sleeves experience high deformation or are damaged, they can be easily replaced. In contrast, replacement of a bit block involves cutting out the damaged block with a torch and welding on a new one, which can result in inaccurate placement of the block on the cutting drum and subsequent problems. The use of sleeves often results in higher productivity and lower maintenance in excavation systems.

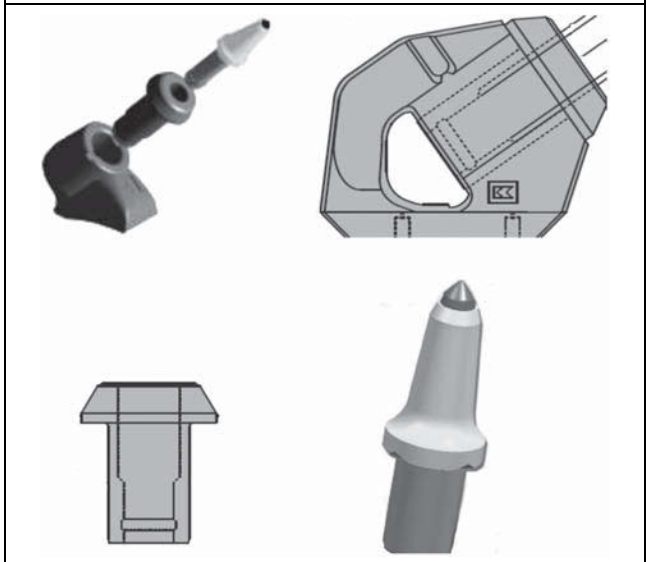
In harder rocks, roller-type cutters, including disc cutters and strawberry or multi-row tungsten carbide cutters, are used. Figure 7.1-4 shows disc cutters with the front- and back-loading mounting assembly that uses the wedge-lock system. Back-loading cutters are used in tunneling applications and allow for cutter changes from within the cutterhead, which is safer than changing cutters at the front of the cutterhead, where the ground is unsupported. The new cutter material and assembly system affords longer life, better accessibility of the cutters (especially in bad ground), and reduced time for cutter change. Figure 7.1-5 shows "strawberry" cutters with multiple rows of tungsten carbide buttons that are used in extremely hard and abrasive rocks and in applications where cutter change is very difficult and longer cutter life is intended. An extensive review of mechanical excavation and rock-cutting tools can be found in Roxborough (1985).

BIT SELECTION

The selection of the right tool for a given application is a function of several parameters, including rock strength and abrasivity, machine type and size, and accessibility of the face. It must be stressed that using the right tool can influence the overall success and the economics of using mechanized excavation in an operation. Figure 7.1-6 is a simple chart for the



A. Drag type or radial bit



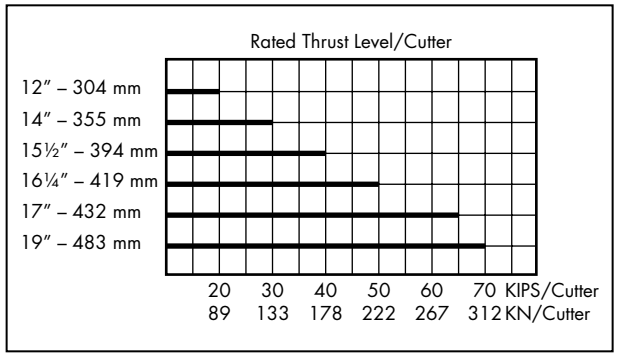
B. Conical pick/bit assembly including bit block, sleeve, and bit

Courtesy of Kennametal Inc.

Figure 7.1-3 Rock-cutting tools for soft- to medium-rock excavation



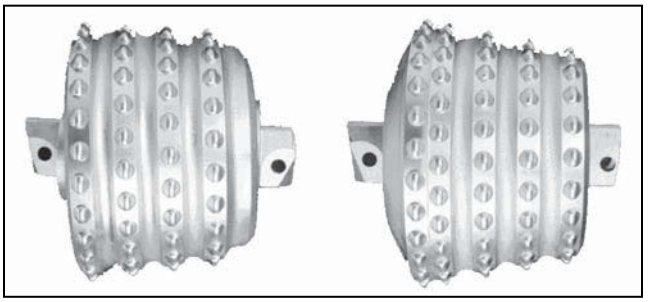
A. Typical disc cutters



B. Typical loading capacity of various-sized discs

(A) Courtesy of The Robbins Company; (B) Source: Askilsrud and Moulton 2000.

Figure 7.1-4 Disc cutters



Courtesy of Sandvik Mining and Construction.

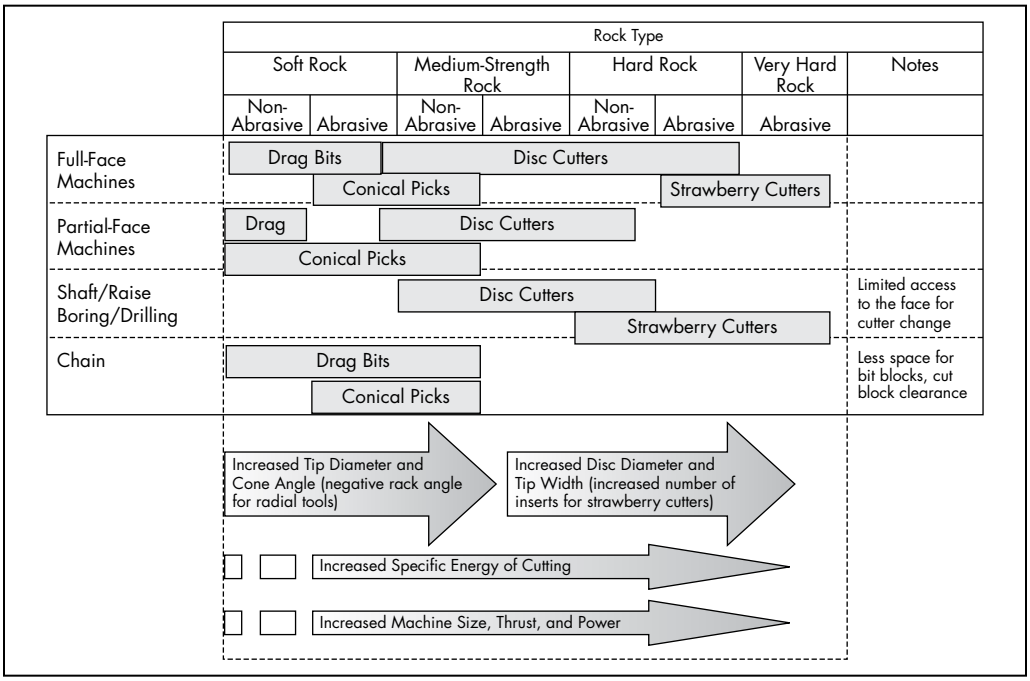
Figure 7.1-5 Strawberry cutters with multiple rows of tungsten carbide buttons

selection of rock-cutting tools. Additionally, selecting a correct strategy for cutter change can have a major impact on the cutting efficiency, machine productivity, and excavation costs. Proper tool selection should result in optimal overall machine performance and minimized excavation cost and not just minimized cutter cost.

Briefly, radial or drag bits are recommended for soft, nonabrasive rocks, and conical or point-attack bits are the most suitable for soft but more abrasive and medium-strength rock, which is typically <100 MPa (<15,000 psi). For harder and more abrasive rocks, disc cutters are preferred, but strawberry cutters with tungsten carbide buttons are also used. A more detailed discussion on selection of cutting tools and bits including criteria for bit selection, cutting force estimation, and bit life estimates can be found in Rostami (1998) and Rostami (2001). Anderson and Rostami (1998) discussed the selection of drag-type bits, and Roby et al. (2009) presented a more recent review of disc cutters and their applications and improvements.

EXCAVATION

Mechanical excavators can be grouped according to various parameters associated with their application, including size (small, medium, or large), direction (horizontal, vertical, or inclined), ground conditions (soil/soft ground, hard rock, or mixed face), shape of the excavation (circular or noncircular), whether surface or underground, and finally the cutterhead system used (partial-face or full-face). Full-face machines are machines where all the cutters are engaged and cut the face simultaneously. Partial-face machines are those where the cutterhead cuts a portion of the face and only portion of the cutters contact the face at a given instant. Unlike the full-face machines, partial-face machines can excavate openings of different shapes and sizes. A third category of machine consists of rock breakers and impact hammers.



Source: Rostami 2001.

Figure 7.1-6 Simplified chart for selection of rock-cutting tools for mechanical excavation

Full-Face Machines

With one exception, full-face machines excavate circular-shaped openings. The best example and the most popular full-face machine is the tunnel boring machine (TBM). Other types of full-face machines include raise borers, shaft-drilling machines, and full-face continuous miners. These four types of machine are described in more detail as follows.

Tunnel Boring Machines

TBMs have enjoyed amazing growth in popularity since 1990 and have set remarkable performance records, the best of which being the excavation of 174 m (~540 ft) of tunnel in one day by a 5-m- (16-ft-) diameter machine through medium-strength sandstone in Australia. A functional TBM is comprised of a cutterhead, a cutterhead support and drive unit, the main frame/shell/beam of the machine, a propulsion system that typically consists of a set of grippers and thrust cylinders, a face conveyance system, a towing section or bridge, and a backup system (Figure 7.1-7).

These machines can be steered and can negotiate a bend with a radius of around 300 m (1,000 ft). The main cutting tools on rock TBMs are disc cutters. Machines range from 1.8 to 15 m (6 to 50 ft) in diameter. TBMs of various sizes have been used to advance numerous water storage tunnels in limestones and dolomites in Chicago, Milwaukee, and other major U.S. cities. A 15-m- (50-ft-) diameter TBM has recently been used in a hydroelectric project under Niagara Falls, Canada. The following types of machines are in use today.

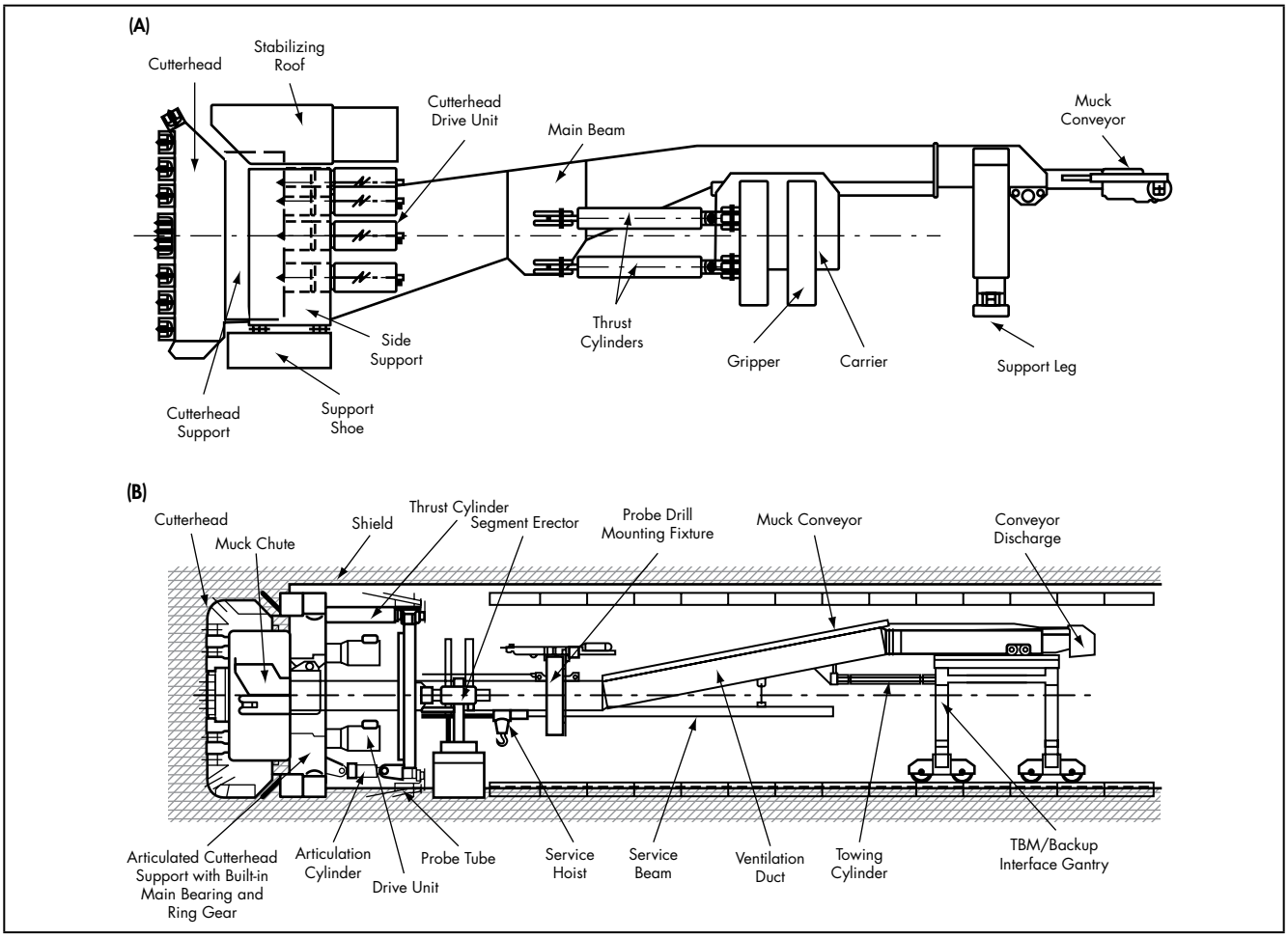
- **Open-type TBMs:** These machines are used for more competent rock, and they push off the walls with their grippers.
- **Shield machines:** Shield machines are used in broken, unstable rock or in soft ground/soil, and they install a segmental concrete lining behind the face to support the ground and provide a solid base for the jacks propelling the machines forward.
- **Double-shield TBMs:** These TBMs have been successfully used in many projects where mixed-face ground conditions are anticipated. These machines can grip on the walls and move forward in good ground, while simultaneously erecting a concrete segmental lining where needed. They can also advance through bad and broken ground by pushing off the segmental lining in a similar manner to regular shield machines. This provides for higher efficiency and flexibility, especially for fast-track projects where the nature of the ground has not been fully explored.

A simplified chart for making a preliminary selection of the type of TBM needed for project based on the rock mass rating for the project ground conditions is presented in Figure 7.1-8.

Shaft Boring Machines

For vertical or inclined openings, shaft boring, shaft drilling, raise boring, or raise drilling machines can be used depending on whether access is available at the top or bottom of the proposed opening or both. A shaft boring machine is a self-contained machine similar to a TBM, but it is used for driving vertical openings. It can sink large-diameter, blind shafts where the ground is reasonably good and the machine can grip on the walls to propel the cutterhead into the face. The cutterhead is powered by onboard electric or hydraulic drives. A vacuum system is used to remove broken muck from the bottom of the shaft and transfer it to the main hoisting system of the shaft.

At the time this chapter was written, shaft boring machines had been used to depths of around 1,000 m (~3,000 ft), but this is not a fixed limit and could be exceeded if needed. The



Courtesy of The Robbins Company.

Figure 7.1-7 Components of (A) typical open-type TBM and (B) shielded TBM

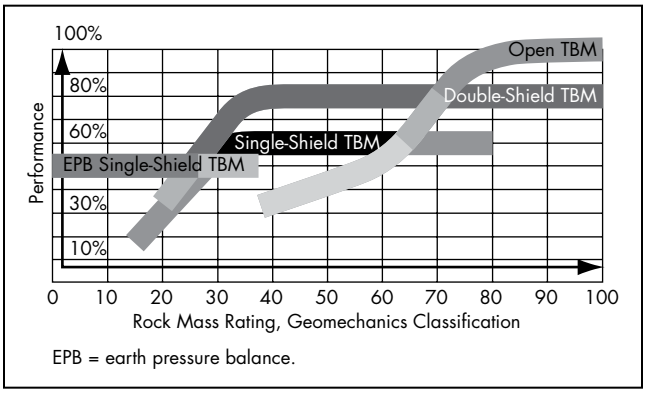
diameter of a shaft boring machine can be up to 10 m (30 ft), with the controlling factor being the velocity of the cutters at the circumference of the opening similar to TBMs. The penetration rate of these machines ranges from 1 to 2 m/h (3 to 6 fph), and various types of ground support can be installed simultaneously in the upper decks of the machine.

Shaft Drilling Machines

Shaft drilling refers to blind shaft excavation where the cutterhead hangs from a headframe and is rotated from the top head through the drill string. The thrust force for advancing the machine is provided by a series of deadweight segments directly behind the cutterhead at the bottom of the drill string. This allows the drill string to be in tension rather than compression (thereby avoiding buckling of the drill string), provides for better control of the straightness of the shaft, and minimizes deflection from the alignment. In effect, the system uses gravity to advance and steer the head. The limiting factor for shaft drilling is the amount of torque that can be transferred through the drill string, so the potential depths and diameters of the shafts are limited by the breakout torque delivered by the drill string (the Euler buckling stress varies inversely as the square of the slenderness ratio [length/diameter], and this imposes the limit on the amount of torque that can be transferred to the head).

Typical drilled shaft diameters range from 1.5 to 4 m (5 to 14 ft). Recently, a shaft drilling operation with a diameter of 6 m (20 ft) was begun for a 450-m- (1,475-ft-) deep shaft in Indiana (United States). The top head is capable of lifting up to 400 t (440 st) and can deliver high torque through a turntable to a cutterhead fitted with disc cutters. Formulas for estimating machine thrust, torque, and power requirements are found in a later section of this chapter.

Smaller-diameter shafts have been successfully drilled to depths of 1,000 m (3,300 ft) (A. Zeni, personal communication). The muck generated at the face is removed using reverse circulation. A typical excavation rate for a shaft-drilling machine is 2 to 3 m/h (6 to 10 fph), and setup time for the machine can be achieved within 3 to 4 weeks. During the operation, the cutterhead is submerged and the entire shaft is filled with drilling fluid to provide support for the shaft walls. When the excavation is complete, the cutterhead and drill string is retrieved and the same headframe is used to lower a steel lining into the drilled shaft. After the lining is in place, it is anchored by pumping concrete into the annular space between the lining and the walls, which displaces the drilling mud. Only a limited number of these machines exist worldwide and only a few specialty contractors have experience using these machines. Figure 7.1-9 shows a typical shaft-drilling machine.



Courtesy of The Robbins Company.

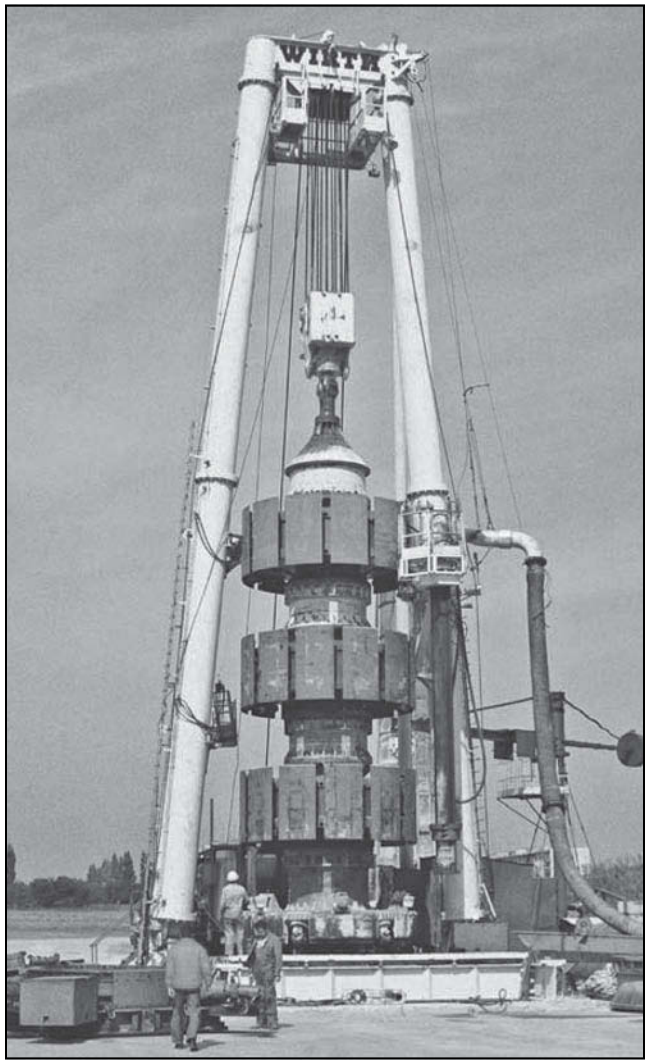
Figure 7.1-8 Preliminary selection and performance chart for various TBMs

Raise Boring Machines

Where access to a lower level is available, vertical and inclined shafts can be excavated by raise boring. This operation is shown in Figure 7.1-10 and includes drilling a pilot hole, removal of the bit, installation of the reamer head, and reaming of the shaft back to the upper level or toward the machine. The pilot hole is typically in the range of 200 to 350 mm (8 to 14 in.) in diameter, depending on the size of the drill string needed to provide sufficient thrust and torque for the intended final size of the bored shaft. The pilot hole is typically drilled using a tricone bit. However, when raise boring from the surface, the pilot hole can be drilled with any drilling machine capable of drilling a pilot hole of the required size. For long raises, use of a down-the-hole rotary percussion drill rig for the pilot hole could result in schedule and cost advantages compared to using a raise borer.

The raise can be vertical or inclined, and recently there have been several applications of this system in a horizontal direction, where a raise-bored tunnel was driven between two access tunnels. The limitations of the raise boring machine regarding size and length of the excavation are the same as for shaft drilling machines. Raise boring machines can be used to drive larger diameters if reaming is performed in multiple passes. This operation can be repeated several times with the size of the reamers selected to conform to the drill string and machine power. Low-profile raise boring machines have been developed that can be set up in a room with around 6 m (20 ft) of head clearance.

During pilot drilling, the muck is removed by circulating drilling fluids. While reaming, the muck drops down to a lower level where it can be removed by a loader. In horizontal raise boring, the muck is picked up by a loader. Ground support for raise-bored openings (for ground that is competent enough to stay open while the reaming operation is in progress) is installed after completion of the reaming, usually by a robotically controlled shotcrete spraying system that is lowered into the hole, or ground support can be installed by lowering a work platform from which the crew can work and install required additional ground support under the protected areas. This system can be considered for ground that is competent enough to stay open while the reaming operation is in progress. Typical rates of penetration for raise borers are from 3 to 5 m/h (10 to 16 fph) for pilot drilling and from 1 to 2 m/h (3 to 6 fph) for reaming. (An equation for predicting penetration rate as a function of rock strength and machine parameters for both pilot drilling and reaming is presented in a later section of this chapter.)

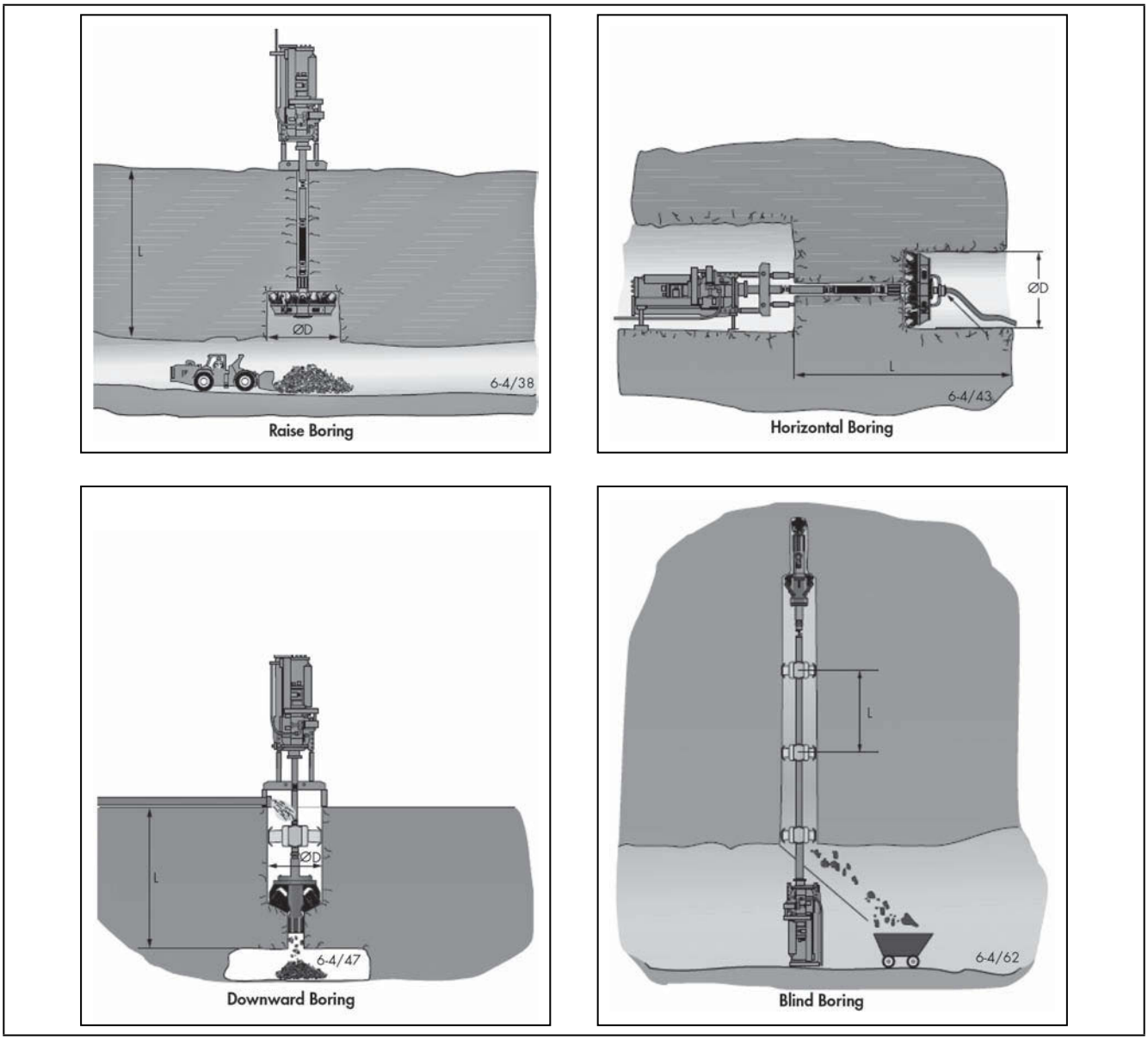


Courtesy of Aker Wirth.

Figure 7.1-9 Shaft drilling machine at surface showing the cutterhead

Raise Drilling Machines

Inclined shafts/raises can also be developed by raise drilling machines. This involves drilling upward using a cutterhead, where the drill string is pushed to drive the head and provide the torque. The muck flows down by gravity, and at the collar it is diverted onto the floor or into a muck car through a pan or a muck chute (Figure 7.1-10). The head can be up to 1.8 m (6 ft) in diameter, and the machine can drill a blind raise of around 100 m (300 ft). The limit on the size and length of section is imposed by the fact that this machine uses the drill string in compression to drive the head upward, in which case the torque that can be transferred to the cutterhead is more limited than that in a shaft drilling (or raise boring) operation. Raise drilling operations are very common in underground metal mines where a large opening is required for initial blasting of the stope. The typical drilling



Courtesy of Sandvik Mining and Construction.
Figure 7.1-10 Types of raise boring operations

rate is around 1 to 2 m/h (3 to 6 fph), similar to reaming from an overhead machine setup.

Stope Boring Machines

The stope boring system or BorPak is used for blind-hole raise boring. It comprises a guided boring machine, a power unit, a launch tube/transporter assembly, a conveyor, and an operator's console (Atlas Copco 2010). The BorPak operates like a micro-boring machine, climbing up the raise as it bores. Cuttings pass through the center of the machine, falling down the raise and launch tube onto a conveyor.

This machine has the potential to bore openings from 1.2 to 2.0 m (3.9 to 6.6 ft) in diameter at inclinations as low as 30°. This system is a miniature TBM, where instead of a gripper, an inflatable packer is activated to hold the walls while the head turns and is pushed against the face. At the end of the boring cycle, the steering shoes are pressed to the walls to hold the head against the face while the packer is deflated and moves up for regripping.

The machine has excavated holes with lengths of up to 150 m (450 ft). A preliminary design for a boring unit that can use a vacuum pickup system for mucking was developed to allow these units to be used for excavation in a horizontal or a slightly declined opening. However, operational units have not yet been manufactured. The BorPak system has been extensively used to excavate slot raises for sublevel stoping in underground hard-rock mines.

Full-Face Machine Cutters

The cutting tools on most, if not all, of the above noted full-face rock-cutting machines are roller-type cutters. TBMs and BorPaks generally use disc cutters. Conversely, some shaft

and raise boring and drilling machines use strawberry cutters to achieve longer cutter life. This is because changing cutters in these operations is difficult if not impossible and can significantly delay heading advance because it is necessary to pull the machine or reamer away from the face. Significant improvements have occurred in recent years with respect to cutter materials, cutter load capacity, cutter life, and computer monitoring and control technology on the cutters that allows for monitoring of cutter rotation.

Machine Transport

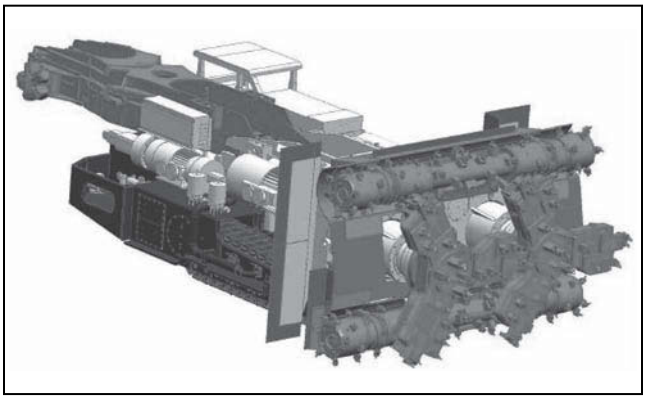
An important issue when planning to use full-face machines is transporting them to a site and then to the workplace. The minimum dimensions of these machines are typically such that it will be necessary to ship them in pieces with erection at the site. It is customary to specify the largest size and heaviest piece of the machine during the procurement phase so that the pieces can be delivered to the site through the available roads and bridges and be hoisted down the shaft if necessary. Most machine manufacturers design and fabricate their machines to comply with size and weight specifications to facilitate transportation and assembly, as well as decommissioning and dismantling the machine when a project is complete. With shield machines, the split cutterhead design and collapsing shield allows them to be withdrawn from within a lined tunnel they have just constructed.

Full-Face Continuous Miners

As noted previously, the only noncircular full-face mechanical excavators are the full-face continuous miners, which are often known as borer miners or "Marietta miners." These machines, which excavate rectangular openings with semicircular ends, are common in underground mining of potash and trona. The machines contain one or two pairs of counterrotating cutting drums that are 2.5 to 3.6 m (8 to 12 ft) in diameter, and depending on the number of drums, the machines can be 5 to 10 m (15 to 30 ft) wide. The triangular area between the circular cuts created by the drums is excavated either by a chain or by special cylindrical-shaped cutterheads (known as "cob cutters" due to their shape).

These machines are highly productive and can produce more than 1,000 t/h (1,100 stph) in trona and potash. They can be equipped with drag bits, but more commonly use conical picks, and they typically have 400- to 1,000-hp cutterhead power for high production rates. Excavated materials fall on the machine's front apron, from which they are pushed onto a chain conveyor that discharges onto a panel conveyor belt that feeds the main conveyor system. Panel belts are typically extensible and fed from belt storage magazines. These machines can excavate 500 m (1,500 ft) of entry per day. The ground support requirement is typically met by installing a single row of roof bolts over the operator's cabin. Some versions of these machines can install roof bolts while the machine is excavating, allowing for continuous production and face advancement.

These machines have high utilization rates of around 40% to 50% and have very high production rates. However, due to the limited applications and number of machines and manufacturers, they are gradually being replaced by drum-type continuous miners and roadheaders. Figure 7.1-11 shows a new generation of borer miner. These machines are slightly larger than their predecessors, reaching around 250 t (275 st) on the single-pair (two-drum) machines.



Courtesy of Sandvik Mining and Construction.

Figure 7.1-11 New generation of borer miner

Partial-Face Machines

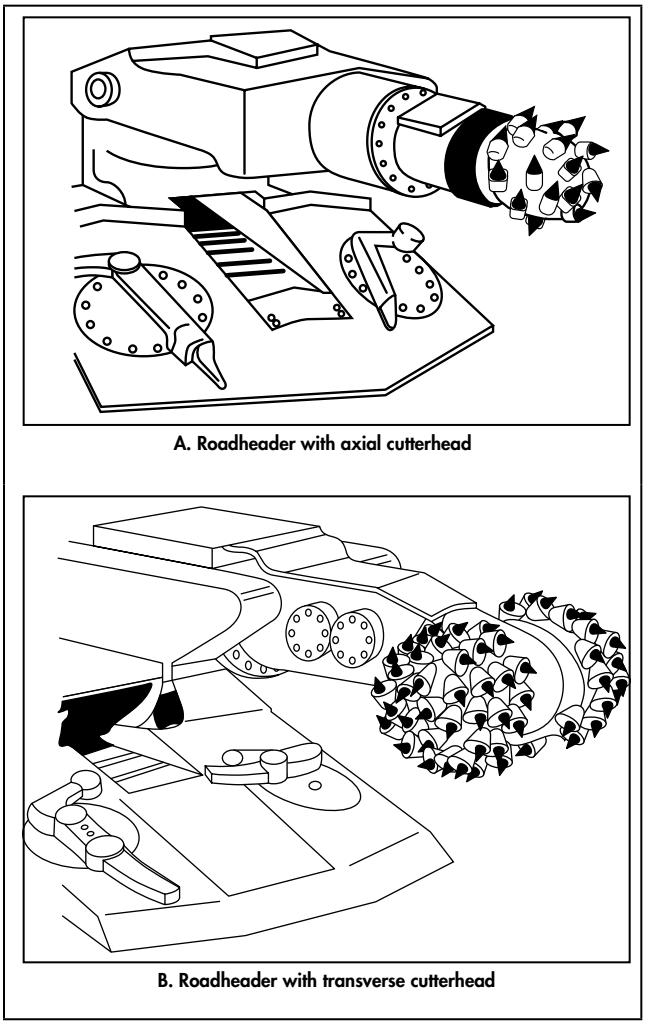
Partial-face machines are used for development and production in a variety of activities in underground or surface mines. These machines have traditionally been used in soft to moderately hard rocks using drag bits and conical picks. Owing to the need for rapid development in underground mines, several hard-rock partial-face machines have been developed and used for excavation of noncircular openings. However, they have only had limited success.

Partial-face machines are the most popular and most frequently used machines for underground mining of softer rock types, such as coal, salt, gypsum, potash, trona, and the softer variations of limestone. This group of machines includes roadheaders, continuous miners, longwall drum shearers, plows, road milling machines, surface miners, trenchers, and some variations of hard-rock mining machines. Bucket-wheel excavators (BWEs) can be considered a partial-face machine, but they are primarily used to excavate and load very soft soil-like material. Similarly, dredges may occasionally excavate rock but are generally used for removing partially consolidated material from the bottom of rivers or lakes. Because they generally do not excavate rock, BWEs and dredges are not covered in this chapter.

Roadheaders

Roadheaders were primarily developed for the driving of tunnels or mine entries. They use drag bits on a cutting head mounted at the end of a boom, which moves side to side and up and down to excavate the face of the tunnel. Along with TBMs, roadheaders are the most popular mechanical excavators in use. Over the years, two types of roadheaders have been developed and used. One, known as an "axial," in-line, or milling machine, has a cutterhead that rotates around the axis of the boom. The other, known as a transverse or ripping machine, has a cutterhead that rotates in a plane perpendicular to the boom axis. Figure 7.1-12 shows both types of roadheaders.

Axial roadheaders are typically favored in softer rock types, whereas transverse machines are preferred in harder rocks. Roadheaders are generally limited to use in rocks with unconfined compressive strengths of 100 MPa (15,000 psi) or less. Roadheaders are mostly used in mine development, especially in evaporites (salt and potash), and may also be used in production. Roadheaders have also been used to drive



A. Roadheader with axial cutterhead

B. Roadheader with transverse cutterhead

Source: Hekimoglu 1991.

Figure 7.1-12 Two types of roadheaders

large-sized shafts as the primary excavation system, where the cutterhead and boom are reconfigured to excavate the floor.

A typical work cycle includes sumping the head into the bottom of the face to begin a new cut, followed by motion of the head laterally and up and down to remove the rest of the face. Roadheaders take advantage of weaknesses in the rock and perform better in jointed rocks. Their performance drops in abrasive rock types due to the need for frequent bit replacement. The production rate of a roadheader is a function of its size, cutting power, weight, rock strength and abrasiveness, and operator skill in maneuvering the boom. The larger, more powerful machines can produce more, but their size requires a minimum opening dimension to allow for efficient operation. A major limiting factor for roadheaders is their ability to react to side forces as the cutterhead moves from side to side. This so-called limit on slewing or arcing forces determines how deep the head can be plunged into the face and how fast it can be moved from side to side in the production cycle before the machine starts to slide on its track.

Roadheaders are divided into three classes: small, medium, and heavy duty. Small roadheaders are typically less than 45 t (50 st) in weight and have cutterhead power of 70 to 110 kW (100 to 150 hp). Medium roadheaders range from 45 to 75 t (50 to 83 st) and have cutterhead power of up to 200 kW (260 hp). Heavy-duty roadheaders typically weigh more than 75 t (83 st) and can have a cutterhead power of 300 kW (400 hp). There are several variations of machines with low profiles for cutting harder rock types and those with an extended reach for cutting softer rock in large-size tunnels with high ceilings. Production rates for roadheaders can range from as little as 10 to 20 t/h (11 to 22 stph) to more than 200 t/h (220 stph).

Machine setup consists simply of connecting it to an electrical cable. Broken rock is removed from an apron on the front of the machine by means of gathering arms or rotating stars and a chain conveyor from which it is transferred to trucks, panel conveyors, or shuttle cars for haulage away from the face. Machines often come with additional equipment, ranging from bolters to steel set erectors, for the installation of ground support. Various types of mucking arrangements have been tried in the past to move the muck from the apron onto the chain conveyor, but the most efficient system appears to be rotating discs, rotating stars, or clam arms. Formulas for estimating performance and production rate of the roadheaders are provided in a later section.

Continuous Miners

Continuous miners (CMs) are the most popular machines in underground coal mines, but they are equally important in salt, potash, trona, and gypsum mines. These machines were invented in the mid-1950s and have evolved into robust and reliable mining machinery. Although several variations of CMs have been used over the years, currently available machines use solid heads that combine three cutting drums to form a cylindrically shaped cutterhead.

These machines operate by sumping at the upper part of the face near the roof and shear downwards. Because of the shape of the cutterhead, CMs cut a rectangular face. CMs can operate in seams as low as 1 m (3.3 ft) to as high as 5 m (15 ft). Cutterheads are typically 3 to 4 m (10 to 13 ft) long and 0.8 to 1.4 m (2.3 to 4.5 ft) in diameter, and they are generally fitted with conical bits for cutting the rock. The machines vary from 60 to 110 t (66 to 120 st) in weight and cutterhead power varies from 200 to 400 kW (260 to 530 hp). CMs are primarily used in room-and-pillar (R&P) operations, although they can be used in shortwall mining as well. In R&P operations, machines are used to drive multiple entries and crosscuts. However, the time required to move the machine between faces cuts into production time and is avoided as much as possible. Some new CMs have onboard bolting systems to reduce the need for maneuvering and relocating of the machines between working faces, which can improve machine utilization and daily production.

Typical R&P (or "CM panels") mining can produce in the order of 4 to 5 kt (4,400 to 5,500 st) per shift. In soft, thick coal seams, production can reach more than 1,000 t/h (1,100 stph) with larger, more powerful equipment. CMs used for driving headgates for longwall panel development can advance a three-entry development panel about 30 m/d (100 fpd) in a typical well-coordinated operation. An important aspect of using the CMs is the dust-control issue, which has resulted in most if not all cutterheads to be fitted with water sprays or so-called "wet" heads. Some of the sprays are phased, meaning that they only work when the bit is engaged and cutting. This allows for lower usage of water and prevents the sprays from creating a turbulence that pushes the dust back. Machines are

also fitted with dust scrubbers that collect the air right behind the head on both sides to reduce the amount of dust that gets into the air circulation.

Longwall Drum Shearers

Drum shearers are the backbone of longwall operations in coal and provide high productivity and high reserve recoveries of up to 80% when used on bedded ore bodies such as coal. Longwall mining accounts for slightly more than 50% of the total underground coal production in the United States. Longwalls in the United States generally use double-drum, ranging-arm shearers. Similar to CMs, drum shearers can work in seams as low as 1.5 m (5 ft) and as high as 6 m (20 ft). Drums are fitted with conical bits arranged in a helical (spiral) pattern. The excavated material falls onto the face (chain) conveyor. The installed power of the arms on the drum shearers has gradually increased over the years and currently machines with cutterhead power of 750 kW (1,000 hp) per drum (or per arm) are available on the market.

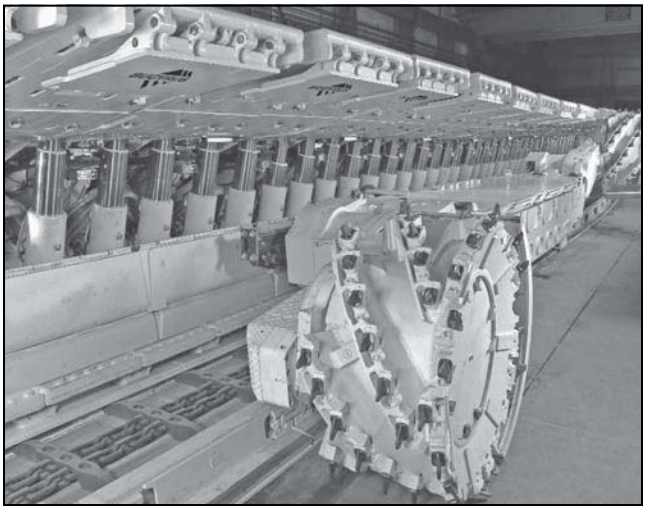
Recent advances in the shield and machine automation have resulted in considerable increases in machine cutting speeds. Cutting speeds of up to 0.83 m/s (150 fpm) have been achieved, while the automated shields will maneuver behind the fast-moving drum to advance the face. With such efficient systems, longwall faces have grown as wide as 450 m (1,500 ft), and panels can consistently advance 15 m (50 ft) per day. At high cutting speeds, drum shearers can produce in excess of 5,000 t/h (5,500 stph) at peak production. However, to avoid overwhelming the face and panel conveyors, drums take half cuts or work at lower speed to allow for a more uniform and sustainable production rate. A typical longwall panel can produce about 10 to 12 kt/d (11,000 to 13,200 stpd) in a 3-m- (10-ft-) high coal seam in two 10-hour shifts. Figure 7.1-13 shows a typical longwall drum shearer.

Drum diameters are chosen to be 0.5 to 0.6 times the seam height. Therefore, diameters range from 0.8 m (2.5 ft) to more than 3 m (10 ft). Drum widths are often limited to 0.75 to 1.0 m (2.4 to 3.3 ft) to avoid cantilever forces on the arm from the weight or cutting forces. Counterrotating drums are used in alternating lead-and-tail drum positions, and the machine is maneuvered back and forth along the headgate or tailgate to advance the face in a "snaking" motion.

To minimize airborne dust and improve visibility, water is sprayed on the cutterhead. Cutting drums are replaced or overhauled after 2 to 3 Mt (2.2 to 3.3 million st) of production (the same is true for CMs). Longwall systems are capable of passing through fault zones. Special drums are used along with stronger bits and lower cutting speeds to handle harder roof and floor formations. Longwall drum shearers are also used in some trona and potash mines, but they produce slightly less than in coal because of higher energy requirements for cutting such formations.

Plow Systems

Coal plows are also used in longwall operations mainly in Europe, and they work by being dragged along the face to take small 100- to 150-mm- (4- to 6-in.-) deep cuts between the headgate and tailgate. The motion of the plow along the face is controlled by a set of high-strength cables. At speeds of between 2.5 and 3.6 m/s (490 to 710 fpm), these systems can be highly productive, especially in low seams, where drum shearers are not very effective. Plow systems can work in seams as low as 0.5 m (2 ft) and as high as 2 m (6.5 ft).



Courtesy of Bucyrus.

Figure 7.1-13 Drum shearer in a longwall face

Machines can maneuver and follow the seam, cutting up or down relative to the pan, which can negotiate vertical seam curvature along the face. To increase productivity, higher-power cutting systems have been installed on some machines. The available models have total installed power ranging from 800 to 1,600 kW (1,080 to 2,160 hp). Production rates can be estimated from the speed of cutting and depth of slice with production rates of more than 1,200 t/h (1,320 stph) possible.

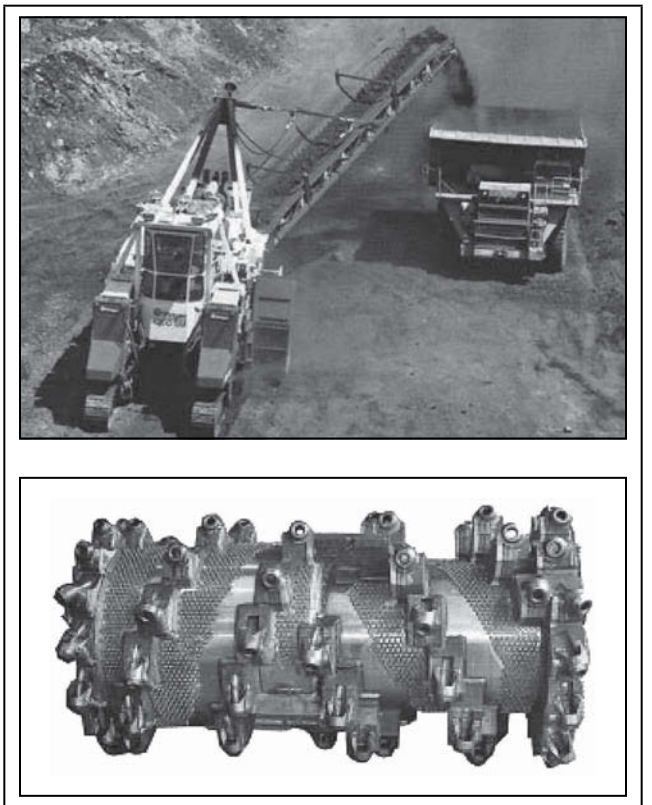
Road Milling Machines

Road milling machines, which are used to remove road pavement prior to resurfacing, have in recent years been adopted for mining operations such as coal, gypsum, oil sand, and limestone. These machines are highly productive and allow for selective mining in flatly bedded ore bodies. The machines primarily consist of a chassis mounted on three to four independently controlled suspension systems that carry a cylindrical cutting drum. The depth of the cut can be controlled by adjusting the height of the legs.

In mining applications, the cutting drums are about 1 m (3.3 ft) in diameter and 2 to 4.2 m (6.6 to 13.8 ft) in length, and they can cut a slice of about 150 to 200 mm (5.9 to 7.9 in.) into the rock. Cutting speed depends on the hardness of the material as well as depth of slice, and speeds can reach up to 1.3 m/s (240 fpm). Machines can load mining trucks or a surface conveyor by means of bridge conveyors. Machine production is typically around 400 to 500 t/h (440 to 550 stph) with a single machine and fleet of four or five trucks, but new records have been set by larger machines in recent years, reaching production rates of around 2,100 t/h (2,300 stph). With recent restrictions on blasting in quarries near urban areas, the use of these machines has become more common.

Road milling machines are often powered with onboard diesel engines with power ratings as high as 1,200 kW (1,600 hp) on machines with gross operating weights up to 180 t (200 st). The cutting drums used in larger mining operations are fitted with conical bits at a higher bit spacing to increase the size of the product and minimize fines. Compared to the milling machines used in typical road applications, the drums on machines used for mining have fewer bits and use less energy to cut the material but leave a slightly rougher

surface. Small machines weighing about 10 to 15 t (11 to 17 st) and fitted with cutting drums as small as 0.5 m (<2 ft) in diameter and 0.5 m (<2 ft) in length can be used to pave underground roadways to remove floor heave and improve the road surfaces. Figure 7.1-14 shows road milling machines in mining operations.



Courtesy of Wirtgen GmbH.

Figure 7.1-14 Road milling machine and milling drum

Partial-Face Machines for Hard-Rock Applications

Underground hard-rock miners seek mechanical excavation systems that can excavate noncircular openings, preferably horseshoe-shaped drifts with a flat floor. Because of the strength and (often) abrasiveness of hard rock, the only feasible cutting tools for hard-rock excavation have been roller-type cutters, particularly disc cutters, which require high thrust forces in the 10-to-30-t (11-to-33-st) range. As a consequence, having several cutters in contact with rock means that there are several hundreds of tons in forces at the face. TBMs can provide reaction forces for such high-thrust forces by applying pressure to the circular walls of the opening because the resulting thrust force is directed along the axis of the tunnel. In contrast, partial-face machines have been limited in the thrust they can deliver at the end of the cutting boom, because excessive force will make the machine slide on its tracks.

Over the years many concepts have been tried, but none has been fully accepted in mining operations. In the mid-1970s, an Atlas Copco Mini Fullfacer was tested at the Craigmont Mine in British Columbia, Canada, and was considered but not tested in Sudbury, Canada.

The Robbins Mobile Miner was developed in mid-1980s and was used in the Mt. Isa and Pasminco mines in Australia, as well as in a tunneling operation in Japan. It consisted of a cutting drum fitted with disc cutters (or multi-row carbide cutters) that could be moved from side to side while the drum rotated to excavate a profile with a flat floor and roof and arched sides (Figure 7.1-15). To provide for the reaction forces, the machine weight was gradually increased, and additional gripping mechanisms were added to the system to the point where, in the smaller version, the gross weight of the machine exceeded 200 t (220 st).

Although the Mobile Miner was considered successful in the excavation of hard and abrasive rock and could offer a consistent advance rate of more than 1 to 1.5 m/h (3.3 to 4.9 fph), it has not been used in any mining application since its use in the aforementioned Australian mines in the early 1990s.

Another machine of this class was developed by Aker Wirth. This continuous mining machine was designed based on the concept of undercutting the rock face using disc cutters pushed sideways to break the rock in shear toward a free face. The machine has four arms with 500-to-600-mm (20-to-24-in.) disc cutters at the end of each arm (Figure 7.1-15). For excavation, one arm starts at the middle of the face and moves toward the center while the other three arms move outward as the cutterhead rotates. The outward cutting arms are designed to move independently and push out of their projected cutting circle to excavate corners of an opening, creating a variety of profiles, such as circle-, horseshoe-, and square-shaped openings. The machine was successfully tested in a sandstone quarry in Germany. However, the field trial in an underground hard-rock mine in Sudbury, Canada, did not produce the anticipated results. No continued performance record is available on the machine.

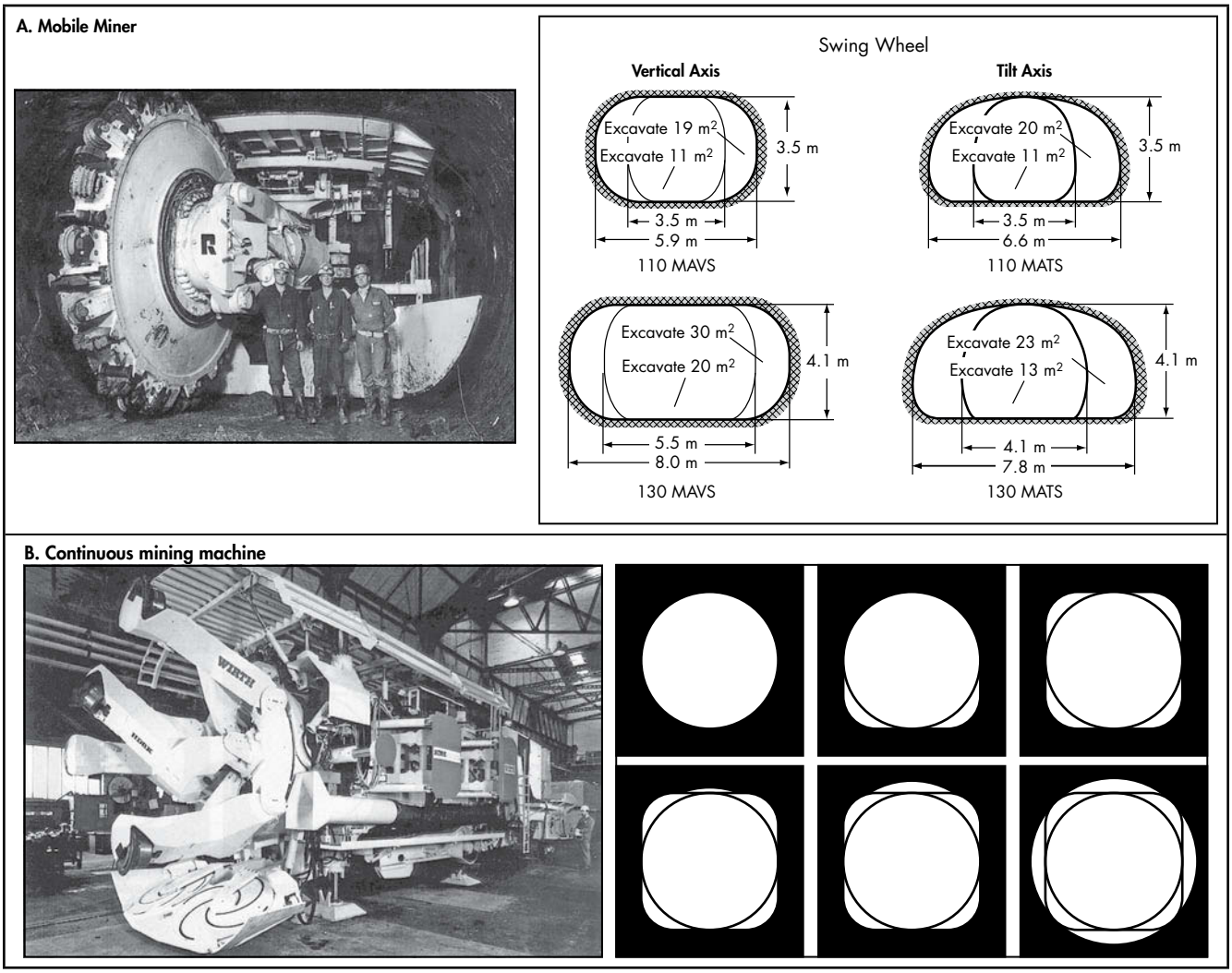
A heavy-duty roadheader has also been tested for possible application in hard-rock mines. This machine, which was developed by Eimco Inc., was an axial roadheader that was modified with higher power, higher weight, and additional gripping system for added stability and control of vibration as well as increased arcing force. The cutting tools on this machine were conical bits with a larger tip and improved bit design to withstand higher cutting forces needed for excavation of hard and abrasive (igneous) rocks. Nevertheless, the trials at the underground hard-rock mine in Sudbury, Canada, seem to have been unsuccessful.

A Voest-Alpine (now Sandvik) roadheader with the ICUTROC cutter system has driven declines for two underground mines in dolomite in the Chicago area and is currently driving perimeter development drifts in one of the mines. The contractor, who owns the machine, has made a number of modifications to the machine to improve productivity. The compressive strength of the dolomite encountered by the machine has ranged from 62 to 120 MPa (9,000 to 17,500 psi).

Other Types of Mechanical Breakers and Excavators

Rock Breakers, Impact Hammers

One group of mechanical excavation equipment that does not fit either of the aforementioned groups and is inherently different in its cutting mechanism is the impact hammer or hydraulic breaker. These tools convert oil flow into the linear and reciprocating motion of a ram that impacts against a replaceable steel rod. Hydraulic breakers are mounted in place of digging buckets on construction equipment such as track or wheel excavators, backhoe loaders, and skid-steer loaders. The breaker is connected to the hydraulic system of the host



(A) Courtesy of The Robbins Company; (B) courtesy of Aker Wirth.

Figure 7.1-15 Hard-rock excavation machines for noncircular openings and their excavation profiles

carrier to receive the input pressure required for operation. The steel rod, whose tip may be a chisel, moil, or blunt tip, delivers percussive force to the material to be broken. Common uses for hydraulic breakers include breaking oversize boulders on orepass grizzlies and scaling to remove loose material on rock faces. Hydraulic breakers are also used for breaking concrete, brick, and asphalt. In some cases, impact hammers have been used to excavate jointed rocks and, in conjunction with water jets, to trim tunnel perimeters.

Miscellaneous Cutting and Excavation Machines

Several partial-face machines not used in mining operations on a regular basis have been developed over time. One such machine is the surface miner by Voest-Alpine. The surface miner is a heavier version of a continuous miner without the height limit that the CMs in underground mines have to comply with. These machines could be used for bench mining and for cut benches as high as 5 to 6 m (16 to 20 ft). The cutting tools of these machines are conical picks, which restricts their cutting medium to rocks with compressive strength below 100 MPa (14,500 psi).

Another machine is the trencher, which can excavate rock trenches as wide as 1 to 1.5 m (3 to 5 ft) and as deep as 10 m (30 ft). The cutting system includes a large crawler-mounted tractor chassis with a cutting boom at the end of the machine. The cutting boom is fitted with a cutting chain laced with conical bits. Trenchers can cut soft to medium-strength rocks.

Similar to trenchers, rock saws are also available in a variety of sizes for different mining applications. These rock saws are used for cutting stress-relief cuts and trenches of 100 to 200 mm (4 to 8 in.) in width and up to 3 m (10 ft) in depth. Rock saws are also fitted with drag bits and conical bits, and they can be used in soft-rock applications, mainly in coal and evaporites.

Diamond-wire saws are also available and very popular in dimension-stone quarries around the world. The diamond- or carbide-impregnated wires are very efficient in cutting large blocks of limestone and marble and are used to a lesser extent in granite quarries. The operation of these saws is very simple and straightforward. Guide holes are drilled on the sides of the blocks in orthogonal directions and the wire is passed through the holes and looped. The wire loop is then driven by a drive

unit and gently tensioned while making a straight cut on the side of the block as it moves out. This operation is repeated on each side until the block is cut from all sides and released.

Another type of rock cutting and excavation system used in mining applications is a water jet. Although the cutting action of water jets does not involve the use of mechanical tools, they are typically grouped with mechanical excavators. The cutting ability of a water jet is controlled by the water pressure, the rate of motion across the rock surface, and the rate of water flow (which is a function of the nozzle size). The jet pressure used in rock cutting typically varies from 10 to 310 MPa (1,450 to 45,000 psi), and the flow rate can range from a few liters per second to thousands of liters per second.

Water monitors, which consist of low-pressure, high-volume water jets, are still used in the mining of alluvial deposits and softer ground, but they are not capable of excavating rock. For rock-cutting applications, higher-pressure, lower-volume jets are preferred. Many water jet systems are available on the market for various applications, and many attempts have been made to use water jets as the primary excavation system, but none has so far proven to be economical. Water jet systems include oscillating jet, vibrating jet, pulse jet, cavitational jet, and abrasive jets.

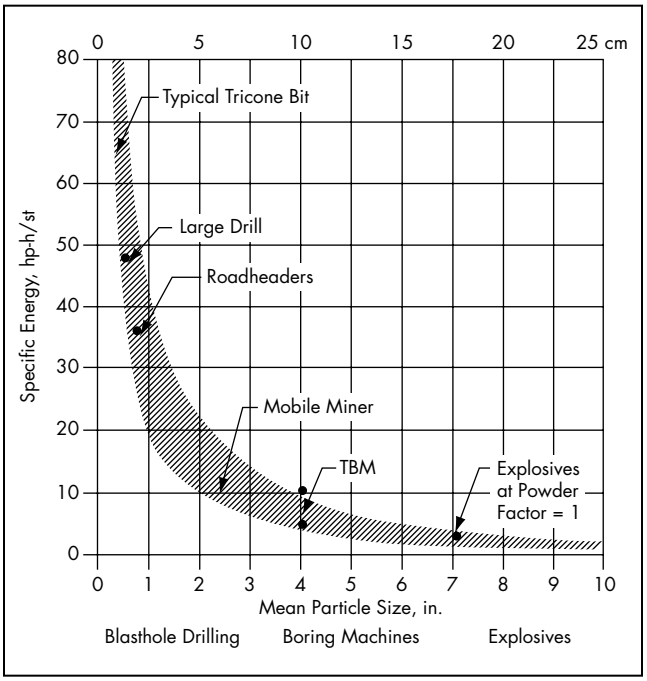
While water-jet cutting dominates the stone-cutting and stone-carving industries, its application in mining is limited to cutting large blocks of granite (in effect replacing line drilling or wire saws) in a few large quarries. In general, water jet systems are effective for high-precision cuts but are not energy-efficient for high-volume excavation. The specific energy of cutting with water jets varies from 100 to 1,000 KW·h/m^3, compared to 0.5-10 KW·h/m^3 for cutting rock using other machines. Thus, the use of water jets in mining and construction applications has been limited to mechanically assisted water jets, which improves the cutting efficiency of partial-face machines, reduces dust generation, and improves bit life. In such applications, lower-pressure jets of 5 to 10 MPa (700 to 1,500 psi) have been employed on various machines such as roadheaders, CMs, and drum shearers.

Lower-pressure jets do not require high-powered pumps, and the piping, seals, and connectors are simpler, cheaper, and easier to maintain. Water jet applications in mining and rock excavation are described in detail in Summers (1995).

Some novel and intriguing systems have been proposed for extraterrestrial or space mining in recent years. One of the most valuable ores is water, which is anticipated to be ice in frozen regolith at the poles and dark side of the moon. Recently some efforts have been made to develop surface mining equipment that can efficiently cut frozen regolith with different degrees of moisture. The cutting system is capable of changing bit spacing to cope with the various strengths of frozen regolith that can be encountered. A prototype model of this system was recently fabricated and tested in material ranging in strength from 3.5 to 50 MPa (500 to 7,000 psi). In practical application, the cutterhead would be mounted on a roadmilling-type machine or on specially designed chariots to excavate the material and load it into containers that could be transported to the base for processing.

PERFORMANCE ESTIMATES

Estimating the rate of production in mining applications and the rate of advance in tunneling or entry development is a crucial step in calculating the mining cost per ton, the cost per meter of tunnel, and the project completion time. In other words, justification of using mechanical excavators depends purely on the production rate and on accurate and reliable performance estimates. There are specific formulas and models available for the estimation of production rate for each machine; however, there is a general guideline calculation of machine performance, based on the SE of the excavation, that can be applied to any mechanical excavator.



Source: Rostami et al. 1994.

Figure 7.1-16 Typical specific energy of cutting for various excavation systems

Advance Rate Prediction Based on Specific Energy

If the SE of cutting for a given rock type and cutting tool and the installed cutterhead power for a machine are known, the instantaneous production rate (IPR) can be calculated as follows:

$$\text{IPR} = \frac{\text{HP} \times \eta}{\text{SE}}$$

where

IPR = instantaneous production rate, m^3/h (yd^3/h)
HP = machine power, kW (hp)
η = mechanical efficiency of the system
SE = specific energy, KW·h/m^3 (hp-h/yd^3)

Figure 7.1-16 shows the SE of cutting for the various sizes of muck particle generated by different drilling and excavation systems (Rostami et al. 1994). Table 7.1-1 provides typical values for SE and η for certain types of machines and rock types. Using this table and the installed cutterhead power of the machine, one can estimate relevant volumetric or mass excavation rates. After the production rate is calculated, the linear rate of advance of the machine can be determined by dividing the IPR by the cross-sectional area, as follows:

$$\text{ROP} = \frac{\text{IPR}}{A}$$

Table 7.1-1 Typical specific energy, mechanical efficiency, and machine utilization for various mechanical excavators*

Machine Types	Rock Type/Strength	Specific Energy, KW·h/m³	Mechanical Efficiency, η, %	Utilization, %	Bit/Cutter Cost, $/t
Tunnel boring machine	Soft (<100 MPa)	5–7	80–85	25–50	2–5
	Medium (100–200 MPa)	8–12			5–10
	Hard (>200 MPa)	12–17			10–15
Roadheader	Soft (<40 MPa)	2–5	80–90	15–40	1–2.5
	Medium (40–70 MPa)	5–8			2–4
	Hard (70–100 MPa)	8–11			4–7
Continuous miner	Soft coal	0.35–0.5	50–65	30–40	0.05–0.08
	Medium coal	0.5–0.8			0.1–0.14
	Hard coal (or soft evaporites)	0.8–1.5			0.15–0.25
	Stronger evaporites	1.5–2			0.25–0.4
Longwall drum shearer	Soft coal	0.3–0.5	60–70	25–40	0.05–0.08
	Medium coal	0.5–0.8			0.1–0.14
	Hard coal (or soft evaporites)	0.8–1.5			0.15–0.25
	Stronger evaporites	1.5–2			0.25–0.4
Road milling machine	Soft (<40 MPa)	0.5–1.0	55–70	35–60	0.05–0.05
	Medium (40–70 MPa)	1.0–1.5			0.10–0.15
	Hard (70–100 MPa)	1.5–2.5			0.20–0.30
Shaft and raise boring and drilling systems†	Soft (<100 MPa)	8–12	75–80	45–60	5–10
	Medium (100–200 MPa)	12–17			10–15
	Hard (>200 MPa)	17–20			15–20

*Notes:
- SE units in metric (KW·h/m³) and English (hp-h/yd³) are nearly the same.
- Estimated values in the table are very general and could vary greatly by ground condition, machine specification, and operator skills.
- Production rates can be impacted significantly by rock mass conditions and jointing.
- Production is also a function of machine mass and rigidity; machine weight typically matches installed power.
- Bit costs are affected by the abrasivity of the rock, design of the cutterhead, and machine weight.

†These results assume the use of strawberry or multi-row carbide cutters. If disc cutters are used, the SE values listed for TBMs are applicable to these machines, but utilization and mechanical efficiency should be slightly reduced.

where

ROP = linear rate of advance, m/h (ft/h)
A = cross-sectional area of opening, m² (ft²)

Obviously, the units used in the calculations should be consistent. The daily advance rate of the machine, which is the daily distance mined in a single heading, can also be calculated by multiplying the linear rate of advance per hour (ROP) by the percentage utilization of the machine per day and the total number of operating hours per day, as follows:

$$\text{AR} = \text{ROP} \times U \times N_s \times H$$

where

AR = daily advance rate
U = machine utilization in percent total time
N_s = number of shifts per day
H = number of hours per shift

Utilization of the excavation machines are estimated based on the ratio of the time that the machine is actually excavating rock to the total time per shift or day.

This estimation system is applicable to any and all mechanical excavation systems from a small drill bit to a large TBM. The system makes some assumptions that may not be valid if there is a mismatch between machine specifications. For example, if the machine has the correct installed power but has low thrust capacity or if it operates within a weak structure that cannot deliver a full load on the cutters, penetration decreases and the specific energy for breakage will increase significantly. Another assumption is that the cutterhead configuration is reasonable for the application; if it is not, specific energies will be underestimated. The given specific energies in Table 7.1-1 assume a reasonably well-designed cutterhead. In general, the production and penetration rates estimated using this method are the maximum achievable rates. Additional information on the SE of cutting can be found in Fowell and McFeat-Smith (1976), Copur et al. (2001), Acaroglu et al. (2006), Bilgin et al. (2006), and Tiryaki and Dikmen (2006).

The following are two examples of using this system for estimation of the production rate of mechanical excavation systems.

Example 1. A 5-m-diameter TBM is chosen to mine a tunnel in medium rock. The machine has 1,000 kW of installed cutterhead power and is assumed to be capable of delivering full cutterhead to the face. Assuming a machine mechanical efficiency of 80% and a utilization rate of 25%, what is the daily advance rate of the machine if the work schedule calls for three 8-hour shifts?

Assuming a 5.7 KW·h/m³ specific energy and machine mechanical efficiency of 80%, one has the following:

$$\text{IPR} = \frac{1{,}000 \times 80\%}{5.7} = 140 \text{ m}^3/\text{h}$$

At 5 m diameter, the area of the face is approximately 20 m² and the linear rate of advance is 7 m/h. With 25% utilization and 24 operating hours per day, the daily advance rate becomes 42 m/d as follows:

$$\text{ROP} = 140(\text{m}^3/\text{h})/20(\text{m}^2) = 7 \text{ m/h}$$

$$\text{AR} = 7 \times 25\% \times 3 \times 8 = 42 \text{ m/d}$$

Example 2. A continuous miner with a cutterhead power of 400 kW is selected to excavate coal in a mine. The coal is of

medium strength and the entries are 3.5 m high by 6 m wide. What is the daily linear footage of entries excavated by the CM? If the machine is used to develop a three-entry panel with 30-m-long, 15-m-wide square pillars, what is the net distance of panel advance? Machine utilization is nearly 40%, and the miners work two 10-hour shifts. What is total tonnage produced on a daily basis if the specific gravity of coal is 1.2 t/m^2?

If one assumes from Table 7.1-1 a 0.8 KW·h/m^3 SE for cutting medium-strength coal and a machine mechanical efficiency of 60%, the following rates can be calculated:

$$\text{IPR} = \frac{400 \times 60\%}{0.8} = 300 \text{ m}^3/\text{h}$$

At a 3.5-m width and a 6-m height, the area of the face, A, equals $3.5 \times 6 = 21$ m^2, and the linear rate of advance is 14.3 m/h (300 m^3/h/21 m^2). The daily linear advance rate then becomes 114 m/d, as follows:

$$\text{ROP} = 300/21 = 14.3 \text{ m/h}$$

$$\text{AR} = 14.3 \times 40\% \times 2 \times 10 = 114 \text{ m/d}$$

The development of one crosscut in the panel requires advance of the entries and connecting them via side entries or crosscuts. To advance the panel 36 m from the start of one crosscut to the start of the next requires an advance equal to the length of one pillar plus the width of one crosscut plus the widths of two pillars. With 30-m-long, 15-m-wide pillars, the total length of excavation in 36 m of panel advance is 138 m ($3 \times 36 + 2 \times 15$) or 3.8 m of entry per meter of panel advance. Thus, with a linear advance rate of 114 m/d, the panel advance rate becomes 29.7 m/d (114/3.8).

Advance Rate Prediction Based on Cutterhead Design

A whole family of estimation methods that are applicable to both full-face and partial-face machines are based on computer modeling of the cutterhead. These models utilize the following:

- Cutting forces acting on individual bits
- Geometry of the cutterhead
- Geometry of the cutting surface
- Bit to rock contact area, the depth of penetration of each bit
- Composition of normal and rolling/drag forces
- Direction of the bits relative to the cutterhead to calculate the overall resultant forces
- Moments acting on the cutterhead while cutting the rock

The calculated values are then compared to the machine capabilities to see how much production can be anticipated from the given machine configurations. The same models are also used for cutterhead design and design optimization to minimize eccentric forces and moments on the cutterhead for full-face machines and can help in minimizing cutterhead vibration on partial-face machines. An optimized design will reduce SE requirements, increase machine productivity, improve bit life, reduce maintenance, enhance machine utilization, and reduce production costs. Overall, it is a worthwhile practice to review the cutterhead design for a particular machine for each mining application to ensure optimal performance. Further discussion on cutterhead design and optimization can be found in Hekimoglu (1995), Rostami and Ozdemir (1996), Rostami et al. (1998), Rostami (2008), Goktan (1995), Cigla and Ozdemir (2000), and Tiryaki et al. (2001).

TBM Advance Prediction

There are several models available for prediction of the TBM performance in hard-rock excavation. One is known as the Colorado School of Mines (CSM) model, which is based on the cutting forces acting on the disc cutters. The CSM model uses rock compressive strength and tensile strength together with cutter tip width and diameter, cut spacing, and TBM specifications to estimate penetration rate. The TBM advance and penetration rate is a function of applied cutter load, rock strength, and the availability of sufficient machine power to rotate the cutterhead. More detailed information on this TBM performance prediction system can be found in Rostami (2008).

An alternative prognosis system was developed by the Norges Teknisk-Naturvitenskapelige Universitat (Norwegian University of Science and Technology) in Trondheim, Norway. The Norwegian model uses specialized rock indices such as drilling rate index, bit wear index, and cutter life index, in addition to joint spacing and orientation and machine specifications to develop an estimated rate of penetration. A more detailed description of the models and examples can be found in Bruland (2000). Overall, either of these models can typically predict machine performance to within ±15%.

Roadheader Production Estimation

Several advance/penetration-rate prediction formulas have been developed for roadheaders. One example is the following formula developed by Bilgin et al. (1996):

$$\text{ICR} = 0.28\text{HP}\,(0.974)^{\text{RMCI}}$$

$$\text{RMCI} = \sigma_c(\text{RQD}/100)^{2/3}$$

where

ICR = instantaneous cutting rate (m^3/h) or simply machine production rate
HP = machine cutterhead power (hp)
RMCI = rock mass cuttability index, MPa
σ_c = rock compressive strength, MPa
RQD = rock quality designation, %

With the ICR, the machine production rate can be estimated, and using the area of the face and the utilization factor as discussed previously, the linear rate of penetration and the daily advance rate can be calculated as was shown in the examples.

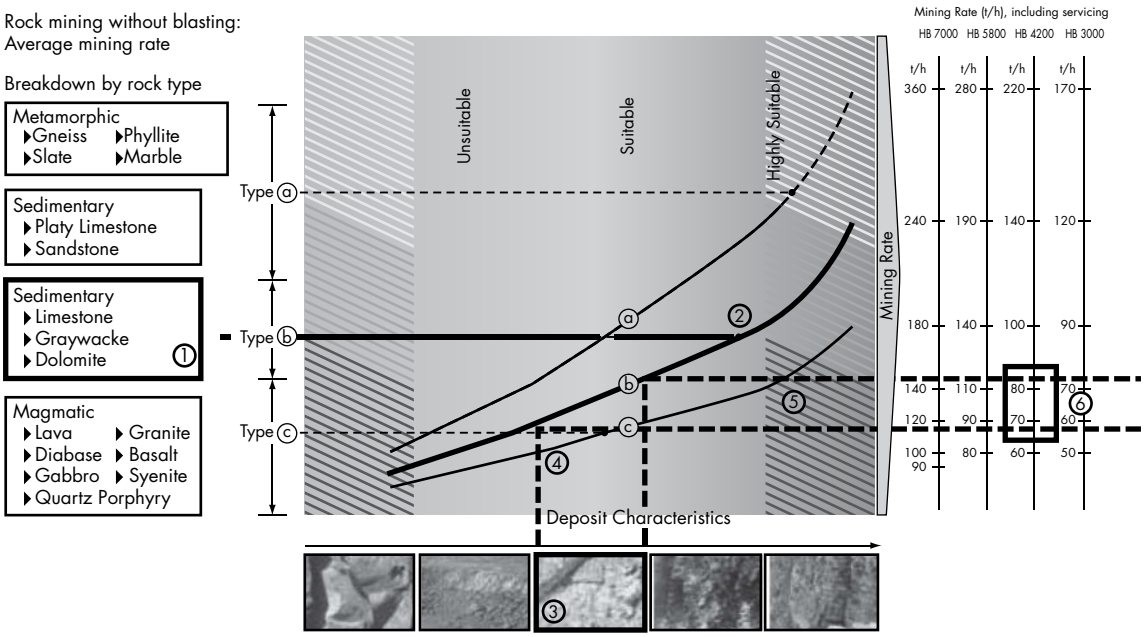
Other performance prediction equations for roadheaders based on compressive strength and specific work required for rock breakage were developed by researchers in Germany (Thuro and Plinninger 1998, 1999). Indeed, roadheader manufacturers generally have their own prediction methods and models.

Impact Hammer Production Rates

Production from impact hammers can be estimated using the chart in Figure 7.1-17.

Estimation of Cutter Life

Cutter and bit life has a direct impact on the economics of employing mechanical excavators. Consequently, the prediction of cutter cost is an important aspect of planning for projects involving hard-rock tunneling. Cutter costs typically range from $0.20 to $0.50 per metric ton for soft-rock mining (e.g., coal, salt, and potash) but could be as high as $1 to $2



Note: Circled numbers represent the steps in a production estimate.

Courtesy of Atlas Copco.

Figure 7.1-17 Impact hammer productivity

per metric ton in roadheader applications in moderately abrasive rocks, $3 to $10 per metric ton for TBM tunneling, and finally in the range of $10 to $20 per metric ton for very hard and abrasive rocks where strawberry cutters are used. A quick estimate of bit life for partial-face machines using conical bits can be found using the following:

$$L_B = 4 \times K_1 \times K_2/\text{CAI}$$

where

L_B = bit life in cubic meters of rock produced per bit change
K_1 = cutterhead speed factor, which varies from 0.8 to 1.2 for high to low cutterhead rotational speeds
K_2 = cooling factor, which varies from 0.9 for dry working condition of the cutterhead to 1.2 where water sprays are used on the head to cool the bits
CAI = Cerchar abrasivity index, which can be simply measured on rock samples in the laboratory

After the volumetric bit life has been estimated, bit cost can be determined from the cost of individual bits and their accessories. A totally different routine can be used to estimate the cost of disc cutters used in tunneling applications. Disc life in linear footage traveled over the face has been shown to have an inverse relationship with the CAI of the rock. That is, the life of a disc in a rock that has twice the CAI is cut by half. The life of the disc also increases linearly with disc diameter and tip width. After the linear footage of disc life is calculated, the overall life of the disc can be determined based on its location on the cutterhead and the number of rotations it can endure, which in turn can be translated to cutter life per hour for a given cutterhead rotational speed. More information on cutter life and cost issues can be found in Deering et al. (1991), Rogers and Roberts (1991), Rostami et al. (1994), Rostami (2001), and Roby et al. (2009).

ADDITIONAL SOURCES OF INFORMATION

A good source of additional information on mechanical excavation are the proceedings of the Rapid Excavation and Tunneling Conferences (RETCs), which are sponsored by SME and the American Society of Civil Engineers (ASCE) and which have been held every two years since 1972. Two journals whose papers are often related to mechanical excavation are the *International Journal of Rock Mechanics, Mining Sciences*, published by Elsevier, and *Rock Mechanics and Rock Engineering*, published by Springer-Verlag Wien. The *Handbook of Mining and Tunneling Machinery* (Stack 1982), which has recently been updated to an interactive CD, is also a great source of information, especially on the background of many of the machines used in the mining and tunneling industry.

Reports and other periodicals from certain government and research organizations are also a good source of information for some of the details on specific tools and applications. Among these are the Reports of Investigations of the Mining Research Group (formerly the U.S. Bureau of Mines) of the National Institute for Occupational Safety and Health (NIOSH) and similar publications by the Commonwealth Scientific and Industrial Research Organisation (CSIRO), Centre for Mining Technology and Equipment (CMTE), and Cooperative Research Centre (CRC) in Australia; the Council for Scientific and Industrial Research (CSIR) in South Africa; Canada Centre for Mineral and Energy Technology (CANMET) in Canada; and SveBeFo in Sweden.

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CHAPTER 7.2

Blasthole Drilling

Jamal Rostami and Douglas F. Hambley

INTRODUCTION

Drilling is an important unit operation in mining. Drilling is required for a variety of mining or mining-related activities including blasting, ground support installation, utility installation, dewatering, exploration, and much more. The term *drilling* is used for smaller-diameter holes, with a few exceptions such as shaft drilling and raise drilling, where larger excavations are created by special machines including cluster drills. This chapter is primarily concerned with mine production. Most boreholes drilled for mine production are blastholes for explosives. However, boreholes drilled for use in solution mining will also be discussed.

The majority of blasthole drilling is performed by two primary methods:

1. Percussive drilling
   - With smaller hole sizes (diameters up to 150 mm [6 in.]), the impact and rotation of the bit used to fracture the rock is transmitted from the drill (top hammer) to the bit through a drill rod or series of drill rods coupled together (referred to as a drill string).
   - With larger hole sizes (diameters from 75 mm to 508 mm [3 to 20 in.]), the hammer is located down-the-hole (DTH) or in-the-hole (ITH) at the bottom of the drill string immediately above the bit, and the impact is transmitted directly to the bit.
2. Rotary drilling
   - Drag bits are used for 75–250 mm (3–10 in.) holes in soft rock.
   - Tricone bits are used for 75–300 mm (3–12 in.) holes in medium and harder rock.
   - Larger roller bits are used for holes larger than 300 mm (12 in.).

A third drilling method, jet piercing, which was formerly used to drill blastholes in taconites and other very hard, abrasive rocks, has largely been superseded by rotary drilling with tricone bits with tungsten carbide (TC) buttons.

PERCUSSIVE DRILLING

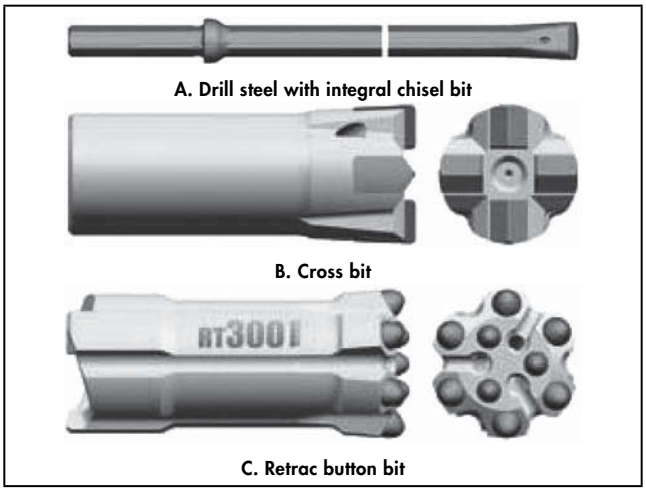
In percussive drilling, the rock is broken by a combination of rotation of the bit and high-frequency percussive impacts transmitted by the bit to the rock. The percussive impact is delivered by either pneumatic or hydraulic pressure. Percussive drills were originally powered by compressed air. Since the mid-1970s, however, hydraulically powered drills have supplanted pneumatic ones, except in the case of small, handheld equipment such as stopers and jackhammers. The advantages of hydraulic drills over pneumatic drills are the fewer moving parts and the significantly higher penetration rates. The depth of percussive drilling with top hammer drills is limited to approximately 76 m (250 ft), due mainly to losses in energy transfer at joints in the drill string. Larger and deeper holes can be achieved using DTH drills.

In DTH drilling, compressed air is supplied through the drill string to drive the hammer, which is located immediately behind the bit. The compressed air activates a hammer that delivers impact directly to the bit. This eliminates the loss of impact energy in joints and is a more efficient mechanism of percussive drilling. Rotation of the bit is provided by the motor at the top and transferred through the drill string. The diameter of the drill string, usually 150 mm (6 in.) or more, helps limit deviation of the hole, which improves drilling accuracy. As an example, at a Canadian uranium mine in 1978–1979, the second author of this chapter supervised the drilling with a DTH drill of 200-mm (8-in.) and 311-mm (12¼-in.) holes to depths of approximately 490 m (1,600 ft), with an error of less than 4.5 m (15 ft) at the target depth.

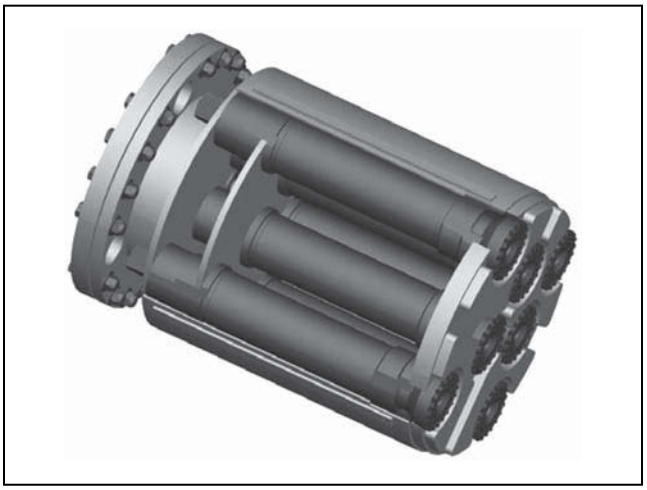
Percussion Drill Bits

Bits for top hammer percussion drills come in various shapes, as shown in Figure 7.2-1. The drill steel with an integral "chisel" bit and a single TC insert is used with handheld jackhammer (sinker) drills in shaft sinking. More common are the hammer-on or screw-on cross-shaped bits with four chisel-shaped TC inserts. However, in harder rocks especially, the chisel-shaped bits are being replaced with button bits, of

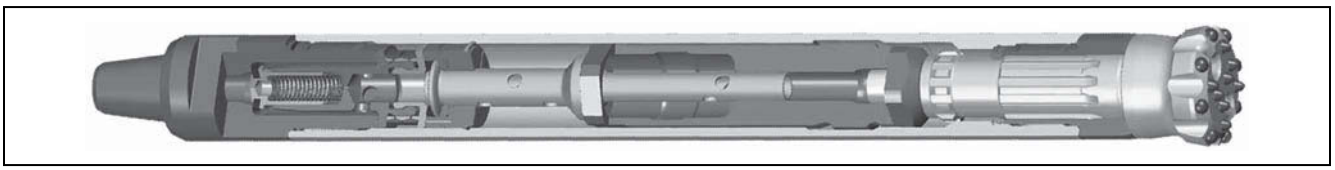
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Douglas F. Hambley, Associate, Agapito Associates, Inc., Golden, Colorado, USA



Source: Sandvik 2009.
Figure 7.2-1 Top hammer percussive drill bits



Source: Atlas Copco 2007.
Figure 7.2-3 Cluster drill head



Courtesy of Atlas Copco.
Figure 7.2-2 Cutaway view of a DTH hammer and bit

which the retrac bits are an example. (Retrac bits are designed to facilitate removal of the bit and steel from the hole, which is crucial in soft or squeezing ground.) The shape of the buttons is selected based on the application and the type of rock to be drilled. Longer, pointed inserts are used for softer rock, while shorter, rounded inserts are used in harder rock. A bit can be initially selected using charts from commercial bit manufacturers; however, optimum bit geometry and button shape is usually determined by trial and error at the job site.

DTH drills were introduced by Ingersoll Rand Company in 1955, and the concept has changed little since then. Figure 7.2-2 shows a cutaway view of a DTH hammer and bit. The weak link in the DTH system is often the plastic foot valve between the piston and the bit, which can break easily if not seated properly.

Cluster drills are extra-large impact drill heads that incorporate a number of DTH hammers and bits. They can be as large as 2 m (6½ ft) in diameter. Depending on the size of the hole, the drills may have various numbers of hammers at the face, with the individual bits receiving impact energy from their individual hammers. The entire drill head is rotated via a drill string so that the hammers will cover the entire face. Cluster drills are often used in very hard and abrasive rocks, where a limited number of holes are planned, or in sinking small, shallow shafts less than 300 m (1,000 ft) deep. Figures 7.2-3 and 7.2-4 show a typical cluster drill.

Percussive Drilling Equipment for Underground Use

Handheld jackleg and stoper drills are lightweight percussive drills powered by compressed air. They are used for drilling small blastholes that are generally 35 mm (1.375 in.) or smaller in diameter. They are typically used to drill holes from 1.8 to 3 m (6 to 10 ft) in length, in 0.6-m (2-ft) increments, as that is the approximate extension of the piston in the air leg; holes up to 3.3 m (12 ft) long can be drilled by a skilled driller. Advance rates are generally inversely proportional to the rock strength and the cross-sectional area of the drill hole. Typical drilling rates for handheld drills range from 0.3 to 0.5 m/min (1 to 1.7 ft/min), depending on the rock mass properties.

Drilling rigs for underground mining applications can be divided into face drilling and production (or long-hole) drilling rigs. Face drilling is performed by mobile rigs equipped with drills mounted on one boom or multiple booms. A two-boom jumbo is shown in Figure 7.2-5. The number of booms and drills depends on the opening dimensions and rock mass properties, the number of holes to be drilled per blast round, and the number of faces to be drilled in a shift. In hard rock metal mining, two-boom or three-boom jumbos are used, whereas in limestone mining, two- or single-boom drill jumbos are common. Hole diameters typically range from 35 mm to 51 mm (1⅜ to 2 in.) with the exception of the reliever holes used in burn cuts, which are somewhat larger. Drill feeds are typically 3.7 to 4.3 m (12 to 14 ft); however, drill feeds of up to 6.4 m (21 ft) are available. Drill jumbos were historically powered by compressed air. Since the late 1970s, however, electric/hydraulic and diesel/hydraulic units have almost completely supplanted the older pneumatic (compressed air) units because of the much faster penetration rates that are achievable with the larger, more powerful hydraulic drills.

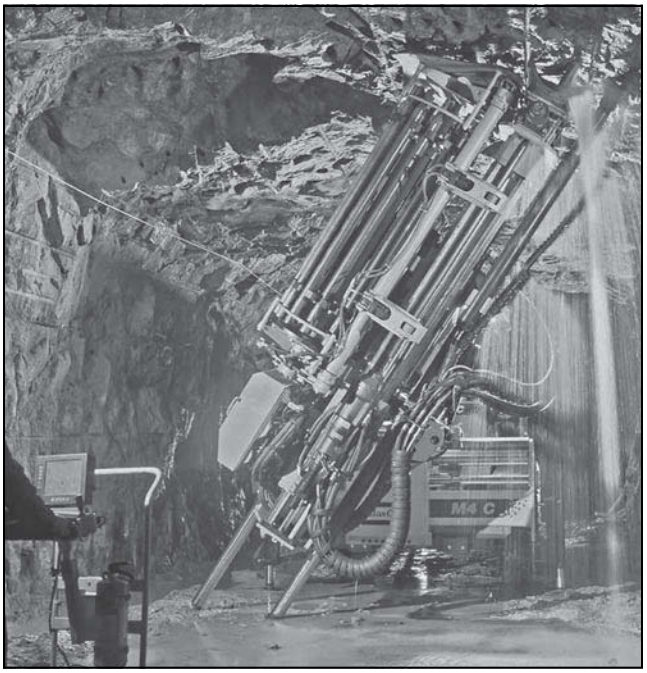
Ring-drilling production drills are used in underground metal mines to drill the long inclined or vertical blastholes (typically >6.1 m [20 ft] long) used in sublevel stoping, sublevel caving, and vertical crater retreat mining. Figure 7.2-6



Source: Atlas Copco 2010a.
Figure 7.2-4 Cluster drill in a shaft excavation



Source: J.H. Fletcher and Company 2007.
Figure 7.2-5 Two-boom jumbo for face drilling



Source: Atlas Copco 2010b.
Figure 7.2-6 Typical ring-drilling production drill for underground metal mining

shows a typical production drill unit in operation drilling an inclined uphole. In such operations, the drilling operation may include both long production blastholes and ground support installation (primarily cable bolts). To ensure good fragmentation, it is imperative for these holes to have the correct length and direction. To achieve this, it must be possible to easily rotate the drill boom to various positions, maintain the proper hole alignment, and minimize the deviation from the proper location. A bit with a flat or concave front face will create a straighter hole than a bit with a convex front face. Retrac bits with long bit skirts were introduced to improve drilling accuracy and reduce hole deviation. Another method to reduce hole wandering is to use larger-diameter drill strings, whereby the small difference between the diameter of the bit and the drill string limits the amount of bending of the drill string and thus keeps the hole straighter. Because of the need to flush drill cuttings from the borehole, the drill string on a noncoring drill cannot be the same diameter as the bit.

For upholes, drills often use an ITH drilling system that delivers the impact directly to the bit. Combined with the larger-diameter drill tube, this allows for much higher accuracy and lower deviation. For example, in a hole that is 30 m (100 ft) long, drill-hole deviation may be as much as 3 m (10 ft), or 10%, when a top hammer and a conventional rod and bit are used. When the same drill is used with a retrac bit and guide tube behind the bit, the deviation can be reduced to 1.5 m (5 ft), or 5%. With an ITH hammer, however, that deviation can be reduced to 0.5 m (1.7 ft), or 1.7%. For very long holes, a stabilizer/reamer behind the ITH hammer will reduce deviation even further.

For shaft sinking applications, specialized jumbo drills can be used. Shaft drill jumbos use the same drilling equipment and bits as other drilling rigs—the only difference is that instead of a drill carriage, they are mounted on a work platform that hangs on a set of ropes. The jumbo is lowered



Source: Atlas Copco 2010c.
Figure 7.2-7 Crawler-mounted top hammer percussive surface drill

to drill at the face and raised a safe distance above the face to minimize damage from flyrock from the blast.

Typical hole diameters for production drilling with regular top hammer systems range from 50 to 100 mm (2 to 4 in.); with the ITH system, the range is from 100 to 254 mm (4 to 10 in.). The drill string diameter for regular rods varies from 38 to 51 mm (1½ to 2 in.), and the lengths of individual drill rods typically range from 1.2 to 1.8 m (4 to 6 ft), but can be up to 3.0 m (10 ft). For tube drilling in conjunction with ITH, drilling rods range from 45 to 87 mm (1¾ to 3½ in.) in diameter and from 1.5 to 3.6 m (5 to 12 ft) in length. (With ring drilling, the length of the rod is constrained by the height and width of the mine opening.)

Penetration rate is a function of the impact energy imparted to the bit, frequency of impacts, feed pressure, rotational speed, drill bit type, and rock mass properties. A typical drill unit is capable of delivering 10 to 25 kW of energy to the drill string with an impact frequency between 20 and 70 Hz (impacts per second). The drill rotational speed ranges from 0 to about 100 rpm, with the maximum for a given hole size defined by an angular speed at the bit perimeter of approximately 0.4 m/s (1.4 ft/s). Depending on the rock mass properties, penetration rates of 0.5–2.5 m/min (1.6–8 ft/min) can be achieved. It should be noted here that the penetration rates given previously in this paragraph do not account for the time required to retract the drill from a hole, reposition the boom, and collar a new hole, which can take from 30–60 seconds per hole, depending on the type of equipment.

For the sizes of blastholes normally encountered underground (35–100 mm [1⅜–4 in.]), jointing doesn't usually affect the ability to drill blastholes unless the rock is very closely jointed, as in a shear zone, or the drill hole is in the zone of back break behind the previous blast in the case of benching. Hence, other things being equal, the penetration rate in underground blastholes can be considered to vary inversely with the rock strength.

Percussive Drilling Equipment for Surface Use

Drilling equipment for surface mining applications can be categorized into two classes—percussive drills for small to medium-sized holes of up to 200 mm (8 in.) and rotary drills for larger sized holes of up to 450 mm (18 in.). Surface drilling units are generally crawler mounted. Older percussive drills (known as *air tracks*) may use compressed air as their power source; however, newer machines use hydraulic drills and are diesel powered. Percussive drilling for surface blasting typically uses holes ranging from 89 to 150 mm (3½ to 6 in.). Both top hammer and DTH drills are used. Figure 7.2-7 shows a crawler-mounted, surface percussive drill.

The lengths of drill rods used in surface drilling are selected to minimize the time for adding or removing the rods. Typical drill rods for percussive surface drilling range from 50 to 150 mm (2 to 6 in.) in diameter, and from 4 to 6 m (13 to 20 ft) long. The weight loading on the bits ranges from 1 to 4 t (1.1 to 4.4 st). Drilling rates for typical surface drills vary from 2 to 80 m/h (6 to 240 ft/h). They increase with increased power of the rig, and decrease with increasing hole diameter and rock mass strength.

Small, handheld drilling equipment has all but disappeared from surface mining applications, except for breaking oversize boulders or drilling sinking cuts for starting a bench in very inaccessible places in the mine. Most surface mines use impact hammers for breaking large or oversized boulders to avoid secondary blasting.

Percussive Drilling Penetration Rate

Typical penetration rates for various sizes and types of percussive drills are presented in Table 7.2-1.

ROTARY DRILLING

With the exception of special applications that use cluster drills, larger holes are typically drilled by rotary drilling. A typical rotary drilling rig used in surface mining is presented in Figure 7.2-8. With rotary drilling, the drill bit is rotated by applying torque at the end of the drill string, which results in removal of chips from the face of the hole. Power for bit rotation and penetration is either diesel or electric.

Since their introduction in the early 1900s by the Hughes Tool Company, tricone bits have been the traditional type of bits used in rotary drilling. They remain the most popular bits for blastholes ranging from 150 to 444 mm (6 to 17½ in.) in

Table 7.2-1 Comparison of different percussive drilling systems and their production rates

Drilling Method	Range of Diameter, mm	Typical Depth, m	Typical Drilling Rate, m/h*	Accuracy, % or cm/m	Application
Top hammer pneumatic jackleg	25–50	<4	60	2–5	Used in small tunnel and drift development, small-scale stoping, surface operations as starter, shallow trenching, bolt installation, etc.
Jumbo	37–100	4–10	100–200	1–3	Tunneling, stoping, fan drilling, bolt installation, and probe drilling
Surface crawler-mounted drills	50–150	5–30	20–150	1–3	Surface and bench drilling
Top hammer hydraulic jumbo drills	37–125	4–20	30–120	2–3	Tunneling, stoping, fan drilling, bolt installation, and probe drilling
Down-the-hole	75–150	10–50	20–80	3–5	Surface and bench drilling

*Drilling rates are for actual drilling and do not include time for drill retraction, boom relocation, collaring, etc. Drilling rates decrease as rock mass strength increases and may increase or decrease depending on the number and orientation of joints/blocks.

diameter. Drag-type bits have gained an increasing share of the market since the introduction of polycrystalline diamond compact (PDC) bits in 1976 (Baker Hughes 2008). PDC drag bits are more expensive than tricone bits, but for relatively deep holes, the time saved due to longer intervals between bit changes may justify the price.

Bit selection is based on hole size and depth, rock type, and operational requirements. Sources of information to guide selection of the proper bit include manufacturers such as Atlas Copco and Sandvik for percussive drill bits, and Smith International and Baker Hughes for tricone and drag bits. Figure 7.2-9 shows typical drag and tricone bits used in rotary drilling. The weight loading on the bits ranges from 10 to 74 t (11 to 81.6 st).

The rate of advance for rotary drilling is a function of the rate of rotation, bit diameter, weight on the bit, and rock mass properties, and is well predicted by the following empirical equation (Calder 1973):

$$P = (61 - 28 \log_{10} S_c)\, W\, R/(250\, D) \qquad (7.2\text{-}1)$$

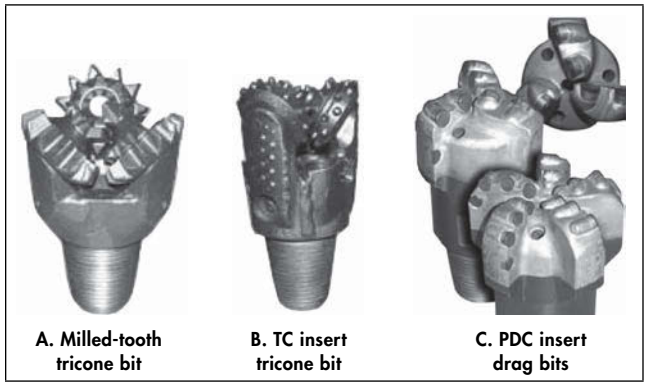
where
P = penetration rate (ft/h)
S_c = rock compressive strength (thousands lb/in.^{2})
W = pulldown weight of drill (thousand lb)
R = drill rotary speed (rpm)
D = hole diameter (in.)

In metric units, Equation 7.2-1 becomes

$$P\,(214 - 98 \log_{10} (0.145\, S_c))\, W\, R/(250\, D) \qquad (7.2\text{-}2)$$



Source: P&H Mining Equipment 2010.
Figure 7.2-8 Large rotary drilling equipment for surface



Courtesy of Bit Brokers International, Logan, Illinois.
Figure 7.2-9 Three types of rotary drill bits

where
P = penetration rate (m/h)
S_c = rock compressive strength (MPa)
W = pulldown weight of drill (t)
R = drill rotary speed (rpm)
D = hole diameter (mm)

Table 7.2-2 provides a general guide to surface blasthole drill productivity in various rock types. It is assumed that the work is performed on a reasonably prepared bench and includes the following operations:

- Moving the drill from prior hole to next hole location
- Jacking/leveling the drill
- Aligning the mast/feed for plumb or desired angle orientation
- Collaring and advancing the first rod
- Adding additional rods and advancing them until the required hole depth is reached
- Tripping out the drill string until it is clear of the hole
- Checking the hole/collar for depth/condition
- Retracting the jacks and making the drill ready to tram

Table 7.2-2 Generalized production rate of drilling units in surface mining

Drilling Conditions	Soft Rock	Medium Soft Rock	Medium Hard Rock	Hard Rock
Rock strength, MPa	70–100	100–175	175–225	225–300
Rock strength, 1,000 psi	10–15	15–25	25–35	35–45
Rock fabric condition	**Drilling rate, m/h***			
Stable, uniformly competent ground conditions	55	33.5	30.5	24.5
Competent rock, fractured collar zone (top 0.6 to 1.2 m of bench)	49.5	30	27.5	22
Rock with closed joints, fractured collar zone (top 0.6 to 1.2 m of bench)	42	25.5	23.5	18.5
Rock with open joints, fractured collar zone (top 0.9 to 2.4 m of bench)	33.5	20.5	18.5	15
Heavily jointed, poorly cemented rock, fractured collar zone (top 1.2 to 3 m of bench)	27	16.5	15	12

Courtesy of W. Hissem.
*To obtain drilling rates in feet per hour, multiply by 3.28.

The need for a stable drill setup with the jacks in place and activated cannot be overemphasized. In addition, for long holes, it is important to place centralizers at intervals in the drill string, especially if the rods are significantly smaller in diameter than the drill holes.

DRILL HOLES FOR SOLUTION MINING

Boreholes drilled for solution mining are not production holes in the same sense as blastholes; however, they are production holes in the sense that they are used to either inject or remove the fluid used to dissolve the rock and have a limited life span. Another difference is that after the holes have been advanced to their ultimate depth, they are cased and cemented and fitted with production tubing, a process known as *well completion*. Several different well-completion types are found in solution mines. The Frasch process for solution mining of sulfur utilizes concentric pipes, with hot water injected in one pipe and the sulfur-bearing pregnant solution removed in another.

In solution mining of salt and potash, concentric pipes are used to develop the injection and recovery wells. When the wells have reached the target depth and the sumps have been excavated, mining proceeds laterally until the two wells are connected. At this point, one well is converted to an injection well and the other becomes the recovery well.

Wells used for solution mining are generally drilled using oilfield equipment and mud-rotary drilling techniques. As with oil and gas drilling, selection of the proper chemistry and density of the mud is important. In evaporite zones, the saturated saline brines used for mud in normal oilfield drilling will dissolve potash minerals and should be replaced with oil-based invert mud. The outside diameter of solution mine wells typically ranges from 219 to 346 mm (8⅝ to 13⅝ in.) and depends on the completion configuration and tubing diameters required to transport the fluids.

NEW DEVELOPMENTS IN PRODUCTION DRILLING

The most important advance in drilling equipment since 1990 has been the development of computer-controlled drilling systems. These systems automatically locate and collar the holes, based on a preprogrammed blast round design, and incorporate real-time monitoring and optimization of the drilling. In surface mining, drill rigs have been fitted with Global Positioning Satellite systems, allowing for better control of the drill-hole locations.

Drills have been fitted with computer control systems that allow for optimization of drilling parameters such as feed rate, weight on bit, and torque to achieve the highest possible drilling rates. These systems allow for uploading of the drilling pattern into their computer via storage devices or, more recently, via wireless communication.

In underground mining, multiboom jumbo drills can be programmed to drill the desired blasthole patterns automatically, through coordination with an automated surveying and guidance system, and simultaneously monitor the drilling parameters and optimize the control parameters. Interfaces also exist that allow for ground characterization while drilling by comparing the current drilling rates and other parameters with stored data for drilling in similar rock types (Atlas Copco 2004; Schunnesson 1998; E. Tanner, personal communication). In this manner, the instrumentation system can provide an estimate of the rock properties.

Directional drilling has also made significant progress and is used with high speed and an advanced degree of accuracy for applications in oil well drilling, coal mine degasification, as well as civil engineering. Direction changes are effected using a bent sub together with a downhole mud motor drive unit to rotate the bits. The mud motor drive works in conjunction with rotary bits and receives the thrust force and torque reaction from the drill string, which in this case does not rotate.

Coil tubing, where the drill string is rolled on a reel and dispensed rapidly as needed, has been used extensively in oil/gas well drilling to eliminate the need for breaking and assembling the drill steels. This technique is only possible with mud motors, since there is no need to rotate the drill string. These systems are capable of making bends with radii as tight as 6–10 m (20–30 ft) in holes 75–100 mm (3–4 in.) in diameter. Because of this capability, coil tubing is commonly employed on workover rigs used to rehabilitate wells used for solution mining and oil production.

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CHAPTER 7.3

Explosives and Blasting

Braden Lusk and Paul Worsey

INTRODUCTION

Noise, dust, and vibrations created by blasting are objectionable to many in our modern society; however, blasting is generally necessary to break large volumes of rock, especially if the rock is medium to high strength. An indication of the importance of blasting is that more than 1 billion kg of explosives are used annually in Australia and 3 billion kg in the United States for rock breakage, more than 85% of which is used in the mining industry. Explosives are the first choice of mining engineers for primary fragmentation and will continue to be so for the foreseeable future.

The blasting industry saw substantial changes since 1985, with many traditional techniques now verging on obsolescence while the explosives themselves have changed dramatically. Since 2000, the use of emulsions and ANFO (ammonium nitrate and fuel oil)/emulsion blends has expanded to meet the needs of almost every mining and construction situation. This is linked with the development of a full understanding of emulsion systems and their limitations and with the development of new techniques that allowed these systems to be optimally used.

This chapter concentrates on modern blasting techniques; explosives and initiation systems; concepts of fracturing and fragmentation; basic blast design for surface and underground mining; common misconceptions concerning blasting; estimation of blasting costs; description and control of blast effects; and trends, automation, and the future of blasting. The environmental effects of blasting and their control are discussed in depth in Chapter 16.3.

EXPLOSIVE AND INITIATION SYSTEM CLASSIFICATIONS

Primary and secondary explosives are distinguished based on their sensitivity to impact, heat, electrostatic shock, and friction. Secondary explosives normally require at least a primary explosive for initiation, whereas primary explosives are used as initiators. Nobel's original design for a detonator utilized mercury fulminate, a primary explosive, for successful and repeatable detonation of dynamite. Most modern detonators still employ primary explosives as the initiator for the base charge (a secondary explosive), which is usually pentaerythritol tetranitrate (PETN). Most commercial detonators use lead azide or diazodinitrophenol as the primary explosive in the detonator; however, other primary explosives include mercury fulminate, lead styphnate, and tetracene. Some manufacturers are now inserting PETN into a stainless-steel delay element directly against the pyrotechnic compounds that burn to create the delay timing. In this confined condition, the PETN can detonate without the use of primary explosives.

Classification of Explosives

Classifications of practical importance to mining engineers include the basic separation of types of explosives and their classification by regulatory agencies, which are represented in each state and territory, for commercial use in the mining industry.

An explosive is defined as a compound or a mixture of compounds, which, when initiated by heat, impact, friction, or shock, undergoes a rapid decomposition, releasing tremendous amounts of energy in the form of heat, gas, and shock. To define the different classifications used by regulatory agencies, an understanding of the distinction between detonation and deflagration is required. Detonation occurs when the rate of reaction in the explosive product exceeds the speed of sound (sonic velocity) in the product, thus creating a shockwave. Detonation velocities for commercial explosives range from 1,500 to 7,830 m/s, which is much higher than the sonic velocities. Deflagration is a process where the reaction occurs at rates much lower than the sonic velocity of the explosive material, so that no shock (primary pressure wave) is produced within the explosive material.

Regulatory authorities classify explosives into the three groups shown in Table 7.3-1. The distinction between high and low explosives by regulatory authorities closely follows the scientific definition; that is, high explosives detonate and low explosives deflagrate. Regulatory authorities identify a third explosive classification, blasting agents, in defining explosive storage requirements. Most blasting agents are capable of

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Table 7.3-1 Regulatory authorities' classifications and definitions

Classification	Definition
High explosives	Explosive materials that can be caused to detonate by means of a blasting cap when unconfined (e.g., dynamite)
Low explosives	Explosive materials that can be caused to deflagrate when confined (e.g., black powder, safety fuses, igniter cords, fuse lighters, and "display fireworks")
Blasting agents	Materials that can be detonated but are not a detonator (e.g., ANFO and certain water gels)

Source: Adapted from CFR 2009.

detonation; however, regulatory authorities reserve a separate classification for these explosive products due to a relatively lower sensitivity than other high explosives. Regulatory authorities define blasting agents as explosive products that cannot be detonated by a Number 8 blasting cap. A Number 8 blasting cap is defined as one having 0.4 to 0.45 g of PETN base charge as discussed in the "Classification of Modern Detonators and Initiation Systems" section. In the mining and blasting industries, an explosive product is termed *detonator sensitive* if it can be detonated by a Number 8 test detonator. According to this definition, blasting agents are not "cap sensitive." The regulatory authorities' classification in Table 7.3-1 determines the requirements for the storage of explosives on-site.

The transport classification is another important explosives classification system that is useful to mining engineers. This classification is primarily used for the transportation of explosives, but certain aspects of it are used by the regulatory authorities for regulation of explosives storage. The transport regulations classify explosives as Class 1 Hazardous Materials (CFR 2008). Class 1 is subdivided into six subclasses, 1.1 through 1.6, from most sensitive to least sensitive, respectively. The transport hazard classification is used for placarding vehicles that transport explosives (and other hazardous materials) on public roads and highways. The mining industry generally uses products from Subclasses 1.1, 1.4, and 1.5. Table 7.3-2 presents the transport classifications for explosives and examples of each class. In addition to the six subclasses, explosives are further separated into different compatibility groups identified by letters following the subclass number. The classification systems are changed periodically as new products emerge and as changes within the industry take place.

The classifications described in Table 7.3-2 are established by the respective regulatory agencies to guide policies in regard to storage and transportation of explosives. The classifications are also useful in the application of explosives. For example, the classifications can be useful for the selection of products for priming blastholes. A majority of mining operations use blasting agents (Subclass 1.5) for production blasting. Blasting agents require high explosive (Subclass 1.1) boosters to function at an optimum level because they are not detonator sensitive. Manufacturers also provide product technical information that includes minimum priming recommendations. This information will be the most reliable source for primer selection. Several options, ranging from dynamites to pentolite cast boosters, are available for priming. When a booster is combined with a detonator, it becomes a primer. Several regulatory authorities and Mine Safety and Health Act regulations concern the proper use, creation, transportation, and storage of primers. Note, especially, that primers are not to be prepared until they are ready to be placed in the blasthole.

Table 7.3-2 Department of Transportation's classification of explosives

Subclass	Definition of Subclass	Examples
1.1	Explosives with a mass explosion hazard	High explosives, boosters, dynamite, TNT, C4
1.2	Explosives with a projection hazard	Ammunition
1.3	Explosives with a fire hazard and a minor blast hazard or a minor projection hazard, but not both	Commercial (display) fireworks
1.4	Explosives with no significant blast hazard; effects largely confined to the package	Specially packaged explosives for prevention of mass detonation (e.g., packaged detonators)
1.5	Very insensitive explosives; mass explosion hazard but little probability of initiation during transport	Blasting agents such as ANFO and emulsions
1.6	Extremely insensitive articles with no mass explosion hazard	None at this point

Source: Adapted from CFR 2008.

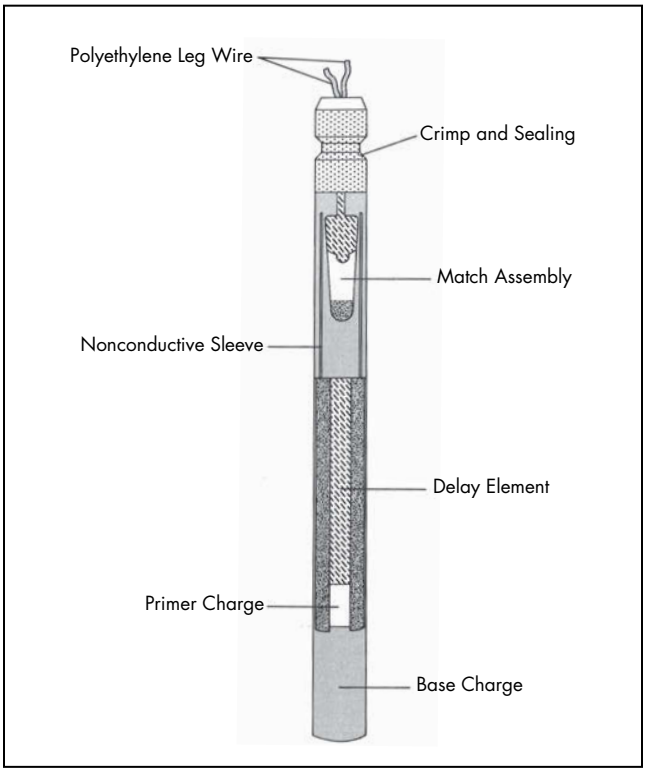
Explosives can also be classified according to their packaging: cartridge versus bulk. Cartridge explosives must be smaller in diameter than the hole. Therefore, until tamped, they will not fill the hole. In contrast, bulk explosives come in bulk containers and are less labor intensive during loading. They also fill the entire hole diameter, making them more efficient to use. The majority of explosives that are used in mining today are bulk explosives, which are blasting agents such as ANFO, emulsions, and blends of ANFO and emulsion. In 2008, dynamite represented only 0.5% of all the explosives used in the United States, and there was only one remaining dynamite manufacturing plant in North America.

Classification of Modern Detonators and Initiation Systems

Three basic types of initiation systems are available for use in commercial blasting: electric, nonelectric, and electronic. There are many variations of each type of system, but all modern initiation systems fall into at least one of these categories. For many years, electric detonators dominated the industry, and cap-and-fuse initiators for many years before that. Electric detonators were used because of their availability and reliability. Other systems were not used as extensively until the introduction of shock-tube initiators. By 1990, shock-tube initiation systems became the majority of initiation products in use. Throughout the 1990s and 2000s, shock-tube initiation systems gained progressively more market share due mostly to their immunity to accidental initiation caused by two-way radio transmissions and stray current sources. With mining operations using radio dispatch systems for hauling and loading cycles, the need to turn off two-way radios in the presence of electric initiation systems was not acceptable. The three major types of initiation systems are discussed as follows.

Electric

Electric products were versatile and allowed for significant advances in blast design through the use of in-hole delays. There is a large variety of electric detonators, but the commercial



Source: Atlas Powder Company 1987.

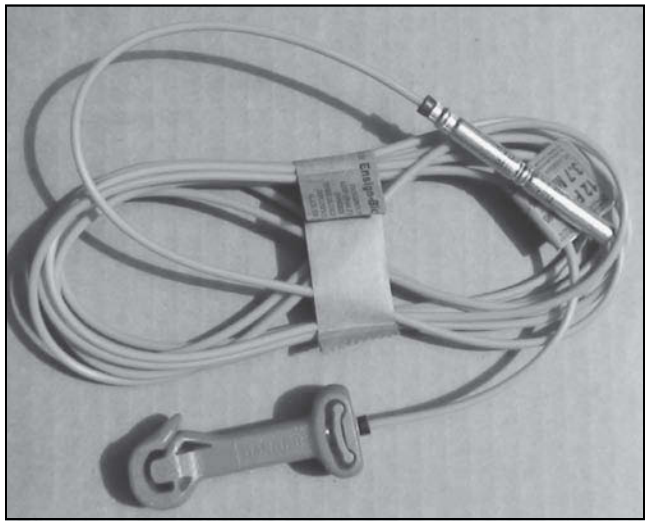
Figure 7.3-1 Inner workings of a typical electric detonator

blasting industry typically utilizes low-firing-current electric detonators. These detonators are initiated when a small amount of current is introduced to a bridge wire within the detonator shell via two leg wires. This bridge wire heats up and ignites a pyrotechnic match head or bridge wire embedded in a pyrotechnic compound. The match head ignites the delay element, which is manufactured to burn for a specific amount of time prior to reaching the primer charge consisting of a primary explosive such as lead azide and a base charge usually made of PETN. Figure 7.3-1 shows the inner workings of an electric detonator as described in this chapter.

Sequential timers were created for use with electric detonators to increase the number of design options available for delaying blastholes. For electric blasting caps, sequential timers allowed for several series of electric delay detonators to be purposefully staggered by electronics within the sequential timer. This allowed for many design options while still only using a minimal amount of individual delay caps. Sequential timers are no longer used extensively, but the idea of programmable detonators probably began with these devices.

Nonelectric

There are many types of initiation systems that could be considered nonelectric: detonating-cord (usually referred to as "det-cord"), cap and fuse, and nonelectric shock-tube detonators. The shock-tube type detonator is now the most common form of nonelectric initiation system. Nonelectric shock-tube detonators are responsible for the demise of electric detonators as the preferred initiation system. Although they are sometimes used in conjunction with a detonating cord, the majority of mining operations using nonelectric detonators utilize a combination of in-hole and surface delays.



Courtesy of Ensign Bickford.

Figure 7.3-2 An EZDet nonelectric detonator. The narrow metal cylinder (top) is the in-hole cap and the plastic connector (bottom) is the surface delay.

There are two common ways to use shock-tube systems. One method uses downhole delays that are completely separate from the surface delay detonators. Nonelectric shock-tube detonators are also produced in a dual-delay system where the downhole delay and the surface delay are attached at opposite ends of a single shock tube (Figure 7.3-2). These provide a similar design advantage to sequential timers. They are offered by all of the major manufacturers for use in blasting. The concept consists of a 200-to-500-ms delay in the hole with a much shorter (17 to 109 ms) surface delay between holes. The surface delay detonators are simply daisy-chained, which allows for a much larger number of delays than with single-delay detonators.

Electronic

Electronic detonators utilize microchip technology to provide delays for blast designs. Most electronic detonators consist of wires, a detonator shell that looks similar to electric and nonelectric detonators, a microchip, a capacitor or capacitors, and a primer charge/base charge similar to electric and nonelectric detonators. In most cases, the detonators will communicate with a logger/machine, and individual detonators can be checked prior to firing. At firing time, the blasting machine sends out a code to initiate the electronic timing devices within the detonators. When the detonators reach their programmed time, the electronics within them transfer the remaining current to the fuse head after which the process is similar to an electric detonator. The difference between electronic and electric detonators is the replacement of the pyrotechnic-delay element by a microchip. This allows for much greater accuracy in firing times. Unlike electric detonators, there can be a great deal of variation on how different manufacturers' systems function.

System Comparisons

Electric detonators and nonelectric shock-tube detonators are manufactured with several delay times. All major delay detonator manufacturers have a specified series of delay timings that they produce. The delay series can be generally separated

into the following two groups: millisecond (MS) series and long-period (LP) series. The MS series usually consists of 20 to 40 delay detonators having a separation of 25 ms between each two successive delay periods. Available are No. 0 delay (0 ms or instantaneous) through No. 20 (500-ms delay). The delay interval is approximately 25 ms but may vary by up to ±10 ms or more. With LP series detonators, most manufacturers have approximately 20 LP delays with varying times between delay numbers. Generally, the time between delay numbers for LP systems is between 200 and 400 ms but varies with the manufacturer. Current product information should be sought prior to making design decisions.

Programmable electronic detonators provide the most flexibility in delay design. Most systems allow for a large number of detonators to be programmed in a circuit to fire at any time between zero and 10 to 20 seconds. System capabilities vary, but most manufacturers claim accuracies to ±1 ms or ±0.1%, whichever is greater. This is a substantial improvement over conventional pyrotechnic detonators, both electric and nonelectric, which are usually only accurate to within ±5%. These accuracy figures are approximate and may not apply to all available systems; however, in general, electronics are at least an order of magnitude more accurate. The most obvious downside to electronic detonators is their higher cost per unit—although some systems have also displayed higher misfire rates and may be more complex to use than shock-tube alternatives. In some cases, better blasting results, and thus better overall processing costs, outweigh the initial high cost of the individual detonators. Many case studies have been performed to show their overall cost advantages, and their use is steadily growing. Electronic detonators are especially cost-effective in larger-diameter holes, where their added cost represents only a very small portion of the investment in drilling and explosives.

FRACTURING AND FRAGMENTATION

The obvious first question in considering blasting is, Why should one use explosives to break rock? There are two answers: ease of fragmentation (i.e., break the rock to make it easier to dig, load, haul, and crush) and ease of stripping (i.e., move overburden or waste rock overlying a seam by explosive means to reduce the mechanical work required to achieve the same result). Of these, fragmentation is the most important item for many mining operations. However, no two mining operations are identical and neither are their fragmentation requirements, where fragmentation refers to the size distribution of the broken rock within the muckpile.

As a result, a vast range of powder factors is used. The average powder factors for production mining in underground operations overall may be up to about twice that of surface operations, simply because underground workings are by definition more confined, and as a result, oversize material is quite difficult to deal with and displacement of rock into a loose muckpile may require more energy. Oversize material is usually avoided by using a higher powder factor (i.e., the explosives required to break a unit measurement of rock).

An additional item that is often misunderstood is the difference between fragmentation and fracturing. Well-fragmented rock has to be closely fractured and "jumbled up" by the associated movement of a blast, so that it can be easily dug. The movement is essential, and as this movement is in progress, bulking occurs as the chunks (or cubes) of rock are randomly reoriented so that void spaces are developed. The voids between the larger rock fragments make it easier for the teeth of buckets to penetrate and dig. The degree of rock mass bulking after a blast can be as much as 50%, depending on the shape and size distribution of the rock fragments. In stark contrast, soil when dug generally only bulks 10%.

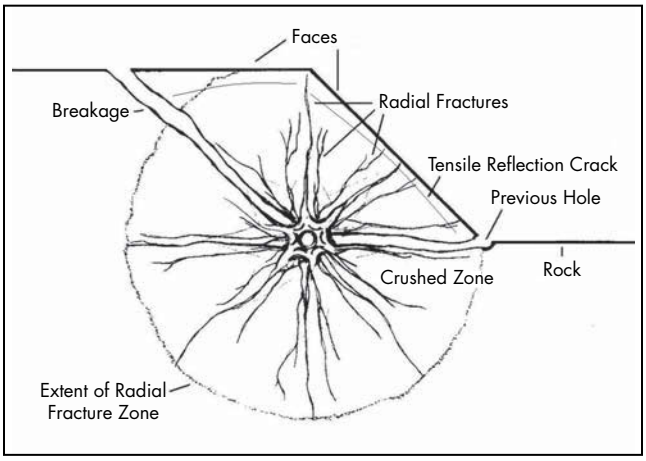


Figure 7.3-3 Processes of rock fracturing during fragmentation

The discontinuities in the rock mass, which includes bedding, jointing, and partings, are the main items that dictate how a rock fragments. The closeness of the separation of these determines the maximum block size in the muckpile. The effect of blasting is to reduce the size distribution of those preblast blocks. In closely jointed ground, there will be little change in the size distribution after blasting, though the rock mass may have been "moved."

The major elements of the fragmentation process include shock, gas production, extension of fractures, and rock mass movement. To start with, the shockwave from the explosive causes the creation of new fractures around the blasthole by the extension of microflaws orientated in a radial direction. Close to the hole there may be an extremely dense zone of small fractures (often referred to as the crushed zone). The wave travels outward, opening up and loosening the existing joint system, and when reflected from a free face, may cause minor tensile spalling. Hot on the heels of the shockwave is the pressurization of the blasthole by the explosion gases. This causes the small fractures around the hole to coalesce and grow due to the created tensile stress field. When these fractures connect with the blasthole, they are opened, and explosion gases are then able to penetrate. As these radial fractures extend, the compression bulbs that are created at their tips cause shorter fractures to slow, making the longer ones dominant. The width of the radial fracture zone caused by the shockwave is extended generally six times by the gas pressure. This is both forward and backward in blasting, preconditioning the rock for the next row of blastholes or the next blast. When the radial fractures reach a free face, major rock movement may occur, as shown in Figure 7.3-3.

The radial fractures may be replaced by a preexisting joint system. The explosion gases then act as a propulsion agent and accelerate the rock forward. This acceleration occurs over a brief period as the size of the initial blasthole is very small, and thus, as the void increases, the gas pressure drops very quickly. As the rock mass moves out it also expands, and this allows the rotation of fragments in the muckpile. Further fragmentation

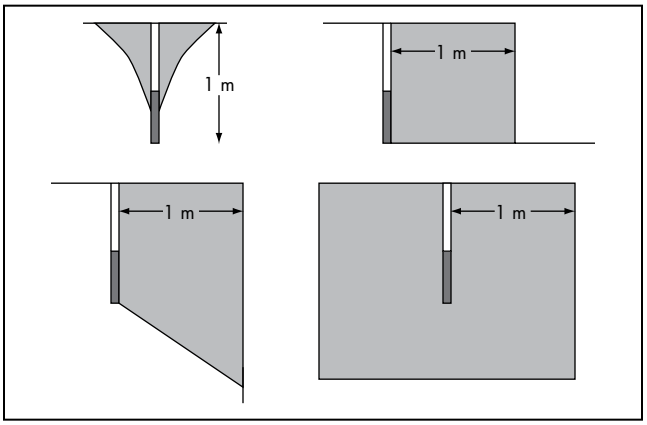


Figure 7.3-4 Effect of increasing degrees of freedom on volume fragmented using identical holes

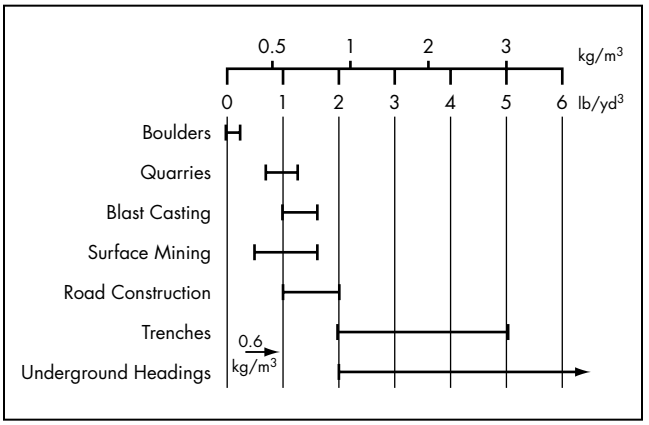


Figure 7.3-5 Typical range of powder factors for different types of blasting

occurs during this process because of the collision of fragments, the grinding of corners, grinding at the interface of the moving muckpile and the floor, and the impact of fragments as they come to rest. In low Young's modulus moderately to highly jointed rocks, significant compression may occur behind the bench during a confined blast, and when the pressure is released, rebound may occur, which can significantly loosen the back of the bench, causing overbreak and large cracks. The fragmentation process is very complex, especially when considering the complexity of geology.

ELEMENTS OF BLAST DESIGN

Importance of Free Faces

In blasting, a free face provides a direction for movement and, thus, a degree of freedom. As the number and extent of free faces located near the blasthole increases, the amount of rock that can be fragmented from that blasthole also increases, as shown in Figure 7.3-4. Because of such geometric considerations, as well as the properties of the rock and discontinuities, proper blast design requires more than simply adopting a reasonable powder factor.

It is simple to see that the easiest means of fragmentation in a repetitive fashion is bench blasting where the holes are parallel to the free face. In most instances, the fact that the face and hole are vertical also enables the easiest method of drilling. Because of this, in the most economical blast designs for both surface and underground, it is best to develop a bench-blasting configuration. The bench is not limited to a vertically down configuration; it can be at any angle (i.e., it is orientated to suit the mining method). Methods that incorporate burns or cratering are now normally only reserved for developing relief (called sinking blasts) for successive bench blasting. Although the term *bench blasting* is generally only used for surface applications, it exists in other forms, with appropriate titles, throughout underground blasting (such as long hole, breasting, and stoping holes) and is the basic production method for both surface and underground blasting.

The aim of good blast design is to produce the desired fragmentation with the minimum of back break and environmental effects such as flyrock, high air blast, and ground vibrations. Flyrock and elevated air-blast levels indicate inadequate confinement, whereas elevated ground vibrations suggest excess confinement. Excessive flyrock, air blast, and ground vibration all indicate inefficient use of explosive energy.

In mining, no single fragment size distribution satisfies every need. For example, in some operations, fine fragmentation is preferable because the rock is eventually going to be ground down to powder, as with copper ore or limestone (for cement). However, in iron ore mining and the quarrying of aggregates, fines (screenings) are usually a less valuable by-product. Also, in large mining operations, larger equipment such as crushers are used, so they can generally handle larger rock than smaller operations.

To get coarser rock, less explosive is used, which means expanding the drill pattern, whereas to get finer rock, more explosives are used, which means a tighter pattern. The amount of explosive used per unit of volume or weight of rock blasted is called the powder factor. Blast design equations and guidelines define the starting point that is designed to provide reasonable, rather than optimum, fragmentation. The equations, although based on different theories and empirical details, yield very similar powder factor results.

The range of powder factors for different types of mining and construction are given in Figure 7.3-5. The range of applications, and thus the range of appropriate powder factors, is large.

Surface Blast Design

The nomenclature used for bench blast design is shown in Figure 7.3-6. The following discussion presents some guidelines for blast design. The guidelines provide initial estimates from which to design a blasting pattern. However, the properties of the rock mass and of the explosive should also be considered. There are guidelines that have been established over the years for blast designs. In the following subsections, guidelines are presented that the authors believe represent common practice.

Hole Diameter and Burden Versus Face Height

This is an important item that is often overlooked. The face-height-to-burden ratio should be at least 3:1 to 4:1 for good fragmentation. Short faces are stiff and resist breakage. Also, the amount of stemming required to seal the hole increases as the face shortens, leaving less room for explosive in the hole, increasing the proportion of the bench interval remote from adjacent explosive columns and the hole utilization and powder factors drop, as illustrated in Figure 7.3-7.

As the face height decreases, because the stemming height is proportional to the hole diameter and burden, the

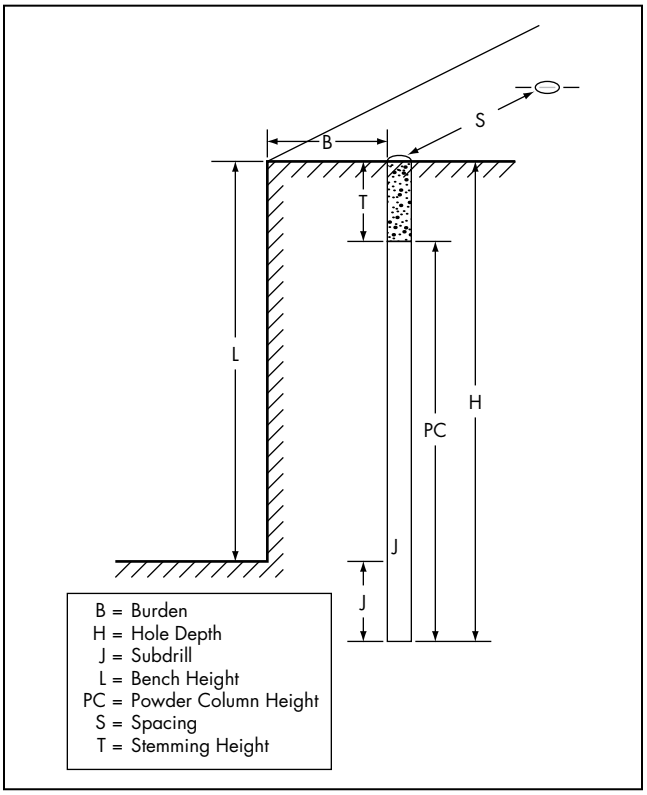


Figure 7.3-6 Nomenclature for bench blast design

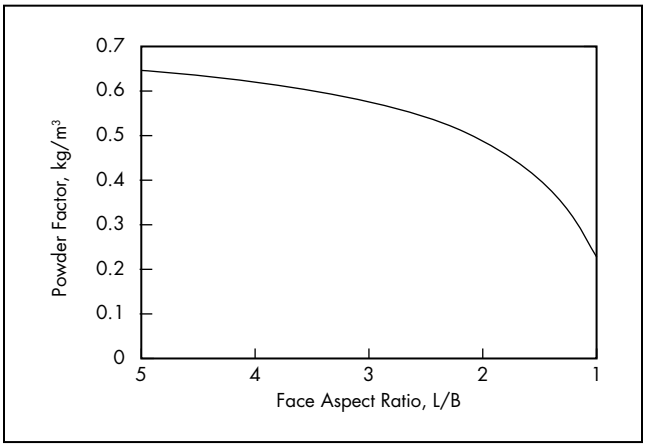


Figure 7.3-7 Effect of face height on powder factor using a constant burden and spacing

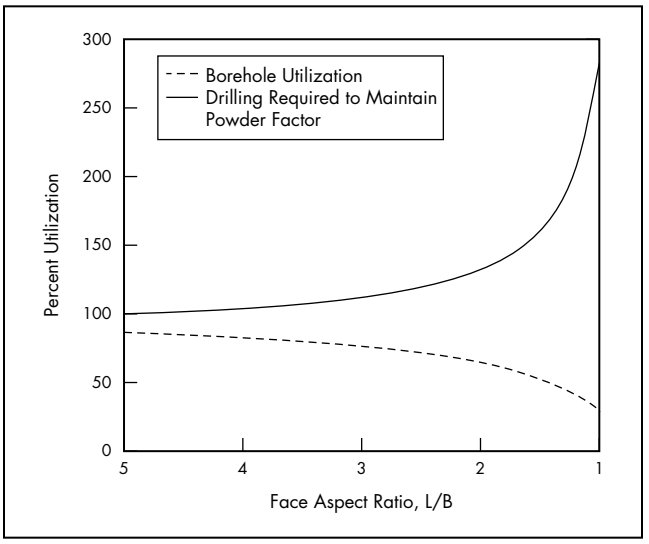


Figure 7.3-8 Effect of face aspect ratios on blasthole utilization and how much extra drilling is needed to maintain powder factor compared to when face height is used

powder column height decreases disproportionally, resulting in reduced borehole utilization (explosive-filled portion). To maintain powder factor, the pattern has to be shrunk, and this requires the drilling of more holes, increasing drilling costs. These concepts are illustrated by Figure 7.3-8.

Hole utilization curves for shots designed using the guidelines are given in Figure 7.3-9, where the upper curve represents holes with subdrilling (metal mining) and the lower curve is holes without subdrilling (limestone quarrying). In many metal mines, ore-body configurations dictate smaller bench heights for ore selection. Strangely, the same inefficient bench heights are often used for large areas and volumes of waste removal where there is no ore, and this does not make sense.

In contrast, when the face height is excessive, drill deviation occurs, which means a variation in the distribution of explosive energy at the bottom of the blastholes. In areas where holes wander excessively away from each other, inadequate breakage will occur, which often leads to pockets of high (or hard) toe. When blastholes come too close to the face, excessive flyrock and air blast can occur (Figure 7.3-10).

A good range for face height is 100 to 120 times the hole diameter. Going to extremes such as 50 or 200 times will most certainly result in problems. The excavation equipment should not dictate the face height unless regulations control the face height.

Burden and Spacing

For bench blasting, most equations produce similar results to the guidelines in Table 7.3-3 for rock in the specific gravity range of 2.2 to 2.8. There are basically two specific gravity divisions for modern bulk explosives. These are 0.8 to 0.87 for ANFO and 0.65 to 1.3 for emulsions and blends. As illustrated in Table 7.3-3, two sets of burden and spacing rules are used that result in a powder factor of approximately 0.6 kg/m^3. These are for rocks of "average strength" and where the required level of fragmentation is medium to coarse.

In bench blasting it is accepted practice that a burden-to-spacing ratio of about 1.15 to 1.4 for bench heights greater than four times the burden should be used on a staggered pattern. The reason for this is that repeatable faces are developed with equal burden on both faces of relief (see Figure 7.3-11).

Subdrilling

Subdrilling is necessary when blasting massive rock where there is no suitable horizontal bedding plane to maintain floor grade. In rocks such as granite, subdrilling is essential to maintain grade, otherwise the floor will rise. The accepted amount of subdrilling is one-third of the burden. Some operations may range from 0.2 to 0.5 times the burden (Morhard et al. 1987), but one-third is a good guideline. Subdrilling can

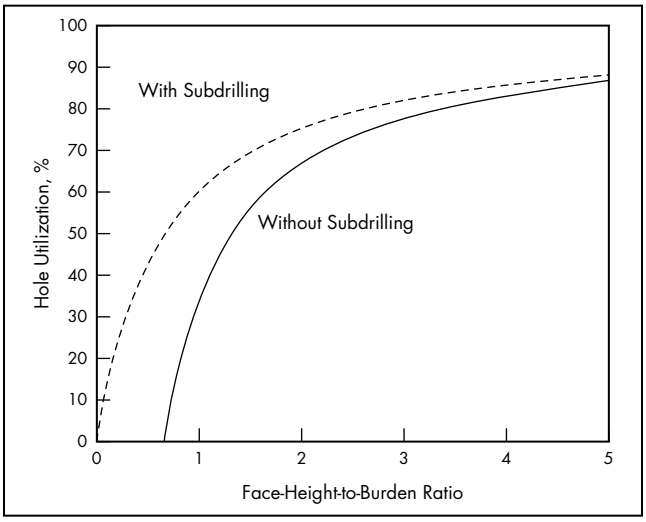


Figure 7.3-9 Hole utilization curves with and without subdrilling

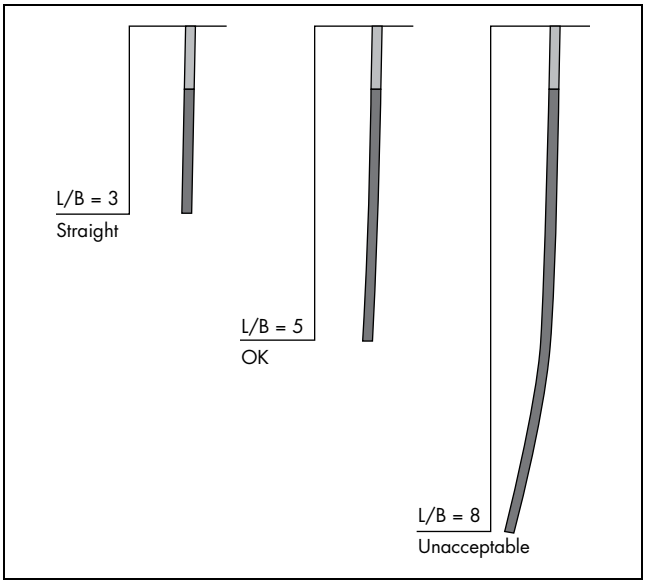


Figure 7.3-10 Effect of drill deviation with increasing face height (deviation is unacceptable with excessive bench height)

Table 7.3-3 Guidelines for burden and spacing as a multiple of hole diameter in bench blasting

	ANFO		Blends and Emulsions	
	Metric	U.S.	Metric	U.S.
Burden	× 25	× 24 (2 ft/in.)	× 30	× 30 (2.5 ft/in.)
Spacing	× 35	× 36 (3 ft/in.)	× 45	× 42 (3 ft/in.)

also be calculated as 5 to 8 times the diameter of the hole. One should beware that excessive subdrilling results in the smashing of the bench below, which makes drilling difficult and generally requires extra stemming to hold this later blast in place. In addition, the explosive energy from excessive subdrilling has nowhere to go, and not only is this energy wasted, but extra ground vibrations are created. The negative effects of subdrilling are best controlled by offsetting holes from one

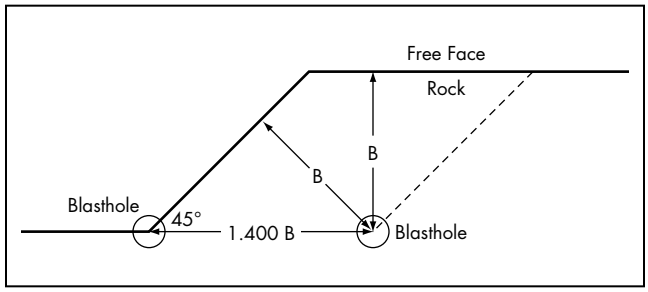


Figure 7.3-11 Plan view of the geometry for a burden-to-spacing ratio of 1.4 (S = 1.4B) showing breakout angle and the burdens for each face being equal

bench by half the spacing and half the burden when laying out the next bench. This hole offset also reduces the risk of drilling into a misfired "butt."

Subdrilling hould not be used in horizontally bedded sediments that have well-defined bedding planes, as these can be used as shear planes at the bottom of the blast to provide easy relief and a smooth floor.

In coal mining when blasting overburden, it is appropriate to use negative subdrilling, where the drill hole stops above the coal to prevent coal loss and excessive fines.

Stemming

Stemming is necessary to confine the explosives column to prevent explosive energy from escaping and reducing the effectiveness of the blast. If the stemming is ejected prematurely, not only is costly energy lost, but also a sonic boom is often generated along with flyrock. Unfortunately, stemming reduces the amount of explosives that can be placed in the hole, thereby, reducing the utilization factor of the hole. For example, with large-diameter blastholes and short benches, hole utilization can be 60% or less, whereas with benches with normal height-to-hole-diameter ratios, the hole utilization is often 85% or more. To minimize the length of a blasthole containing stemming, stemming materials should be carefully chosen for their size and gradation. The optimum stemming material has a diameter of about one-eighth the hole diameter for small- to medium-diameter holes. This is to maximize the potential for interlocking and also to avoid bridging when pouring the material in the hole. For example, a 150-mm (6-in.) hole would call for 19-mm (¾-in.) stemming. Stemming material locks best when it is "clean" (i.e., free of fines) rather than well graded. The guidelines for stemming are

$T = \frac{2}{3}$ B for clean stemming material ⅛ of the hole diameter
$T = \frac{4}{3}$ B for drill cuttings

Based on the above-mentioned relationships, drill cuttings are only half as effective as clean gravel when used as stemming. It is especially important to keep stemming to a minimum for large-diameter holes, as large lengths of face at the top of the blast may otherwise have no explosives to fragment them, and serious oversize (sometimes the size of small houses) may occur.

Powder Column Length

The powder column length is simply the amount of hole left over after subtracting the stemming. The hole utilization is the percentage of the hole used for powder. Normal hole utilization values of 80% or more can be expected.

Decking

Decking is the separation of the explosives column in a blasthole into two or more parts with stemming between them. The thickness of the deck material should be 6 times the hole diameter for dry holes and 12 times the hole diameter for wet holes. Decks are used for the following reasons:

- The first reason is to fill voids so that excessive explosive is not used. Even medium-sized voids can result in the excessive concentration of explosive energy. It is normal to measure the rise of decking in the hole until a normal rise is established and then to reprime and continue loading. In the case of voids, the same in-hole delay is used for each deck. Also a mud seam or other weak spot in the rock column has to be decked through to avoid flyrock.
- The second reason is to reduce the kilograms per delay when blasting close to residential structures to ensure that regulated limits are not exceeded. In this case, the stemming deck is used to divide the explosive column into two smaller decks. A delay (typically 25 ms) is used between decks so that two small "thumps" are provided rather than one large one.
- The third reason is to reduce the amount of explosive in the hole. In this case, air decks are used where an air void is placed either above or below the explosive. This is a way of decreasing the explosive load without the use of excessive stemming, which tends to lead to blocky ground on top of the muckpile and oversize.

Powder Factor

The powder factor is usually determined by taking the amount of rock to be fragmented to grade and dividing it by the weight of explosives used. It can be calculated for each hole or for a complete shot/round. In some mining, the powder factor is often quoted as weight of explosive per unit weight of rock (kilograms per metric ton); however, in certain quarry markets in the United States, the powder factor is quoted in tons per pound, but in construction, the weight of explosive per unit volume of rock is used (kilograms per cubic meter). The difference between construction blasting and mining is that in construction the excavated volume is the quantity of importance, whereas in mining the excavated tonnage of ore, and so forth, is of concern. The volumetric powder factor is also used in mining in connection with contract excavation overburden removal and waste stripping. For normal surface-mining operations, such as quarrying, a powder factor of 0.6 kg/m^3 is a good initial estimate. The weight of explosive, W, per hole is calculated as follows:

$$W = PC \times C_\rho$$

where

PC = powder column height
C_ρ = column density

Column density, C_ρ, is dependent on explosive diameter and explosive density and can be found using a manufacturer's chart or using one the following equations:

$$C_\rho = (D_e^2 \times \rho)/1{,}275 \text{ kg/m}$$

where

D_e = diameter of the explosive
ρ = density of the explosive

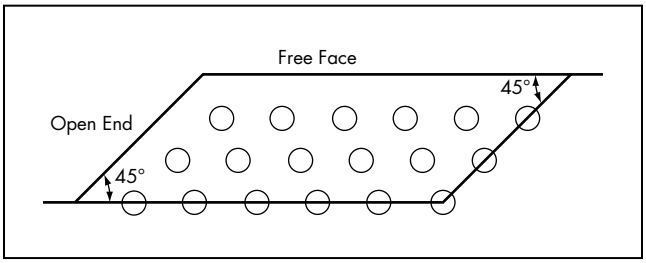


Figure 7.3-12 Staggered pattern (three rows with six holes per row)

Both equations are approximate because the diameter of the drill hole is often larger than the diameter of the drill bit. Also, drill bits wear considerably over their lifetime, and it is not usually possible to measure the diameter of the hole at the depth of the explosive column.

Choice of Patterns

A number of different patterns can be chosen, including square, staggered, en echelon, and diamond. It is normal to use a staggered pattern in bench blasting, firstly, because the hole in the row behind is blasting into more solid rock rather than a weakened pocket and, secondly, because holes tend to break at 45° to the free face, making a square end to a bench almost impossible (Figure 7.3-12). In some cases staggered patterns are avoided where angled holes are utilized. The staggered pattern creates logistical problems with drilling where the drill operator is forced to realign the boom angle rather than backing up perpendicular to the face and drilling subsequent rows with the same angle.

A square pattern is more appropriate for a confined shot, such as sumps used for dewatering the next bench level. A square pattern may also be appropriate in sinking cuts when developing a new level. In construction blasting, a square or rectangular pattern helps maintain straight sidewalls, especially in road cuts.

Number of Rows

The number of rows depends on what one wants to achieve. For small production shots, a single row may be appropriate if it is easy to maintain good face profiles. If it is difficult to maintain good face profiles, then multiple-row shots mean fewer drilling and loading issues. In quarry blasting, it is normal to shoot three rows or more, with only special loading consideration necessary for the front row. Multiple-row shots also afford less exposure of workers to highwall falls and failures. If employing more than three rows, the blast starts "piling up on itself," resulting in a high muckpile unless the timing pattern compensates for this. Otherwise to compensate for this, a higher powder factor is required. Excessive rows without the compensation of an increased powder factor often result in excessive back break, especially at the crest of the bench, and a muckpile that may eventually be higher than the original face height. Taking the number of rows to the extreme, one has a trench shot, which requires a powder factor of at least 1.2 kg/m^3 for successful fragmentation and reasonable digability.

Delays

There are two main types of delay in a blast pattern. These are the hole-to-hole (also known as "inter-hole") delay and the row-to-row (or "inter-row") delay. The optimum hole-to-hole

delay for fragmentation has been determined by a number of different researchers, including the U.S. Bureau of Mines (Stagg and Nutting 1987). The general consensus is that the delay is 3 ms/m of burden. However, maximum fragmentation is not always desired. In some cases, maximum throw is of primary importance, such as in blast casting (e.g., the explosive casting of overburden from above a coal seam).

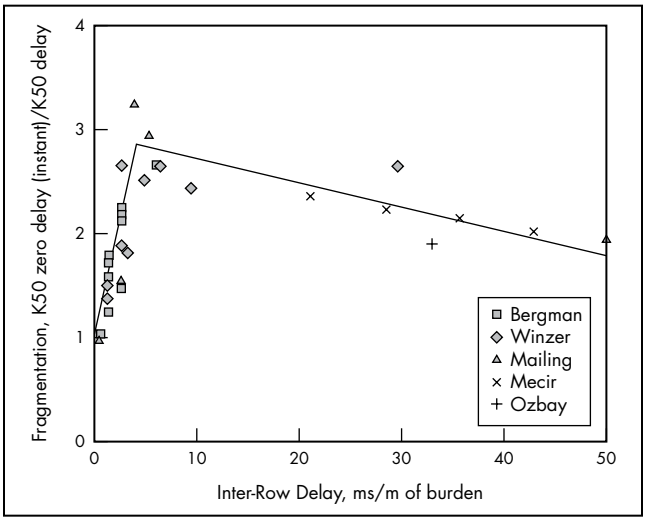
The effect of hole-to-hole delay on fragmentation is shown in Figure 7.3-13, where fragmentation is expressed as the ratio of the median fragment size (K50) for zero delay to that for increased delay between holes. The data were combined from five different sources. As shown in Figure 7.3-13, fragmentation improves up to a delay of approximately 3 ms/m of burden and then gradually worsens.

When using delays, the accuracy of the delay time is also important. As shown in Figure 7.3-14, delay timing with conventional detonators is subject to scatter, and accurate detonation delay times are most likely to be achieved when using electronic detonators. Because of the potential for inaccuracy in detonator timing, it is recommended to use a delay interval within a row larger than the 3 ms/m optimum. Large delay intervals between holes result in reduced fragmentation.

Insufficient delay between holes is also not recommended and generally results in excessive throw (Figure 7.3-15), increased back break/wall damage, and inadequate fragmentation. The firing of neighboring holes together results in splitting of the rock between them and the premature propulsion of the rock mass forward, resulting in poor fragmentation. In blasting, it is normal to use a spacing-to-burden (S/B) ratio of 1.15:1 to 1.4:1, which means that delays are a necessity for fragmentation but a penalty in blast casting where burden and spacing are often reversed to achieve less fragmentation but more throw. The splitting effect is shown in Figure 7.3-16.

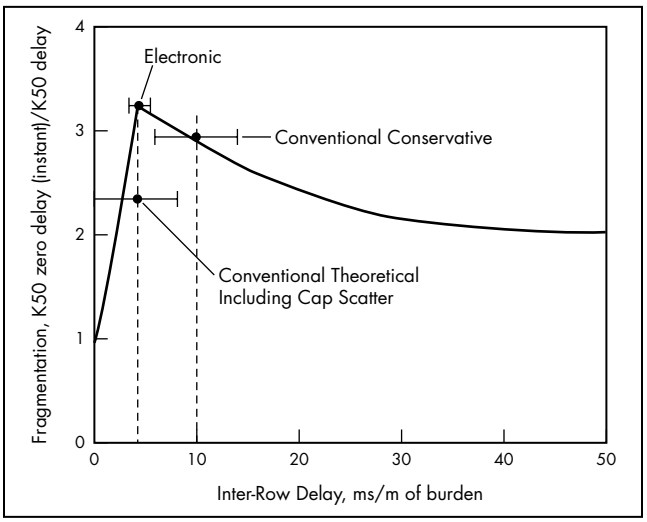
The dramatic loss in fragmentation before the "sweet spot" means surplus energy, which is consumed in throw, as shown in Figure 7.3-15.

The row-to-row delay to provide good movement and fragmentation is a minimum of 3 ms/m of burden. As the number of rows increase to more than three, this value should be increased to provide good movement and to reduce back break. Often significant "back splatter" (rock thrown backward on the bench) is a sign of insufficient delay between rows or excessive burden distance.



Source: Grant 1990.

Figure 7.3-13 Effect of hole-to-hole delay on fragmentation



Source: Adapted from Grant 1990.

Figure 7.3-14 Effect of timing inaccuracies on fragmentation, highlighting the need for increased delays using conventional initiation systems

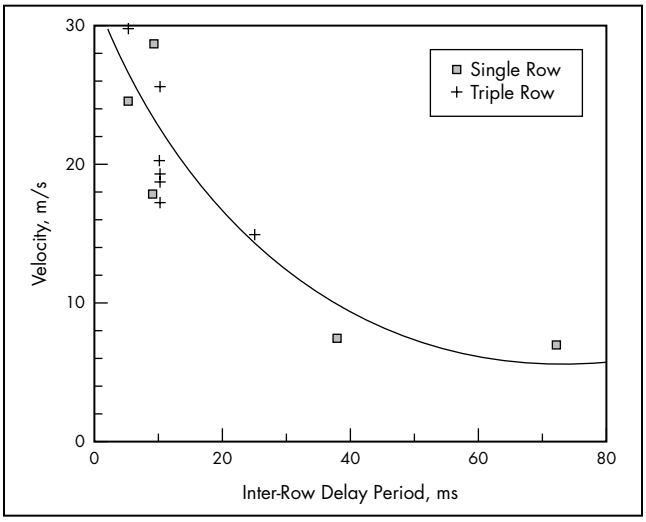
Application of Basics of Blast Design to Underground Blasting

Underground blasting uses the same principles as surface bench blasting, except that with underground blasting there is less space, and thus oversize cannot be so easily handled. Therefore, with underground blasting it is common to use double the powder factor to ensure fine breakage. Modern underground blasting can be separated into four types: headings, long-hole stopes, mass blasts, and pillar recovery.

Headings

Heading blasting is used for tunnels, development drifts, adits, and room-and-pillar (R&P) mining. Heading blasting has no free face parallel to the blastholes; therefore, it must create one. This is done by separating the blast into two parts: the cut and the main blast. The cut is designed to provide a free face perpendicular to the face to provide relief for the rest of the blast (bench blasting using horizontal holes). These first holes are collectively called the "cut," of which there are the following three principle types used in modern underground blasting: the burn cut, the V-cut, and the fan cut (often called slashing). These three types of cuts are illustrated in Figures 7.3-17 through 7.3-19, respectively.

In the burn cut (Figure 7.3-17), uncharged open holes that are often larger in diameter than the regular blastholes are used to provide relief for the loaded holes. For proper fragmentation, rock must be allowed to swell up to 50% in volume. For this to occur, the center-to-center distance between the loaded and relief holes must be of the order of 1.5 times the diameter of the relief hole. When the broken rock has swelled, it is propelled from the resulting enlarged hole by the blast gases, creating a free face. As successive holes fire, the "free face" is enlarged and greater standoffs can be employed using the same design criteria. This continues until the desired burden is reached. Because the same sized burn cut is used for



Source: Grant 1990.

Figure 7.3-15 Effect of hole-to-hole delay on face velocity

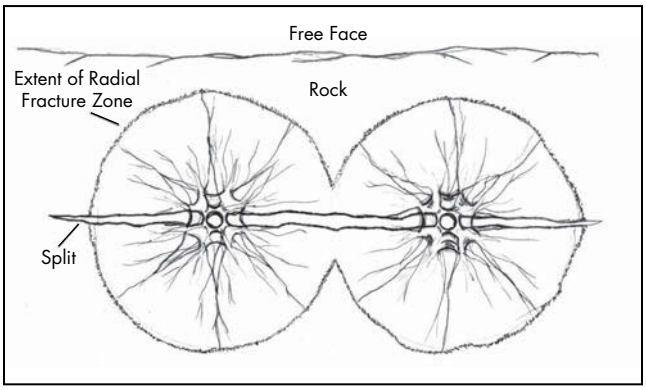


Figure 7.3-16 Effect of simultaneously fired holes splitting at S/B ratios less than 2:1 (fracture zones coalesce and provide a split before fracturing reaches the free face)

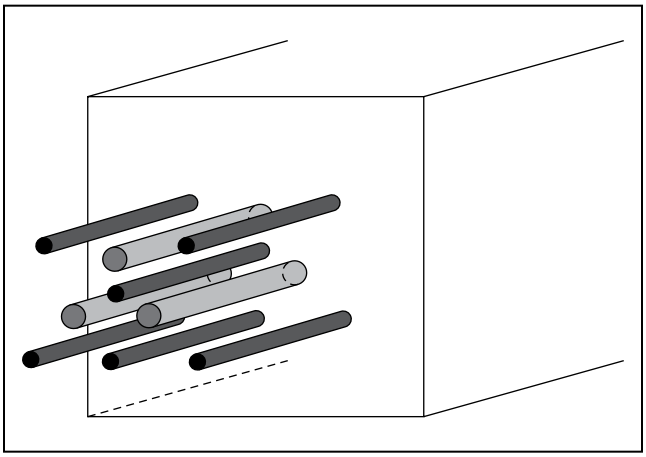


Figure 7.3-17 Example of a burn cut (rock's eye view). Light-gray (larger) holes are empty and dark-gray (smaller) holes are charged.

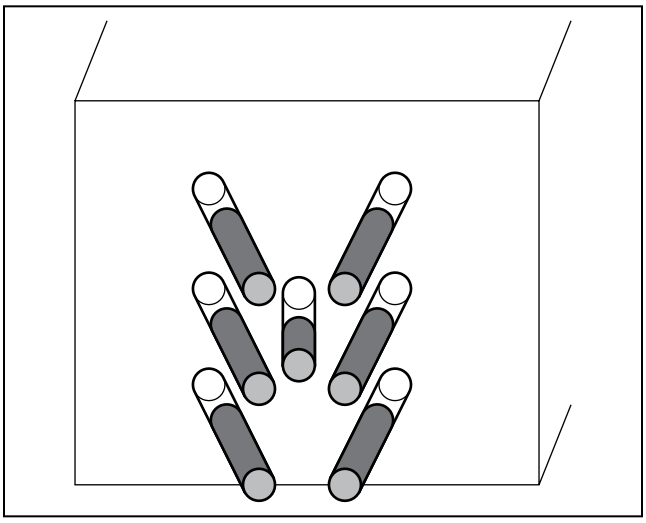


Figure 7.3-18 Example of a V-cut with a center "buster hole" (rock's eye view). This "V" is designed for modern bulk explosive, and the backs of the holes are purposely separated to avoid dead pressing.

a small-diameter tunnel or a large R&P heading, the powder factor drops from around 6 kg/m^3 for a very small heading to 1.2 kg/m^3 for a large heading. Because burn rounds use parallel holes drilled straight into the face, they can be used on any width face.

V-cuts (Figure 7.3-18) use angled holes and can only be used in headings whose width is four or more times the depth of pull. Traditional V-cuts do not work well with modern explosives because the bottoms of V-holes come close together. In such cases, dynamites tend to sympathetically detonate, and emulsions and ANFO can easily dead press. A blasting agent is dead pressed when it is compressed/crushed enough to collapse a sufficient quantity of voids, thus losing sensitivity and resulting in a failure to fire. To overcome this, the bottoms of the V-holes are usually spaced 0.3 to 0.4 m apart. When using V-cuts, it is common to find a huge wedge of rock well in front of the muckpile when reentering the heading after blasting. To avoid this, "mini" V-cuts have been used, but these are very wasteful and they can be replaced by less-expensive buster holes, which may be fired before or after the V.

Fan cuts (Figure 7.3-19) have for some reason lost popularity. This is probably because their advance rate is less or because they are more complicated to drill. Because burn cuts end up with considerable fines, they are normally used in metal mining, whereas in underground quarries it is normal for V-cuts to be used because fines are generally a waste in crushed-stone operations. One measure of a blast's effectiveness is the length of the hole sockets left after the blast, signifying a reduction in maximum pull. In most cases it is not possible to totally eliminate sockets because this is the same phenomenon known as subdrill in surface bench blasting, and a maximum socket length of one-third of the burden can therefore be expected.

Long-Hole Stope Blasts

Long-hole stope blasting is used in narrow to wide vertical and subvertical ore bodies. It has replaced most of the short-hole techniques, and it has been made possible by the development in the 1970s of equipment that can drill long, straight holes. It is used to achieve high productivity but without the

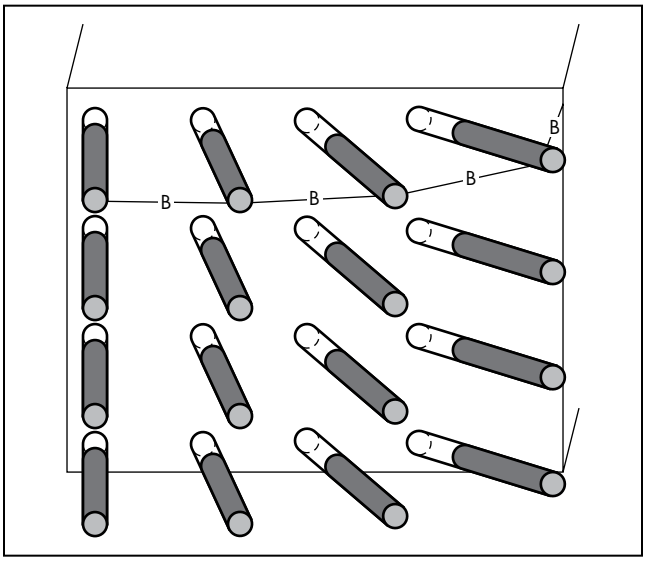
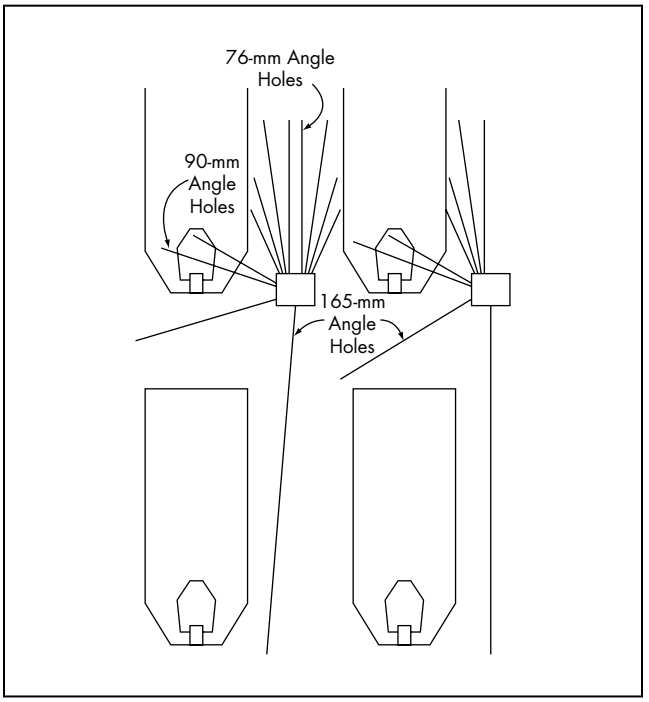


Figure 7.3-19 Example of a fan cut/round showing swing of holes (rock's eye view). The bottoms of the holes are at a constant burden between rows.



Source: Irvine 1982.

Figure 7.3-20 Mass blast design illustrating development drifts in pillars, partial drill patterns, and the use of abandoned stopes for swell

associated development costs and is basically a form of underground bench blasting. However, it relies on having a free face to blast into, which is usually provided by a raise that has been previously drilled between levels. These raises, in turn, rely on long-hole drilling, usually from the upper level and the use of multiple relief holes or vertical crater retreat (VCR). The technique of VCR started in the mid-1970s (Lang 1977). The use of large-diameter holes and down-the-hole hammers now ensures long-hole accuracy.

Mass Blasts

Mass blasts are large blasts that might take months to drill and weeks to load and that provide production for as much as a year. They require volume for expansion, and development drifts do not usually provide this. They are often used in older sublevel stoping operations in massive ore bodies, where large openings already exist and a high percentage of valuable ore deposit remains in horizontal and vertical pillars between the stopes. Blasting is often accomplished through the drilling and placement of explosives in rings of long holes located in development drifts excavated through the center of pillars, as shown in Figure 7.3-20.

An important factor in design is the adequate distribution of explosives. In a typical ring, the holes are designed to be at the appropriate spacing at their furthest extent. However, in the development tunnel the holes are rather close together. Consequently, many holes are often not loaded along their full length, as shown in Figure 7.3-21.

In addition, there are usually a limited number of delays available when using traditional initiation systems. Because of this, whole rings are usually shot on one delay. To limit splitting between holes, resulting in decreased fragmentation, the far ends of the holes are usually designed such that the spacing is at least twice the burden while the required powder factor is still maintained.

In mass blasts the production of air blast is also massive. As a result, fully activated sequential timing is required, otherwise detonating cord can be ripped apart and large-scale misfires may occur.

In the past several years, there have been record-sized mass blasts made possible through the use of electronic detonator systems that overcome many of the disadvantages outlined previously.

Pillar Recovery

For many mining techniques, a significant proportion of the resources are left at the end of the mine's life, especially in areas of high grade. In R&P mining, it is often possible to recover pillars, although the resulting large spans of unsupported roof may be too dangerous to allow. Often, cemented fill may be used to provide artificial pillars or cable bolting used to provide roof support. As pillars are taken out, stress redistributions occur, making neighboring pillars potentially unstable for drilling. As such, pillar scaling is required, and pillars should be drilled at least two pillars away from any pillar-extracted area. Pillars are best drilled from the side furthest from extraction (i.e., from the side of supported ground and least stress). Horizontal holes fired in rows parallel to the vertical sides of the pillar provide the best fragmentation. The initiation sequence is most efficient from one side to the other. Remote-controlled loading is the preferred method of removing the broken reserves.

In mass blasting and pillar removal, holes may be required to be loaded for some considerable time before they are blasted. It is, therefore, important that the explosive selected will both perform after being loaded for the required time and be compatible with the ore (some sulfide bodies are reactive, requiring special explosive).

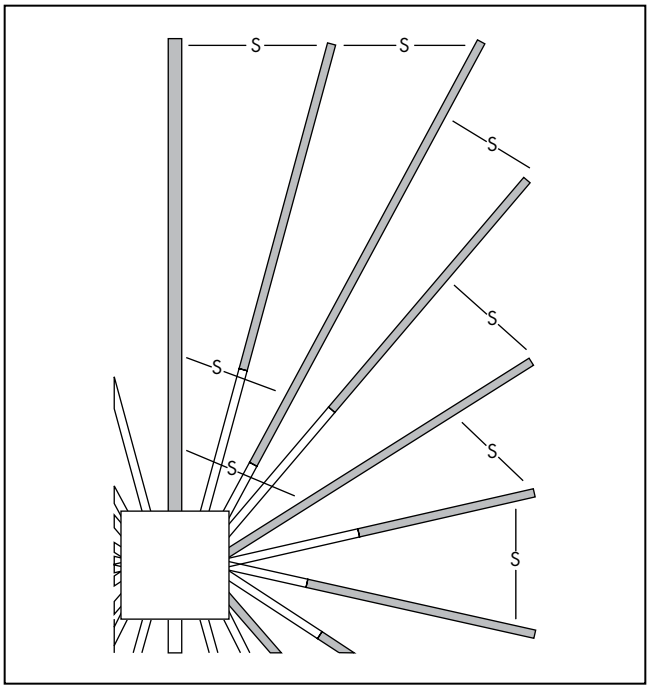
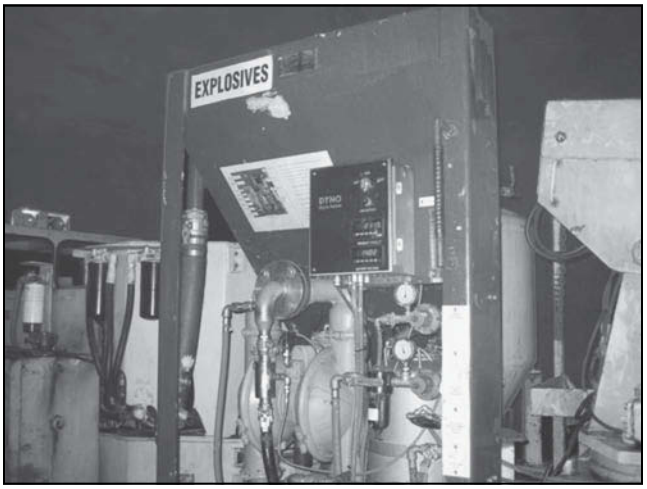


Figure 7.3-21 Loading of holes in a fan to provide a more uniform powder distribution. The maximum spacing of charges is not exceeded throughout the ring.



Courtesy of Clay McNail, The Doe Run Company.

Figure 7.3-22 A modern repump emulsion system (Dyno Miner underground explosive loading system from Dyno Nobel) set on a Getman charging rig

Modern delivery systems underground comprise pumped emulsion systems and blown ANFO. High-shear pumps may be used to ensure that emulsions stay in vertical holes. Repump emulsion systems (as shown in Figure 7.3-22) allow for the rapid loading of high-energy bulk explosives, make optimum use of expensive drilling, and result in a significant reduction in the number of holes required. This type of system is now widely used in underground mines.

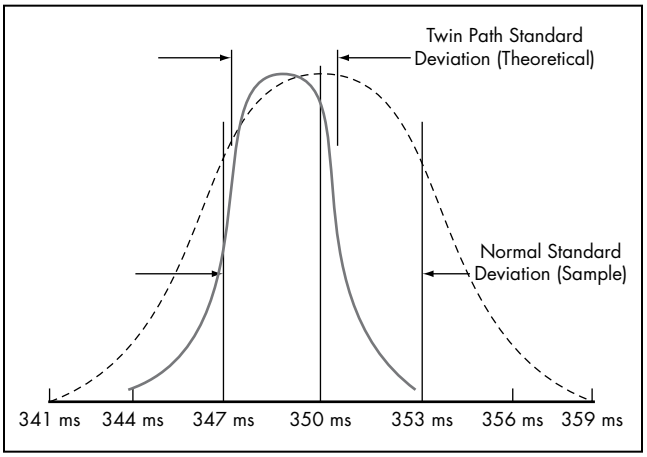
COMMON MISCONCEPTIONS

There are four common misconceptions about blasting. The first misconception is "high-gas" versus "high-shock" explosives. This concept gives the impression that some explosives are all gas and some are all shock. In all commercial explosives, the fraction of energy liberated in the form of shock is low compared to the gas energy. Thus, high-gas explosives should be called "higher-than-average-gas" explosives. In addition, blastholes have a fixed volume, so the bulk (volume) strength of an explosive can be used rather than its weight strength. For example, even though emulsion contains 15% to 22% water, which is inert, a hole filled with emulsion will have a higher gas pressure than one filled with ANFO because of the significantly higher density of the emulsion.

The second misconception is that one should not put a detonator in a hole. The assumption is that a cutoff could result, leaving an uninitiated detonator in a primer assembly in the hole, which could then be set off during excavation. Such occurrences were common when blasting caps were initialed by a slow-burning safety fuse, but these occurrences are rare when blasting with short-period electric detonators or electronic detonators. With such initiation systems, the problem of cutoffs is caused by not using a fully activated circuit (where all of the detonators are activated before the first hole detonates). Blasting circuits that are not fully activated are usually caused by a combination of bad blast design and overly large blasts.

The third misconception is that all holes have to be double-primed and that the top primer needs to be on a higher delay (Figure 7.3-23). Holes are usually double-primed for the following reasons: the hole is unusually long, the downline may be damaged during the stemming operation, rock slippage is expected along bedding planes and along low-angle discontinuities, and there is a large investment in each hole (the extra detonator and cartridge explosive represent a very marginal investment compared to the cost of a loaded hole). The mining industry has traditionally assumed that a blasthole has to be initiated from the bottom—it does not. However, if a rock-shift cutoff occurs, the toe will be hard to excavate. Therefore, for single priming, initiation at the bottom of the hole is the common practice. In top and bottom initiation, the top detonator is rarely put on the very top of the column but rather at a distance from the top. Putting the backup primer on a delay more than 25 ms longer than the bottom primer increases the chance of holes firing out of order with all the associated problems.

The fourth misconception is that in large-scale mining with a shovel, the face height is limited by the length of the stick on the shovel. This is a very simple and easy mistake—one digs the muckpile resulting from the blast and not the face. Thus, it is the height of the muckpile that should not exceed the stick height. The height of the muckpile is not necessarily the height of the face. For instance, in quarry blasting, the height of the muckpile is often only a third of the face height for higher benches with a limited number of rows. Mine operators often argue that when loading with shovels, the blast pile needs to be concentrated rather than spread out, as shovels are not as mobile as front-end loaders. However, with proper blast design, face heights can be doubled, even tripled while still maintaining the same muckpile height. In large-scale operations, the trend over the years has been toward drilling larger-diameter holes, because they are more economical; however, face heights have not increased accordingly, resulting in reduced face-height-to-burden ratios and stiffer faces, resulting in poorer fragmentation. In addition, because the



Courtesy of Ensign Bickford.

Figure 7.3-23 Effect of dual capping on standard deviations in nonelectric detonator delays

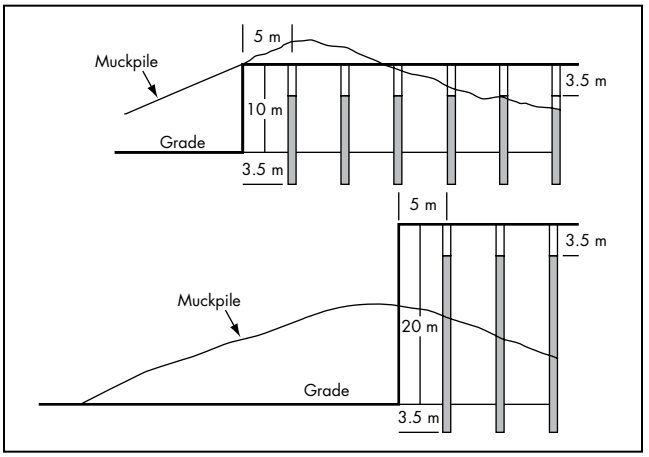


Figure 7.3-24 Example of increased efficiency from doubling face height and halving the numbers of rows

face heights have not increased proportionally to drill diameter, hole utilization has decreased by as much as 30%. This means significantly higher drilling footage to get the job done.

In the example shown in Figure 7.3-24, increasing the face-height-to-burden ratio from 2 to 4 results in a slightly lower muckpile, drilling is reduced by 23% (13% due to reduced hole footage and 10% due to an expanded blasthole spacing to maintain the powder factor), and the proportion of powder above grade is increased from 65% to 82.5%.

Mining engineers will often accidentally end up with "the tail wagging the dog" if they are not careful. The engineer cannot let one factor govern design, as economy and production will subsequently suffer.

THE COST OF BLASTING OPERATIONS

Unfortunately, many operators consider that the optimum cost of blasting is the lowest one that they can get away with. However, blasting is not an independent or standalone operation; it affects many other downstream operations, such as digging and haulage, as well as, more importantly, crushing/milling and handling oversize. Crushing and milling can be a bottleneck for many operations, and typically these parts of the operation cost much more per ton of rock than drill-and-blast. Reducing blasting costs can easily result in reduced fragmentation, which makes crushing and milling slower. Conversely, increasing the investment in blasting by increasing the powder factor generally results in an increase in fragmentation. This will normally result in an increase in throughput. Obviously, this is a complex subject, and because many mines and quarries are configured differently, with different needs and objectives, site-specific analysis is necessary. The lowest blasting cost may not (and usually does not) equate with the lowest overall mining cost. More and more companies are becoming interested in mine-to-mill cost studies, where it is necessary to track material blasted using different designs and account for costs in all stages.

Table 7.3-4 shows the costs that should be considered when analyzing a blasting operation. The table provides a starting point for gathering cost information on blasting costs and can be used to assign an estimated cost to each cell in the table. For example, explosive cost can be estimated on a per-hole basis by using the design equations given previously. After the weight of explosive per hole is established, explosive cost is calculated using the price per kilogram obtained from a manufacturer. Engineers should seek quotations from manufacturers when estimating explosives costs, which can vary greatly depending on quantity purchased, shipping distances, and proximity to manufacturers. When a critical mass of data is collected, trends should become evident and optimization analysis can begin.

Direct costs are almost directly proportional to powder factor. As in blasting, one deals with loaded holes, and the only realistic way to increase powder factor without changing the explosives is to use more loaded holes. In many quarrying operations, fines represent a waste material that not only results in lost revenue but also results in costly waste-handling costs. The opposite applies in some types of metal mining operations where fines are readily used. In both extremes, the production of fines has to be balanced with that of oversize.

After basic and complete information is gathered concerning a blasting operation, analysis can be undertaken to allow informed decisions to be made on issues such as product selection. Explosive selection is an important task requiring thorough consideration.

An old axiom predominates: "if drilling costs are low, then use ANFO, but if drilling costs are high, then use emulsion for better hole utilization with the denser emulsion." However, with the introduction of blends, replacing emulsions is a better cost alternative in dry holes of 100 mm (4 in.) in diameter or more. In many situations, the only reason not to use blends is their limited water resistance or minimum hole diameter requirements (see the manufacturers' technical data sheets). The costs of operational decisions regarding blasting should be thoroughly evaluated. For example, emulsions may provide a higher-density product where less drilling is required. However, in most rock types, higher powder factor will be required to compensate for the water in emulsions and the poorer energy distribution inherent to these expanded patterns. A simple analysis can determine the breakeven point where a certain percentage of wet holes can be most cost-effectively loaded using ANFO, an emulsion, or a blend.

The importance of blasting economics is best illustrated using examples, even though costs vary with time and location. The following examples use costs typical for the

Table 7.3-4 Summary of actual blasting costs

Drilling Costs	Loading Costs	Explosives Costs	Indirect Costs
Amortization or lease of drill rig	Dewatering	Explosives	Clearing the blast area—loss of production
Fuel	Accessories	Boosters	Stabilization and cleanup
Operator	Stemming materials	Initiation system	Compliance, litigation, and insurance
Lubrication	Transportation	Product enhancers (stemming plugs, additives, etc.)	Explosives storage
Maintenance	Loading equipment		Parts inventory
Drill steels	Maintenance		Drilling and fragmentation assessment
Drill bits	Labor		Permits
			Documentation

Midwest United States in 2006. The costs were provided by a drill-and-blast contractor. In the examples, 100-mm holes are drilled on a 12-m-high bench in limestone. The costs of bulk explosives delivered and loaded in the hole are $0.68/kg for ANFO and $1.06/kg for gassed emulsion. Costs of $4.50 per hole for the initiators and $3.00 per hole for cast boosters are also budgeted. The cost of contract drilling is $6.23/m. Comparative costs are estimated for the tonnage that would be produced from a 100-hole blast using ANFO. The rock tonnage and powder factor are held constant for both blasts. The number of holes in the emulsion and blend shots are reduced accordingly to achieve this.

The cost of drilling affects the economics of a dry pattern, as shown in Figure 7.3-25. For the cost of drilling at $6.23/m, the total shot cost is only 3% lower for ANFO at $13,172 compared with emulsion at $13,563. This small differential decreases to zero as drilling costs increase to $7.45/m, at which point the cost of using ANFO or gassed emulsion is the same at $14,650. As drilling costs exceed $7.45/m, emulsion is cheaper on a per-shot basis, but the margin of difference in this range of drilling cost is not very large. If a 70/30 ANFO/emulsion blend were used instead, then the cost would be reduced to below $12,000, which would give an overall cost savings of nearly 11% over ANFO. A 70/30 ANFO/emulsion blend is not sufficiently waterproof to be loaded into wet holes. Drilling costs must be less than $2.26/m before ANFO becomes more economical than a 70/30 blend.

The previous example shows the effect of drilling costs on a per-shot basis for dry holes. Additional analysis is required to determine the economics of encountering wet holes. On some shots, a percentage of holes may be wet while others are dry. Likewise, in certain mines, some shots may be entirely dry and others entirely wet. In these situations, it is common practice to drill an ANFO pattern and then deal with wet holes as they are encountered by loading a water-resistant (and thus more expensive) product. Wet holes cannot be loaded with ANFO and usually require bulk emulsion or, more commonly today, a high emulsion percent blend. This increases cost in two ways: first, a more expensive explosive is used, and second, because of the greater density, more explosives weight is required in the hole to bring the column up to the correct level.

Figure 7.3-26 graphically illustrates the effect on cost of having to use emulsion in wet holes on a pattern designed for ANFO. The solid line represents the cost of a shot designed for emulsion. The cost is constant for dry or wet ground. The dashed line represents the cost of encountering wet holes in a pattern designed for ANFO. As the percentage of wet holes increases, the cost increases because the more expensive emulsion product is required for the wet holes. As can be seen, the cost of loading emulsion in an entirely wet pattern that was drill-designed for ANFO represents a dramatic increase in cost (dashed line). The example also illustrates that if only 10% of the holes are wet, it is more economical to use emulsion with a drill pattern designed for emulsion. The bottom line is where a 70/30 ANFO emulsion blend is used to replace the emulsion (in the expanded/wet-hole pattern, solid line) where dry holes are encountered. In this case, the same weight of explosive is used in each hole as the densities of blend and emulsion are very similar and the pattern is already a "wet pattern" (i.e., expanded pattern suitable for emulsion). It is plainly evident that the cost of using a blend (bottom line) as the baseline explosive is far more attractive than that using ANFO, and it is cheaper than using only emulsion. This is why the use of blends continues to increase.

Another important economic factor is the size of drill appropriate for an operation. This is a simple question that may sometimes produce a rather complex answer because of issues such as the proximity of neighbors and ground vibration limits. Even without considering these issues, all equipment at a mine should be matched in productivity so that no one piece is overutilized (thereby providing some slack for expansion and to cover production problems) or severely underutilized. For example, a modern hydraulic 100-mm-diameter drilling rig is capable of providing more than 2 Mt of production per year, and a 150-mm-diameter drilling rig can provide 5 to 6 Mt/yr in limestone. Obviously, buying a 150-mm drill for a 0.75-Mt/yr quarry will result in severe underutilization of the equipment, and an 89-mm-diameter drilling rig would make more sense. In addition, this quarry would not be expected to have a very large primary crusher, so that problems with oversize could be expected with the expanded pattern size and consequent less uniform energy distribution of a 150-mm-diameter drilling rig.

Shot service affords many economic advantages, not just in the direct cost of blasting, but also, because these companies specialize in blasting, they have a far higher level of expertise. Blasting contractors, who often provide their own vibration-monitoring equipment and insurance, can usually reduce blasting problems, improve fragmentation and wall conditions, and reduce or eliminate explosives storage on-site. Relying on companies with expertise in blasting allows the management of small companies to concentrate on other items specific to their mining operation.

DESCRIPTION AND CONTROL OF BLAST EFFECTS

The use of explosives in mining creates several effects that can be detrimental to the operation if they are not properly controlled through proper design and application. Most operations are concerned with controlling flyrock, ground vibrations, air blast, overbreak, and so forth. This section provides some basic troubleshooting for the blast effects mentioned and shows connections between the design elements and respective blast effects.

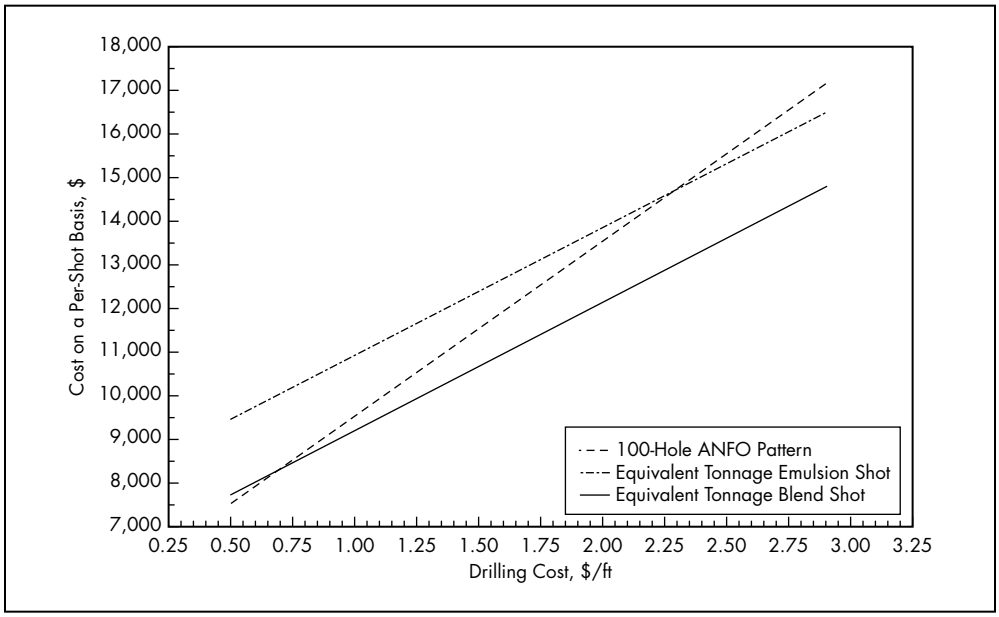


Figure 7.3-25 Effect of drilling costs on economic selection of ANFO versus emulsion and blend

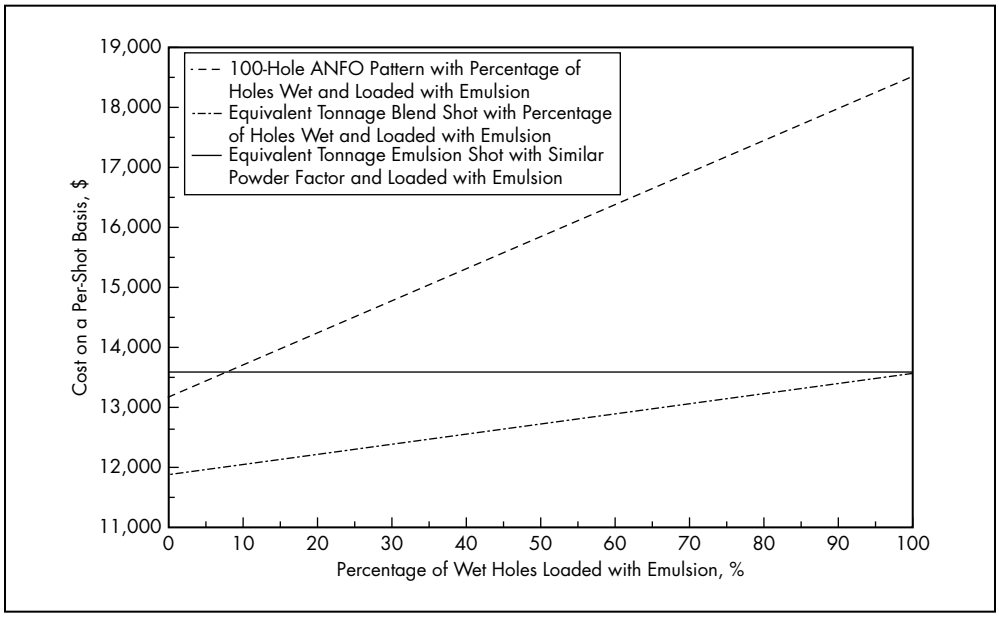


Figure 7.3-26 Economic analysis for using a 70/30 blend on an expanded pattern using emulsion for wet holes

By using proper blast design, the energy in an explosive product is harnessed to produce the desired result. Mining operations usually wish to use the majority of the energy to break rock and move it, thus fragmenting it into pieces of manageable size. The optimum scenario is to minimize the explosive energy that is converted into undesirable blast effects while maximizing the use of energy to provide the desired fragmentation and movement.

Flyrock is a significant issue that has plagued the mining industry since the inception of blasting as a method for excavation. The definition of flyrock is "any rock from a blast that goes where it is not intended to go." Various regulatory agencies have developed their own specific definitions for flyrock for the protection of residents. For example, Kentucky regulation KRS 351.330 states that "flying rocks shall not be allowed to fall greater than one-half the distance between the blast and a dwelling house, public building, school, church, commercial, or institutional building…" (Kentucky Revised Statutes 2009). A majority of injury-causing blasting accidents have been caused by flyrock that occurred when the blaster intentionally fired the shot. In contrast, it would be very difficult to find cases where individuals were injured by

ground vibrations or air blast from commercial blasting. For this reason, flyrock remains a focus for safe design. There are several potential sources of flyrock. In most cases, flyrock is generated from specific points in a blast where the burden is insufficient. Reasons for such occurrences include insufficient stemming, poor drilling, irregularities in the face, and a high charge density caused by loading through voids. In cases where flyrock must be well controlled, the practice of drilling holes adjacent to muckpile *before* it is excavated should be avoided as the reduced control of face hole burden can and has produced faceburst and long-range flyrock incidents. Other cases exist where a satisfactory design burden becomes insufficient. For example, mud seams or clay seams and loading explosives through these seams is a potential cause of flyrock. Identifying mud seams and voids is much easier during the drilling cycle, and such occurrences must be communicated to the shot blaster by the driller.

Accurate drilling and recording of drill logs is imperative to eliminating flyrock. In fact, poor drilling or poor driller communication are usually to blame for the majority of the blast effects. Modern tools, such as face and hole profilers and drill automation, can be used effectively to ensure consistent and appropriate burden, but nothing can replace a properly trained and experienced driller.

Operators generally want to minimize overbreak. In surface operations, overbreak/back break can create unstable highwall conditions. Likewise overbreak/fracturing to the bench below can cause problems when drilling and preparing lower benches. Overbreak in underground operations can lead to unstable conditions and more extensive and/or more complicated roof-control programs. Overbreak in bench blasting is usually caused by either too much powder in the last row, firing of choked/buffered or otherwise overconfined blasts next to walls, or by insufficient delay timing. In underground operations, overbreak usually occurs from overloaded holes too close to the planned perimeter of the opening. A number of perimeter-blasting techniques can reduce the amount of overbreak to an acceptable level. In surface operations, the application of presplit blasting is commonly used to generate stable highwall conditions. Other common techniques include line drilling, buffer blasting, and smooth walling. The concept is similar for each technique and generally involves closely spaced holes with lighter explosive loads than the main part of the blast. One of the most important aspects of a successful overbreak control program is the knowledge and analysis of geology. Many geologic features can enhance or reduce the effects of the perimeter blasting techniques.

Ground vibration and air blast are common blast effects that are almost impossible to completely eliminate. There is generally some form of both generated from every mine blast. Regulations govern the amount of vibration and air blast permissible at nearby structures. Although state and local ordinances concerning air blast and ground vibration vary, a majority of these regulations will closely resemble the content of Office of Surface Mining blasting regulations.

If scaled-distance limitations require a lower amount of explosive per delay than can physically fit into a blasthole with stemming, decking must be employed. Holes can be loaded where two or more powder columns are separated by inert stemming material and fired on separate delays. This is a common practice for surface blasting where nearby structures impose explosive weight restrictions per delay due to scaled-distance calculations.

Improper stemming or insufficient burden often leads to high air blast, which is usually accompanied by flyrock. Other causes of excessive air blast include the use of detonating cord on the surface and gas venting from the face. The need to use detonating cord on the surface has been superseded by modern advances in initiation systems. Air blast overpressure levels can be estimated using scaled-distance relationships. More complete coverage of ground vibration and air blast can be found in Chapter 16.3.

Another potential cause for excessive ground vibration, air blast, and flyrock is improper timing. Delays are used in blasting to provide optimum fragmentation and movement. There must be sufficient delay between holes to provide adequate relief for subsequent holes. A delicate balance exists between sufficient and excessive delay. In some cases, too much delay can remove the burden of holes that are in rows further away from the bench crest, leaving the holes underburdened and causing flyrock and high air blast. If insufficient delay is utilized, holes risk becoming overburdened and high ground vibrations can result. Overburdened holes can also generate flyrock in the form of stemming ejection. Similarly, if insufficient timing is used, the gas pressure is likely to eject the stemming and crater the blasthole. In bench blasting, insufficient timing on the back row is evident by fragmented rock heaped on the bench behind the blast. In a properly timed bench blast, the bench behind the blast should be clean and clear of shot rock.

Gas fumes are produced as a normal by-product of blasting operations regardless of the types of the explosive materials used. Common gases produced by explosives include ammonia, carbon monoxide, carbon dioxide, oxygen, water vapor, nitrogen, nitrogen oxides, sulfur dioxide, and hydrogen sulfide. Sulfur dioxide and hydrogen sulfide can be generated if sulfur is used in the explosive formulation. Currently, no products in the United States use sulfur as an ingredient. Every explosive product fumes on detonation. These fume volumes are generated under ideal conditions and do not consider the effects of adverse field conditions, including water, dilution of explosive products, and low-order detonation as a consequence of underpriming. These conditions can cause excessive fumes and can create other operational problems. ANFO is particularly susceptible to fume generation because of water infiltration. When blasts are confined, such as in trenching or lift-shot applications, fumes are likely to seep through the strata to nearby neighborhoods and developments. Blasters should be wary of these conditions and apply preventive measures to ensure that fumes do not cause dangerous situations. For example, when blasting in wet conditions near residential areas, a waterproof low-fume class product is necessary.

With all of the blast effects discussed in this section, it is apparent that attention to detail is of utmost importance. Designing a proper blast involves relatively simple calculations. However, the application of this design in a mine environment is much more difficult. Assurance of proper burden, spacing, loading, and timing is imperative to a successful blast.

TRENDS, AUTOMATION, AND THE FUTURE

Blasting techniques are constantly evolving with technology. In general, changing trends in the explosives industry are most evident to shot service companies and explosives manufacturers.

The explosives product of choice for many years has been ANFO, which replaced dynamite as the dominant explosives

product many years ago. Dynamite is still a viable product for a few niche applications, but it has only a very small share of the market. Most operations have also moved away from bags and cartridges to bulk. Explosives supply contractors have highly automated bulk trucks that are capable of creating variable blends of ANFO and emulsions. Advances in bulk truck technology allow the storage and transportation of oxidizers that become blasting agents when they are gassed by the bulk truck during loading into the blasthole. Because of the regulatory authorities' storage regulations, this ability has prompted many companies to adopt this technology. Several shot service companies have started using these gassed emulsions almost exclusively. Even large underground operations utilize bulk emulsions.

Small quarries and surface mines often use shot service contractors for off-site storage of explosives and complete blasting services. To keep a shot crew employed and to justify the purchase of new-technology bulk-loading equipment, an operator needs to be undertaking a lot of blasting. For smaller operations, full shot service not only offers direct monetary efficiency, but it also reduces the issues associated with compliance, insurance, dealing with complaints, and so forth. Even quarries producing 6 Mt/yr often use shot service for delivery of bulk explosives products from the manufacturer to the bench while using their own shot crew to help load, connect, and fire the blast.

The same can be said for drilling, and as such, there has been a rapid increase in the amount of drilling that is contracted out. For a smaller operation, this can be far more cost-effective in the long haul. One added reason for the sudden increase in contract drilling between 2003 and 2006 was the introduction of noise limitations on drilling by the Mine Safety and Health Administration. The only way that drilling manufacturers have been able to reduce driller noise exposure in an acceptable manner is to provide soundproofed cabs. However, this requires drill operation to be automated with drill steel carousels and so forth. Also, because hydraulic drills are less noisy, mines have been forced to convert to hydraulic drills and abandon their air-track drills. The high purchase cost of new equipment has forced them to investigate the contract drilling option. Quarry companies with multiple sites now often own high-productivity hydraulic drill rigs, which are shared and transported among their operations. This in turn has forced them to shoot bigger, more infrequent blasts, which has lent itself to the economics of shot service. Shot service is now the norm in the United States.

Advances in drilling technology have allowed for the use of Global Positioning System (GPS) locators for drill holes. This ability allows engineers to design blast patterns in real time with the advance of the mine. If used properly, a GPS has the potential to increase the quality of many blasting programs. In underground situations, drilling jumbos now have GPS technology to start from a reference point and drill a round autonomously. The drill operator is still required to ensure quality control and to adapt patterns to changing conditions, but the drilling cycle is becoming more automated and efficient.

The most recent advances in initiation systems have taken place in electronic detonators. More operations are utilizing electronic detonators and are benefiting from this newer technology. Cost has been the major disadvantage, but costs are gradually becoming competitive with nonelectric systems. The advantages of electronic initiation in fragmentation and control are becoming more evident, and many operations are beginning to quantify the economic advantages of switching to electronics. It is likely that electronics will take an increasing share of the market in the coming years.

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PART 8

Ground Mechanics

CHAPTER 8.1

Introduction to Ground Mechanics

Russell Frith and Mark Colwell

The nature of the geotechnical environment and the resultant geomechanics (i.e., how the ground responds) during excavation is one of the primary influences on mining—for example, whether to commence mining in the first place, the choice of mining method, the design of the mine layout, or the need for ground control measures during and subsequent to mining. After a deposit of ore or coal has been found and a market for its subsequent sale identified, understanding the various mining constraints, as a result of the nature of the geotechnical environment, becomes a key mining consideration throughout the entire life of the project.

There is little doubt that mining efficiencies are materially affected by geotechnical aspects of the mining environment. For example, as the immediate rock mass surrounding an ore body becomes less competent, ground support requirements to maintain a suitably stable and safe excavation will inevitably increase. Increased ground support translates to increased mining costs but, more importantly, will decrease the rate of roadway or tunnel advance. In almost every underground mining scenario, the rate of tunnel development is a key business performance parameter.

In surface mining, slope angles are strongly linked to the geotechnical nature of the overburden or waste. The less competent the overburden, the lower the slope angle must be to maintain an adequately stable and safe pit wall. As the slope angle decreases, the amount of overlying material that needs to be removed to access each ton of mining product correspondingly increases. This, in turn, increases the mining cost and so detracts from business performance.

As early as possible in the mine feasibility assessment process it is vitally important to understand and fully consider the interrelationships between the local geotechnical environment and the mining process. In both surface and underground mining, the geotechnical environment is a material mining consideration, well summed up in the following quotation by Napoleon Hill: "Nature cannot be tricked or cheated. She will give up to you the object of your struggles only after you have paid her price."

Part of "her price" is understanding and adequately controlling the geotechnical environment in all of its facets. This section of the handbook covers many such aspects in a technical sense, while this chapter is intended to provide a general overview with some appropriate examples.

MINING FEASIBILITIES

The objective of every mining feasibility study is to make a prudent capital investment with the expectation of making a suitable return over time. As with all things in life, there are no guarantees when making such an investment, but the focus should be on achieving the highest possible level of confidence that the feasibility predictions on which the decision is made become reality. This is no trivial exercise, and ignoring or underplaying the geotechnical environment's influence on mining is a serious feasibility oversight. Plenty of failed mining projects around the world substantiate this particular point.

The irony of such failed projects is that eventually most deposits become economically viable due to either improved mining technology or significantly higher commodity prices as a result of increased demand and/or increasing scarcity of the resource. Investing in mining projects prematurely has both short-term and long-term adverse consequences. As the custodians of the earth's natural resources, miners have a duty to invest only in those projects that make good economic sense at the time of mining. Understanding the impact of the geotechnical environment on mining is a key part of this.

Larger mining companies are typically owned by shareholders, many of whom invest via either private equity or through retirement funds. Mining companies have a duty to invest the shareholders' hard-earned savings wisely, with the feasibility study being the primary vehicle for such decision making. Different companies take varying approaches to the mining feasibility process. Although this chapter is not aimed at discussing how financial viability is measured, in general terms, mining feasibilities should have three basic stages: concept study, prefeasibility assessment, and feasibility assessment.

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Concept Study

This stage simply establishes that certain minimum requirements are met to allow a feasibility study to take place. For example, it is important to know the likely quality of the deposit and, in the case of the geotechnical environment, whether any obvious "showstoppers" would immediately preclude any form of economic mining.

For example, in underground longwall mining at high depth (>500 m), the presence of a thick and strong (>100 m thick) dolerite sill just above the target coal seam, with the potential to prevent overburden caving, would be an obvious geotechnical showstopper to efficient and indeed safe longwall extraction, as longwall mining relies heavily on the regular caving of the overburden. A dolerite sill of such thickness so close to the extraction horizon would prevent regular caving of most of the overburden and instead would store significant amounts of potential energy through spanning across one or more longwall panels. With such a feature present in even a small number of boreholes, it would obviously raise the questions as to whether to proceed any further and, if so, whether other mining methods would be better suited to mining the resource.

Prefeasibility Assessment

Fortunately geotechnical showstoppers are rare, and most projects at least get to the prefeasibility stage. The aim of the prefeasibility assessment is to develop several credible mining options that could conceivably be used to exploit the deposit. These can be compared on a net present value (NPV) basis, as well as associated technical risk, with the outcome a preferred mining option to take into the feasibility assessment.

Feasibility Assessment

With a preferred mining option identified, the focus of the feasibility stage is to evaluate the option in sufficient cost and technical detail so as to provide a suitable basis for a prudent investment decision. If the decision is made to invest, the project proceeds to the construction and final operational phases. If not, it remains as a prospect that may be revisited in the future when technology and market forces are more favorable.

In geotechnical terms, the credibility of the feasibility study process relies heavily on two aspects that need to be in place:

1. Availability of a suitable amount of relevant geotechnical information. Clearly one cannot make quality decisions without quality data, and this encompasses both the type and quantity of such data.
2. The use of appropriate geotechnical assessment or design methods that establish reliable mine design parameters and identify geotechnical risks and their mitigation. For example, there is little point in designing tunnel or roadway strata support in great detail at this stage when all that is required is a credible estimate of likely average support quantities to gauge the impact of such support on the rate of advance and mining costs.

The type of geotechnical data collected for a given mining prospect clearly varies according to the type of mining being evaluated and, most importantly, the geotechnical assessment methods used. It is vital at the project's start that likely assessment methods are defined and that exploration protocols are targeted to the collection of the necessary input data. Keeping in mind that such projects may be subject to peer review by individuals and groups using different methods of assessment (some proprietary), the need to be clear at the outset on the required range of geotechnical data is critical. Because much geotechnical data emanate from either logging or testing of fresh bore core, the ability to revisit incomplete boreholes later in the feasibility study can be limited.

In terms of the density of the available geotechnical data, no hard and fast rules exist, and different mining companies apply different corporate standards. One could argue that the various standards of resource reporting (e.g., *The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* [AusIMM 2004]) should provide some specific guidance in this regard. Most standards recognize that as a resource goes from *inferred* to *measured* or a reserve moves from *probable* to *proven*, the level of geological and geotechnical knowledge and associated confidence must increase. This makes logical sense.

The SME Guide for Reporting Exploration Information, Mineral Resources, and Mineral Reserves (SME 2007) provides the following quotation in regard to the reporting of mineral reserves:

> Mineral Reserves are those portions of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Competent Person making the estimates, can be the basis of a viable project after taking account of all relevant mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors.
>
> The term "economic" implies that extraction of the Mineral Reserve has been established or analytically demonstrated to be viable and justifiable under reasonable investment and market assumptions.

In discussing coal reserves, the 2007 *SME Guide* makes the following comments in regard to geotechnical matters:

> Geological similarity between neighbouring coal deposits can greatly simplify demonstration of a new Coal Resource, as well as reduce the technical and economic study requirements needed to demonstrate a Coal Reserve next to an operating mine. Geological similarity must be demonstrated by means of drill holes, mapping or other deposit-specific geoscientific evidence to a suitable level of confidence required to declare Measured and/or Indicated Resources. Mere inference of the continuity of coal thickness and quality from an operating mine onto a neighbouring block or property is not sufficient to declare Measured and Indicated Resources and subsequently a Reserve.
>
> It is the responsibility of the Competent Person to ascertain that there is sufficient information to demonstrate geological similarity and to determine which additional factors must be taken into account to demonstrate technical and economic feasibility with a reasonable level of confidence.

Clearly then, the definition of adequate mine feasibility input data requirements should be determined by a competent person and not governed by statute or standards. Obviously, the person responsible for undertaking the geotechnical assessment

of a mining prospect should have a leading role in determining what constitutes suitable quantities of geotechnical data, as opposed to what is required for simply estimating the volume of the reserve base.

One point worth making is that if the economic measure of a mining project is its NPV, the bulk of that value relates to the first 15 years of mining, beyond which mining outcomes have a significantly diminished effect on the investment decision. Therefore, on the basis that all exploration budgets have limits, it makes sense, particularly with very large projects, to ensure that the initial mining areas for the majority of the NPV, and hence investment risk, contain a higher quantity of exploration data than those that are longer term.

Geotechnical design methods will be discussed in more detail later in this section, specifically that all methods, regardless of how they are named (e.g., analytical, numerical, or empirical) are in fact or certainly should be empirical in nature at some level because they refer back to, and rely on, previous mining experiences. In terms of the feasibility stage of a project where the objective is to make an investment decision with confidence, a highly valuable feasibility finding is demonstrating that another mining project achieved certain mining outcomes in geotechnically similar conditions to the one under consideration, which is far more valuable than predicting future ground support densities or mining geometries to high levels of accuracy. By definition, this means that well-constructed empirical methods of geotechnical assessment and design within an overall assessment strategy should have a key role in mining feasibilities in which a mineability estimate is the key focus.

All mining investment decisions contain some level of residual or undefined risk, particularly in the area of geomechanics, because, unlike most other engineering disciplines, the engineer does not have control over the nature of the material involved (i.e., soil or rock). Because this is controlled by Mother Nature, it is impractical to know at the feasibility stage the full range and distribution of mining conditions to be faced. Furthermore, much of the capital investment will be sunk before the true impact of the geotechnical environment is experienced in actual operations, at which time abandoning the project would be costly in terms of both cash losses and business reputation. Therefore, mining feasibility studies should be undertaken on a prudent rather than overly optimistic geotechnical basis, because experience shows that, as more geotechnical information and mining experience becomes available, the geotechnical outlook generally either worsens or, at a minimum, becomes more complex.

MINING OPERATIONS AND STRATA MANAGEMENT

When a mining project proceeds to an operational status, the geotechnical focus changes to

- Striving to improve mining efficiencies within geotechnical constraints,
- Promoting safe mining practices, and
- Identifying and controlling those geotechnical conditions that were previously unknown but are encountered during mining.

Whether formally or informally, this is all covered by what is termed colloquially in the Australian underground coal industry as the *strata management process*, which is now defined in local mining legislation.

The strata management process has been implemented, in fact, since historic mining operations. Even though the earliest miners had little or no geotechnical data or sophisticated measurement methods, they could still use their own judgment (typically gained from previous successful and unsuccessful mining experiences) as to whether a mine area was safe and hence whether to continue mining or not. Unfortunately, a combination of inexperience and/or an optimistic outlook can lead to poor judgments, the result being inadvertent unsafe mining practices. Personal judgment (via observation) in isolation is not always sufficient to optimize mining practices and maintain suitable stable and safe mine workings. Geotechnical design and continual operational control of the geotechnical environment need to work hand in hand in order to achieve the highest standards of both mine safety and economic performance.

In the 21st century, unsafe mining, while morally and legally unacceptable, is also not good business. One has only to consider the rigor to enforce recent corporate safety initiatives of the major mining houses (e.g., BHP Billiton's Fatal Risk Control Protocols [BHP Billiton 2005]) to prove this point. It is also gratifying to know that different countries are tending to cooperate with each other at the highest levels in the effort to improve mine safety. The Australia/China Coal Mine Safety Demonstration Project is one such example of this (Queensland Government 2009).

The strata management process during operational mining has three basic elements:

1. Geotechnical design
2. Exploration and input data
3. Implementation and management of residual risks (i.e., those risks remaining despite all defined controls being in place)

Each of these will be discussed in detail, but it is worth making the initial point that the overall process hinges (or should) around geotechnical design. Geotechnical design requirements will govern what information needs to be collected as part of the exploration program and subsequent mining activities.

Similarly, geotechnical design outcomes, particularly the residual uncertainties involved, will influence what measures need to be taken to safely implement the design and ensure its ongoing suitability during operations. A pillar design with a high factor of safety (FoS) will require little or no ongoing operational vigilance (other than related to the immediate skin of the pillar or ribs) to ensure that it remains suitably stable. However, if a wide mine roadway, for example, needs to be excavated with a relatively low FoS against roof instability occurring, implementing an effective monitoring and response regime to ensure that roadway's ongoing stability becomes quite critical.

As a result of this realization, geotechnical design will be discussed first followed by exploration, then implementation and management.

Geotechnical Design and Design Methods

Design is an intrinsic component of effective engineering, including geotechnical engineering, in mining. Whether underground or surface, a mine represents an engineering structure where persons are required to work and shareholder value is to be realized. It is common sense that mining ventures will almost certainly be safer and more profitable if

controlled by credible mine design according to the specific set of geotechnical circumstances faced.

The Coalbrook disaster of 1960 in South Africa, where 437 miners lost their lives in the world's largest coal pillar collapse, demonstrates the inherent consequences of inadequate geotechnical design (van der Merwe 2006). In 1960, few if any credible coal pillar design methods were available to mine operators, and the occurrence of such a tragedy was probably inevitable. The same is not true today when duty of care requires that credible and prudent geotechnical design be applied to mining operations.

Undertaking geotechnical design in mining does not in itself guarantee that the required outcome will be achieved in practice. However, the likelihood of achieving it increases considerably with the effort of undertaking a meaningful and risk-based design evaluation.

Mine design in general should consist of three primary considerations:

1. Safety
2. Reserve recovery
3. Rate, or unit cost of production

The fundamental dilemma is that, if taken for granted that all mine design needs to result in a safe working environment, a compromise may be required between reserve recovery and rate/cost of production. For example, leaving pillars behind clearly reduces reserve recovery, but they can improve general mining conditions and thus promote safety and assist mining rates. It is not always possible for any given mining environment to allow 100% of the ore to be mined safely and cheaply, particularly in underground mining. Therefore, a strategic decision needs to be made as to which parameter (reserve recovery or rate/cost of production) is to be compromised in favor of promoting the other.

There are no universal rights and wrongs in this, because much will depend on the primary objective of mining. Any commercial mining operation will want to extract as much of the reserve as possible but only within a strict cost range. However, in a number of government-controlled mining industries, the focus is on extracting all of what is considered a precious national resource, almost regardless of cost. Both of these are legitimate mining strategies. This is where prudent geotechnical mine design has a role to play, ensuring that the balance between safety, reserve recovery, and mining efficiencies is kept within tolerable levels.

Specifically, the primary objective of geotechnical mine design is to provide mine operators with either benign or, more typically, inherently manageable conditions during operations. The design is ineffective if the manifestation of geotechnical problems is either so rapid that mine workers cannot take effective remedial action or vacate an area (e.g., coal mine bumps or rock bursts), or so sensitive to the actions of mine operators that minor mistakes on their part result in catastrophic failures.

It is obvious that effective geotechnical design cannot be undertaken without reliable design methods. This section will provide some of the fundamental requirements for a design method to be considered credible:

- It must be of direct relevance to the problem being addressed.
- It must provide risk-based design outcomes to allow mine decision makers to consider their mining options and associated risks.
- Wherever possible, the method should refer to and make use of known documented mining experiences (either locally or industry-wide), either through the use of empirical design methods or the back-calibration of analytical or numerical methods.
- The method should have a proven track record via previous use or at least be published in the public domain.
- The basis of the design and methodology must be understandable by mine-site personnel, particularly decision makers.
- The method should be transparent and fully auditable.
- Design should always be undertaken by suitably competent personnel.
- Design outcomes must fit with the mining process in use and be able to be installed as intended in a safe manner.
- The design method must include relevant geotechnical parameters.

Mining personnel can judge for themselves as to whether geotechnical design in use at their mine meets these credibility requirements. However, it is important to include geotechnical parameters in the design process; the potential limitations of relying on "rules of thumb" can be found in various sources.

For example, it may be a general rule of thumb that the length of a rock bolt should be one-half to one-third the heading width. However, without considering the ground stresses, structural competency of the host strata, the required excavation serviceability and service life, or indeed the type of rock bolt being employed, it would be imprudent to heavily rely on such a rule for design purposes when considering rock bolt length. It may be convenient to use such general rules in mining as initial estimates, but current duty-of-care considerations require a far more professional approach.

Mining legislation may also stipulate that a geotechnical design process is in place, even if the word *design* is never actually used. The following is an excerpt from the Australian New South Wales Coal Mine Health and Safety Regulation 2006 (NSW Government 2006) about the Strata Failure Management Plan (Section 32 of the regulation):

> [The plan] must make provision for the following matters:
>
> a. the estimation of the geological conditions likely to be encountered in roadway development
> b. the assessment of the stability of roadways to be developed in those geological conditions
> c. the recording of geological conditions that may affect roadway stability
> d. the development of support measures that will provide roadway stability in those geological conditions
> e. calculations (including maximum roadway width and the minimum dimensions of coal pillars) to determine the probability of instability to be assigned to any coal pillar, consistent with the pillar's role or roles over its life.

Without ever using the terminology of *geotechnical design*, this legislation clearly states that coal pillars and roadway strata in coal mining need to be controlled based on a logical design process rather than simple rules of thumb or local mining precedent. Furthermore, the design process needs to be proactive in an attempt to predict geological conditions before they are

encountered, rather than reactive via defining control measures once discovered. That is not to say that unexpected geological conditions will not be encountered during development, simply that the mine must make all efforts to minimize the potential for this happening, because safety hazards are inevitably heightened when unexpected ground conditions are intersected.

Clearly, mining legislation varies around the world, and no hard and fast rules exist for the prescribed need for geotechnical design. However, it is important for practicing mining personnel to be fully aware of what the local mining rules prescribe in this regard and so ensure that minimum geotechnical design requirements are met.

At a basic level a good deal of geotechnical design in mining can be encapsulated by the following equation that incorporates the well-known concept of FoS, even if a particular design methodology does not directly utilize the concept as the design output:

$$\text{FoS or risk-based measure of stability} = \frac{\text{load-bearing capacity}}{\text{applied load}}$$

The point is that most geotechnical problems in mining can be simplified for the purpose of illustration to (1) a driving force, (2) a resistive force, and (3) a risk-based outcome that can be used in making informed and prudent decisions.

On many occasions, using a form of this equation for actual geotechnical design results in inevitable complications, and in these situations the design approach should typically provide a credible range to work within as part of a management process. Ground support associated with mine roadways (as opposed to permanent tunnels) is a classic example. In this situation, instead of the outcome being an FoS, it is a credible range for the level/type of support that can be installed in conjunction with an appropriate monitoring program to manage the risk.

Where an FoS or risk-based design approach is adopted, everything to the right of the equal sign should be predominantly within the domain of the geotechnical engineer, whereas everything to the left should be ultimately that of the mine-site decision maker who is the custodian of acceptable levels of risk. In this instance, it is important that geotechnical inputs and risk-based design outputs do not get confused or indeed considered as one, specifically as their custodians are different. The geotechnical designer should provide mine-site decision makers with various mining options, each containing a defined level of risk. It is the responsibility of the decision maker to consider those options along with non-geotechnical considerations and come to a holistic risk-based decision.

It is understandable that mine operators wish to minimize costs and maximize resource recovery in their efforts to improve mining efficiencies. However, geotechnical input parameters should never be modified as part of this endeavor. The only design variable that allows optimization of the design outcome is the level of acceptable risk. This ensures that mine-site decision makers are fully informed of the potential ramifications when they make their decisions, particularly the need to institute specific operational controls to manage the inherent residual risks.

Generic design methods in geotechnical engineering are commonly referred to as

- Empirical—based on mining experiences, ideally analyzed in a statistically rigorous manner;
- Analytical—involving some form of closed-form solution utilizing discrete equations;
- Numerical—using computer codes;
- Local experience—referring to what has worked previously at a given mine; and
- Rules of thumb.

The choice of geotechnical design method rests with the engineer undertaking the design. Publications such as industry standards on quality control in engineering design (e.g., AS 3905.12-1999, *Guide to AS/NZS 9001:1994 for Architectural and Engineering Design Practices*) mandate processes but not *how* to undertake design or what methodologies should be used. This makes sense as every geotechnical designer brings his or her experience into the design, and it is important to utilize those methods that best complement that experience base. This may include the use of several independent methods to address the same problem, which is commonly considered to be good practice.

Local experiences at a given mine site can be a source of significant design guidance but as an isolated or limited number of case histories only, because reliance on such guidance can lead to problems when geotechnical conditions depart significantly from previous local experiences. It is only by giving due consideration to the primary geotechnical drivers at work in case studies that geotechnical design can attain high levels of reliability; and for this, one is forced to move to either empirical, analytical, or numerical design.

It is beyond the scope of this chapter to provide a detailed comparison of these three design methodologies. In fact, it comprises a book in itself if one wishes to delve deeply into the specifics of individual design methodologies. However, what is worth stating is that regardless of whether a design method is described as empirical, analytical, or numerical, all are by definition empirical at some level. Empirical design methods are self-evident in this regard. However, the highly common practice of calibrating analytical or numerical design methods to known or measured outcomes in the field is an obvious and unequivocal form of empirical design.

It would be nice to believe that geotechnical engineering has progressed to a point where we have at our disposal infallible geotechnical models that can make accurate predictions for how strata will behave under given conditions and how best to go about controlling the resultant manifestation. The reality is that no *first principles* design approaches come close to achieving this, and all design methods need to or should refer back to known experiences prior to making future predictions.

In relation to empirical modeling, Salamon (1989) states the following:

> The main advantage of this approach is its firm links to actual experience. Thus if it is judiciously applied it can hardly result in a totally wrong answer. Also in our legalistic world it has the added advantage of defensibility in a court of law. After all it is based on actual happenings and is not just a figment of imagination.

It is important for a mine-site employee with any form of responsibility for geotechnical design to be satisfied that designs used in operations are credible, and one significant aspect of this is the credibility of the design method in the first place.

If it is accepted that almost all current geotechnical design methods are empirical at some level, this gives the mine-site employee a basis for interacting with the geotechnical

engineer. For example, it would be unreasonable to expect a general mining engineer to hold a sensible discussion on the application of the associated flow rule for plasticity within a numerical code. However, it is reasonable and, in fact, important that mining engineers charged with operational decision making in the area of geomechanics are conversant with the empirical or experience base that underpins the use of any given geotechnical design method at their mines.

In the same way that mining legislation develops over time via learning from mining outcomes (usually accidents), so geotechnical design methods should always be based on and continually updated from relevant mining experiences. Without reference to such experiences, the credibility of geotechnical design requires significant scrutiny before being implemented in mining operations where the safety of workers and the financial security of the mine are at stake.

One final aspect of geotechnical design is having a clear view about the design output. Geotechnical matters rarely completely define the entire focus of geotechnical design, because in the final analysis it is all about supporting a mining process. Therefore, for geotechnical design to be as effective as possible, the geotechnical engineer must be geotechnically competent and have a sound understanding of the mining process and mine operators' needs. For example, mine operators may need one or more of the following non-geotechnical aspects of mining included and considered by the geotechnical design process:

- Exposure of mining personnel to geotechnical risks and associated safety hazards
- Service life of the excavation or mine opening
- Business consequences of inadequate ground control during mining
- Whether the mining work is a critical path activity (i.e., detracts directly from mine production levels)
- Costs involved
- Mining equipment to be used and its limitations
- Ability to install remedial measures if problems occur
- Dilution of run-of-mine product or grade control
- Needs of the mine budget and forecasts

This list clearly demonstrates that an effective geotechnical designer in mining must have a good working knowledge and understanding of the various needs of the mining process and the people who manage and operate the mine. Conversely, it is important for mine operators to clearly define their requirements so the geotechnical engineer can duly consider them as part of the engineer's work.

Exploration and Input Data for Geotechnical Design Purposes

The subject of geotechnical data for design purposes is vast and varies according to the geotechnical issue being addressed (e.g., shaft wall stability, open-pit slope stability, or pillar design for underground mining). Each geotechnical design approach will have its own particular input data requirements and, as stated earlier, it is vital that the exploration program provides the geotechnical designer with the data he or she needs.

Geotechnical data for assessment and design purposes inevitably comes from a variety of sources; no single preferred source of data is preferred. However, the following are typical sources of input data for geotechnical design purposes:

- Surface boreholes: core, chip logs, observations, geophysics, in-hole measurements
- Surface geophysics: seismics, magnetics
- Surface mapping: highwalls, cliffs, lineaments
- Underground mapping: roadways, previously mined-out areas, falls of ground, adverse mining conditions
- Underground boreholes (i.e., upholes and in-seam drilling)
- Underground monitoring data: stress measurement, deformation monitoring (e.g., extensometers, tiltmeters), borescope or fiberscope observations, seismic monitoring systems, and so forth
- Mining experiences: falls of strata, trapped mining equipment, safety statistics, production statistics
- Published technical literature: technical papers, industry bodies, documented case histories
- Informally obtained information: from adjacent mines, mining personnel, third-party consultants

One common theme about geotechnical data collection is consistency of method. Whether it be something as simple as a laboratory test for the uniaxial compressive strength of rock or the calculation of a rock mass rating parameter, consistency is critical. No matter who is undertaking the test or evaluation, the numerical outcome should not be sensitive to the individual concerned. This leads into the area of standards, protocols, and the competency of persons, and it is therefore no surprise that groups such as the International Society for Rock Mechanics (ISRM) have gone to great lengths to develop and publish standards for the characterization of the geotechnical environment. This is summarized in what is known as *The Blue Book*, available from the ISRM (2007).

Quality decisions in geotechnical matters are strongly linked to the quality and quantity of the input data that supports the decision. Therefore, there is great mining value in committing resources to the collection and (just as importantly) the formal storage of geotechnical data relating to the mining project. This is a vast subject area.

Implementation and Management of Residual Risks

After a geotechnical design is signed off, it is transferred to mine operators for implementation. This is one area in which mining is different from, for example, civil construction activities and where many geotechnical problems can occur.

In civil construction projects, the designer (or design team) is usually intimately involved throughout the entire construction project on a day-to-day basis. The designer has the ability to monitor the construction as it proceeds, review and sign off various stages, and collect samples for testing to confirm that his or her original design assumptions have been achieved in practice. A key feature of major civil construction is that it is built consistent with the intent of the design and the assumptions made in its derivation.

In mining, such a process in which the designer is involved throughout the construction phase is far less common and is rarely carried out in practice. Many mines do not have full-time geotechnical professionals on-site, and even if they did, design is often carried out by third parties that do not reside on-site. Therefore the designs are handed over to mine operators for implementation without the day-to-day involvement of the geotechnical design person or team.

The strata management plan in Australian underground coal mines provides a reasonable framework to illustrate the important elements that ensure the associated residual risks are adequately addressed. Once again, Section 32 of the NSW Coal Mine Health and Safety Regulation 2006 (NSW Government

2006) addresses the terms of provisions that relate to the implementation of geotechnical designs in the mine:

f. the preparation and distribution of support plans that clearly describe the following:
   (i) the type of support,
   (ii) the dimensions of the support,
   (iii) the locations where the varying types of supports are in use,
   (iv) the distance between supports,
   (v) the maximum distance roadways can be advanced before support is installed,
   (vi) the means of roadway support required to be installed, in a manner that they may be readily understood by those required to install the roadway support,
g. other information necessary to enable an employee to install support according to the requirements of the management plan,
h. safe, effective and systematic work methods for the installation and subsequent removal when required, of roadway support (including support in connection with carrying out of roof brushing (or baring down) operations or the recovery of plant),
i. the availability of adequate plant and resources to effectively install or remove roadway support,
j. the monitoring of the stability of roadways after development and support installation,
k. the training of employees in support design principles, support plan interpretation, placement and removal of support, understanding the need for and the importance of the various support systems, and recognition of indicators of change that may affect roadway stability,
l. the recording of strata failures that have the potential to cause serious injury to people,
m. a description of the following features and any special provision made for them
   (i) any multiseam workings
   (ii) any mining that has the potential to cause windblast or rapid stress change
   (iii) any mining at depths of less than 50 metres
   (iv) any coal pillars with a pillar width-to-pillar height ratio of 4:1 or less
n. a prohibition on people entering an underground place that is not supported in accordance with the management plan, unless the person does so for the purpose of erecting support, in which case temporary support must be used,
o. a prohibition on mining taking place unless there is sufficient support for the place in accordance with the requirements of the management plan,
p. a statement that nothing in the management plan is to be read as preventing the installation of more strata support or support installation at more frequent intervals than is required by the management plan.

The excerpt about the strata failure management plan illustrates the number of areas that need to be addressed in order for geotechnical designs to be implemented effectively during mining operations. Specifically, it addresses all of those areas in which a well-founded geotechnical design could become compromised and fail in practice. This includes

- Competency of mining personnel in geotechnical matters related to their job functions;
- Need for effective communication with mining personnel about what is precisely required of each geotechnical design;
- Need for the mine to provide adequate resources and fit-for-purpose mining equipment;
- Need for safe systems of work, including consideration of temporary support;
- Need for formal monitoring of mining excavations over time to ensure ongoing and adequate stability or conversely to identify the need for additional support in a timely manner;
- Recording of failures in order to aid the learning process for future mining;
- Identification of issues that have a high associated safety threat and hence the need for special mining provisions;
- Recognition of the right of all mine workers to install strata support necessary to ensure the safety of their workplace; and
- Need to control access to mine areas to ensure that mine workers are not exposed to unacceptably high geotechnical safety threats.

Many of these subject areas have been in place in the mining industry for many years. However, the focus in the 21st century is to incorporate them into a formalized management process rather than simply an informal set of customs and practices, because a formalized and controlled process will yield better overall outcomes in all areas.

SUMMARY COMMENTS

Because the technical aspects of mining geomechanics are many and complex, geotechnical engineering in mining is a specialist discipline. Although mining engineers specializing in mine planning/scheduling or in operational management roles are not expected to be as knowledgeable about geotechnical matters as the geotechnical engineer, such mining professionals should at least have a working knowledge of the relevant subject areas and be able to ask questions of the geotechnical engineer, which add value to the project.

The commonality between all geotechnical disciplines is process rather than technical content. It does not matter whether it is surface or underground, or coal or metalliferous, optimum control of geotechnical hazards in mining can only be achieved if a comprehensive process is in place, covering exploration activities, geotechnical design, and the safe implementation and management of geotechnical design as part of operations.

As importantly, in any system of enabling legislation (which geotechnical matters almost always are), the judgment as to whether a mine has met its duty-of-care obligations will usually be based on the specifics of the processes and procedures that are utilized rather than specific design methodologies employed. Geomechanics is an earth science, and mining is always exposed to vagaries and uncertainties due to the semichaotic nature of the geotechnical environment. Therefore the outcome is not guaranteed, the focus being on minimizing the potential for unexpected occurrences via the application of best practice geotechnical methods within a prudent risk-based management process.

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CHAPTER 8.2

Soil Mechanics

Ivan A. Contreras and Jedediah D. Greenwood

INTRODUCTION

Soil is a three-phase (solid, liquid, and gaseous) medium consisting of weakly bonded or unbounded material grains and voids filled with water and air. Mineral grains are more strongly bonded in rock than in soil, although in reality there is no sharp boundary between soil and rock. In particular, properties of weak rock bear many similarities to those of soil and are included in the discussion in this chapter. *Soil mechanics* refers to the fundamental principles that govern the behavior of soils as related to the design, construction, and performance of engineering works. The most significant engineering properties of soil and rock are permeability, compressibility, strength, and durability.

Principles of soil mechanics are applied toward the solution of a variety of problems, including the following:

- Seepage or water flow through soils
- Settlement of foundations and fills on compressible soils
- Swelling and slaking of clay shales in foundations
- Stability of slopes and underground openings
- Earth pressures against retaining walls
- Stability of slopes and excavations
- Design of earth dams and retention ponds
- Bearing capacity of foundations and mine pillars on weak soils
- Tunneling in soils, weak rock, or fault zones

SOIL AND ROCK PROPERTIES

Soil and weak rock possess several characteristics that differentiate them from other engineering materials. The following sections discuss the various properties specific to soil and rock in addition to the manner in which these properties are typically determined by way of laboratory and in-situ testing.

Characteristics of Soils

Fine-grained soils interact with water in such a way that there is no resemblance between their dry and wet behavior, and their properties are largely influenced by the physicochemistry of the soil–water system. Fine-grained saturated soils are generally cohesive as a result of the net balance between attractive and repulsive inter-particle forces.

Soft or weak sedimentary rocks such as shale and other *rock–soil transitional materials* such as weathered rock or residual soils not only have strengths that may approach those of highly overconsolidated soils but also have behavior similar to soils, inasmuch as it is influenced by the physicochemistry of the solid–water system. Thus the principles of soil mechanics can be also used to explain the behavior of so-called *weak rock* whose pore water response, volume change behavior (e.g., swelling), and shear strength characteristics are similar to those of soil.

Stiff soils and weak rocks commonly contain fissures and joints that affect permeability, compressibility, strength, and durability of the soil/rock mass. Thus the principles used in rock mechanics for evaluating a discontinuous medium are also applicable to these materials.

The concept of *effective stress* is one of the most important principles of soil mechanics. Both soil compressibility and soil strength are controlled by effective stress. The concept establishes that, for a saturated soil–water system, the effective stress σ' applied to the soil skeleton is equal to the total stress σ externally applied minus the existing pore water pressure u:

$$\sigma' = \sigma - u \tag{8.2-1}$$

The maximum shearing resistance (shear strength) that a soil can develop is a function of the effective stress acting on the soil. The pore water pressure is positive when u is in compression (greater than atmospheric pressure). An increase in the pore water pressure for a given total stress reduces the effective stress and, consequently, the shear strength of the soil. The pore water pressure is negative, on the other hand, when u is in tension (less than atmospheric pressure). In the latter case, which occurs for saturated and partially saturated soils above the groundwater table, the effective stress is greater than or equal to the total stress. The concept of effective stress applies not only to soils but also to rock and other materials with continuous voids formed by pore spaces or joints.

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Saturated materials that are soft enough to undergo significant volume change upon application or removal of pressure (soil and many weak rocks) exhibit time-dependent consolidation or swelling, which is a manifestation of the principle of effective stress. Upon application of an increment of pressure $\Delta\sigma$ to a saturated soil, its volume does not decrease significantly until water is expelled from the pore spaces. Because the soil skeleton is more compressible than water, the pressure increment is initially applied to the water; thus $\Delta u \approx \Delta\sigma$, the excess pore water pressure. With time, as the water is squeezed from the pore spaces, the volume decreases, and the water pressure reduces to its original value ($\Delta u = 0$). The pressure increment is progressively transferred from the water to the soil skeleton, until finally the pressure increment becomes fully an effective pressure, $\Delta\sigma' = \Delta\sigma$. This time-dependent process is termed *consolidation* for the volume decrease resulting from an increase in effective stress, and *swelling* for the volume increase resulting from a decrease in effective stress. Excess pore water pressure dissipates rapidly for permeable materials such as sands, and slowly for low-permeability soils such as clay.

For a large mass of low-permeability soil, the process is so slow that significant changes in volume or water content may not occur over the period of loading, so the material remains undrained, and its strength and stiffness can be assumed to be unchanged from the initial condition before the load increment was applied. Upon application of load and over a long period of time, drainage occurs and the soil consolidates, with corresponding increases in strength and stiffness. Upon unloading, however, the soil swells with time, and strength and stiffness decrease.

In contrast, most sandy soils subjected to static loads are assumed to be loaded slowly enough that significant excess pore water pressure Δu does not build up; therefore the soil is drained, and the pressure increment is applied directly to the soil skeleton throughout the loading increment ($\Delta\sigma' = \Delta\sigma$; $\Delta u = 0$).

Index Properties of Soil and Weak Rock

The spatial distribution and properties of soil, rock, fluids, and air at a site should be investigated as they relate to local geology and information collected from field exploration. The results of field exploration should be used to log the distribution and changes in material properties, to index soil properties by means of in-situ tests, and to collect disturbed (remolded) samples for laboratory index testing or undisturbed samples for elaborate laboratory tests.

Soil and weak rock are classified for engineering purposes in terms of *index properties*, which are by definition significant (i.e., engineering behavior can be inferred from them), simple and quick to measure, reproducible, and easy to express (such as with a numerical value).

Grain size is an important property of all soils. A soil is considered *fine grained* (clay and silt) or *coarse grained* (sand and gravel) if its predominant grain size is smaller or greater, respectively, than the smallest particle diameter that can be identified with the naked eye (equivalent to passing through a No. 200 U.S. standard sieve, or 75 μm).

With little to no clay or silt, coarse-grained soils are generally cohesionless, and their properties remain practically unchanged whether the soil is unsaturated or fully saturated. Coarse-grained soils are indexed in terms of their grain size and grain-size distribution.

The Atterberg limits serve as an index test for measuring the characteristic of fine-grained soils, referred to as *soil plasticity*. The Atterberg limits define the range of water contents (percent ratio between the weight of water and the weight of solids) at which fine-grained soil passes from a liquid state (liquid limit) through a plastic state (plastic limit) and finally into a solid state (Atterberg 1911). The *shrinkage limit* is defined as the water content at which fully saturated soil cake stops decreasing in volume. Any further moisture loss simply starts emptying the soil pores. High-plasticity soils, having a large plastic range, are subject to high swelling and shrinkage.

Both grain size and plasticity form the basis for the categories in the Unified Soil Classification System shown in Table 8.2-1. These index tests are performed on disturbed samples, which do not retain the structure and density of the soil in situ.

Field Exploration

In order to collect disturbed samples for index-property testing and to evaluate in-situ soil properties (i.e., permeability, compressibility, and strength), it is necessary to perform a field exploration program. Index properties are determined on disturbed samples, because the properties are independent of sample disturbance. In contrast, in-situ properties are determined on relatively undisturbed samples in the laboratory or in the field, because the properties are highly dependent on sample disturbance.

The field exploration program generally includes soil borings with in-situ tests including but not limited to the following:

- Standard penetration test
- Cone penetration test
- Marchetti dilatometer test
- Pressuremeter test
- Field vane test

Standard Penetration Test

The standard penetration test (SPT) involves driving a 2-in.-(50.8-mm-) diameter split-spoon sampler into the ground using a 140-lb (63.5-kg) hammer dropped freely from a height of 30 in. (762 mm). The sampler is attached to drill rods and lowered to the bottom of the clean drilled hole. After the sampler reaches the bottom of the hole, the number of hammer blows is counted until it achieves three successive penetrations of 6 in. (150 mm). The number of blows to advance the sampler the first 6 in. (150 mm) is disregarded because of likely disturbance at the bottom of the hole. The number of blows "*N*" to advance the sampler for the second and third 6-in. (150-mm) increments into the ground is the standard penetration resistance of the soil.

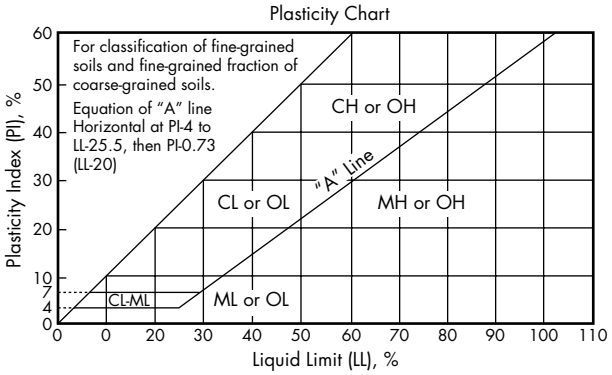
During the SPT, the amount of energy actually transmitted to the drill rods by the hammer is influenced by several factors. The ratio of the energy actually transmitted to the rods to that theoretically delivered by the hammer is commonly known as the *energy ratio* or *efficiency* of the hammer system. In general, most correlations are based on N values corresponding to an efficiency of 60%, designated N_{60}. The energy correction factor C_E is given by the equation

$$C_E = \mathrm{ER}/60 \tag{8.2-2}$$

where ER is the energy ratio. The energy transmitted to the drill rods can be determined by direct measurement in the field

Table 8.2-1 Unified Soil Classification System

Criteria for Assigning Group Symbols and Names Using Laboratory Tests[a]			Soil Classification Group Symbol, Name[b]
Coarse-grained soils: >50% retained on No. 200 sieve			
Gravels: >50% of coarse fraction retained on No. 4 sieve	Clean gravels: <5% fines[c]	$C_u \geq 4$ and $1 \leq C_c \leq 3$[e]	GW, Well-graded gravel[f]
		$C_u < 4$ and/or $1 > C_c > 3$[e]	GP, Poorly graded gravel[f]
	Gravels with fines: >12% fines[c]	Fines classify as ML or MH	GM, Silty gravel[f,g,h]
		Fines classify as CL or CH	GC, Clayey gravel[f,g,h]
Sands: ≥50% of coarse fraction passes No. 4 sieve	Clean sands: <5% fines[d]	$C_u \geq 6$ and $1 \leq C_c \leq 3$[e]	SW, Well-graded sand[i]
		$C_u < 6$ and/or $1 > C_c > 3$[e]	SP, Poorly graded sand[i]
	Sands with fines: >12% fines[d]	Fines classify as ML or MH	SM, Silty sand[g,h,i]
		Fines classify as CL or CH	SC, Clayey sand[g,h,i]
Fine-grained soils: ≥50% passes No. 200 sieve			
Silts and clays: liquid limit <50	Inorganic	PI > 7 and plots on or above "A" line[j]	CL, Lean clay[k,l,m]
		PI < 4 or plots below "A" line[j]	ML, Silt[k,l,m]
	Organic	$\frac{\text{Liquid limit (oven dried)}}{\text{Liquid limit (not dried)}} < 0.75$	OL, Organic clay[k,l,m,n] or silt[k,l,m,o]
Silts and clays: liquid limit ≥50	Inorganic	PI plots on or above "A" line	CH, Fat clay[k,l,m]
		PI plots below "A" line	MH, Elastic silt[k,l,m]
	Organic	$\frac{\text{Liquid limit (oven dried)}}{\text{Liquid limit (not dried)}} < 0.75$	OH, Organic clay[k,l,m,p] or silt[k,l,m,q]
Highly organic soils		Primarily organic matter, dark in color, and organic odor	Pt, Peat



Source: Adapted from ASTM D2487-00.

a. Based on the material passing a 3-in. (75-mm) sieve.

b. If field sample contained cobbles, boulders, or both, add "with cobble, boulders, or both" to group name.

c,d. Gravels or sands with 5% to 12% fines require dual symbols such as "GW-GM well-graded gravel with silt" or "SP-SC poorly graded sand with clay."

e. $C_u = \frac{D_{60}}{D_{10}}, C_c = \frac{(D_{30})^2}{D_{10} \times D_{60}}$

f. If soil contains ≥15% sand, add "with sand" to group name.

g. If fines classify as CL-ML, use dual symbol GC-GM or SC-SM.

h. If fines are organic, add "with organic fines" to group name.

i. If soil contains ≥15% gravel, add "with gravel" to group name.

j. If Atterberg limits plot in hatched area, soil is a CL-ML, silty clay.

k. If soil contains 15% to 29% plus No. 200, add "with sand" or "with gravel," whichever is predominant.

l. If soil contains ≥30% plus No. 200, predominantly sand, add "sandy" to group name.

m. If soil contains ≥30% plus No. 200, predominantly gravel, add "gravelly" to group name.

n. PI ≥4 and plots on or above "A" line.

o. PI <4 or plots below "A" line.

p. PI plots on or above "A" line.

q. PI plots below "A" line.

Table 8.2-2 Correction factors for the standard penetration test

Factor	Term	Equipment Variable	Correction
Energy	C_E	Donut hammer	0.5–1.0
		Safety hammer	0.7–1.2
		Automatic-trip donut-type hammer	0.8–1.3
Borehole diameter	C_B	2.5–4.5 in. (65–115 mm)	1.0
		6 in. (150 mm)	1.05
		7.9 in. (200 mm)	1.15
Rod length	C_R	<10 ft (3 m)	0.75
		10–13 ft (3–4 m)	0.80
		13–20 ft (4–6 m)	0.85
		20–32 ft (6–10 m)	0.95
		32–100 ft (10–30 m)	1.0
Sampling method	C_S	Standard sampler	1.0
		Sampler without liners	1.1–1.3

Source: Adapted from Skempton 1986; Robertson and Wride 1998.

Table 8.2-3 Relationship between relative density of sand and strength of clay

Relative Density of Sand			Strength of Clay		
N_{60}*	Relative Density, %	Descriptive Term	N_{60}*	Unconfined Compressive, tons/ft²	Consistency
0–4	0–15	Very loose	<2	<0.25	Very soft
4–10	15–35	Loose	2–4	0.25–0.50	Soft
10–30	35–65	Medium	4–8	0.50–1.00	Medium
30–50	65–85	Dense	8–15	1.00–2.00	Stiff
>50	85–100	Very dense	15–30	2.00–4.00	Very stiff
			>30	>4.00	Hard

Source: Cording and Cepeda-Díaz 1992.
*Standard penetration resistance corresponding to an efficiency of 60% blows/ft.

using dynamic pile testing equipment. Alternatively, energy corrections have been summarized based on factors such as hammer type, borehole diameter, rod length, and whether or not the sampler uses a liner. Correction factors corresponding to these factors are shown in Table 8.2-2.

The value of N corresponding to 60% efficiency, N_{60}, is given by the equation

$$N_{60} = N\, C_E\, C_B\, C_R\, C_S \tag{8.2-3}$$

where

N = standard penetration resistance measured in the field
C_E = energy correction factor
C_B = correction factor for borehole diameter
C_R = correction factor for rod length
C_S = correction for sampling method (with or without liners)

N_{60} indicates the relative density of in-situ granular soils and the consistency of fine-grained soils. Table 8.2-3 shows the relationship between the relative density of sand and the strength (consistency) of clay.

The relationship between N_{60} and unconfined compressive strength q_u in the table is a crude estimate of clay strength. The significant scatter in the values of unconfined compressive strength for a given N_{60} is attributed to the disturbance induced by the SPT sampler and procedure. As a result, the relationships in the table should be used with caution.

Similarly, the approximate relationship between N_{60} and the relative density of sand should be used with caution. When the sand is located below the water table and the water inside the casing is allowed to fall below the piezometric level, the sand at the test location may become quick and transform into a loose state, resulting in a low value of N_{60}. On the other hand, boulders or cobbles larger than the diameter of the spoon may give high values of N_{60}.

Because the standard penetration resistance increases with increasing effective overburden stress, an overburden-stress correction factor is generally applied (Seed and Idriss 1982). This correction factor, C_N, is typically applied for granular soils when performing liquefaction assessments and is calculated using the equation

$$C_N = (100\ \text{kPa}/\sigma'_{vo})^{0.5} \tag{8.2-4}$$

where σ'_{vo} is the effective overburden stress in kilopascals at the considered depth. C_N should not exceed 1.7.

The standard penetration-blow count corresponding to a combined efficiency of 60% and a normalized effective stress of 100 kPa is given by the equation

$$(N_1)_{60} = N_{60}\, C_N \tag{8.2-5}$$

This blow count is used in several SPT correlations and in liquefaction assessments.

Cone Penetration Test

The cone penetration test (CPT) has been extensively used in recent years to provide in-situ geotechnical information relatively quickly and inexpensively. It has gained wide acceptance in geotechnical practice since the late 1980s, and additional capabilities continue to be developed. CPT results are less operator-dependent than are SPT results, which makes the CPT more attractive because of its repeatability and the possibility of good correlations between test data and geotechnical parameters. However, the CPT has limitations in gravelly soils, cemented soils, and bedrock, which are unlikely to be penetrated with the equipment.

The test involves pushing a conical tip with a 60° apex angle into the ground at a constant penetration rate of 0.8 in./s (20 mm/s). During penetration, measurements are made of the cone-tip resistance (q_c), side friction of the cylindrical shaft behind the cone (f_s), and pore water pressure (u_2) generated by penetration. These measurements, collected virtually on a continuous basis, are used to infer lithological profiles based on interpreted soil-behavior types. They are used to assess liquefaction, slope stability, foundations, and groundwater flow characteristics.

The cone is also used to perform dissipation tests. A dissipation test is performed at the desired depth by stopping cone penetration and measuring the change in pore water pressure with time. The rate of pore water pressure dissipation depends on the horizontal coefficient of consolidation c_h, which in turn depends on the compressibility and permeability of the soil.

Cones equipped with geophones are commonly used to measure arrival times of shear and compression waves

in addition to the three basic measured parameters. These are commonly referred to as *seismic cones*. The shear-wave source usually consists of a beam pressed against the ground and a hammer with an electronic trigger. Shear waves are generated at the ground surface by hitting the beam ends with a hammer in the direction of the long axis. During a pause in penetration, typically at 3.3-ft (1-m) intervals, shear waves are generated at the ground surface and the arrival time of the shear waves to the geophone is measured. The shear wave is used to estimate shear deformation modulus (small strain) and to perform liquefaction assessments. Compression waves are generated and interpreted in a similar manner.

Some CPT equipment is outfitted with devices to obtain soil and water samples. The device is generally pushed into the ground using an offset sounding. Penetration is then stopped at the desired sampling depth. Soil and water samples are taken only at discrete locations.

New CPT equipment includes additional sensors for subsurface exploration. These include sensors to measure lateral stress, electrical resistivity, heat flow, radio isotopes, and density. They also include sensors for environmental applications such as temperature, conductivity, pH, and laser-induced fluorescence.

Dilatometer Test

The Marchetti, or flat-blade, dilatometer test (DMT) is another device often used to measure in-situ geotechnical properties of soils. It is primarily used to determine stiffness parameters and undrained shear strength of cohesive soils, but also has applicability for granular materials and in determining in-situ horizontal stresses. Soil stratigraphy and depth to the water table can also be determined from the test. Similar to the CPT, bedrock and gravelly or cemented soils may cause problems for penetration.

The test device consists of a flat blade 0.5 in. (14 mm) thick, 3.7 in. (95 mm) wide, and 8.7 in. (220 mm) long. A flexible stainless-steel membrane 2.4 in. (60 mm) in diameter is located in the center of one side of the blade. The blade is pushed into the ground with a CPT rig or drill rig and stopped at regular depth intervals (usually 7.9 in. [20 cm]), after which the membrane is inflated with nitrogen gas and then deflated. The operator measures and records the pressure required to produce expansion of the membrane into the soil at two preset deflections. The first reading is taken when the membrane starts to lift off; the second is taken after a deflection of 0.4 in. (1 mm). An optional third reading is taken when the membrane reaches the original lift-off position to measure pore water pressure.

Pressuremeter Test

The pressuremeter test (PMT) is used to determine in-situ stress–strain characteristics in soil and weak rock. The PMT is typically used in materials that are too stiff for the DMT. The PMT provides information about the stress–strain properties and strength of the soil or rock by expanding the walls of a drill hole while the radial pressure and corresponding deformation are measured.

The test device consists of an assembly of three cells with flexible walls. The upper and lower cells are guard cells; the middle cell is used as a measuring cell. The assembly is lowered into a drill hole similar in size to the pressuremeter apparatus and the cells are then inflated. The measuring cell is inflated incrementally using water pressure. The water pressure is measured to indicate the radial pressure against the soil or rock, and the volume of water is measured to indicate radial displacement during the increment. The guard cells are also inflated to ensure a nearly cylindrical two-dimensional (2-D) state of stress with the purpose of reducing the end effects on the measuring cell. Some corrections are applied to account for expansion of the measuring tubing under pressure.

The measured cell pressure and cell volume are used to generate a stress–strain curve for the in-situ soils. The slope of the pseudo-elastic portion of the curve is used to compute the pressuremeter modulus, which is a measure of the stiffness of the soil or weak rock.

Field Vane Test

The field vane (FV) test is widely used to measure the undrained shear strength of soft clays. The test device consists of a four-bladed rectangular vane fastened to the bottom of a vertical rod. The standard vane has a height/diameter ratio of 2:1, with a height of either 4.3 or 5.1 in. (110 or 130 mm) and a diameter of either 2.2 or 2.6 in. (55 or 65 mm). Some models have a protective casing or sheath for the vane and others do not. The vane is pushed into the ground to the desired test depth and rotated at a rate of 0.1°/s by a crank handle while the torque is measured. The vane is then advanced to the next depth and the process is repeated. Tests are typically conducted at intervals of 1.6 ft (0.5 m). The shearing resistance is computed from the torque and the dimensions of the vane. If the vane is rotated rapidly several revolutions, the soil remolds and the remolded strength can be computed.

Table 8.2-4 Coefficient of permeability

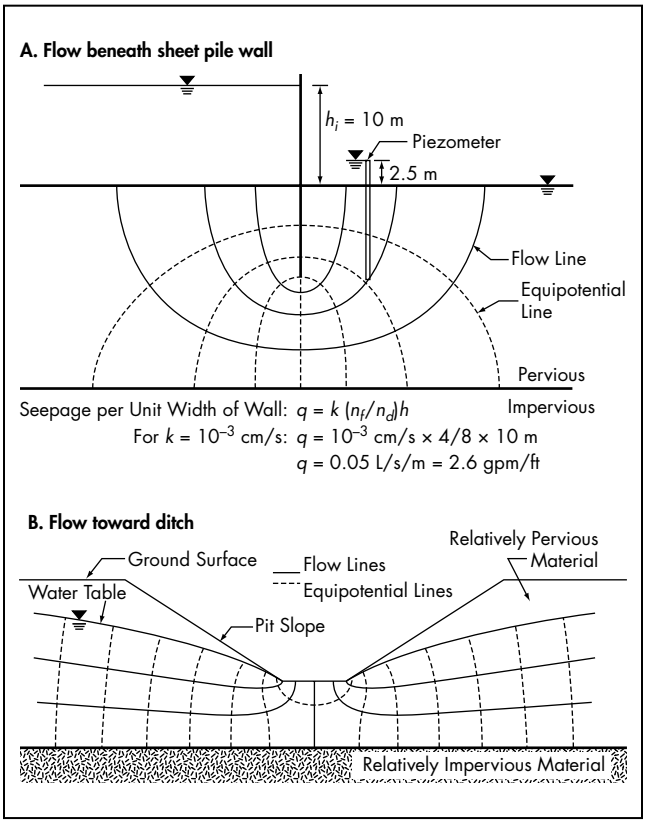
Material	k, cm/s
Clean gravel	1 to 10^2
Coarse to medium sand	10^{-2} to 1
Fine sand	10^{-4} to 10^{-2}
Silt	10^{-6} to 10^{-4}
Natural clays	10^{-8} to 10^{-6}
Highly overconsolidated clays and shales	10^{-12} to 10^{-8}

Source: Cording and Cepeda-Díaz 1992.

Permeability

Permeability k, expressed in units of length per time, indicates the discharge velocity with which water under a unit hydraulic gradient flows through soil and rock. Procedures are available for measuring permeability in the field and in the laboratory. Results depend on the direction of the water flow during the test because soil and rock generally have direction-dependent permeability or anisotropy. In a bedded deposit, horizontal permeability k_h usually exceeds vertical permeability k_v. For example, for homogeneous soft clay, typical ranges for horizontal permeability are $k_h = 1.0–1.5\ k_v$; for varved clay, typical ranges are $k_h = 1.5–5.0\ k_v$. Much greater ratios of k_h/k_v apply to bedded deposits having alternating layers of clay and sand.

The coefficient of permeability k is a function of the size, shape (tortuosity), and number of pores or joints (channels) in the soil/rock mass. Table 8.2-4 presents typical values of k for various soil and weak rock materials. For a given soil or weak rock, k decreases with decreasing values of void ratio e (volume of voids per volume of solids) and therefore with increasing effective stress. The slope of the e–log k relationship is



Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-1 Flow nets for medium soil with homogeneous, isotropic permeability

generally referred to as $C_k = \Delta e/\Delta \log k$. For soft clays, C_k = 0.5–2.0; for shales, $C_k < 0.5$.

For granular soils, Hazen (1892) performed a number of experiments from which he established an empirical formula to estimate the permeability of saturated filter sands. The relationship is $k = C_H(d_{10})^2$ where k = coefficient of permeability (m/s), C_H = Hazen empirical coefficient, and d_{10} = particle size from a sieve analysis for which 10% of the particles are finer (in units of millimeters). Figure 8.2-1 shows permeability and grain-size data from Louden (1952). Caution should be exercised in using the Hazen empirical correlation because of its simplicity and because the C_H coefficient (average value of 0.01) can vary by orders of magnitude. Other more complex relationships (such as the Kozeny–Carman semi-empirical/semi-theoretical formula [Kozeny 1927; Carman 1938, 1956]) have been proposed that are more accurate, but require more input parameters.

Permeability is particularly sensitive to heterogeneities such as stratification, fissures, and joints. Thus their presence determines either the minimum specimen size required in the laboratory for a representative sample or the need for in-situ testing to obtain valid measurements. Disturbance of the soil during boring or sampling can also affect permeability measurements. In many cases, laboratory-measured permeability is not comparable to in-situ permeability. In the field, the local permeability surrounding a borehole can be assessed by a water pressure test in which a quantity of water is pumped into or extracted from a section of a borehole under a given pressure. More comprehensive results, indicative of the performance of a larger portion of an aquifer, are obtained by pump tests in which piezometric levels are monitored in several borings, with time, as a nearby well is pumped.

Darcy's empirical equation (Darcy 1856), initially established for the flow of water through coarse-grained soils, has also proven to be valid for water flow through fine-grained soils, including heavily overconsolidated clays and shales (Mesri and Cepeda-Díaz 1987). Darcy's law states that the discharge velocity v_s in a direction s is equal to the coefficient of permeability in the s direction k_s times the hydraulic gradient i:

$$v_s = k_s\, i \tag{8.2-6}$$

where the dimensionless parameter i is equal to the change of total hydraulic head Δh in the distance interval Δs along the macroscopic or apparent flow path s ($i = \Delta h/\Delta s$). Therefore, the time rate of flow q (volume flow rate) through an area A perpendicular to the direction of the flow is given by

$$q = k_s\, iA \tag{8.2-7}$$

which is used to directly evaluate one-dimensional (1-D) steady-state flow.

A modified version of the previous equation is used to calculate the quantity of flow for 2-D steady-state flow through soil. The solution to the mathematical problem can be obtained graphically in the form of a scaled flow net (Cedergren 1967). For a homogeneous, isotropic medium ($k_h = k_v$), the flow net consists of two mutually orthogonal families of curves (flow lines and equipotential lines) forming curvilinear "square" cells within the boundaries of the problem, thus also forming both a number of flow channels n_f and a number of equipotential drops n_d for a given head loss h_l (Figure 8.2-1). Since each flow channel carries the same flow rate as the others and continuity of flow (conservation of mass) must be satisfied, fluid motion can be described at the individual-square-cell level where the hydraulic gradient i times the discharge area of the channel is equal to h_l/n_d per unit of width and thus iA (per unit of width for the entire flow net) = $(h_l \cdot n_f)/n_d$. For 2-D steady-state flow, therefore, the time rate of flow ($q = k_s iA$) is given by

$$q = k_s(n_f/n_d)h_l \tag{8.2-8}$$

per unit of width perpendicular to the plane of the flow net. A flow-net solution enables the hydraulic gradient and the pore water pressure to be calculated at any location within the hydraulic boundaries considered. Thus, pore water pressures developed as a result of water seepage can be determined along potential surfaces of sliding in order to evaluate the stability of a soil mass.

In materials such as jointed rock or bedded soils, permeabilities are not uniform. As a result, the flow net is distorted, and the flow lines and equipotential lines do not form squares.

Computer modeling programs have simplified the computation of groundwater flow regimes, and many programs allow computed pore water pressures to be imported into slope-stability analyses.

Compressibility and Volume Change Characteristics

The compressibility or expansibility of a soil or weak rock can be separated into several components:

- Initial "elastic" deformation
- Time-dependent hydrodynamic volume change (primary consolidation or swelling)

- Time-dependent volume change (secondary consolidation or swelling) and/or time-dependent deformation (creep)

Initial "Elastic" Deformation

Immediately after the application or removal of load, a fine-grained saturated soil deforms without any change in volume or water content, referred to as the *undrained condition*. In an unsaturated soil or a coarse-grained soil, the initial elastic deformations may include some volume changes and distortion.

Assuming that the material has not yet failed, the amount of strain for a given pressure increment can be estimated using a modulus of deformation that is equivalent to a modulus of elasticity E, but represents a tangent or secant to a stress–strain curve that is usually nonlinear. The deformation modulus E_u of a sample is determined in the laboratory from unconfined or triaxial compression tests in which the specimen is subjected to increasing axial stress σ_1 while the all-round confining pressure σ_3 is kept constant. The slope of the curve relating the stress difference $\sigma_1 - \sigma_3$ to the axial strain ε represents the modulus of the sample, and $(\sigma_1 - \sigma_3)_{max}$ is the compressive strength. The unconfined compressive strength is generally referred to as q_u. For most soils and weak rocks, E_u = (50 to 300) q_u.

For the modulus of deformation, values measured in-situ are expected to differ from those measured in the laboratory. For most soils, sample disturbance reduces soil stiffness significantly, and thus laboratory measurements are generally lower than in-situ measurements. In contrast, for soil and rock masses that contain fissures or joints, a laboratory sample usually does not contain a representative number of fissures or joints, and thus laboratory measurements are generally higher than in-situ measurements.

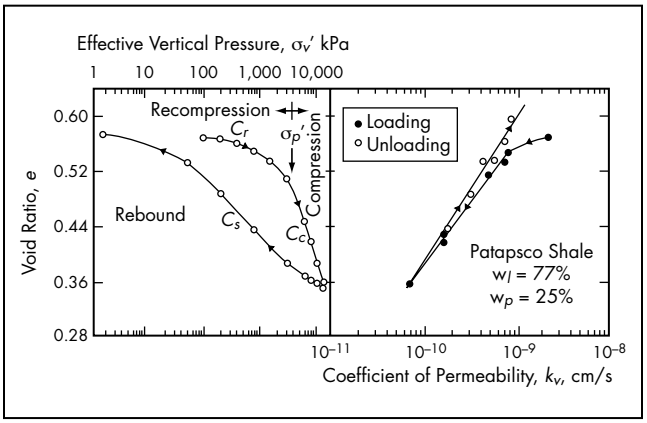
The greater the stiffness of the soil or rock, the more significant is the initial deformation with respect to the overall deformation experienced with time by the soil or rock mass.

Primary Consolidation or Swelling

In a saturated fine-grained soil, the initial elastic deformations described previously occur with very little volume change in the soil matrix. With time, as the load remains in place, water is squeezed out of the soil matrix and the soil volume decreases. This process is called primary consolidation. In soft clays, primary consolidation is responsible for most of the settlement that the clay undergoes upon application of a load. Compression that occurs with time, after the effective stress σ' has reached its final value, under either isotropic loading or 1-D deformation conditions, is called secondary consolidation and is a creep phenomenon.

Upon unloading, as water is sucked into the soil, soil volume increases and effective stress decreases. This time-dependent process is called primary swelling. Swelling that occurs with time after σ' has reached its final value, under either isotropic unloading or 1-D deformation conditions, is called secondary swelling.

The magnitude of settlement or heave during primary consolidation or swelling is determined for fine-grained soils in the laboratory by means of the 1-D consolidation test, in which a sample is restrained so that lateral strains are zero as vertical load is applied. As each load decrement or increment is applied, water is allowed to flow into or out of the soil through porous stones at the top and bottom of the sample, and the vertical deformation of the specimen is measured with time. The change in the void ratio of the sample is proportional to the vertical displacement. The end of primary consolidation or swelling, at time t_p, occurs when the excess pore pressure that is generated upon application of each load increment has completely dissipated, which can be determined graphically.



Source: Cepeda-Díaz 1987.

Figure 8.2-2 Typical plots of void ratio for soil and weak rock as a function of effective vertical pressure (stress-volume) and as a function of coefficient of permeability

For a homogeneous soil layer of thickness H and initial void ratio e_o, settlement or heave ΔH is given by

$$\Delta H = \{(e - e_o)/(1 + e_o)\}H \tag{8.2-9}$$

where e is the void ratio at any particular vertical stress σ'_v and/or time t.

The magnitude of settlement or heave at the end of primary consolidation or swelling is generally obtained for fine-grained soils from a plot of the void ratio e versus the logarithm of the effective stress σ'_v applied to the sample. Figure 8.2-2 shows the typical e–log σ'_v relationship. For a heavily overconsolidated clay, it consists of three parts: recompression, compression, and rebound (Patapsco shale). The preconsolidation pressure σ'_p = 38 tons/ft^2 (40 MPa). The slope of the curve, $\Delta e/\Delta \log \sigma'_v$, for each distinct portion is identified as the recompression index C_r, compression index C_c, and swelling index C_s. In the recompression range, the corresponding settlement is significantly smaller than that resulting when σ'_{vf} (final σ'_v) is in the compression range. Loading of soft clays, whose preconsolidation pressure σ'_p is relatively small and compression index C_c is relatively large can result in large settlements when the applied pressures exceed the preconsolidation pressure. The magnitude of heave can be estimated using the rebound curve whose slope is C_s. Figure 8.2-2 also shows direct measured permeability data at various σ'_v and the corresponding e in the form of an e–log k_v plot for parallel examination with the e–log σ'_v plot.

The time rate of settlement or heave, which is controlled by the rate at which pore water can be expelled (or taken in), is a function of the permeability k and the compressibility (or swellability) of the soil, $a_v = \Delta e/\Delta\sigma'_v$, as expressed by the coefficient of consolidation (or swelling) $c_v = k(1 + e)/a_v\gamma_w$, according to the Terzaghi theory of consolidation (Terzaghi et al. 1996). The mathematical solution of the theory provides the relationship between the average degree of consolidation (or swelling) U in % = $\Delta H/\Delta H_p$ and a dimensionless time factor T whose numerical value is equal to $c_v t/H^2$, where t is real

Table 8.2-5 Values of C_α/C_c for geotechnical materials

Material	C_α/C_c, ± 0.01
Granular soils including rock filling	0.02
Shale and mudstone	0.03
Inorganic clays and silts	0.04
Organic clays and silts	0.05
Peat and muskeg	0.06

Source: Terzaghi et al. 1996.

time and H is the maximum drainage distance. Therefore, after c_v is evaluated (usually by fitting Terzaghi's theory to the observed deformation time data in the 1-D consolidation test), the settlement ΔH at any time $t < t_p$ during the consolidation process can be predicted. For example, according to the Terzaghi theory of consolidation (or swelling), the time factor for U = 95% is T_{95} = 1.12. Therefore, the (real) time for a 95% average degree of consolidation (or swelling) to occur is $t_{95} = 1.12\ H^2/c_{cv}$. For a maximum drainage distance H = 20 ft (6.1 m) and a coefficient of consolidation (or swelling) of 10 ft^2/yr (0.93 m^2/s), the time for end-of-primary to occur t_p is approximately $t_{95} = 1.12\ (20\ \text{ft})^2/(10\ \text{ft}^2/\text{yr}) = 45$ years.

In soils containing lenses of sand with a clay or silt, the effective drainage distance is difficult to determine, and hence the time to achieve consolidation or swelling is difficult to estimate. One of the most effective means is to monitor the settlement of test fills.

The horizontal coefficient of consolidation c_h can be estimated in situ using CPT dissipation test results. During a pause in penetration, the decay of measured pore water pressure is recorded and an empirical correlation between c_h and t_{50} (time to 50% consolidation) has been proposed by Robertson et al. (1992). The values of c_h and c_v for settlement calculations can be related by means of the anisotropy relationship.

Secondary Consolidation or Swelling

The slope of the observed e–log t curve for $t > t_p$ is called the secondary compression index C_α for consolidation and the secondary swelling index $C_{\alpha s}$ for swelling. Since C_α/C_c is constant for any one soil, including shales, its value can be used in combination with the e–log σ'_v relationship to compute the secondary settlement at any $t > t_p$ (Mesri and Godlewski 1977). Table 8.2-5 shows typical values of the C_α/C_c ratio for various geotechnical materials. The relationship between $C_{\alpha s}$ and the OCR (final overconsolidation ratio after swelling $\sigma'_{max}/\sigma'_{vf}$) can be also obtained and used to compute the secondary swelling for any time $t > t_p$.

Shear Strength

The shear strength s or shearing resistance of a soil or rock is a function of the normal effective stress on the shear plane at failure σ'_{nf}. Thus s is controlled by the existing normal effective stress before shear ($\sigma'_n = \sigma_n - u$) and the excess pore water pressure induced during shearing up to failure u_{sf}, $\sigma'_{nf} = \sigma'_n - u_{sf}$. For $u_f = u + u_{sf}$, σ'_{nf} can be expressed as

$$\sigma'_{nf} = \sigma - u_f \tag{8.2-10}$$

During shearing, as a result of particle rearrangement, loose coarse-grained cohesionless soils and normally consolidated clays tend to compress, whereas dense cohesionless soils and overconsolidated clays and shales tend to increase in volume (dilate). Excess pore water pressures develop when water cannot move in or out of the pores fast enough to accommodate the volume changes induced during shearing. In this case, the soil is sheared under so-called undrained conditions.

Effective stress analyses can be used to evaluate the strength of a soil for either drained or undrained conditions. However, to use an effective stress analysis in the undrained case, the pore water pressures generated at failure must be known. This information is difficult to obtain or predict. Therefore, for soft clays subjected to short-term loading, an analysis for the undrained case is used, in which the undrained shear strength s_u is determined from unconfined compression tests, undrained triaxial tests, or in-situ vane shear tests.

Sand

With the exception of dynamic loading, clean coarse-grained cohesionless soils can be treated as shearing under drained conditions as a result of their high permeability. Their shear strength can be expressed as

$$s = \sigma'_{nf} \tan \phi' \tag{8.2-11}$$

where ϕ' is the drained friction angle (or friction angle in terms of σ' at failure). For granular soils, ϕ' is primarily a function of the relative density of the soil. Thus for loose sand $\phi' \approx 28°$ and for loose sand $\phi' \approx 44°$.

Loose saturated sands subject to dynamic loads such as earthquakes are apt to develop positive pore water pressures that reduce the effective stresses to zero and liquefy the sand.

Fine-Grained Soils

For fine-grained soils, their relatively low permeability is such that significant buildup of excess pore water pressure is expected to occur during the loading and unloading operations associated with construction. In a short period of time, the excess pore water pressures that are generated do not dissipate, and the clay does not change in volume or water content. Thus soft clays can be treated as undrained, with an undrained shear strength s_u that does not change from its original value. This condition is described later in the "Undrained Shear Strength" section.

Use of the undrained shear strength is simple, with application to several significant engineering problems. However, many failures have resulted from its inappropriate use, particularly in situations where negative pore water pressures were generated initially and then dissipated, resulting in reduced shear strength during the period in which the undrained shear strength was assumed to be applicable. This is most likely to occur for stiff, overconsolidated clays. In addition, the presence of sand seams or fissures in a clay can cause more rapid dissipation of negative pore water pressures than would be otherwise anticipated, leading to failure earlier during the construction period than expected. Thus, when using the undrained shear strength, it is important to recognize those conditions for which its use is unconservative.

For these reasons, it is always good practice to investigate the stability of fine-grained soils for both the short-term period (during and after the end of construction) when undrained conditions prevail and the long-term period when the material is drained.

Shear Strength

The drained shear strength that a given clay or shale can actually mobilize (peak, fully softened, or residual strength) depends on whether the in-situ material is intact, is fissured, or contains discontinuities (such as bedding planes or shear planes) along which significant movements have occurred or will occur under the displacements imposed on the soil mass.

Peak shear strength. If in-situ fine-grained material is relatively intact, the peak shear strength is expected to be mobilized, and thus the failure envelope for the effective stress range considered can be characterized in terms of the effective-strength parameters c' and ϕ' by

$$s = c' + \sigma'_{nf} \tan \phi' \qquad (8.2\text{-}12)$$

where c' is the cohesion intercept. The greater the magnitude of the preconsolidation pressure σ'_p , the greater is the cohesion intercept c'. For normally consolidated clays, $c' = 0$. Values of ϕ' are typically in the range 22° ± 10° and reflect the capability of a random arrangement of clay particles (of particular size and shape) to interact and interlock with each other.

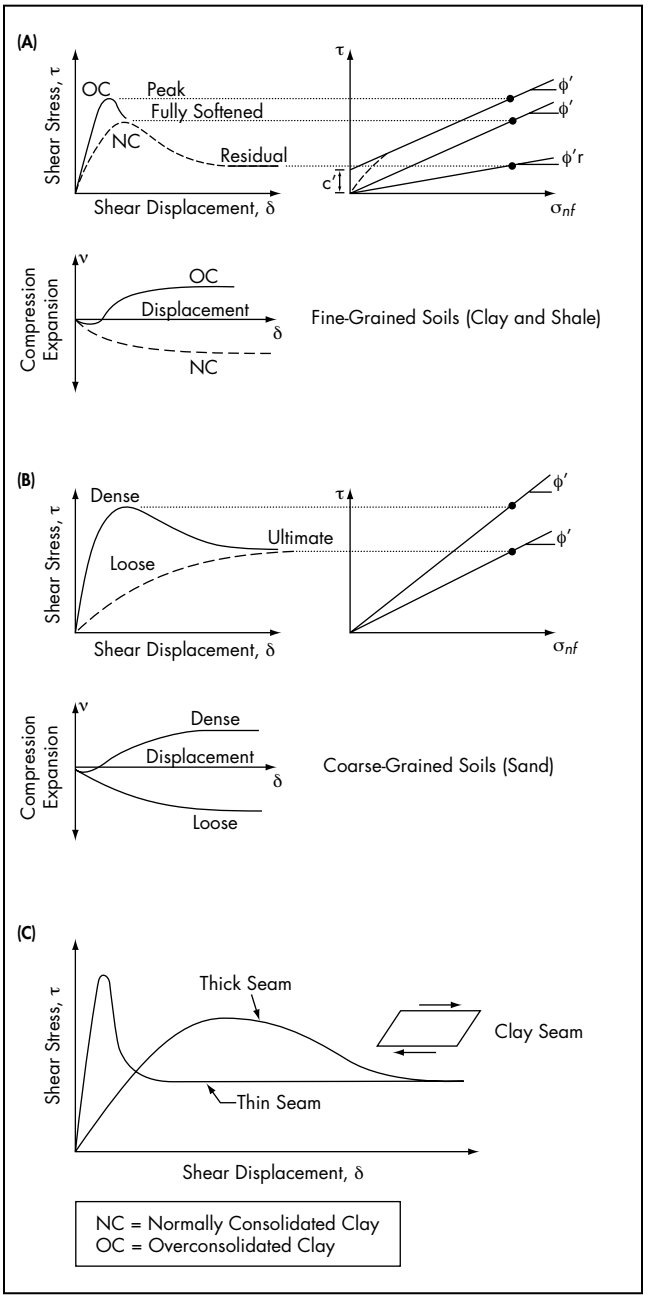
For overconsolidated clays, the typical shear stress–strain and strength curves shown in Figure 8.2-3A show that the peak shearing resistance is mobilized with no major structural rearrangement. Near the maximum shearing resistance, the changes in volumetric strain v are small and mainly in compression.

Fully softened shear strength. Additional shear displacement beyond the peak (intact) shear strength is accompanied by softening (volume increase as the disaggregated clay takes up water) and corresponding decrease in shearing resistance as c' decreases. By the time the softening process has been completed, only the fully softened shear strength is mobilized ($c' \approx 0$), and the material behaves as a normally consolidated clay. Long-term stability of cuts in overconsolidated, fissured clays has been determined to be related to the fully softened shear strength.

Skempton (1977) suggested that, for a first-time slide, the shear strength of a stiff fissured clay at failure can be approximated by the fully softened strength. After reviewing the failure of Carsington Dam (United Kingdom) and slides in highway cuttings and embankments, Skempton and Vaughan (1995) stated that "residual strength can indeed play a part in first-time slides in clay fills and cuttings." Stark and Eid (1997) concluded that residual shear strength can influence the stability or behavior of slopes that have not undergone prior sliding. They indicated that mobilized shear strength can be estimated as an average between the fully softened and residual. More recently, Mesri and Shaien (2003) showed that, for many first-time slope failures, part of the slip surface is at the residual condition.

The typical shear stress–strain and strength curves for a given sand in both dense and loose states (Figure 8.2-3B) show to what extent the behavior of dense and loose granular soils is similar to that of overconsolidated and normally consolidated clays, respectively. For nonplaty granular soils, there is no possibility of particle alignment during shear, and thus the ultimate shear strength of the dense sand (which loosens after reaching its peak strength) corresponds to the maximum shear strength of the loose sand (which densifies during shear).

Figure 8.2-4 shows plots of drained fully softened secant ϕ' as a function of liquid limit, clay-size fraction, and effective normal stress for clays and shales (Stark and Eid 1997).

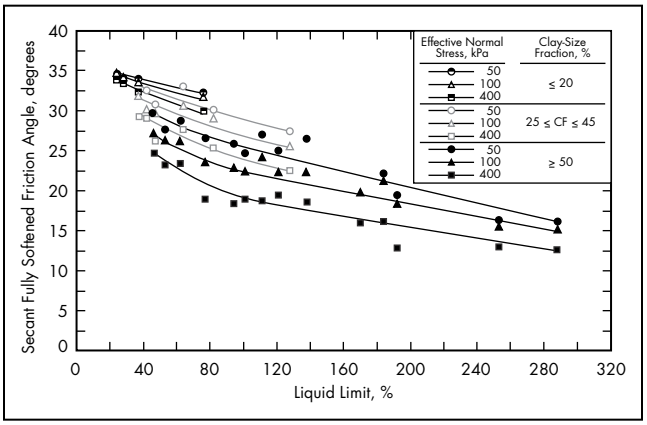


Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-3 Typical plots of stress vs. displacement (strain) and strength characteristics for soils and weak rock

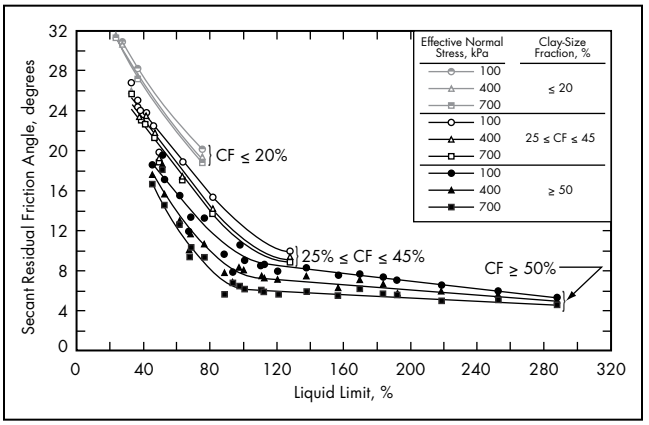
Higher liquid limits and higher clay-size fractions yield lower shear strengths, with ϕ' ranging from 34° (liquid limit $w_l \leq 25\%$ and clay-size fraction CF ≤ 20%) to about 15° (liquid limit $w_l \geq 280\%$ and clay-size fraction CF ≥ 50%).

Residual shear strength. Additional shear displacement, if concentrated along a narrow shear zone, causes further decrease in shearing resistance, below that of the fully softened case, as the plate-shaped particles in the relatively narrow shear zone increasingly align in the direction of shear. In this configuration, the soil or rock is able to mobilize only the residual shear strength or minimum s that it can ever have, or



Source: Stark and Eid 1997.

Figure 8.2-4 Plots of drained fully softened friction angle vs. liquid limit for triaxial compression mode of shear



Source: Stark and Eid 1994.

Figure 8.2-5 Plots of drained residual friction angle vs. liquid limit

$$s = \sigma'_{nf} \tan \phi'_r \qquad (8.2\text{-}13)$$

The residual friction angle ϕ'_r is an index property of the natural soil composition, and reflects the size and shape of the soil particles (Mesri and Cepeda-Díaz 1986). Figure 8.2-5 shows the correlation of drained residual secant friction angle ϕ'_r as a function of liquid limit, clay-size fraction, and effective normal stress for clays and shales developed by Stark and Eid (1994). It can be seen that higher liquid limits and higher clay-size fractions yield lower ϕ'_r values, with ϕ'_r ranging from 31° (liquid limit $w_l \leq 25\%$ and clay-size fraction $CF \leq 20\%$) to about 5° (liquid limit $w_l \geq 280\%$ and clay-size fraction $CF \geq 50\%$).

In materials that contain thin seams of soft material or preexisting planes of weakness, such as bedded shales or shear zones, residual shear strengths are reached with much smaller displacements than those required to reach residual strength for a thicker material. Figure 8.2-3C compares the different shear displacements observed in direct shear tests on thick and thin seams of clay (Nieto 1974). In practice, residual strengths are likely to be more representative of thin-bedded materials, such as a bedded shale, rather than of massive clay with indistinct bedding. Small displacements along the bedding planes can rapidly reduce peak strength to the residual value. Additionally, any excess pore water pressures generated in the bedding seam during shear rapidly dissipate so drained conditions may prevail. If the soil or rock mass presents evidence of discontinuities along which significant movements have already occurred, such as bedding or shear planes, then only the residual shear strength is expected to be mobilized and the failure envelope can be characterized in terms of the residual friction angle ϕ'_r.

Undrained shear strength. The undrained shear strength of a clay can be obtained in terms of c' and ϕ' as long as u_f can be determined; however, this is often difficult in the field and sometimes infeasible in the laboratory. Therefore, the undrained shear strength s_u is usually determined or approximated from direct and simpler measurements such as FV test or CPT, laboratory unconfined compression tests, or unconsolidated undrained triaxial tests on undisturbed samples.

Cone penetration testing can also provide a valuable measure of undrained shear strength by means of the following equation:

$$s_u = (q_t - \sigma_{vo})/N_{kt} \qquad (8.2\text{-}14)$$

where

q_t = corrected cone tip resistance
σ_{vo} = in-situ total vertical stress
N_{kt} = empirical cone factor

N_{kt} can be determined on a site-specific basis by collecting and testing undisturbed samples in the laboratory. For small projects where extensive laboratory testing is not possible, typical values of N_{kt} can be taken as 16 ± 2. Some researchers have proposed a correlation between N_{kt} and the plasticity index, but other researchers have found no such correlation.

The *unconfined compressive strength* ($q_u \approx 2s_u$) is a measure of the consistency of fine-grained cohesive soils and has been used for classification purposes (Table 8.2-3).

Back-calculation of the mobilized shear strength $(s_u)_{mob}$ in a number of slope and foundation failures in various soft clay deposits indicates that, for the cases investigated, $(s_u)_{mob} \approx 0.22\,\sigma'_p$ (Mesri and Godlewski 1977). Typically, s_u determined from laboratory unconfined compression tests is less than $(s_u)_{mob}$ largely because of sample disturbance. Corrections are also applied to the measured laboratory undrained shear strength to account for field conditions that are not exactly duplicated in the tests, such as mode of shear, strength aniso tropy, influence of shearing rate, and progressive failure.

Undrained shear strength depends on the mode of shear and differs in different directions in the ground. The values of undrained shear strength mobilized in different directions can be measured in the laboratory by means of shear tests that simulate distinct modes of shear. The most widely used tests are the following:

- Triaxial compression (TC)
- Triaxial extension (TE)
- Direct simple shear (DSS)

Figure 8.2-6 shows the relevance of these tests along a circular failure surface in the field. The relationship between the undrained shear strength and the effective stress before shear can also be investigated by means of TC tests on undisturbed samples that are first consolidated to the various effective stresses selected and then sheared under undrained conditions; such tests are usually called consolidated undrained TC tests.

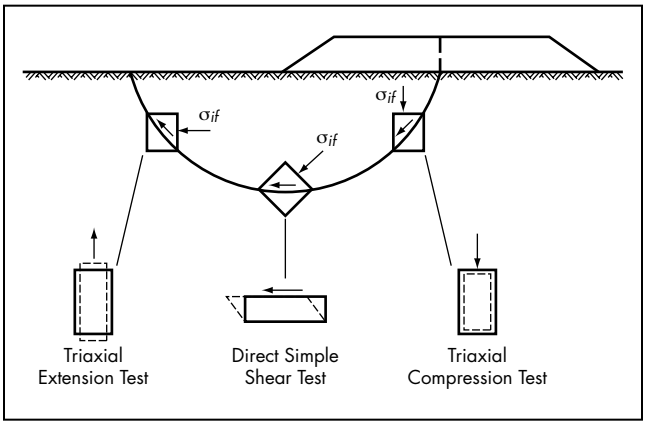
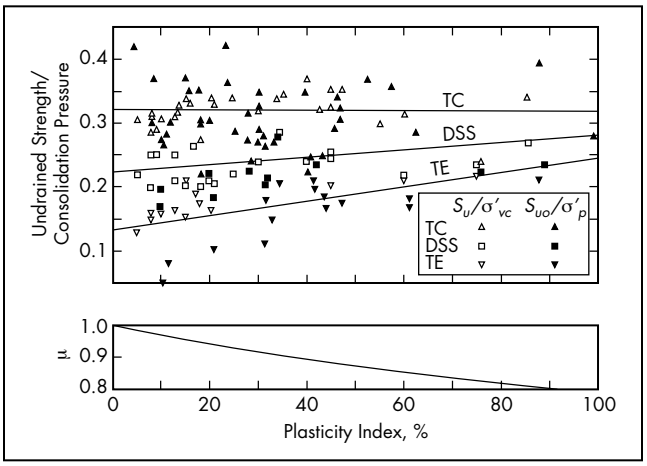


Figure 8.2-6 Undrained shear strength tests



Source: Terzaghi et al. 1996.

Figure 8.2-7 Plots of normalized undrained shear strength from laboratory triaxial tests vs. plasticity index

The TE test, which is similar, involves a sample that is consolidated to various effective stresses and then sheared due to a decrease in the vertical load. The DSS test also involves undrained shearing; its failure mode lies between those from the TC and TE tests.

Figure 8.2-7 shows an empirical correlation between the undrained shear strength s_u from triaxial and DSS tests (normalized by the consolidation pressure σ'_{vc} or the preconsolidation pressure σ'_p) and the plasticity index I_p (Terzaghi et al. 1996). At low plasticity values, the difference between the three failure modes is significant, with the undrained shear strength ratio for the TC failure mode being about 2.3 times the ratio for the TE failure mode. As plasticity increases, the difference between the three failure modes decreases until it is very small at I_p = 100%. The TC failure mode appears to have little or no correlation with I_p. The time to failure in laboratory shear tests is generally measured in hours, whereas mobilization of the undrained shear strength in the field requires weeks. The time to failure for instabilities is likely to increase with I_p. A correction factor has been developed to account for these differences and is included in the figure.

Drainage conditions and shear strength parameters in the field. The undrained shearing of highly overconsolidated clays and shales leads to a decrease in s with time as the induced excess negative pore water pressures gradually dissipate. Thus, for this type of material, the long-term drained condition is generally the least-favorable condition. For normally consolidated clays, on the other hand, the undrained shear strength (corresponding to a given σ'_n before shear) may be considerably lower than the corresponding drained shear strength as a result of the positive excess pore water pressures induced during undrained shearing. During foundation loading, for example, the least favorable condition for normally consolidated clays and slightly overconsolidated clays (OCR < 2) is generally the short-term undrained condition, and thus using the undrained shear strength for this latter type of material is usually conservative.

For unloading slope cuts and other excavations, excess pore water pressures resulting from unloading are superimposed on the excess pore water pressure generated by undrained shearing. This can result in a net negative excess pore water pressure, even for normally or slightly overconsolidated clays. With time, as the excess negative pore water pressure dissipates, shear strength decreases. Thus, the undrained shear strength is unconservative if used to estimate long-term strength of excavations, particularly for highly overconsolidated clays or shales.

For bedded shales or weak rock with joints, the strength along the discontinuity, rather than the unconfined compressive strength of the material, controls many of the failures. In some cases, the strength along the bedding surface or other discontinuity can be estimated using a peak friction angle, which may include a dilatant component to account for irregularities on the surface. However, the residual friction angle should be used if irregularities have already been sheared off or if they will be lost with small displacements.

For failure surfaces that cut at an angle to the discontinuity, the peak strength of the intact material may be applicable. However, combined failures, both through intact material and along discontinuities, often occur in shale and other weak rock, and the interaction of a discontinuity with the intact portion of the weak rock decreases strength to less than predicted for failure of the intact material alone.

Creep

Creep refers to the time-dependent deformation of soil or rock resulting from internal rearrangement of particles in response to the application of a sustained stress difference $(\sigma_1 - \sigma_3)$, which is generally smaller than the stress difference of the soil at failure $(\sigma_1 - \sigma_3)_f$, where σ_1 and σ_3 are the major and minor compressive principal stresses, respectively. Deformation during undrained creep results from shape distortion as the soil mobilizes a constant shearing resistance in response to shear stresses applied upon loading or unloading.

Creep models have been applied toward a variety of engineering problems, such as the closure of and loading on tunnels, chambers, and pillars in creep-sensitive materials such as salt, shale, and fault zones. Highly stressed and creep-sensitive ground encountered in underground excavations is described in tunnel-worker terminology as *squeezing ground*, inasmuch as it leads to a gradual closing of the opening under practically undrained conditions.

Undrained creep behavior is closely related to the drained creep behavior associated with secondary consolidation and swelling, inasmuch as the mechanisms that cause volume change for drained loading (at a constant effective stress) cor-

respondingly cause excess pore water pressures for undrained (constant volume) loading.

A fundamental approach to creep behavior, involving the response of soil to isotropic and deviatoric stresses, is quite complex and still in the development stage. For this reason, creep predictions have been traditionally based on semi-empirical creep models obtained through attempts to best fit the observed laboratory creep data of a variety of soils. Singh and Mitchell (1968, 1969) determined the interrelationship among creep strain rate $\dot{\varepsilon} = d\varepsilon/dt$, stress level with respect to a reference strength $D = (\sigma_1 - \sigma_3)/(\sigma_1 - \sigma_3)_f$, and time t. The interrelationship is widely applicable to both drained and undrained creep, and is expressed by

$$\dot{\varepsilon} = \frac{d\varepsilon}{dt} = A_1 e^{\alpha D}\left(\frac{t_1}{t}\right)^m \tag{8.2-15}$$

where t_1 is the reference time. The m parameter is the slope of the linear $\log \dot{\varepsilon}$ –$\log t$ relationship existing for any intermediate stress level D (approximately 30%–90% of the failure stress) for which m is also a constant. In other words, m establishes the rate of decrease of the strain rate with time. The α parameter gives the dependency of strain rate on stress level for $D \approx$ 30%–90% of the failure stress for a given time ($t_1 < t_2 < \ldots t_n$) for which α is also a constant. The parameter A_1 is the extrapolated value of strain rate at the reference time t_1 for $D = 0$. The creep potential of a clay is closely related to the value of m, which is in the range 0.5–1.3 for a variety of soils. As m decreases, creep deformations become more significant and the soil becomes more susceptible to creep rupture under sustained loads. For any one soil, there seems to be a maximum shear stress that the soil can sustain without creep rupture, called the upper yield strength.

Expressing the strain–stress model as a hyperbola provides a better physical representation of the creep parameters. The parameters in the hyperbolic model are the ratio of undrained modulus to undrained shear strength E_u/s_u and the axial strain at failure ε_f (Mesri et al. 1981) as given by

$$\varepsilon = \frac{2}{(E_u/s_u)_1}\,\frac{D_1}{1-(R_f)_1 D_1}\left(\frac{t}{t_1}\right)^{\lambda} \tag{8.2-16}$$

where $R_f = 1–[2/(E_u/s_u)\varepsilon_f]$ and all the pertinent parameters are evaluated at initial time t_1.

The creep parameter λ that controls time or strain-rate effects also correlates very roughly with E_u/s_u. Specifically, $\lambda \approx$ (1/6,000) E_u/s_u. Typical values of E_u/s_u are in the range 100–600 for clays and shales, and thus the values of λ are typically in the range 0.01–0.10 ($\lambda = 1 - m$).

Laboratory creep tests on samples of clay gouge can be used to determine creep rates and pressures anticipated around a tunnel in squeezing ground, as discussed later in the "Tunneling" section.

Deterioration

Deterioration of overconsolidated clays and weak argillaceous rocks in response to weathering, swelling, and associated softening is perhaps one of their most important engineering characteristics. Unstable shales, for example, have plagued the petroleum industry for many years; the washout experienced by shale formations in response to borehole drilling can be tolerable in some wells but uncontrollable in others. Deterioration of shale mine roofs has been a major cause of the time-dependent roof collapses that have troubled the coal-mining industry.

Disintegration of shale into relatively small pieces upon exposure to different humidity-moisture environments, including submersion, is usually referred to as slaking. For tunnels and other underground excavations, the slaking characteristics of shale affect stand-up time, overbreak, support loads, and construction and support procedures required to minimize deterioration. Successive disintegration and breaking away of loose material from the tunnel crown section results in a fresh rock surface that is exposed each time to a further cycle of weathering. Slaking also progresses along surfaces of fractures behind rock slabs, contributing to time-dependent instability of large blocks.

Wetting and drying are also known to significantly increase the expansion of exposed clay and shale strata (and thus the heaving of overlying structures) as compared to the magnitude of swell and associated softening caused only by unloading with access to water. The additional heave and expansion occurs in response to the further breakdown of diagenetic bonds and alteration of the shale structure. The rate of disintegration of the upper shale layers also controls the rate at which shale slopes are "worked down." The colluvium of disintegrated shale (which could be tens of feet or meters thick) often forms an unstable mass that slides along its contact with the underlying intact shale. Thus the overall strength and stability of slopes is frequently controlled by the strength of the softened and deteriorated material.

During drying, shale is often subjected to heterogeneous and anisotropic shrinkage that can cause cracks or microfissures to open, providing conduits for future moisture redistribution. Upon access to water, shear stresses resulting from differential swelling may also cause local failures and shale disintegration. Moreover, pore-air compression, resulting from immersion of unsaturated shale in water, can apply internal pressures that may eventually exceed the local tensile strength of the material and cause additional shale slaking.

The resistance of shale to deterioration upon exposure to drying and wetting cycles is frequently estimated by subjecting representative specimens to the standard slake durability test. In this test, 10 oven-dried lumps of shale 0.08–0.13 lb (40–60 g) each are rotated (20 rpm) for 10 minutes in a partly immersed drum of 0.08-in (2.0-mm) mesh. The slake durability index is the oven-dried weight percentage retained in the drum after two (oven drying–wetting) standard cycles. Shale slake durability is characterized as described in Table 8.2-6.

The deterioration response of shales encompasses a broad range because of the expansive assortment of shale compositions and the wide variety of field-humidity and moisture environments to which they are subjected. As a result, index tests and classifications based on limited, often severe testing environments do not permit a full evaluation of shale performance under field conditions. This is particularly true in underground excavations in which moisture changes may be caused by relatively small humidity changes rather than alternate soaking and drying.

A study of the deterioration of two dozen natural shales exposed to controlled temperature and relative humidity conditions revealed the following (Cepeda-Díaz 1987). In general, shales with natural water content w_o smaller than the shrinkage limit w_s are fissile, whereas shales with natural water content higher than w_s are massive. Fissile shales

Table 8.2-6 Shale slake-durability classification

Slake-Durability Index	Descriptive Term
>98	Very high
95–98	High
85–95	Medium high
60–85	Medium
30–60	Low
<30	Very low

Source: Cording and Cepeda-Díaz 1992.

(group F) deteriorate by opening along weak planes of stratification, followed by separation of thin slabs perpendicular to the stratification. In contrast, massive shales (group M) lack a preferred orientation of discontinuity and often break down to chunks and chips along random cracks, fractures, and slickensided surfaces. Fissile shales generally have natural water content of <15% and liquid limit w_l of ≤100% (using ball-mill disaggregated samples passing through a No. 200 U.S. standard sieve). Beyond this range of values for w_o and w_l, shales are generally massive.

Fissile and massive shales can be further subdivided into three groups (1, 2, and 3) depending on moisture-change response and deterioration behavior upon exposure and thus related to their equilibrium relative humidity (ERH). ERH is the relative humidity (RH), at a given temperature, in which freshly exposed specimens of a given shale remain at their natural water content. At RH > ERH, intact shale gains moisture from the air environment; at RH < ERH, intact shale loses water to the environment. Shales in the three groups react differently to humidity:

1. Shales in group 1 deteriorate when subjected to RH > ERH and include shales with $w_l > 40\%$ and $w_o < 0.2w_l - 8\%$ but natural water content ≤15%.
2. Shales in group 2 (75% ≤ ERH ≤ 97.5%) deteriorate when subjected to RH ≠ ERH and include shales with $w_l > 10\%$ and natural water content other than those characteristic of groups 1 or 3.
3. Shales in group 3 (ERH > 97.5%) deteriorate when subjected to RH < ERH and include shales with $w_l > 10\%$ and $w_o > 0.2w_l - 2\%$.

Shales that deteriorate when subjected to RH > ERH (groups 1 and 2) generally have natural water content lower than that corresponding to $w_o = 0.2w_l$. In contrast, shales that deteriorate only when subjected to RH < ERH (group 3) generally have natural water content higher than $w_o = 0.2w_l$. Thus it is consistent that shales with natural water content equal to both $w_o = 0.2w_l = w_s$ remain intact upon exposure to any RH environment. At such a moisture and structural state, the total volume change associated with moisture gain or loss upon environmental exposure may be relatively small, and thus the shale does not experience a significant amount of anisotropy and heterogeneous deformation—that is, it does not deteriorate.

For underground excavations such as tunnels and mines where field environmental control can be considered, the following guidelines are suggested for preventing the deterioration of various types of exposed shale strata (Cepeda-Díaz 1987).

- Shales in group 1 should always be maintained at environments drier than ERH and preferably not wetter than 75% RH. Dry environments not only avoid the damaging effect of humidity fluctuations at high RH but also discourage the chemical alteration of any iron sulfides that are present in the shale. Iron sulfides are known to oxidize in the presence of water and oxygen to form limonite and sulfuric acid. The sulfuric acid reacts with any calcite in the rock to produce carbonic acid and gypsum with resulting large increase in volume and shale disintegration.
- Shales in group 2 (75% ≤ ERH ≤ 97.5%) should always be maintained at relative humidities near ERH and preferably not exceeding ERH ± 2%.
- Shales in group 3 (ERH > 97.5%) should always be maintained at environments wetter than ERH, and thus very often close to 100% RH.

Because shales in group 3 rarely deteriorate upon immersion in water at their natural water content, the use of sprayers to keep the exposed shale strata from drying may be a suitable alternative to high-humidity control. For shales in groups 1 and 2, any direct contact with water should be suppressed to avoid severe deterioration. Independent of the initial type of shale deterioration upon initial environmental exposure, wetting as a result of water inflow, immersion, or spraying (rain in the case of open cuts) is likely to induce severe deterioration if the exposed shale strata has been allowed to dry before wetting.

MECHANICS OF BEHAVIOR

For many problems in both soil and rock mechanics, the stresses or interrelation of stresses and deformations in the soil or rock mass must be determined. A first step in approaching these problems is to consider the boundary conditions—the geometry of the mass and the distribution of loads or displacements applied to the mass.

These types of problems have traditionally been divided into two groups (e.g., Terzaghi 1943): (1) those in which the stresses in a region or a surface in the mass have reached a limit or failure condition, and (2) those in which the stress levels are below the limit condition, and changes in stress within the medium are a function of the deformations.

- **Limiting stress conditions.** The loads or the state of stress can be determined from the equilibrium of a portion of the mass, assuming that the stresses along a surface or region within the medium have reached their limit—the strength of the material. Slope-stability analysis, earth-pressure theory, and bearing-capacity problems concern the evaluation of the loadings upon the soil or rock mass when certain surfaces within the mass are at a limiting level of stress.
- **Stress-deformation conditions.** In many problems, it is assumed that changes in stress are a function of deformations. The simplest assumption is that stresses are proportional to strain—that is, the material is elastic. Closed elastic solutions are available for relatively simple boundary conditions, and often can be used to obtain a first approximation of the mechanics of the problem. More sophisticated stress-deformation models may assume that the material has a nonlinear stress–strain relationship that may even approach a limit at some strain level, and that energy losses occur as loads are cycled. Solutions for these models, as well as for elastic models with more complex boundary conditions, require the use of a computer and numerical methods such as finite-element or discrete-element models.

Stress-deformation relationships are used to evaluate not only foundation settlements but also the interaction between the soil medium and a structure—such as an anchored bulkhead, braced excavation, or tunnel lining—whose stresses and deformations are a function of relative stiffness with respect to the soil mass.

Another group of stress-deformation problems is that in which settlements or heave occur with time as water flows into or out of the soil matrix in response to changes in the state of stress—the process of swelling or consolidation, respectively. In soft soils, volume changes occurring as water flows into or out of the soil are the major cause of settlement, whereas deformation of the soil at constant water content is usually a minor portion of the total displacement.

LIMIT EQUILIBRIUM

Limit equilibrium is a common approach for analyzing soil- and rock-mechanics problems. The following sections discuss how this method is used to analyze retaining walls, stability of slopes, and foundations.

Analysis of Active and Passive Earth Pressures

Lateral extension or compression of a soil mass decreases or increases, respectively, the lateral soil pressures until a limit is reached. The limits are termed the *active and passive earth pressures* and represent, respectively, the minimum and maximum possible natural lateral stresses that can exist in the ground. The active pressure P_a denotes the earth pressure acting behind retaining walls; the passive pressure P_p is the maximum pressure that can act at the toe of a wall embedded in the soil.

Figure 8.2-8 shows the active and passive pressures resulting from extension and compression of an infinite soil mass with a horizontal ground surface.

The magnitude of the active and passive stresses acting on a vertical plane is obtained from Mohr's circle of stress, recognizing that the vertical stress is equal to the overburden pressure z, and the soil element is at its limit, and therefore the circle of stress is tangent to the strength envelope, which is assumed to be a straight line, $s = c + \sigma \tan \phi$.

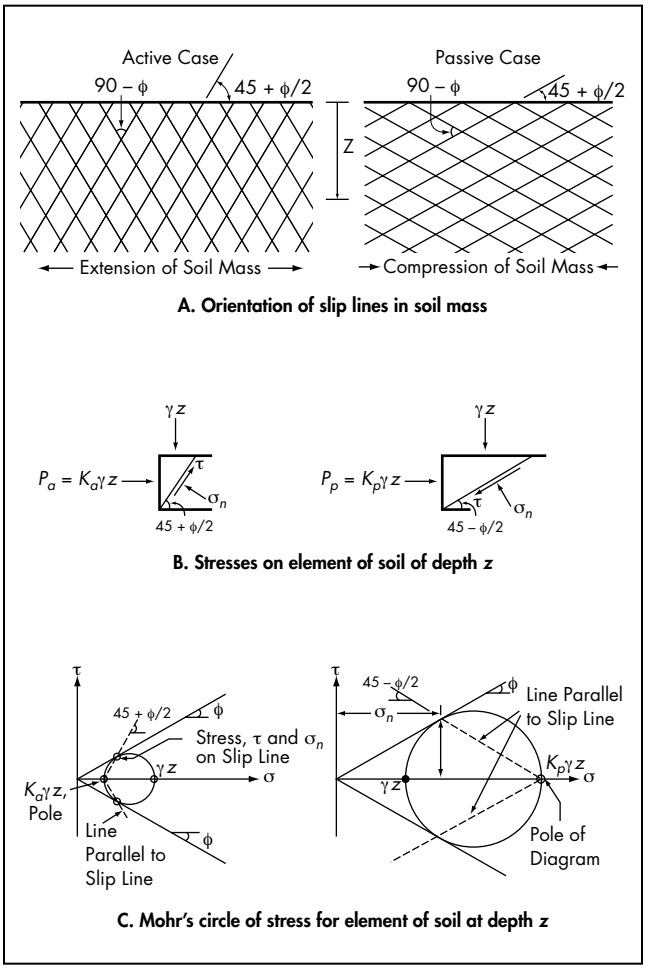
Rankine Analysis

Rankine (1857) provided the first solution to this problem. The principal stresses at the limit condition are related as follows:

$$\sigma_1 - \sigma_3 \tan^2 (45^\circ + \phi/2) + 2\, c \tan (45^\circ + \phi/2) \qquad (8.2\text{-}17)$$

For lateral extension of the soil mass, the lateral stresses are reduced to their minimum value, and σ_3 equals the lateral stress, which is the active earth pressure. For a horizontal ground surface, the coefficient of active pressure $K_a = 1/\tan^2 (45 + \phi/2)$. For lateral compression of the soil, the lateral stresses are increased to their maximum value, and σ_1 equals the lateral stress, which is the passive earth pressure. For a horizontal ground surface, the coefficient of passive pressure $K_p = \tan^2 (45 + \phi/2)$.

The more general Rankine solution is for active and passive pressures that develop in an infinite slope. For an infinite slope, the vertical force on a plane parallel to the ground surface is equal to the weight of the overburden. The active and passive earth pressures are the stresses acting on a vertical plane. They are oriented parallel to the ground surface and thus have both a normal and shear component. Figure 8.2-9 illustrates the condition for active earth pressure. The magnitude and direction of the active earth pressure and the orientation of the shear planes can be obtained from Mohr's circle.

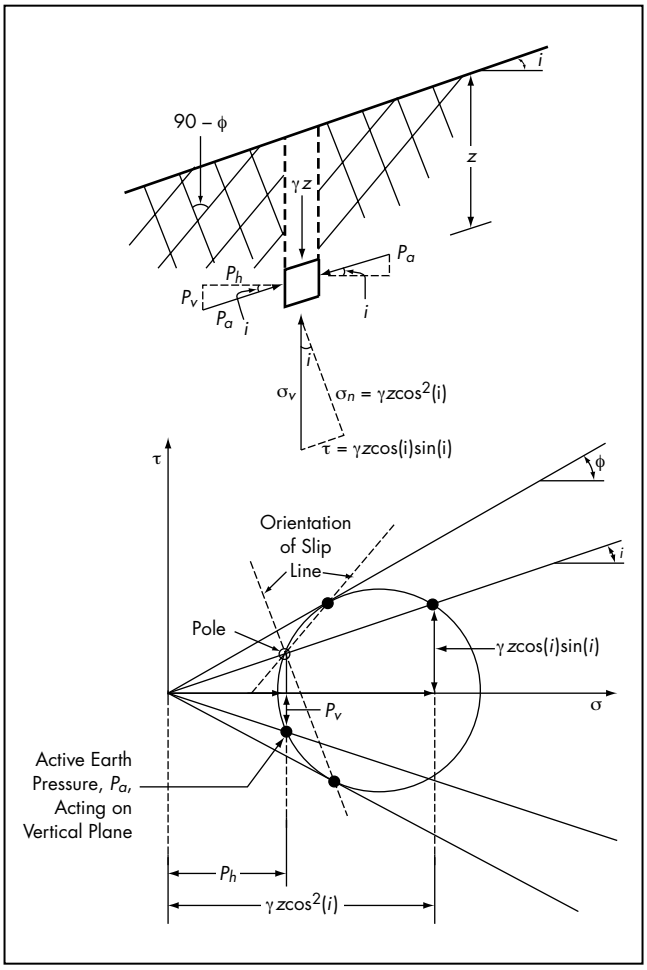


Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-8 Rankine analysis of active and passive earth pressures for a vertical plane

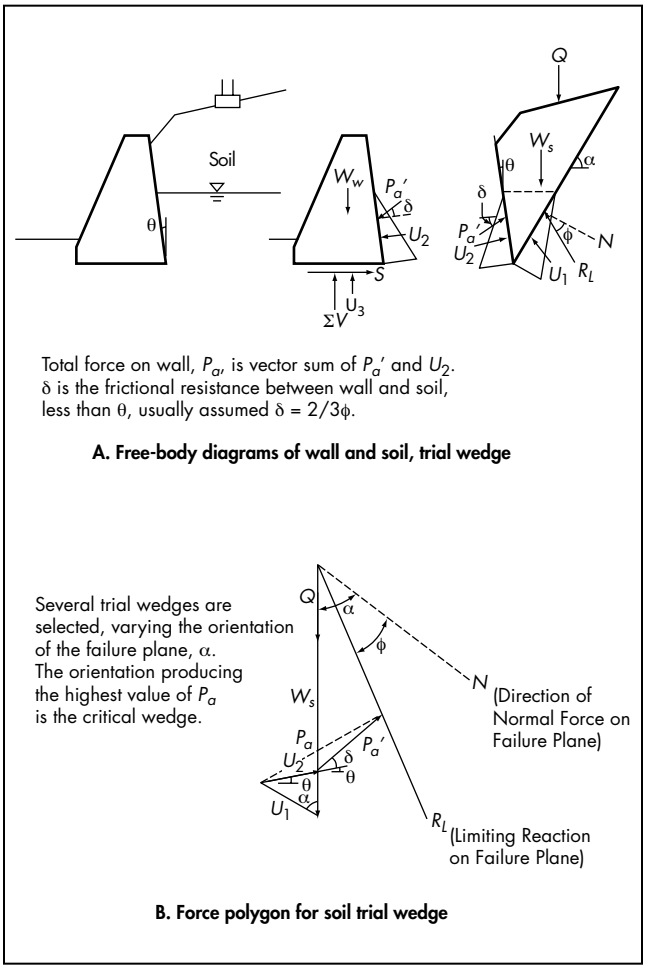
If the entire soil mass is in a limiting state of equilibrium, as is assumed in the Rankine case, then the active and passive stresses acting on the vertical plane for a frictional material increase linearly with depth. The total active or passive force over a given height z acts at the lower third point and has a magnitude equal to $z/2$ times the active or passive pressure at depth z. This condition is obtained for a retaining wall when outward deflections at the top of the wall are on the order of 0.1%–0.5% of its height. Thus the validity of designs based on an assumed active state of stress behind the wall depends on the outward deflections that the retaining structure actually undergoes.

For walls in which the upper portion of the wall is not allowed to deflect outward, the distribution of stresses on the wall does not increase linearly with depth, even though movements of the lower portions of the wall may be sufficient to reduce the total forces on the wall to the active forces. This is the case with braced walls in which the upper portion of the wall is restrained early in the excavation process, usually in order to limit ground movements that could adversely affect structures and utilities adjacent to the wall. As a result, the center of pressure for the forces acting on the wall is at a higher



Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-9 Rankine analysis of active earth pressure for an infinite slope



Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-10 Coulomb wedge analysis of active earth pressure

elevation than the lower third point, although the total magnitude of the forces can be estimated using Rankine or Coulomb assumptions. Field measurements of strut loads for braced excavations are used to develop apparent earth-pressure envelopes for design. The envelopes, which encompass the variations that can be expected in strut loads due to construction procedures, provide an almost rectangular (constant) pressure distribution with depth and a magnitude of the total force under the envelope that is approximately 1.3 to 1.75 times the active Rankine or Coulomb pressure (Peck et al. 1974).

Rankine analysis, in which a ground surface of infinite extent is assumed, either horizontal or sloping, can be applied to real problems in which the ground surface is not of infinite extent, as long as the stresses on the vertical plane can be assumed to act parallel to the ground surface. Rankine analysis usually produces a conservative estimate of either the active or passive pressures on a wall because it usually does not fully account for the shear stresses that act on the wall.

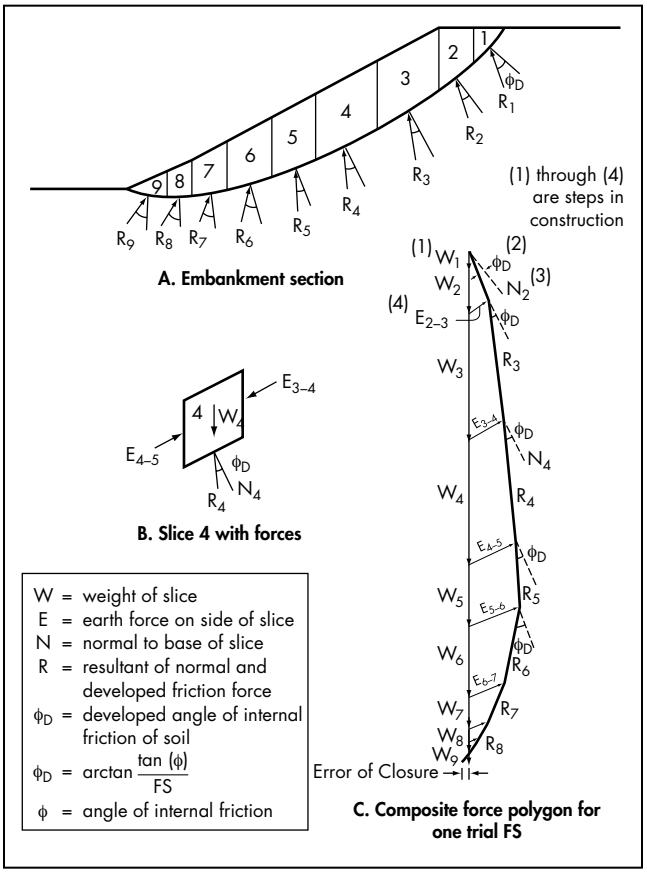
The magnitude and location of the total active or passive forces acting on the back of a retaining structure can also be determined using the assumptions involved in the Coulomb wedge analysis, described in the next section. Coulomb analysis is more general than the Rankine method in that any orientation of the forces on the back of the retaining wall can be assumed, and surcharges and irregular backfill slopes can be handled.

Coulomb Wedge Analysis

A more general solution for limiting conditions in a soil mass can be obtained by Coulomb (1776) analysis, which evaluates the forces acting on a wedge of soil. It is assumed that a planar failure surface has developed in the soil mass and that the soil wedge slides along a planar surface forming the back of the wall. The analysis is commonly used to evaluate active earth pressures behind retaining walls.

Figure 8.2-10 illustrates the procedure for a frictional material. As in most limit equilibrium problems, analysis is handled in a series of steps:

1. A free-body diagram of the wedge of soil is drawn, showing all forces acting on the boundaries of the wedge as well as the body forces (weight W_s and acceleration forces) acting in the free body. A failure plane forms one of the wedge boundaries. A trial failure plane is selected, oriented at angle α to the horizontal.
2. A force polygon is constructed to include all forces acting on the free-body diagram. The summation of forces is obtained either graphically or by trigonometry,



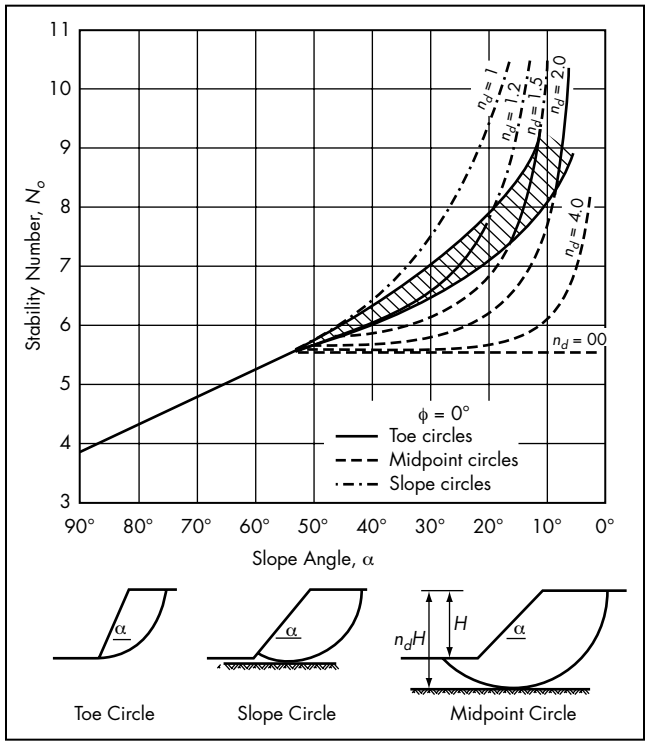
Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-11 Method-of-slices analysis of slope stability

considering the geometry of the force polygon. Water pressures U_1 and U_2 acting on the boundaries of the wedge are included as external forces. Although the summation of forces equals zero, the assumption of a planar failure surface may result in the summation of moments not equaling zero.

3. Because the orientation of the failure surface is usually not known, several trial wedges are analyzed, each with a planar failure surface at a different orientation α. The largest value of the active force or the smallest value of the passive force represents the critical condition in which the first planar failure surface would develop.
4. For soil or rock containing preexisting planes of weakness of known orientation, the trial failure plane is set parallel to the plane of weakness for cases in which its orientation is close to the orientation of the critical failure plane. In this case, it is not necessary to search for the orientation of the critical failure plane by analyzing multiple trial wedges.

Coulomb analysis can be used for general retaining-wall problems, but with some limitations. For passive pressure, when there is a significant component of wall friction, a planar failure surface does not closely approximate the actual failure surface in the field, and thus Coulomb analysis is unconservative (i.e., it overestimates the passive pressure). In such cases, the critical plane is curved and passive pressure can be analyzed as described in the following section.



Source: Taylor 1937.

Figure 8.2-12 Stability chart for slopes in cohesive soil

Application of Wedge Analysis to Slope-Stability Problems

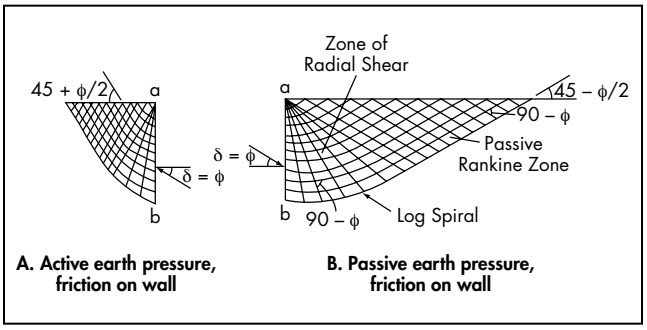
Figure 8.2-11 illustrates the procedure for determining the stability of a slope by analyzing multiple wedges or slices of the potential sliding mass. Multiple slices are treated by summing forces in the same manner as for the Coulomb wedge described previously. A slice is related to the adjacent slice by equating the force acting between the two slices.

Several techniques exist for evaluating the forces between slices. In general, forces acting between vertical slices are assumed to act parallel to the slope, as is the case for an infinite slope. Where one slice or wedge has a significant vertical displacement with respect to an adjacent wedge or slice, the forces acting between the two may approach their maximum obliquity, at ϕ degrees to the normal. The mobilized or developed friction angle ϕ_D (required to produce a factor of safety of 1) is determined from trials using the graphical construction for the composite force polygon shown in Figure 8.2-11C. The factor of safety (FS) is the ratio of tan ϕ to tan ϕ_D.

In most slope-stability analyses, the slope is assumed to be 2-D. For an actual three-dimensional slope (that does not extend a large distance in the third dimension), 2-D analysis gives a conservative (lower than actual) value of FS. For analysis of a 3-D slope, back-analysis of shear-strength parameters assuming that the slope is 2-D overestimates the shear strength.

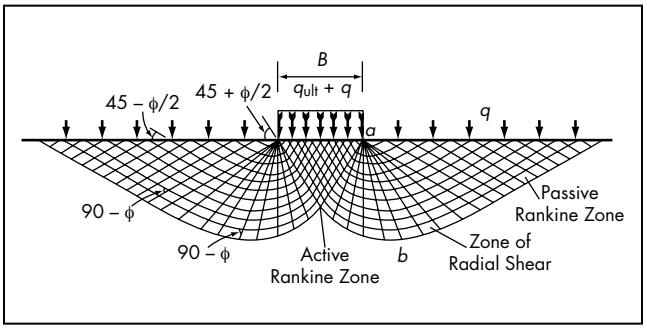
Circular Failure Surface for Stability of Cohesive Material

The maximum height of a slope or a cut in a cohesive material, in which the undrained shearing resistance s_u is assumed to apply, can be determined from the stability chart of Figure 8.2-12 (Peck et al. 1974). The factor of safety, FS, of the slope is



Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-13 Active and passive earth pressures, curved failure surfaces



Source: Cording and Cepeda-Díaz 1992.

Figure 8.2-14 Bearing capacity of a shallow footing

$$FS = N_o s_u/\gamma H \tag{8.2-18}$$

Use of the undrained shear strength s_u is unconservative if the slope is excavated in overconsolidated or fissured clays or in clays containing sand and silt seams. In such cases, effective shear strength parameters should be used, which assume that the material has frictional resistance and, in some cases, a small effective cohesion.

Curved Failure Surfaces for Passive Earth Pressure

Coulomb analysis produces unconservative results for the passive case when the passive force on a wall acts at a significant angle δ to the normal. In this case, a curved failure surface should be assumed in order to determine the passive earth pressure. The failure zone consists of a zone of radial shear and a passive Rankine zone (Figure 8.2-13). For a frictional material, the shape of the failure surface in the zone of radial shear approximates a logarithmic spiral. Use of a log spiral simplifies calculations because the limiting reactions acting on the log-spiral failure surface meet at a point and can be ignored in the analysis by summing moments about that point. Figure 8.2-13 shows the pattern of slip lines for the case when the angle δ on the wall is at its maximum value, equal to the angle of friction, in which case the plane of the wall becomes a slip line for the soil mass. The active-earth-pressure case is also shown in Figure 8.2-13 for completeness.

General Bearing-Capacity Relationship

Relationships for the bearing capacity of soils loaded by foundations have been developed from the theory of plasticity. The solutions of Prandtl (1921) and Reissner (1924) were obtained for a rough, rigid strip footing on the surface of a weightless, perfectly plastic material (Figure 8.2-14). The pattern of the failure surfaces is similar to the combined active and passive zones shown in Figure 8.2-13. The ultimate bearing capacity q_{ult} is related to the undrained shear strength of the soil c and the surcharge pressure q as follows:

$$q_{ult} = cN_c + qN_q \tag{8.2-19}$$

where N_c and N_q are dimensionless bearing-capacity factors that are a function of the friction angle ϕ:

$$N_q = \exp(\pi \tan \phi) \text{ and } \tan^2(45^\circ + \phi/2) \tag{8.2-20}$$

$$N_c = (N_q - 1)\cot \phi \tag{8.2-21}$$

The general bearing-capacity relationship (Terzaghi 1943) is an approximate solution, obtained by superposing on Equation 8.2-15 an additional relationship for a frictional soil with weight but without cohesion intercept ($c = 0$) or surcharge ($q = 0$). The superposition produces an error on the safe side (≤20%) for friction angles of 30°–40°, and, of course, no error for a friction angle of zero (Vesic 1975). The additional term is proportional to the shear capacity developed from normal stresses in the weighted soil mass below the footing, and is therefore proportional to the size B of the mass and is also a function of the friction angle ϕ.

The resulting general bearing-capacity relationship for a long strip footing of width B is given by

$$q_{ult} = cN_c + qN_q + \tfrac{1}{2}B\gamma N_\gamma \tag{8.2-22}$$

where N_γ is a dimensionless bearing-capacity factor that is a function of ϕ and must be evaluated numerically. It can be approximated by the expression

$$N_\gamma = 2(N_q + 1)\tan \phi \tag{8.2-23}$$

with an error on the safe side (5%–10%) (Vesic 1975).

Contrary to some earlier discussions in the literature, Vesic (1975) notes that the shape of the failure zone and the magnitude of the ultimate bearing capacity are not strongly affected by the presence or lack of friction at the base of the footing. He further notes that the ultimate bearing capacity is not strongly influenced by the stress distribution at the base of the footing, which can range from a uniform pressure for a flexible foundation to a variable stress distribution created by a rigid footing (or punch) displacing a constant amount into the soil mass. However, if the base of the footing rests on a thin soft layer above a stronger layer (such as for underclays beneath a mine pillar), significant reductions in bearing capacity can result by changing from a rough to a smooth footing or from a rigid punch to a uniform pressure acting at the base of the footing.

For foundations wider than about 4 ft (1.2 m) in sands and other frictional materials, the bearing-capacity factors are quite high and bearing-capacity failures develop only for very high foundation loads. In these cases, the capacity of the foundation is controlled by the allowable settlement.

Adjustments to the general bearing-capacity relationship have been made for other footing shapes and for extension of the failure planes into the soil surcharge above the footing base.

For cohesive, frictionless materials, a linear approximation of the bearing-capacity relationship (Skempton 1951) is given by

$$q_{ult} = (1 + 0.2D_f/B)\,(1 + 0.2B/L)s_u N_c \quad (8.2\text{-}24)$$

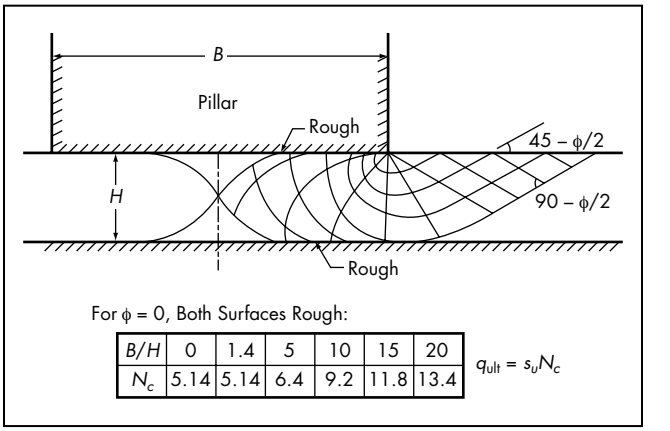
where

D_f = depth of the surcharge above the footing base, equal to $2B$ for values of $D_f > 2B$
B = footing width
B/L = ratio of width to length for the footing
$N_c = 5.14$

The ultimate bearing capacity q_{ult} is the pressure in excess of the pressure applied by the surrounding soil surcharge γD_f.

This relationship is used to estimate the allowable bearing capacity q_a for foundations on clay. The allowable bearing capacity is computed by dividing the ultimate bearing capacity q_{ult} by FS, which is typically 3 for foundations. In addition, the foundation can accept a pressure equal to the surrounding soil surcharge pressure γD_f as follows:

$$q_a = q_{ult}/3 + \gamma D_f \quad (8.2\text{-}25)$$



Source: Mandel and Salençon 1972.

Figure 8.2-15 Squeezing a layer of underclay out from beneath a mine pillar

Bearing Capacity of Mine Pillars on Underclays

In many coal mines, underclays are present beneath the coal pillar and the floor of the mine. Stresses in the pillars are often high enough to cause a bearing-capacity failure. Several types of bearing failure can occur. The pillar can punch into the underclay, or a general bearing-capacity failure can develop, with large heave occurring in the floor adjacent to the pillar. Typically, underclays are in layers that are thin compared to the pillar width, in which case the slip surfaces used to determine the general bearing-capacity relationship cannot form and the underclay must be squeezed out between the base of the pillar and the underlying stronger rock layers (Figure 8.2-15).

As a result, the ultimate bearing capacity is greater than for a deep deposit of the same material. Vesic (1975) and Mandel and Salençon (1972) show that, as the ratio of pillar width to soft-layer thickness increases, the ultimate bearing capacity for a cohesive, frictionless soil increases from 1 to approximately 2.5 times the ultimate bearing capacity for a deep deposit of the same soil.

Experience with coal pillars overlying underclay shows an interesting interaction between the pillar and the underclay. As the underclay is squeezed laterally outward from beneath the pillar, the pillar is subjected to tensile stresses, causing vertical fractures to develop in the pillar. Squeezing of the underclay not only causes closure of the opening but also can lead to collapse of the pillar and further closure of the mine opening. Formation of vertical tensile fractures in the pillar demonstrates the inability of the pillar to sustain shear stresses that tend to develop between the pillar and the underclay as the clay squeezes outward. Mandel and Salençon show that the increase in bearing-capacity factor N_c is much less significant if there is no shear resistance on one of the layers bounding the thin failing layer.

Shear stresses that develop at the base of a pillar have a much greater effect on the bearing capacity of a thin soil layer than a deep layer. Failure of a footing above a deep layer is by a general downward movement of the soil beneath the footing. In contrast, failure of a pillar above a thin weak layer is by soil squeezing out laterally between the pillar above and the stronger layer below. Vertical fractures in a pillar cause the pillar to break into what is effectively a group of narrow pillars, thus decreasing the ratio of pillar width to layer depth, possibly reducing the shear stresses along the pillar–underclay contact and decreasing the bearing capacity.

Time effects also occur in the failure of coal pillars overlying underclay. Squeezing of the underclay occurs with time as a result of clay creep under high stress and loss of strength of the underclay beneath the floor due to swelling in response to excavation unloading. Pillar settlement and floor heave may begin during active mining, but often the time-dependent effects cause continued failures, floor heave, and large settlements long after excavation has been completed. Large and relatively abrupt subsidence has taken place over abandoned room-and-pillar workings many years after the facilities were abandoned.

SETTLEMENT OF FOUNDATIONS

In engineering applications, the settlement of foundations is a major design consideration, and foundation systems are designed to keep settlement within tolerable limits. The following sections briefly discuss approaches to estimating the anticipated settlement for foundations constructed on various soil and rock materials.

Soft Clays

When load from a footing or fill is placed on a layer of soft clay, most settlement takes place with time as water is squeezed from the soil pores and the volume of the voids decreases. The settlement of a foundation on soft clay can be estimated by determining the change in effective stress on the clay layer and the effective stress existing before loading, and then determining the change in the void ratio by using an appropriate slope on a plot of void ratio versus change in effective pressure (e vs. log σ'_v). The slope to be used can be estimated from laboratory tests or other field tests. For a material loaded above its maximum previous consolidation pressure or preconsolidation pressure σ'_p, steeper slopes corresponding to C_c apply. For stress changes below σ'_p, flatter slopes corresponding to C_r apply.

Stresses resulting from a footing load tend to spread out with depth below the footing, reducing the imposed stress. In order to estimate the stresses at depth, an elastic distribution of stresses is usually assumed. Newmark charts, available in most soil-mechanics texts, provide a means of graphically

determining the stresses at any depth below a footing. An approximate estimate can also be obtained by assuming that the stresses spread on a $2V$:$1H$ slope from the edge of the footing.

Sands

The standard penetration test and its corresponding N values are generally used to predict the allowable contact pressure to which a granular soil can be subjected by a foundation in order not to exceed tolerable settlements. Compression of saturated coarse-grained soils occurs rapidly as a result of their high permeability.

For footings with widths $B > \sim 4$ ft (1.2 m), the allowable pressure q_a expected to produce a total settlement of <1 in. (25.4 mm) and corresponding differential settlements of ≤0.5 in. (12.7 mm) can be approximated by

$$q_a = 0.11\ N \tag{8.2-26}$$

where

q_a = allowable pressure, tons/ft^2
N = smallest average value of the standard penetration resistance

The value of N is within a depth interval equal to B below the foundation level after corrections are made, if necessary, to take into account the effect of overburden pressure and the position of the water table (Peck et al. 1974).

The ratio of differential settlement to total settlement is significantly less for very wide footings or raft foundations than for regular-size footings, and thus the allowable pressure for rafts is that corresponding to a total settlement of 2 in. (50.8 mm), which can be approximated by $q_a = 0.22\ N$. For very narrow footings ($B < 4$ ft [1.2 m]), a small increase in pressure may lead to sufficient settlement to consider failure to be a bearing-capacity failure. Therefore, the margin against bearing-capacity failure for these footings should also be investigated as explained previously.

Weak Rock and Stiff Soils

The compressibility of foundations in weak rock and stiff soils can be evaluated using the theory of elasticity, assuming an appropriate deformation modulus for the material. The Boussinesq equation gives the displacements and stresses in an elastic mass due to a point load applied at the ground surface. The equation is integrated to determine displacements and stresses due to loads distributed over some area at the ground surface. Newmark developed a graphical method for determining stresses or displacements for any assumed distribution of pressures at the ground surface. For the simple case of a uniform pressure p acting over a circular area of radius a on the surface of an elastic medium, the surface displacement d at the center of the circle can be estimated as

$$d = 2pa(1 - \mu^2)/E \tag{8.2-27}$$

where

μ = Poisson's ratio
E = in-situ deformation modulus

For rock containing natural fractures, the stiffness obtained in a laboratory test on an intact sample should be reduced in order to obtain the in-situ deformation modulus. Parameters that index the degree of fracturing of the rock mass, such as the rock quality designation (RQD) or the ratio of field seismic velocity to laboratory sonic velocity, can be used to estimate the amount by which the in-situ deformation modulus should be reduced from the laboratory modulus. Plate-bearing tests (typically with plate diameters of 3 ft [1 m]) can be used to determine the in-situ modulus at selected locations in the field.

TUNNELING

Tunnels are constructed in many mining and civil-engineering applications. Among the important considerations in their design are ensuring the safety of personnel and equipment and minimizing the impact of tunneling on nearby infrastructure. The following sections discuss some of these engineering considerations in tunnel design.

Tunneling Conditions

The following should be considered when constructing a tunnel in soil or weak rock (Peck 1969):

- The mining company's ability to
  - Safely excavate and support the tunnel and
  - Limit damage to third parties—in particular, to control ground movements that could affect nearby buildings and utilities;
- The tunnel's capability to perform its intended function over the life of the project—in particular, an adequate lining.

Long, circular tunnels in soil are commonly constructed by shoving circular shields through the ground, excavating the soil at the face of the shield. The shields provide initial support to the perimeter of the tunnel heading and permit the initial tunnel lining to be erected in the tail section of the shield without exposing the soil surface.

Stability of the tunnel heading and prevention of damage to ground settlements are major concerns in soft-ground tunneling.

Design of the tunneling process is more difficult when ground conditions vary across the tunnel face or along the length of the tunnel alignment. A single shield may not accommodate all conditions efficiently, and it may be necessary to change tunneling procedures or use ground-modification techniques such as dewatering or grouting to advance the tunnel.

Mixed-face conditions, in which rock and soil are encountered in a single tunnel face, create difficulties in excavating the rock while supporting the soils, which are often unstable. Bedding and layering of soils create variations in permeability that allow water to perch on top of low-permeability layers and flow into the tunnel. To minimize such flows requires placement of dewatering wells on very close spacings adjacent to the tunnel.

Soil-tunneling procedures must also be used for tunneling in rock when a mixed face of soil and rock is encountered at the contact between rock and the overlying soil overburden, or when fault zones are encountered in the rock. Commonly, water in the soil and permeable-rock fractures contributes to the instability of these zones. Although rapid and efficient tunneling may not be possible in the soil, it is desirable that the methods used to get through the soil zones allow safe, steady progress. This is more likely if the possible existence of such conditions is anticipated during tunnel design.

The initial tunnel lining must accommodate loads applied during construction as well as stresses produced by ground loads. In shielded tunnels, initial support commonly

Table 8.2-7 Tunnelman's Ground Classification System

Classification	Behavior	Typical Soil Types
Firm	Heading can be advanced without initial support, and final lining can be constructed before ground starts to move.	Loess above water table; hard clay, marl, cemented sand, and gravel when not highly overstressed.
Raveling • Fast raveling • Slow raveling	Chunks or flakes of material begin to drop out of the arch or walls sometime after the ground has been exposed, due to loosening or to overstress and "brittle" fracture (ground separates or breaks along distinct surfaces, as opposed to squeezing ground). In fast raveling ground, the process starts within a few minutes.	Residual soils or sand with small amounts of binder may be fast raveling below the water table and slow raveling above. Stiff fissured clays may be fast or slow raveling depending on the degree of overstress.
Squeezing	Ground squeezes or extrudes plastically into the tunnel, without visible fracturing or loss of continuity or perceptible increase in water content. Ductile, plastic yield, and flow occur because of overstress.	Ground with low frictional strength. The rate of squeeze depends on the degree of overstress. Occurs at shallow to medium depth in clay of very soft to medium consistency. Stiff to hard clay under high cover may move in a combination of raveling at excavation surface and squeezing at depth.
Running • Cohesive running • Running	Granular materials without cohesion are unstable at slopes greater than their angle of repose (±30°–35°). When exposed at steeper slopes, they run like granulated sugar or dune sand until the slope flattens to the angle of repose.	Clean, dry granular materials. Apparent cohesion in moist sand or weak cementation in any granular soil may allow the material to stand for a brief period of raveling before it breaks down and runs; termed *cohesive running*.
Flowing	A mixture of soil and water flows into the tunnel like a viscous fluid. The material can enter the tunnel from the invert as well as from the face, crown, and walls, and can flow for great distances, completely filling the tunnel in some cases.	Below the water table in silt, sand, or gravel without enough clay content to give significant cohesion and plasticity. May also occur in highly sensitive clay when it is disturbed.
Swelling	The ground absorbs water, increases in volume, and expands slowly into the tunnel.	Highly preconsolidated clay with plasticity index 30, generally containing significant montmorillonite.

Source: Adapted from Terzaghi 1950, Peck 1969, and Heuer 1974.

consists of steel ribs and lagging, steel-liner plate segments, or concrete segments. Significant loads can develop during erection and subsequent grouting or expansion of the segments against the ground. If the ground has been excessively loosened or if it forms an irregular perimeter in poor contact with the lining, the resultant nonuniform loadings and bending may result in significant damage to and even buckling of the lining. As the shield advances, ground loads are transferred to the installed lining. Elastic analyses and experience show that the ground and the lining interact significantly. If the lining is in full contact with the ground around its perimeter, then active ground loads applied to one portion of the lining perimeter cause a small outward deflection and buildup of a passive ground reaction on other portions of the perimeter that will support the lining and permit it to act as a stable arch. If the lining is relatively flexible with respect to the soil mass, then the bending moments that develop in the lining will be quite small.

Tunnelman's Ground Classification

Because tunneling may temporarily expose the ground overhead or on vertical surfaces, the capability of the soil to stand until support can be placed is of great concern. A major factor involved in controlling the stand-up time of soil is its short-term cohesion or cementing, as well as the amount of soil surface that is exposed ahead of the supports at any time. It is often difficult to anticipate tunnel behavior from the standard index properties obtained in the soil-exploration program. Sampling often disturbs cohesive granular soils or partially cemented soils, causing their in-situ cohesive characteristics to be lost. Another major factor involved in controlling stand-up time is groundwater flow. Flow into the tunnel creates high seepage-pressure gradients near the tunnel face that can destabilize the soil, causing it to flow into the tunnel.

The so-called Tunnelman's Ground Classification System is a qualitative description of the types of soil behavior encountered during tunneling. Table 8.2-7 presents the system as proposed by Terzaghi (1950) and modified by Peck (1969) and Heuer (1974).

The classification is more than a description of the properties of the soil because it describes behavior under some of the conditions encountered in a tunnel. Construction methods, construction equipment, and tunnel geometry all influence ground behavior.

The stand-up characteristics of soil are the primary factors that define the categories. The initial cohesion or cementing of a soil differentiates between firm, raveling, and running ground. Groundwater seepage can transform running ground into flowing ground. Swelling and squeezing ground are most likely in shales or clays of high plasticity. Protection of the shale from disturbance and access to water minimize any swelling pressures that develop. Thus, early placement of a continuous lining against potentially swelling surfaces is usually desirable. If pressures still are high, then procedures to allow the ground to displace in a controlled manner can be devised.

Squeezing Ground

The severity of squeezing ground in a tunnel can be estimated from the ratio of overburden pressure γH to the undrained shear strength s_u of the clay (Peck 1969). Ratios greater than 5 produce the equivalent of bearing-capacity failure of the clay surrounding the tunnel, with rapid squeezing into unsupported areas in the tunnel face or into voids around the shield and lining. Compressed-air pressures used in the tunnel are subtracted from the overburden pressure to reduce the ratio to acceptable levels. Although squeezing conditions are not as severe for ratios less than about 4, some time-dependent movement and loading of supports takes place for ratios between 1 and 4.

Squeezing relationships can be evaluated not just for soft clays in shallow tunnels but also for deep rock tunnels in shales and fault zones. In mine pillars or abutments where mining nearby has resulted in transfer of large stresses to the pillars, the stresses that cause failure or time-dependent movements are greater than the overburden pressure and are a function

Table 8.2-8 Strength (back-calculated and estimated) and creep parameters for various classes of shale

Ground Class	Description	ϕ, degrees	q_u, psi	λ, %	$E_s/(\sigma_1-\sigma_3)_f$	R_f	Diametral Closure, % at 3 months
CL I	Sandstone and siltstone, widely spaced joints	50	9,000	—	200–500	0.1	
CL II	Siltstone to shale, moderately jointed	45	2,000–4,000	1–2	250–300	0.1–0.2	0.2 to 0.4 (steel ribs)
CL III	Shale, closely jointed with some thin shears	35	500–1,000	5–7	250–300	0.5	1 to 2 (steel ribs)
CL IV	Shale, closely jointed and sheared, shear zones contain crushed and soft materials	25	200–300	7–10	200–250	0.8–0.9	0.8% (concrete segments)
CL IVb	Shale, wide shear zones of crushed and soft materials	25	0–100	12–15	150–200	0.9	4% (steel ribs)

Source: Cording and Cepeda-Díaz 1992.
Note: Steel ribs developed yield stresses (40 psi radial pressure) in Class III–IVb ground.
Concrete segments were effectively stiffer and developed approximately 100 psi radial pressure.

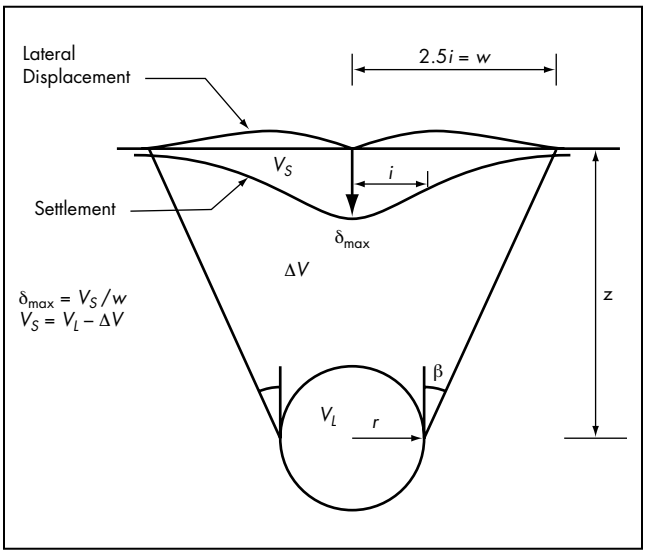
of the degree to which adjacent material has been removed in addition to other factors. The time rate of closure and buildup of pressures on linings in squeezing ground can be evaluated using creep relationships such as given in Equation 8.2-12. Using this equation to describe the stress–strain–time relationships, creep closure around a circular tunnel instantly excavated in a hydrostatic stress field was analyzed and compared with the results of field measurements of the Stillwater Tunnel in Utah (United States), excavated at a depth of 2,000 ft (600 m) in shale containing fault zones and closely fractured zones subject to squeezing (Phienweja 1987). Creep parameters were estimated not only from laboratory creep tests on shale and sheared clay gouge but also from measured closure and lining pressures in various rock-mass classes. Table 8.2-8 lists the creep parameters and typical values of pressure and deformation that developed for the ground classes. The slope of the (log of tunnel closure) – (log of time) plot was initially greater than λ as the headings advanced but approached λ with time. Pressures that develop on a lining installed soon after excavation are also strongly influenced by λ.

Estimating Ground Movements Around Shallow Tunnels

In any tunnel, the ground in the heading must be controlled to prevent instability before and during installation of the tunnel lining. When advancing shallow tunnels in soil in the vicinity of structures and utilities, a further requirement is to limit ground movements to prevent unacceptable movements and damage. Ground control measures must be selected with due consideration to the stand-up time of the soil. Two types of ground loss can occur: (1) large loss of ground, often in a sudden, uncontrolled, or catastrophic manner; and (2) the regular, smaller ground movements that occur throughout the length of the tunnel drive.

Large ground losses usually occur by means of running, flowing, or squeezing of soil into the tunnel face, but can also occur by means of soil flow through the tunnel lining or collapse of an unstable lining. Tunneling procedures should be selected to minimize the risk of large ground losses for the anticipated ground conditions.

Regular ground losses occur around the shield and lining, during excavation and installation of the lining, throughout the tunnel drive. For a shielded tunnel, most regular ground losses occur when the shield cuts a perimeter that is several inches (tens of millimeters) outside the outer perimeter of the installed support. Ground losses are minimized by a shield that has minimum overcutting and is easily steered on line and grade, and by rapid and complete expansion or grouting of the lining behind the tail of the shield.



Source: Cording and Cepeda-Díaz 1992.
Figure 8.2-16 Settlement trough formed by ground loss in a tunnel

The volume of ground loss in a tunnel is distributed through the soil mass and results in a settlement trough of volume V_S that is less than the volume of ground loss V_L by the amount of volume expansion ΔV in the ground (Figure 8.2-16). The denser the soil and the larger the ratio of tunnel depth to diameter, the greater is the volume expansion, which reduces the volume of the surface settlement trough for a given volume of ground loss around the tunnel. In soft clays, V_S is approximately equal to V_L, but exceeds V_L if drainage or disturbance of the clay surrounding the tunnel results in consolidation with time.

The half-width w of the settlement trough can be estimated using procedures described by Peck (1969) and Cording and Hansmire (1975). For a settlement trough shaped like a normal distribution curve with an inflection point i, $w = 2.5\,i$. Typically, for sands, the angle β is in the range 10°–30°, and for clays, β is in the range 25°–45°. With these relationships, the magnitude and distribution of surface settlements and the damage to structures can be estimated for given volumes of ground loss.

Controlling Ground Movements

A variety of tunneling procedures and shield designs can be used to control the face as well as limit movements around the shield and lining. Open-face shields or shields using

mechanical diggers support the face with muck that is excavated and with additional breasting boards or plates positioned against the face. Wheeled excavators must control the face by rotating the wheel only sufficiently to excavate a volume of soil equivalent to the volume of the tunnel that is advanced. For both of these techniques, additional measures such as compressed air or dewatering may be used to prevent inflow of soil, and it may also be necessary to prevent running by use of techniques such as chemical grouting of the soils ahead of the face. Pressurized face shields may be used in lieu of compressed air to stabilize the face; the shields use air, water, a slurry, or excavated muck to provide positive pressures in a chamber at the face of the shield to balance the water and earth pressures in the soil ahead of the face. Even with this technique, additional ground control measures may be necessary to prevent loss of ground in the face.

Procedures used to modify ground and groundwater conditions before tunneling include dewatering, chemical or cement grouting to penetrate the ground, jet grouting to form soil–cement columns around the tunnel, and freezing. Structures that would be damaged by settlements can be directly supported by underpinning or tied to reduce lateral extension of the structure. Compaction grouting can be used as required during tunneling to redensify ground that has loosened above the tunnel before it causes surface settlement. Compaction grouting involves placing a low slump grout at high pressures above the tunnel, usually after the shield has passed, in holes drilled down from the surface or up from the tunnel (Baker et al. 1983).

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CHAPTER 8.3

Slope Stability

Charles A. Kliche

INTRODUCTION

Because of the discontinuous nature of rock, the design of stable rock slopes is as much an art as it is applied engineering. Experience is as important as the proper use of the theories of soil and rock mechanics, structural geology, and hydrology.

Many computerized tools are now available to rock slope design engineers. It is important that these engineers understand the basic theory of rock slope stability (or instability) before they attempt to use many of the computerized methods—especially before they attempt to interpret and apply the results.

The civil engineering field uses mainly soil mechanics principles for slope-stability analysis. This field is mostly concerned with slopes cut in soil-like materials. The analysis is used mainly for foundations, buildings, or dam sites; road cuts; cut-and-cover tunneling; irrigation ditches and channels; tailings dams; and mine waste or lean ore repositories.

On the other hand, the mining engineering field uses mainly rock mechanics principles to analyze the stability of slopes cut in rocks. Rock mechanics is more complicated than soil mechanics for the following reasons (Brawner and Milligan 1971): rock materials are heterogeneous and usually anisotropic; strength parameters relating to rock masses are infinitely variable and difficult, if not impossible, to determine precisely; generalized models and theories of rock behavior are complex, as are the mathematics involved; field conditions are extremely difficult and often impossible to duplicate in the laboratory; field testing is usually complicated and time-consuming and is almost always very expensive.

Rock slope engineering is the application of rock mechanics principles and structural geology principles to the stability of a slope cut in rock. It is a specialized branch of geomechanical engineering. It includes not only kinetic analysis (possible modes of failure) and kinematic analysis (stability of the failure modes) but also probabilistic analysis, specialized numerical methods, slope stabilization methods, groundwater analysis, geologic data collection and interpretation, slope-monitoring methods, and specialized rock-blasting techniques.

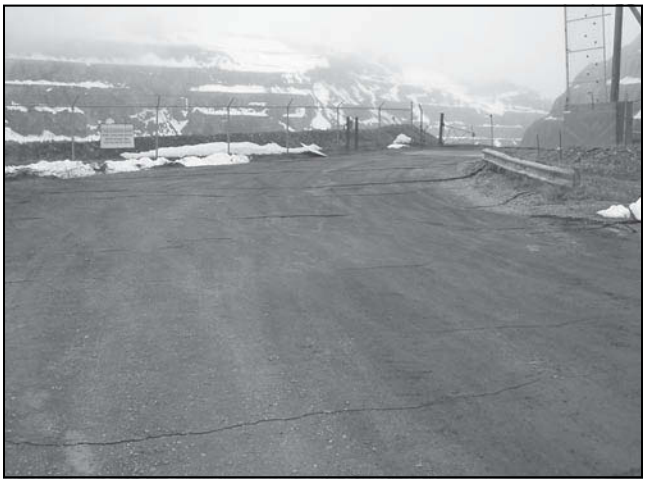
Stability Criteria

From the standpoint of simple mechanics, the stability of a slope is the ratio of the strength of the material (resisting forces, stresses, or moments) to the stresses in the slope tending to destabilize it (driving forces, stresses, or moments). If the destabilizing stress exceeds the strength, the slope is unstable, and the factor of safety is less than unity. Conversely, if the strength exceeds the destabilizing stress, the slope is stable, and the factor of safety of the slope is greater than unity. This ratio, termed the *safety factor* or *factor of safety*, has been the basis for stability analysis in civil engineering for many years. The safety factor to which the slope should be designed is a function of the risk involved, namely, the condition of the slope and the exposure.

Because of the variability of rock properties, uncertainty in the measurement of these properties, and the influence of quasi-random events such as earthquakes and rainfall, the stresses and strengths used in stability are estimates of populations with significant distributions rather than single values. For this reason, an alternate approach to defining stability is to use the reliability (probabilistic) method, whereby the probability of whether or not a slope will be stable is calculated from the distribution of input values (i.e., the distribution of parameters such as cohesion, internal angle of friction, water pressure, transitory forces, etc.).

Slope instability does not necessarily mean slope failure from the operational standpoint. It is not uncommon for a slope to become unstable, with the resulting displacement being <1 m (3 ft). An example of this type of instability is the appearance of tension cracks along the crest of a slope without overall slope failure (Figure 8.3-1). Whether an unstable slope results in significant cost to the operation depends on the rate of movement, the type of mining operation, and the relationship of the unstable material to the mining operation. Unstable areas with displacement rates of >100 mm/d (4 in./d) have been successfully mined by truck and shovel operations. On the other hand, a few millimeters of displacement of the rock under a crusher, conveyor, or building may require extensive

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Courtesy of Homestake Mining Company.
Figure 8.3-1 Tension cracks at the crest of a highwall

repair and stabilization. When the rate of displacement is such that it disrupts the operation or the movement produces damage to mining facilities, it is considered an operational slope failure. A similar economic concept was used by Varnes (1978) to distinguish between creep and landslides. He restricts the lower limit of the rate of movement of landslide material "to that actual or potential rate of movement which provokes correction or maintenance."

Safety

Many rock slopes move to varying degrees during the course of their operational lives. Such movement indicates that the slope is in a quasi-stable state, but this condition may continue for many years, or even centuries, without failure occurring. However, in other cases, initial minor slope movement may be a precursor for accelerating movement followed by collapse of the slope. Because of the unpredictability of slope behavior, movement monitoring programs can be of value in managing slope hazards, and they provide information that is useful for the design of remedial work.

Slope movement is most common in open-pit mines, and many mines continue to operate safely for years with moving slopes that are carefully monitored to warn of deteriorating stability conditions. Other slopes that undergo long-term movement are landslides that may creep for hundreds of years resulting in accumulative movement of tens of meters. Such movement may comprise an approximately uniform creep rate, on which may be superimposed short periods of more rapid movement resulting from such events as earthquakes, unusually high precipitation periods, and human activities. Creep has been defined by Varnes (1978) as "extremely slow distributed movements in both rock and soil." Human activities that can contribute to slope instability include adding a surcharge weight to the crest of the slope, excavations at the toe of the slope, transitory earth stresses (blasting and vibrating equipment), and changing the groundwater conditions by dam filling or irrigation.

When a rock or soil mass is disturbed, either by the actions of people or by natural events, it undergoes a redistribution of stresses, resulting in a change in shape. This readjustment is reflected in displacements, deflections, pressures, loads, stresses, and strains that can be detected and measured. Many of the same measurement methods and instrumentation techniques can also be used to investigate the mechanical properties of the mass, the interaction between the mass and any associated artificial structures, and the effectiveness of remedial measures proposed to correct defects in either the mass or the structures.

Monitoring is the surveillance of engineering structures either visually or with the aid of instruments (Brown 1993). The objectives of a rock slope monitoring program are (Call 1982)

- To maintain a safe operation for the protection of personnel and equipment;
- To provide advance notice of instability, thus allowing for the modification of the excavation plan to minimize the impact of the instability; and
- To provide geotechnical information in order to analyze the slope failure mechanism, design appropriate remedial measures, and/or conduct a redesign of the rock slope.

Slope Geometry

Three main components of an open-pit slope design have been defined (Wyllie and Mah 2004): (1) the overall pit-slope angle from the toe of the excavation to the crest, which incorporates all ramps and benches between; (2) the interramp angle, which is the slope, or slopes, lying between each ramp or ramp segment that depends on the number of ramps and their widths; and (3) the face angle of individual benches, which depends on the vertical spacing between benches, or combined multiple benches, and the width of any berms (or "catch benches") included therein (Figure 8.3-2). The interramp angle is the slope angle produced normally by a number of benches. Where there are haul roads, working levels, or other wide benches, the overall slope angle will be flatter than the interramp angle. And, normally, the bench face angle will be the steepest angle, often approaching vertical. It is important in slope design to consider these components. For example, in the case of bedding dipping into the pit at 40°, as shown on Figure 8.3-2, the daylighting plane shear criteria would result in a design angle of 40°. If this angle were used for the overall slope angle, haul roads cut into the slope would undercut the bedding and result in interramp instability. In addition, there would be almost no catch benches left, as the bench face angle would be steeper than the bedding.

The Detailed Slope-Stability Study

The detailed slope-stability study can be divided into two parts: field work, and office and laboratory work.

Field Work

Field work is the data collection phase of the detailed slope-stability investigation. It should include

- Compiling and editing topographic maps;
- Detailing of geologic mapping (1:1,000 to 1:500)
  - Mapping formation boundaries on about 1:1,000 scale
  - Mapping structural features within the boundaries; using a geological compass to map the structures; putting the mapped structures on the survey map;
- Ascertaining the location of groundwater discharge points in the study area. Determining discharge amounts (monitoring at least twice per year for at least 2 years);
- Collecting rock samples for laboratory testing;

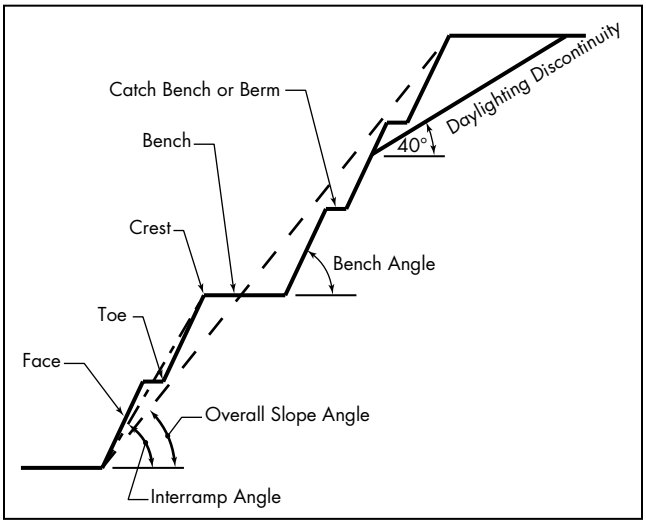


Figure 8.3-2 Highwall slope configuration with 40° daylighting discontinuity

- Conducting a core drilling program (core recovery should be at least 90%), if needed;
- Drilling, and providing instrumentation for, any needed groundwater monitoring wells; and
- Conducting any necessary geophysical studies.

Office and Laboratory Work

Office and laboratory work is the data reduction and interpretation phase of the detailed slope-stability study. This phase should include

- Development of systematic working cross sections: radial for open pits, axial for long pits or road cuts;
- Discontinuity evaluations;
- Laboratory testing of samples collected from the field
  - Testing of intact rock specimens (uniaxial, triaxial, maybe direct shear)
  - Testing of discontinuity surface samples (direct shear);
- Evaluation of the stability of proposed slopes by sector;
- Determination of the seismicity of the region; and
- Stability analysis sector-by-sector, based on available data and sound engineering judgment. If the stability (based on factor of safety, probability of failure, numerical modeling, or other acceptable technique) along a cross section is below the acceptable level, take one or more of the following steps: (1) accept the risk of failure, (2) flatten the slope, (3) use artificial support, and/or (4) drain the slope, if appropriate. Steps 2 through 4 are normally within the responsibilities of the geotechnical engineer, whereas Step 1 would be a management decision.

GENERAL MODES OF SLOPE FAILURE IN ROCK MASS

Numerous primary, and what will be called secondary, modes of slope failure in rock masses have been identified (Hoek and Bray 1981; Kliche 1999; Wyllie and Mah 2004; Call 1992). These modes include

- Plane shear (planar and wedge) failure,
- Rotational failure (including curvilinear or spiral failure surface),
- Toppling failure (including complex planar plus toppling),
- Rockfall,
- Stepped path (stepped plane and stepped wedge),
- Flexural thin-bed failure, and
- Two-block (active/passive blocks).

In order to make a quantitative estimate of the stability of a slope, analytical models amenable to mathematical solutions must be used. The requirements for stability analysis using any of these models are the slope and failure geometry; material, hydrological, rock mass and strength properties; and any assumptions regarding stress distributions and strength.

Geologic Model

The influence of rock sample size on strength has been widely discussed in geotechnical literature, and it is normally assumed that there is a significant decrease in rock strength as the sample size increases. A general discussion of this size–strength relationship can be found in Hoek (2007).

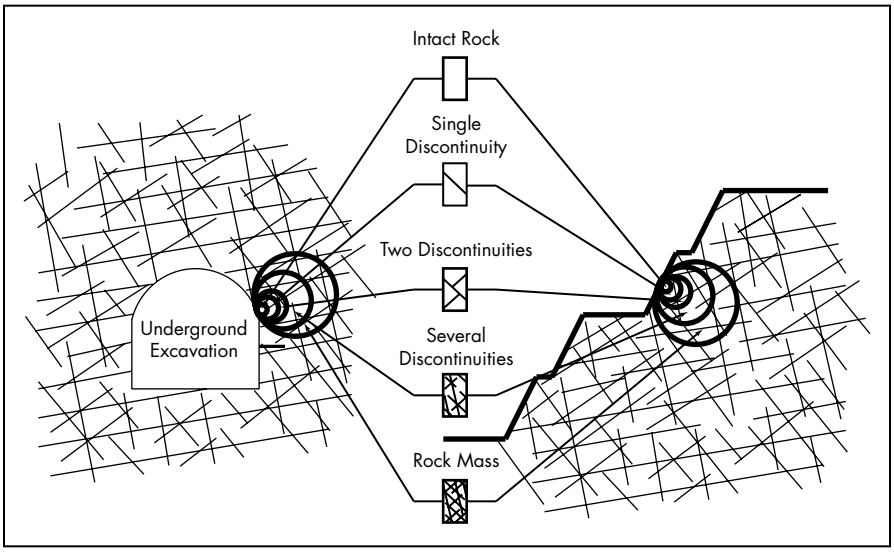
The rock of a slope can be considered to consist of the following components (Figure 8.3-3).

- **Intact rock:** The primary unbroken rock as determined from a piece of core cut for compression testing. The term *rock substance* has also been used for the unbroken rock.
- **Discontinuities:** The term *discontinuity* refers to any of a number of different types of defects in the rock fabric. Discontinuities, representing weakness planes in the mass, control the engineering properties by dividing the mass into blocks separated by fractures such as faults, joints, foliations, and bedding. Joints are the most common defect in rock masses; faults are the most serious. All discontinuities have the following physical properties: orientation, spacing, width of opening, intensity, and roughness.
- **Rock mass:** The combination of intact rock and fractures considered as a unit. Soil could be considered a special case of rock mass.
- **Major structures:** Geologic features such as faults or shear zones along which displacement has occurred and that are large enough to be mapped and located as individual structures. There is actually a continuum between fractures and major structures, but the differentiation is useful for design purposes.
- **Minor structures:** These include fractures, joints, bedding planes, foliation planes, and other defects in the rock mass, generally of lesser importance with respect to rock slope instability.

Collecting and Using Geologic Structure Data

The two most important rock mass factors that control slope stability are (1) discontinuity strength, and (2) the location and orientation of the discontinuities with respect to the slope. This is because the major source of slope instability in rock slopes is failure along preexisting discontinuities. Other important factors include the rock material type, the discontinuity type, discontinuity (joint) intensity, the discontinuity surface roughness, the amount of attachments across the discontinuity, and the type and nature of filling material (if any) between the two planes of the discontinuity.

Geologic structures, which are important for open-pit slope stability, can be divided into rock fabric, which is defined as geologic structures that are too numerous to be evaluated



Source: Adapted from Hoek 2007.

Figure 8.3-3 Transition from intact rock to rock mass

individually and are, therefore, treated statistically in slope design analysis; intermediate structures, which include only major structures that are larger than a lower-limit baseline; and regional structures, which are major structures that are of a regional scale, and are often faults with measured length >100 m (>330 ft) (Nicholas and Sims 2000).

The recording and analyzing of the orientation of discontinuities is normally the first step in the investigation of a slope for the formation of potentially unstable blocks of rock (Hoek and Bray 1981; Wyllie and Mah 2004). Since patterns of jointing are invariably preferentially developed over reasonably large areas, the degree of stability of slopes of the same pit will vary, since some slopes will be formed in rock with adversely oriented joints and others not.

Figure 8.3-4 shows a lower hemisphere stereographic plot of what is known as the Markland test for the possibility of planar-type failure, wedge-type failure, and toppling-type failure. For plane shear failure (or wedge-type failure) to be kinematically possible, the following conditions must be met (Hoek and Bray 1981; Piteau and Martin 1982):

1. The dip of the discontinuity (or plunge of the line of intersection of two discontinuities) must exceed the angle of friction, ϕ, for the rock surface.
2. The discontinuity (or line of intersection of two discontinuities) must daylight in the slope face.
3. The dip of the discontinuity (or plunge of the line of intersection of two discontinuities) must be less than the dip of the slope face, ψ.

These conditions can be expressed by the following relationship, and are also illustrated in Figure 8.3-4:

$$\psi > \beta > \phi$$

where

ψ = dip of the slope face
β = dip of the discontinuity or plunge of the line of intersection
ϕ = angle of friction for the rock surface

For toppling failure to be kinematically possible, the discontinuity set must dip steeply back into the slope face (undercutting condition) and the strike of the discontinuity set must be approximately parallel to the strike of the slope face.

The pole to a plane in space is the normal to that plane, projected to intersect the lower half reference of the stereographical projection. Pole plots, in which each plane is represented by a single point, are the most convenient means of examining the orientation of a large number of discontinuities (Wyllie and Mah 2004). The plot provides an immediate visual depiction of concentrations of poles representing the orientations of discontinuity sets and, along with the orientation of the slope face, will indicate which failure modes are possible.

Plane Shear Failure Geometries

The plane shear failure mode refers to a situation in which displacement occurs along one or more geologic structures and the failure mass is considered to be a rigid block or a number of blocks. These geometries are shown in Figure 8.3-5. A brief description of each of the plane shear modes follows.

- **Planar.** The planar mode is the simplest geometry consisting of a single plane striking nearly parallel to the slope (within 15°–20°). The structure must have a dip flatter than the slope angle, must be long enough to reach the surface or a tension crack, and the rock–rock apparent friction (incorporating rock friction, roughness, and cohesion) must be less than the dip of the discontinuity. Since the stability analysis is two dimensional, the width of the failure must be great enough that the end results are negligible, or there must exist boundary structures that define the lateral extent of the failure. Figure 8.3-6 shows a steeply-dipping discontinuity daylighting in the bench face upon which planar failure occurred.
- **Step path.** The step path geometry occurs where there is a fracture set dipping into the pit in the orientation given for the planar mode, but no individual fracture is long enough to form a single, continuous failure plane. Sliding

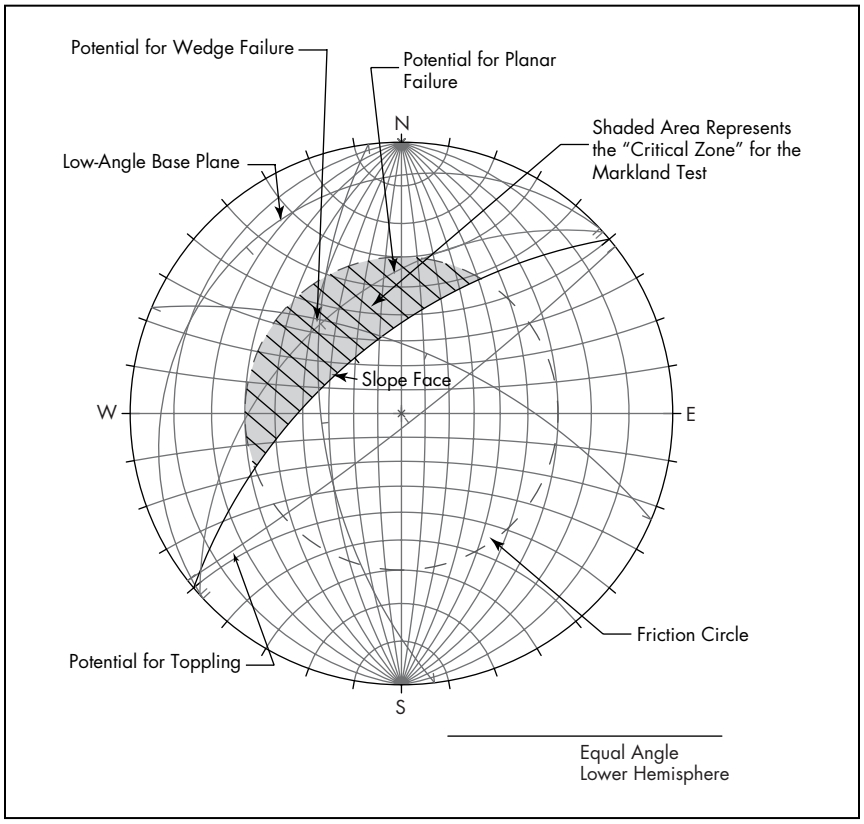


Figure 8.3-4 Stereographic projection of wedge-type failure, planar-type failure, and toppling failure modes

is assumed to occur along fractures in the master joint set, and separation along fractures approximately perpendicular to the master joint set or tensile failure of the rock between the master joints.

- **Wedge.** The wedge failure geometry is the result of two planar geologic structures intersecting to form a detached tetrahedron of material. Sliding can occur down the intersection or on one of the two planes, with separation on the other plane. In some cases the sliding on one will demonstrate elements of rotation rather than simple translation. Figure 8.3-7 shows a wedge-type failure at an open-pit mine.
- **Step wedge.** The step wedge is similar to the simple wedge except that one or both of the failure surfaces are step paths. The stepped-path follows a complex path along numerous intersecting discontinuities.
- **Two-block.** The two-block is a two-dimensional plane shear geometry where there are two plane shear structures dipping into the pit, with a third structure dipping back into the wall that divides the failure into an active and a passive block.
- **Slab.** Where there is bedding or foliation parallel to the pit, slope instability can occur even though the structures are not daylighted. The possible failure mechanisms are crushing at the toe, a two-block geometry formed by joints at the toe, and buckling.

Nonplanar Failure Surfaces

Nonplanar failure surfaces include rotational shear and general surface failures.

In a soil or weak rock mass slope where there are no geologic structures that control the failure, with rotational shear the most unstable failure surface is approximately a circular arc. The radius and location of the most unstable circle (the critical circle) depends on the material properties and must be found by iterative solutions of trial circles. The stability of the circular arc is usually analyzed by one of the many methods of slices (Bishop's method, the Ordinary Method of Slices, or Janbu's method, for example).

The general surface is a mixed mode failure in which part of the failure surface is structurally controlled and part is failure through the rock mass. An example would be a nondaylighted plane shear. Any of the many methods of slices may be used to analyze the stability of the general surface.

Other failure modes include block flow, toppling, and rockfalls and raveling.

Compared with underground rock mechanics, in block flow failure mode the stresses in a pit slope are low and do not exceed the rock mass strength. Thus, most slope instability is controlled by geologic structure. However, in deep pits there is the possibility that the stresses in the toe of the slope would be sufficient to result in the crushing failure of the rock mass, particularly if there was a high horizontal stress. This mode of instability is referred to as block flow by Coates (1981).

A conceptually possible variation of the block flow would be a situation where the rock mass under confinement in the slope wall yields plastically. The resulting deformation would be plastic flow such as occurs in a glacier. At the surface of the pit slope, where there is no confinement, secondary sliding block failure would occur similar to the calving of a glacier.

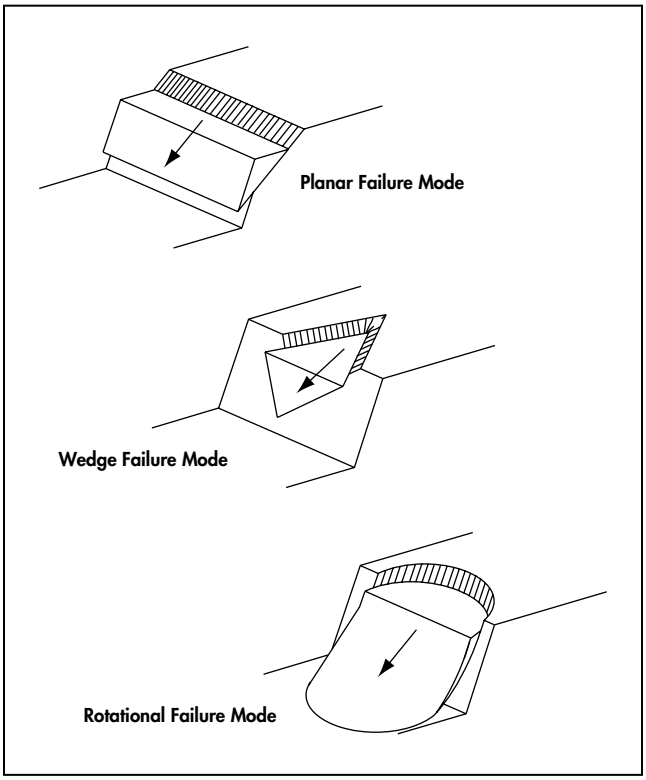


Figure 8.3-5 Typical shear failure modes (planar and rotational)

This is a possible explanation for situations where instability occurs in a relatively flat slope and the back analysis indicates unusually low shear strength.

Toppling is possible whenever a set of well-developed or throughgoing discontinuities dips steeply into the slope (Piteau and Martin 1982). In this type of failure, long, thin columns of rock formed by the steeply dipping discontinuities may rotate about a pivot point located at the lowest corner of the block. Therefore, the governing factor for toppling of a block is the location of the weight vector of the column with respect to the pivot point on the block. If the block thickness-to-height ratio is less than the tangent of the dip of the base plane ($t/h < \tan\beta$), then the resultant force due to the weight of the block will occur outside the toe of the block, and an overturning moment will develop about the pivot point. Toppling is most commonly observed as a secondary failure mechanism resulting from displacement caused by another mode of instability (Figure 8.3-8).

With rockfalls and raveling, bench faces are normally cut as steeply as the excavator being employed can dig them. As a result, individual blocks in the face are at or close to limiting equilibrium, and disturbing forces can dislodge them. The primary disturbing forces are gravity, freeze–thaw, transitory earth motion (vibrations), and water from rainfall. The action of these disturbing forces can dislodge individual blocks, producing a rockfall. The dislodging of large numbers of blocks is termed *raveling*. Weathering can also produce raveling by the deterioration of the material supporting the blocks. In principle, the stability of individual blocks could be analyzed, and there are commercial computer programs available to do so. However, there is no practical method of conducting stability analyses for raveling on a pit scale. The design approach, therefore, is to provide for adequate catch benches.



Courtesy of AngloGold Ashanti.

Figure 8.3-6 Steeply-dipping discontinuity daylighting in the bench face upon which planar failure occurred



Source: Kliche 1999.

Figure 8.3-7 Wedge-type failure

Slope-Stability Assessment Techniques

Techniques for slope-stability assessment are discussed in this section.

Limit Equilibrium Technique

Limiting equilibrium methods are the most common analysis techniques used for the solution of rock slope stability problems. At limiting equilibrium, by definition, all points along the potential failure surface are on the verge of failure. At this point in time, the driving forces (or stresses, moments) just equal the resisting forces (or stresses, moments), and the factor of safety equals unity. Where the resisting forces of a slope are greater than the driving forces, the factor of safety is greater than unity and the slope has a higher probability of remaining stable; when the resisting forces are less than the driving forces, the slope is unstable. Failure modes that can be analyzed by limiting equilibrium techniques include plane shear (planar failure, wedge failure), rotational shear, and toppling failure.



Courtesy of Mine Safety and Health Administration.

Figure 8.3-8 Toppling-type failure

The stability of rock slopes that can be modeled by plane shear at limiting equilibrium depends on the shear strength generated along the sliding surface. For all shear-type failures, the rock can be assumed to be a Mohr–Coulomb material in which shear strength is expressed in terms of cohesion, c, and friction, ϕ , by the equation

$$S = c + \sigma \times \tan$$

where

S = shear strength along the failure plane
c = cohesion generated along the failure plane
σ = normal stress across the failure plane
ϕ = angle of friction for the rock surface

Limit equilibrium analysis can apply to a wide range of conditions and can incorporate additional forces such as surcharge (W_s), horizontal water pressure (V), normal or uplift water pressure (U), vibratory forces ($k_h \times W$, where k_h is the horizontal seismic coefficient), and rock bolt forces (T) applied at some angle inclined upward from horizontal (+θ).

Figure 8.3-9A shows a slope containing a throughgoing discontinuity with area, A, and dip, β, and a vertical tension crack. The slope is partially saturated such that the tension crack is half-filled with water, and the water table daylights in the slope face at the same place as the failure surface. No surcharge weight is to be added to the slope and vibratory forces will be neglected. A tensioned rock bolt force (active case), inclined at angle +θ and anchored below the failure plane, will be added to the slope.

Figure 8.3-9B shows the free-body diagram of the rock wedge with the driving forces and resisting forces resolved. If the resisting forces are equated to the driving forces at limiting equilibrium (FS = 1.0), the following equation for factor of safety (FS) can be developed:

$$\mathrm{FS} = \frac{c \times A(W\cos\beta - U - V\sin\beta + T\sin(\theta+\beta))\tan\phi + T\cos(\theta+\beta)}{W\sin\beta + V\cos\beta}$$

where

c = cohesion generated along the failure plane
A = surface area of the failure plane
W = weight of the sliding mass
β = inclination (dip) of the throughgoing discontinuity
U = uplift water force on the sliding plane
$= \frac{1}{2}\gamma_w \times h_w$
V = horizontal water force on the tension crack
$= \frac{1}{2}\gamma_w \times h_w^2$
h_w = height of water in the tension crack
T = rock bolt force
θ = angle of inclination of rock bolt force (+ is ↑ from horizontal)
ϕ = angle of friction along the throughgoing discontinuity

For the *active* rock bolt case, the assumption is that the component of T parallel to the failure plane tends to increase the resisting forces along the potential failure plane; whereas, for the *passive* rock bolt case, the component of T acting parallel to the failure plane will be mobilized as the failure mass moves and, therefore, the horizontal force T will tend to decrease the driving forces.

This analysis is valid for the model of block sliding on an inclined plane at limiting equilibrium. If, on the other hand, the block is of such height-to-width ratio that it may tend to rotate about its lowermost corner (topple instead of slide), the FS can then be calculated as the ratio of the resisting moments to the driving moments:

$$\mathrm{FS} = \frac{\Sigma(\text{resisting moments})}{\Sigma(\text{driving moments})} = \frac{W\cos\beta \times (t/h)}{W\sin\beta \times (t/h)} = \frac{(t/h)}{\tan\beta}$$

where t/h is the block thickness-to-height ratio.

For limiting equilibrium of rotational-type failures, the model generally applied is one of many methods of slices. These models generally differ only in the assumptions needed to make the equations statically determinate.

Rotational-type ("circular") failure may occur in slopes composed of soil or soil-like material (i.e., overburden repositories, waste dumps, fills, dragline spoil piles, etc.). Much of the development of rotational failure analysis methods was carried out in the 1950s and 1960s, resulting in the development of several easy-to-use stability charts (see Hoek and Bray 1981; Perloff and Baron 1976; Naval Facilities Engineering Command 1986). Moreover, several very good commercial computer packages are available that offer deterministic, probabilistic, sensitivity, and back-analysis capabilities using one or more of a number of built-in analysis techniques.

For rotational failure to occur, the total shear stress along the failure surface must be equal to or greater than the shear strength along the surface. The stability analysis of rotational failure is carried out using the limit equilibrium procedure similar to that discussed earlier. However, the application of this procedure to rotational failure involves the division of the slope (potential failure zone) into a series of slices that are usually vertical but may be inclined to coincide with certain geological features (Wyllie and Mah 2004). The base of each slice should be wholly contained within a soil layer to make calculations a bit easier. The forces acting on each slice are the normal forces on the base, the interslice normal forces, the interslice shear forces, the weight of each slice, and the shear force along the base (Figure 8.3-10). This results in $5 \times n - 2$ total unknowns for n slices in rotational failure analysis. If the rotational failure problem is to be solved in the x–y plane (using a unit width of slice through the assumed failure), and if it is assumed that the slope is in equilibrium, then the total

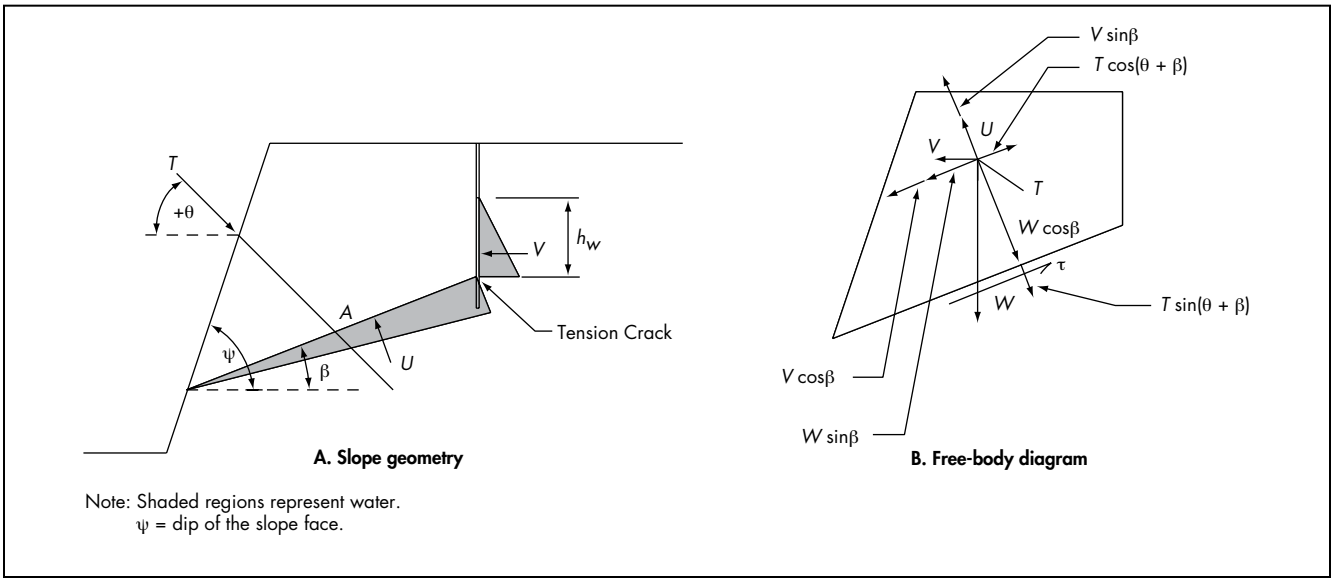


Figure 8.3-9 Plane shear failure geometry showing a slope with a throughgoing discontinuity

number of equations available is the $3 \times n$ equations of statics equilibrium. The methodologies differ mainly in the assumptions made to reduce the equations in order to allow for the solution for the FS (another unknown and assumption is that the FS does not vary slice-to-slice).

Probabilistic Method

The probabilistic method, used most commonly with the Monte Carlo technique, uses known or assumed probability distributions of critical or sensitive parameters to examine the possible effect that variations in these parameters may have on the overall design factor of safety. That is, by simulating different values of the critical parameters (random variables) within the measured distributions (friction, cohesion, dip, dip direction, water level, joint continuity, etc.), a distribution of factor of safety values can be generated for the slope. This can then be used to answer the critical question, "What is the probability that the designed slope will have a factor of safety less than, for instance, 1.3?"

Typical probability density functions (PDFs) for some of the more important variables affecting slope stability are shown in Figure 8.3-11. Plotted on the abscissa (x-axis) is the range of measured values (e.g., cohesion, friction angle, strike); plotted on the ordinate (y-axis) is a ratio representing the normalized number of occurrences within each class interval (i.e., number of occurrences of a dip measurement within the range of 28°–30° divided by the total number of dip angle measurements). This normalization procedure results in the sum of the area under the PDF equaling 1.0. Since the area under the PDF equals 1.0, it can now be used as a means to estimate the probability of occurrence of any value, x, within the range of observed data.

Some of the more important statistical terms frequently used in conjunction with this method are defined here. (See Everitt 2006, for example, for a more-detailed description of the terms.)

- **Random variables**, which include, for example, cohesion, friction angle, joint length, and joint spacing.
- **Probability distribution** describes the range of possible values that a random variable can attain and the probability that the value of the random variable is within any (measurable) subset of that range.
- **Sample mean** indicates the center of gravity of a probability distribution.
- **Sample variance** is defined as the mean of the square of the difference between the value of x_i and the mean value, $\bar{x}$.
- **Standard deviation** is given by the positive square root of the variance, s^2. In the case of the commonly used normal distribution, about 68.3% of the data will fall within ±1 standard deviation, and approximately 95.4% will fall within ±2 standard deviations (Figure 8.3-12).
- **Coefficient of variation** is a normalized measure of dispersion of a probability distribution. It is defined as the ratio of the standard deviation, s, to the mean.
- **Normal distribution** is a common type of probability density—the normal probability density (usually referred to as simply the normal distribution)—and is a mathematical function that takes the form of the familiar bell-shaped curve (Figure 8.3-12).

Since the normal probability density cannot be integrated in closed form between every pair of limits a and b, probabilities relating to normal distributions are usually obtained from a special table. The table pertains to the standard normal distribution, namely the normal distribution with a mean (μ) of 0 and a standard deviation (σ) of 1 (Miller and Freund 1985). In other words, one cannot directly solve the equation of the normal distribution for the value along the abscissa (z-value) given the corresponding accumulated area under the curve by integration, so special tables must be used that give close approximate solutions for the value of z given $F(x)$.

Therefore, if a probability between the limits of 0 and 1 is generated (by using a random number generator), then this probability can be used to generate a corresponding z value. And then the appropriate random variable value from the data set (x) can be determined from the equation

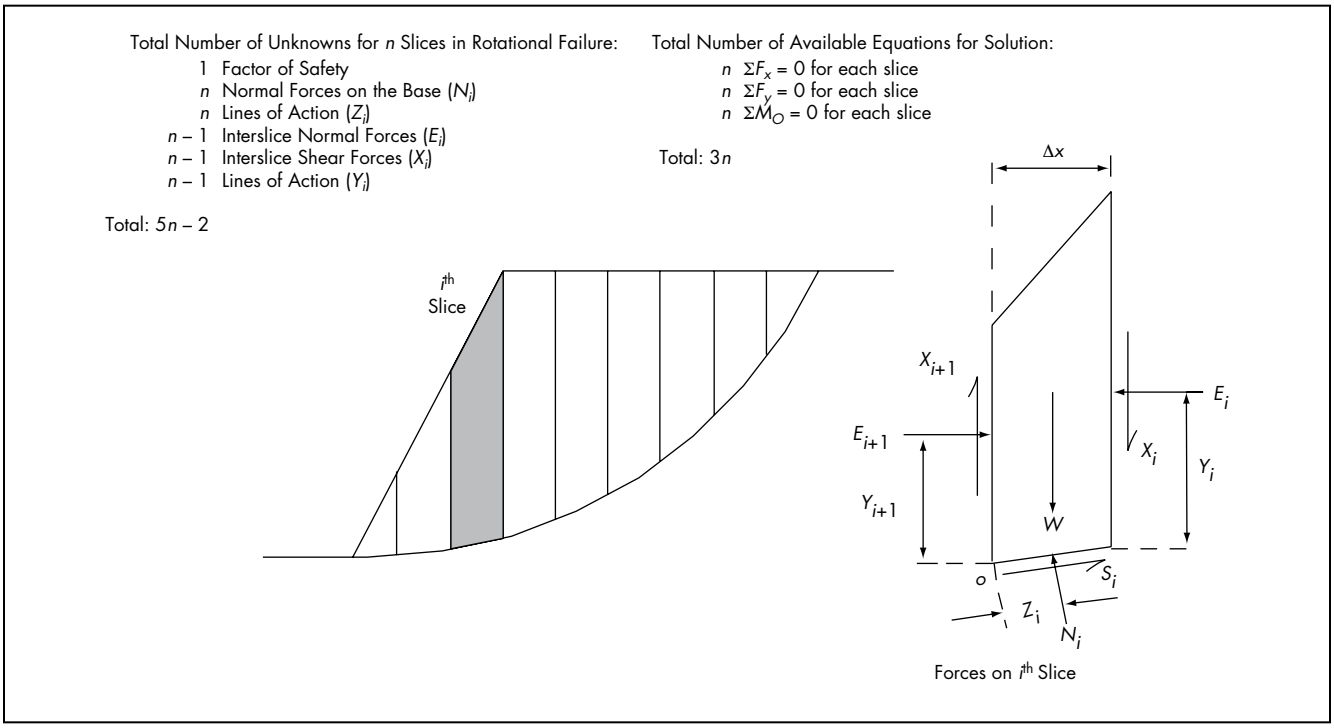


Figure 8.3-10 Statics for the methods of slices for rotational failure

$$z = \frac{x - \mu}{\sigma}, \text{ or } x = \mu + \sigma z$$

In addition to the normal distribution, other distributions commonly used in probability analysis include the lognormal distribution, the beta distribution (of which the uniform distribution is a special case), the exponential distribution, the multivariate normal distribution, and the Weibull distribution.

The Monte Carlo procedure, named for the administrative area of the same name in Monaco on the French Riviera in which Le Grand Casino is located, is a procedure, often computerized, that uses random or pseudorandom numbers to sample from probability distributions and, if a sufficiently large amount of samples are generated and used in calculations such as the factor of safety, then a distribution of values of the end product (factor of safety) can be constructed.

And finally, *numerical models* are defined as the computer programs that attempt to represent the mechanical response of a rock mass subjected to a set of initial conditions such as in-situ stresses and water levels, boundary conditions, and induced changes (e.g., slope excavations). The result of a numerical model simulation is typically either equilibrium or collapse. If the result obtained is equilibrium, the resultant stresses and displacements at any point in the rock mass can be compared to measured values. If the result obtained is collapse, the predicted mode of failure is demonstrated.

The rock mass is divided into elements (or zones) for numerical models. Each element is assigned a material model and properties. The material models are idealized stress/strain relations that describe how the material behaves. The simplest model is a linear elastic one, which uses the elastic properties—Young's modulus, Poisson's ratio—of the material. Elastic–plastic models use strength parameters to limit the shear stress that an element (or zone) may sustain.

The zones may be tied together, termed a *continuum model*, or separated by discontinuities, termed a *discontinuum model*. Discontinuum models allow slip and separation at explicitly located surfaces within the model.

For slopes, the factor of safety is often defined as the ratio of the actual shear strength to the minimum shear strength required to prevent failure. The methodology used in many of the numerical modeling techniques (finite element or finite difference) in order to compute the factor of safety is to reduce the shear strength until collapse occurs. The slope's factor of safety is then the ratio of the actual strength to the calculated reduced shear strength at failure.

To perform slope-stability analysis with the shear strength reduction technique, simulations are run for a series of increasing trial factors of safety (F). Actual shear strength properties, material cohesion (c) and material friction angle (ϕ), are reduced for each trial according to the following equations:

$$c_{\text{trial}} = \left(\frac{1}{F}\right)c$$

$$\phi_{\text{trial}} = \arctan\left(\frac{1}{F}\right)\tan\phi$$

If multiple materials and/or joints are present, the reduction is made simultaneously for all materials. The trial factor of safety is increased incrementally until the slope fails. At failure, the factor of safety equals the trial factor of safety (i.e., $FS = F$).

According to Lorig and Verona (2004), the shear strength reduction method has two main advantages over the limit equilibrium technique for slope-stability analysis. First, the critical slide surface is found automatically, and it is therefore not necessary to specify the shape of the surface (e.g., circular,

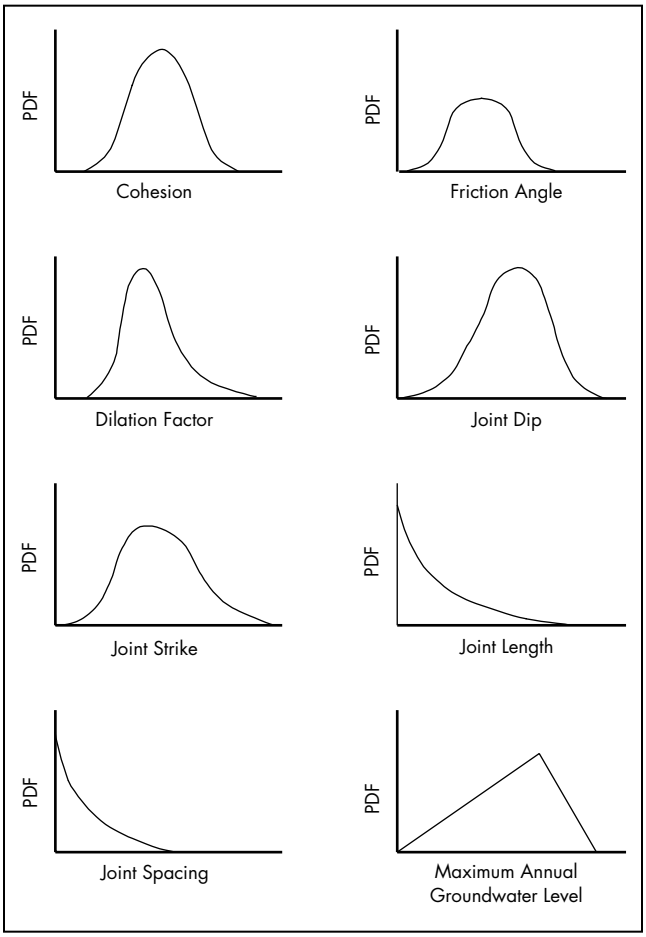
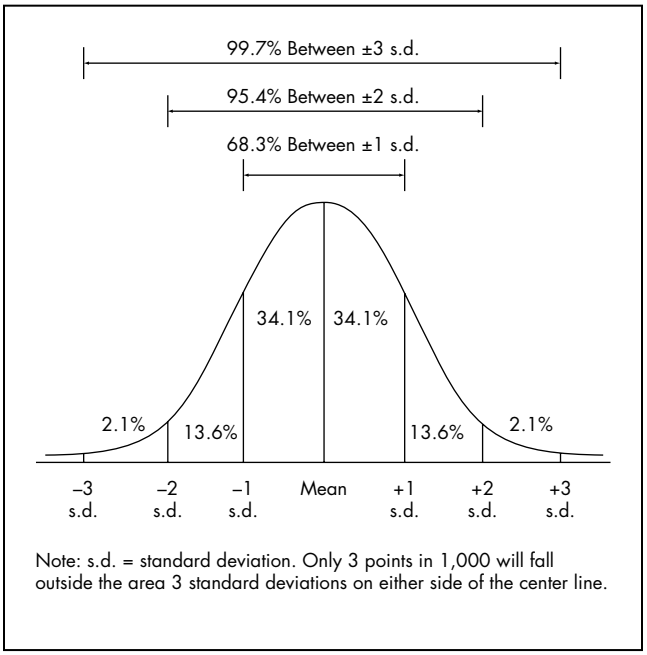


Figure 8.3-11 Typical PDFs for variables affecting slope stability



Source: Syque.com, n.d.

Figure 8.3-12 Normal probability distribution

curvilinear, spiral) in advance. Second, numerical methods automatically satisfy translational and rotational equilibrium, whereas not all limit equilibrium methods do so. Therefore, the shear strength reduction technique will normally determine a factor of safety equal to or slightly less than the limit equilibrium–obtained one.

Figure 8.3-13 illustrates a slope analyzed by the shear strength reduction method with a finite-difference computer code.

STRESSES AND STRENGTH

Numerical analysis techniques have shown that major stress concentrations can be produced where there are rocks of differing stiffness in the slope. Stiffer rock units carry more load and thus have a stress concentration. Of particular concern for slope stability is the development of high shear stresses in the vicinity of the contact between rocks of differing stiffness.

In-Situ Stress

The stability of deep excavations (i.e., very high mine-pit walls) depends on the strength of the rock mass and the stresses induced in the rock. These induced stresses are a function of the shape of the excavations and the in-situ stresses that existed before the creation of the excavations. The magnitudes of preexisting in-situ stresses vary widely, depending on the geological history of the rock mass in which they are measured (Hoek and Brown 1980a, 1980b). Theoretical predictions of these stresses are considered to be unreliable, and measurement of the actual in-situ stresses is necessary for major mine highwall excavation design (Hoek 2007).

Simple gravitational loading would produce a vertical stress equal to the weight of the overlying material as expressed in the following equation:

$$\sigma_v = \gamma \times z$$

where

σ_v = vertical induced stress, t/m^2 (st/ft^2)
γ = unit weight of rock, typically around 2.7 t/m^3 (0.0373 st/ft^3)
z = depth below the surface, m (ft)

The horizontal stresses at some depth, z, below the surface are much more difficult to estimate than the vertical stresses. According to elastic theory, the horizontal stress would be a function of the vertical stress and Poisson's ratio. For the common value of 0.25 for Poisson's ratio, the horizontal stress (σ_h) would be 0.33 ($k = \nu / 1 - \nu$) times the vertical stress (σ_v). This relationship, developed by Terzaghi and Richart (1952), was widely used in the early days of rock mechanics, but proved to be quite inaccurate and has since been supplanted with the more realistic relationship (Hoek 2007):

$$k = 0.25 + 7E_h\left(0.001 + \frac{1}{z}\right)$$

where

k = ratio of horizontal to vertical stresses
E_h = average deformation modulus of the upper part of the earth's crust measured in a horizontal direction (GPa). This direction of measurement is particularly important, especially in layered sedimentary rocks, in which the deformation

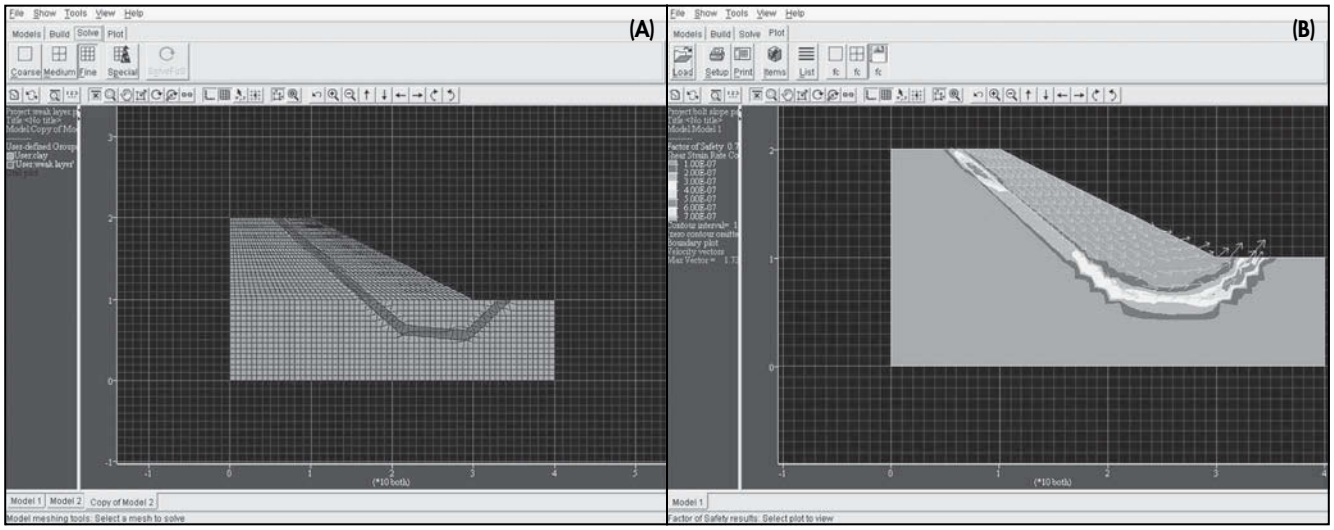


Figure 8.3-13 Shear strength reduction method: (A) grid mesh model and (B) movement vectors and failure surface

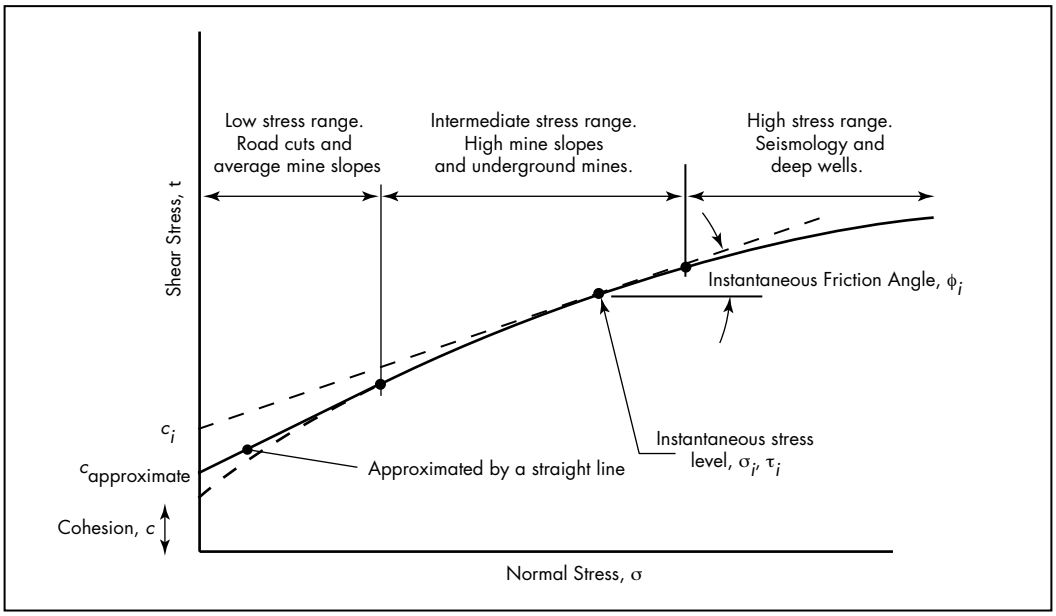


Figure 8.3-14 Hoek–Brown curvilinear failure envelope showing instantaneous friction and cohesion

modulus may be different in the different directions.

z = depth below the surface (m)

This equation, known as Sheorey's equation (Sheorey 1994), gives the ratio of horizontal to vertical stresses, k, for different deformation moduli, E_h. The model predicts a high ratio, k, at shallow depths and it decreases at depth.

Measurements of in-situ stress in underground mines have demonstrated that the horizontal stress can be greater than the vertical stress as a result of active or residual tectonic stress. Additionally, the horizontal stress is rarely equal in all directions.

In the absence of in-situ stress measurements or other indications of a high horizontal stress, the most convenient assumption is that the horizontal stress is equal to the vertical stress (Call 1992). If this assumption is made, then for normal, shallow rock slopes the Mohr–Coulomb failure envelope can be considered to be linear and the strength attributes (ϕ and cohesion, c) defined by the criteria can be considered to be relatively constant; for intermediate depths (high rock slopes) the values of ϕ and c will vary with normal stress (depth); and for great depths (deep underground excavations) the values of ϕ and c again can be considered as being relatively constant, but similar to a plastic material. This is shown on Figure 8.3-14, the Hoek–Brown curvilinear failure envelope, which is discussed later.

Shear Strength of Discontinuities

In analyzing the stability of a rock slope, the most important factor to be considered is the geometry of the rock mass behind the slope face (Hoek and Bray 1981). The next most

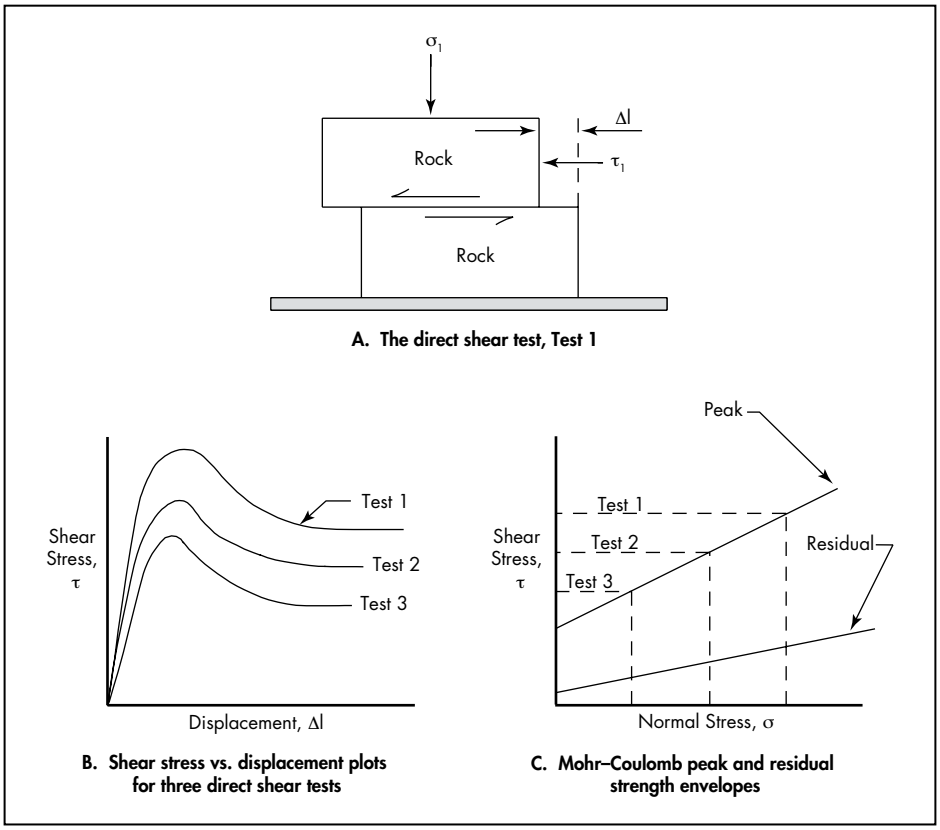


Figure 8.3-15 Peak and residual Mohr–Coulomb strength envelopes

important factor is the shear strength of the potential failure surface. All rock masses contain discontinuities such as joints, bedding planes, shear zones, and faults. For rock-strength purposes, the most serious types of discontinuities are faults and shear zones—those discontinuities that have demonstrated some sort of previous movement. At shallow depth, where in-situ stresses are low, failure of the intact rock mass is minimal and the behavior of the rock mass is controlled by sliding along the discontinuities (Hoek 2007). Shear strength along the discontinuity is controlled by rock type, the surface roughness of the discontinuity, the amount of attachments across the discontinuity, normal stress across the discontinuity, and the type and nature of filling material (if any) between the two planes of the discontinuity.

Shear Strength of Smooth Discontinuities

In rock slope design, rock is assumed to be a Coulomb material in which the shear strength of the sliding surface is expressed in terms of the cohesion (c) and the friction angle (ϕ). Two general failure criteria are in common use for rock slope design: (1) the Hoek–Brown failure criterion (Figure 8.3-14), and (2) the Mohr–Coulomb failure criteria (Figure 8.3-15). For a detailed discussion of these two failure criteria and their applications, refer to Chapter 8.4.

Shear Strength of Rough Discontinuities

Rock slope engineers generally recognize two orders of joint wall roughness (Patton 1966) that affect either the movement characteristics or strength properties of the joints (some practitioners recognize three). Figure 8.3-16 is a side view of a rock surface showing first-order surface asperities on the bottom portion of the figure and second-order surface asperities on the top portion. Waviness of a mean joint surface is regarded as a major, or first-order, type of asperity. These asperities are considered to be of such dimensions that they are unlikely to shear off; for practical purposes they appear as undulations of the joint surface. Waviness is considered to modify the apparent angle of dip but not the joint frictional properties (friction angle and cohesion); and in the stability analysis, effects of waviness are considered as influencing the direction of shear movement of the sliding block.

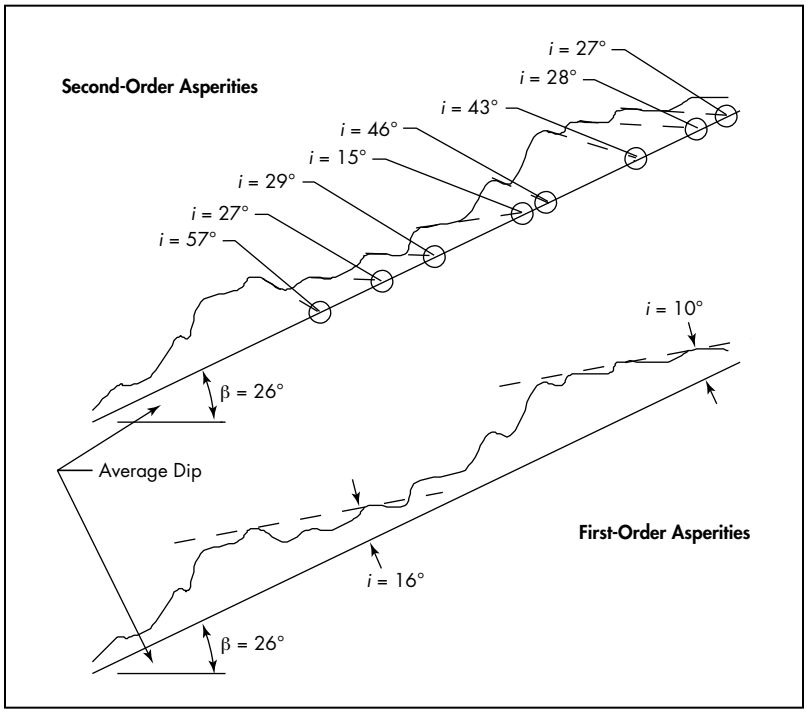
Minor, or second-order, asperities are designated as roughness. Asperities of roughness are considered to be sufficiently small that they are likely to be sheared off during movement along the joint plane (Piteau 1970). Increased roughness of the joint walls results in an increased friction angle along the joint. However, the effects of first- or second-order asperities have been found to be greatly reduced by gouge or other infilling materials.

Using the Mohr–Coulomb failure criteria, the shear strength of rough discontinuities can be represented by

$$\tau = c + \sigma \times \tan(\phi + i)$$

where

τ = shear stress along the discontinuity
c = cohesion across the discontinuity
σ = normal stress across the discontinuity
$\tan(\phi + i)$ = friction along the discontinuity taking into account the rock–rock friction and the average asperity angle, i



Source: Kliche 1999.

Figure 8.3-16 First- and second-order discontinuity roughness

Factors Contributing to Changes in Stress and Rock Strength

Varnes (1978) grouped the factors that contribute to the initiation of slope failure in rock or soil into two main categories: (1) factors that contribute to increased shear stress, and (2) factors that contribute to low or reduced shear strength.

Factors that contribute to increased shear stress include the following:

- The removal of lateral support. This is a common cause of mine highwall failures due to equipment excavating at the toe of an unstable slope.
- The addition of surcharge to the slope. Surcharge may be added to a slope by actions such as the construction of a fill; the construction of mine waste dumps, ore stockpiles, or leach piles; the weight of buildings, other human-made structures, trains, or excavators; and the weight of water from leaking pipelines, sewers, canals, and reservoirs.
- Transitory earth stresses, which include vibrations from earthquakes, blasting, heavy machinery, traffic, pile driving, vibratory compactors, and so on. The shock wave from an earthquake exerts a temporary additional stress on a slope that can cause instability. This has been demonstrated by the number of landslides triggered by earthquakes (Glass 1982), although this record is misleading with regard to rock slopes, as saturated soil slopes are subjected to liquefaction, which would result in much greater displacement at lower seismic loading. Thus it is appropriate to include the affect)of dynamic stresses in the stability analysis of slopes. The classic method of including the effect of earthquakes in stability analysis is the pseudostatic approach whereby the maximum site acceleration that could be produced by an earthquake is input into the stability analysis as an additional horizontal driving force ($k_h \times W$, discussed previously in the "Limit Equilibrium Technique" section).
- A slow increase in the slope angle through regional tilting.
- The removal of underlying support of the slope, which may happen because of undercutting of banks by rivers, streams, or wave actions; mining, quarrying, road construction, and similar actions; loss of strength or failure in underlying material such as clays; and the squeezing out of underlying plastic material.
- Lateral pressure, most commonly from water in pore spaces, cracks, caverns, or cavities. Other sources of lateral pressure include the freezing of water in cracks, swelling of soils as a result of hydration of clay or anhydrite, and the mobilization of residual stresses.
- Volcanic processes, such as swelling or shrinking of magma chambers.
- Tectonic activities, which may alter the stress fields on a very large scale, thereby causing an increase or shift in the direction of geostatic stresses.
- Processes that created the slope. These may include creep on the slope or creep in weak strata below the foot of the slope.

Factors that contribute to low or reduced shear strength include the following:

- Factors stemming from the initial state or inherent characteristics of the material. These factors include the presence and orientation of discontinuities, slope orientation, the existence of massive beds over weak or plastic materials, and the alternation of permeable beds and weak impermeable beds.
- Changes in shear strength due to weathering and other physiochemical reactions. These changes can include

softening of fissured clays; physical disintegration of granular rocks due to the action of frost or by thermal expansion and contraction; hydration (or dehydration) of clay materials; drying of clays, which results in cracks; drying of shales, which creates cracks on bedding and shear planes; and removal of cement within discontinuities by solution.
- Changes in intergranular forces due to water content and pressure in pores and fractures, which may result from rapid drawdown of a lake or reservoir, rapid changes in the elevation of the water table, rise of the water table in a distant aquifer, or seepage from an artificial source of water. The primary affect of groundwater pressure in reducing the stability of rock slopes is the resulting decrease in effective shear strength of discontinuities.
- Changes in structure can be caused by remolding clays or clay-like materials upon disturbance, by the fissuring of shales and preconsolidated clays, and by the fracturing and loosening of rock slopes due to the release of vertical or lateral restraints upon excavation.
- Miscellaneous causes can include weakening of a slope due to progressive creep or due to the actions of roots and burrowing animals.

DATA COLLECTION

Collecting adequate and appropriate data for stability analysis is a key aspect of slope design. Obtaining incorrect results from slope-stability analysis is predominantly the result of failing to analyze the critical failure mode or not having the suitable estimates of the input parameters such as rock strength or the geometry of geologic structures. With the use of computers, our ability to construct mathematical models and perform the calculations exceeds our ability to collect adequate input data for the models. There are two aspects to the problem of data collection: sampling and measurement.

As an example, consider the specific task of determining the uniaxial compressive strength of the rock mass. This is usually measured by conducting a series of compressive strength tests on cylinders, 50–75 mm (2–3 in.) in diameter and 100–200 mm (4–8 in.) in length, cored from the rock mass. These tests should be conducted according to ASTM International standards (ASTM 2008). The population of interest (referred to as the target population by statisticians) includes all the cylinders of the given size contained within the potential slope instability mass. Because of stress considerations, the volume may very well extend some distance beyond the projected limits of the slope instability boundary. It is obvious that all of the target population could not be tested, so the strength distribution must be estimated using the test results from some small, hopefully representative, sample population.

The availability of samples for testing is determined by access, which would be the ground surface, the pit wall, underground workings, and drill holes. Where there is no preexisting pit or underground workings, and the ground surface is covered by alluvium, access to samples is restricted to drill holes. These accessible samples are referred to as the sampled population. The samples that are actually collected and tested are referred to as the sample population. To make valid statistical inferences about a population, every member of the population in question must have an equal likelihood of being sampled, and the tested samples must be an unbiased representation of the population. An example of sampling bias is the determination of rock quality designation (RQD) from rock cores from vertically drilled boreholes that do not intersect an important vertical joint set. The RQD will be biased due to underrepresentation of that important joint set.

In the case of geologic structure data collection, parameters such as orientation, length, and spacing are geometric rather than scalar, and cannot be measured at a point. This results in a window problem, particularly in the case of fracture length. If the fracture is larger than the observation window, such as a bench face, the length cannot be directly measured. This is why surface mapping is preferable to drillhole data where the core diameter is the window. There is also an orientation bias, as a linear sampling window such as a drill hole does not intersect fractures parallel to the window.

Data collection should be well organized, with specific objectives regarding the use of the data and the quantity required. Collecting data for data's sake should be avoided, as it will result not only in information that is not used, but the possible omission of information needed. Ongoing data reduction is important in order to determine whether a sufficient quantity of appropriate data is being collected.

The following general procedure is suggested for geologic data collection prior to, or during, a slope-stability investigation (Kliche 1999):

1. Determine the boundaries between geologic materials with different properties: weathering can be used as criteria (e.g., color changes, hammer impact); look for differences in sedimentation (e.g., grain size distribution); consider differences in joint intensity (e.g., high versus less).
2. Determine structural features: folds; faults and faulting systems; bedding in sedimentary rocks; schistosity and cleavage in metamorphic rocks; discontinuities (e.g., joints, shear zones, bedding planes, faults).
3. Map discontinuities (Figure 8.3-17 shows a typical discontinuity survey data sheet): location (coordinates—along a line survey, or X-Y-Z values); determine material type (i.e., host rock); note the type of discontinuity (fault, shear zone, bedding plane, etc.); determine the orientation of the discontinuity (dip and dip direction or dip and strike); determine the persistence or the continuity of the discontinuity (i.e., look at the continuity of the discontinuity and the joint length); determine joint intensity (number of joints per unit distance normal to the strike of the set) and the degree of separation between the joint faces; determine the openness or closed nature of the discontinuity. If it is open, look for the presence of filling material such as gouge or transported material. If filling is present, look at the type: clay, granular material, crystalline material, or veins. Measure the thickness of the filling material; determine the waviness and the roughness (i.e., first and second order of asperities).
4. Conduct a sampling program: obtain samples of intact rock, both weathered and unjointed; obtain samples of the discontinuity surface for shear testing (care must be taken so as to not disturb the two surfaces of the discontinuity); if filling material is present, obtain representative samples (may require soil mechanics tests for strength parameters).
5. Determine groundwater conditions: locate springs or seeps; determine the permeability of the rock and the

DISCONTINUITY SURVEY DATA SHEET

GENERAL INFORMATION

Date: ____/____/____ Site: _____

Rock Material Information _____

_____

Rock Mass Information _____

_____

Comments _____

Locality Type: _____
1. Natural exposure
2. Construction excavation
3. Mine bench
4. Tunnel
5. Other (specify)

LINE SURVEY INFORMATION

Line Number	Line Trend	Line Plunge	Initial Northing	Initial Easting	Initial Elevation

DISCONTINUITY SURVEY DATA

Line No.	Line Dist.	Dip Dir.	Dip	Structure Type	Continuity	Filling Type	Waviness	Surface Roughness	Rock Type	Rock Hardness	Water

Structure Type	Continuity	Filling Type	Waviness	Surface Roughness	Rock Hardness	Water
0. Fault zone	1 = High	1. Clean	Length	1. Polished	1 = Soft	Yes or no
1. Fault	5 = Low	2. Surface stain	(in meters)	2. Slickensided	10 = Ex. hard	
2. Joint		3. Noncohesive		3. Smooth		
3. Cleavage		4. Cohesive		4. Rough		
4. Schistosity		5. Cemented		5. Defined ridges		
5. Shear		6. Calcite		6. Small slope		
6. Fissure		7. Chlorite, talc		7. Very rough		
7. Tension crack		8. Others				
8. Foliation						
9. Bedding						

Source: Kliche 1999.

Figure 8.3-17 Discontinuity survey data sheet

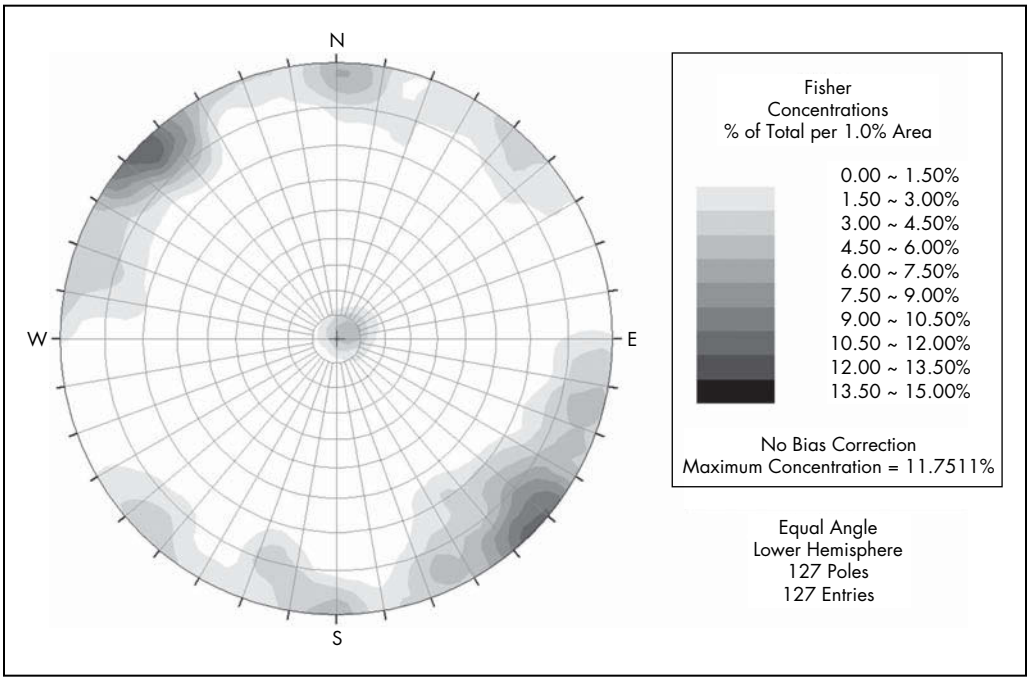


Figure 8.3-18 Contoured pole density plot

joints (joint conductivity is usually the most important—may need drawdown or recovery tests); measure the discharge of springs or seeps, at least twice per year.

Geology and Major Structure

Conventional geology provides the distribution of rock types and alteration, and the location of major structures. Geologic data should be in the form of a surface map, cross sections, and level maps. It is preferable to have two sets of documents: the factual sheets that show only the actual observations, and a set of interpreted maps and cross sections.

For the design of final pits, a geologic map of a trial pit design and cross sections normal to the pit wall should be constructed.

Rock Fabric

Rock fabric is the orientation, length, and spacing of fractures. These are the geometric attributes used in stability analysis and in characterizing the rock mass. On a pit scale, the number of fractures such as joints are too numerous to map. Fracture mapping, therefore, consists of measuring the attributes of a subset of the total fractures and characterizing the population with distributions of the attributes.

Detailed mapping reveals that the orientation of fracture sets (dip and dip direction or dip and strike) has a normal or bivariate normal distribution. Since the orientation is vector quantity, it is properly a spherical normal distribution. This can be seen by observing the contoured pole density clusters of Figure 8.3-18.

The measurable aspect of joint size is the trace length, which is the intersection of the joint and the mapping surface. The negative exponential appears to be the best distribution for trace lengths on the basis of fit-to-mapping data and theoretical considerations. Models such as the circular disk and the Poisson flat have been postulated to describe joints in three dimensions. These models can be used to correct for the observation window limitation.

Several common mapping methods are available (Call 1992):

- **Fracture set mapping.** This is a modification of conventional joint mapping where fracture sets are identified by eye, and the orientation, length, and spacing are recorded. If joints or other structure orientations have been recorded during regular geologic mapping, they can be compiled and used in slope design.
- **Detail line.** The detail line method is a systematic spot-sampling method in which a measuring tape is stretched along the bench face or outcrop to be measured. For all the fractures along the tape, the point of intersection with the tape, orientation, length, roughness, filling type, and thickness are recorded (Figure 8.3-17). To get an adequate representation of the fabric, at least 100 fractures should be mapped. This is the least subjective method, as individual fractures are recorded, and it provides the most detailed length and spacing data. It is relatively inefficient, however, as more observations are made on closely spaced fracture sets than are required for adequate statistical representation.
- **Cell mapping.** In this method, mapping surfaces such as a bench face are divided into cells. Normally, the width of the cells is made equal to the height of the cells. Within each cell, the fracture sets are identified by eye, and the orientation, length, and spacing characteristics are recorded. Cell mapping is a combination of fracture set mapping and detail line, with the efficiency of visual identification of fracture sets and some of the more rigorous measurements of detail line.
- **Oriented core.** Oriented core provides information on fracture orientation and spacing, but the length of fractures cannot be directly measured. This technique is used when the rock types of interest are not exposed. It

is also used to determine whether the geologic structural domains, mapped on the surface, extend back or downward for some distance (Nicholas and Sims 2000).

Rock Strength Properties

Since the spatial variability of rock properties is large, the potential for sampling error is greater than the measurement error. For this reason, it is preferable to use simple test methods for a number of samples than to use an expensive precise method on one sample.

For the shear strength of discontinuities and fault gouge, the direct shear test is recommended as it is a simulator of field conditions. Since the shear/normal failure curve may be nonlinear, it is important to use normal stress values for the test that represent the expected range of normal stresses for potential failure geometries in the slope. The tests at each normal stress should be run with sufficient displacement to obtain both the peak shear strength and the residual shear strength, as the residual shear strength usually is a better estimate of in-situ strength than the peak strength.

For intact rock, unconfined compression and Brazilian disk tension tests are recommended. In addition to obtaining the compressive and tensile strengths, the uniaxial compression tests can be gauged with strain gauges to obtain the Young's modulus and Poisson's ratio for the intact rock.

Index tests such as the point load can also be used to evaluate the spatial variability of intact rock strength.

For the rock mass where direct testing is not possible, indirect methods such as the rock structure rating, rock mass rating, or the rock tunnel quality index and back analysis must be used.

For a more complete discussion of rock strength properties, refer to Chapter 8.4.

Hydrology

The most important groundwater parameter for stability purposes is the groundwater pressure distribution within slopes. This distribution can be obtained in two ways: (1) by direct measurement of pressure using piezometers, or (2) by determining pressures from an analysis of the hydraulic properties of the rock mass (e.g., geology and permeability characteristics). The most satisfactory approach is usually to measure groundwater pressures with piezometers at representative locations, and to correlate these data with analytical studies based on a thorough understanding of the geology and on selected permeability or conductivity measurements of representative soil and rock strata.

In slope stabilization using water pressure control, the focus is on reducing groundwater pressure. Typically, any stabilization involves installing pressure control devices—dewatering pumps, drainage galleries, or drain holes—at a number of locations and reducing the water level (i.e., the pressure), in the material around and between those devices. The process focuses on obtaining a major water pressure reduction in an acceptable time period.

Standard hydrologic procedures such as piezometers and pump tests can be used to obtain the current pore pressure distribution and the permeability for predicting changes in pore pressure with time and changes in pit geometry. Simple techniques, such as measuring the water level in drill holes, can be used. Two factors need to be considered, however: (1) Water behavior in rock slopes is a fracture flow phenomenon, and porous media analysis, while useful at a regional scale, may be a poor predictor of pore pressure at pit slope scale; and (2) The critical factor in slope design is the pore pressure rather than the quantity of water. A low permeability rock mass may yield very little water and appear "dry" yet have significant pore pressure.

Stress Measurements

A number of techniques are available for measuring in-situ stress. Generally, each technique should be evaluated prior to its use at a particular mine site. During early site investigations, when no underground access is available, the only practical method for measuring in-situ stresses is by hydrofracturing (Haimson 1978), in which the hydraulic pressure required to open existing cracks is used to estimate in-situ stress levels. After underground access is available, overcoring techniques for in-situ stress measurement (Leeman and Hayes 1966; Worotnicki and Walton 1976) can be used with such techniques as the Commonwealth Scientific and Industrial Research Organization (CSIRO-) type hollow inclusion cell and, provided that sufficient care is taken in executing the measurements, the results are usually adequate for design purposes.

DESIGN

Steps in slope design include

1. Defining design sectors;
2. Conducting a bench design analysis to determine the maximum interramp slope;
3. Conducting interramp design analysis using economic criteria for the selection of interramp angles; and
4. Evaluating the resulting overall slope for potential instability and modifying the design if required.

Slope design is an interactive process because a trial pit is required to select design sectors, but the development of a trial pit requires slope angles. Frequently, the way this is handled is to generate a trial pit shell using critical parameters based on experience (bench height, bench angle, berm width, interramp angle, overall pit angle). As more geotechnical, geological, hydrological, and operating information comes in, the trial pit is refined, over and over, until the final pit design is established. Rarely, however, is there a "final pit design" until the end of mining, as the design parameters will change as additional or more current information comes in to the designer.

Design Sectors

To conduct stability analyses and develop optimum slope angles for input into pit design, the proposed pit must be divided into design sectors that are sections of the pit with similar geologic and operational characteristics (Figure 8.3-19).

The first criterion for the selection of design sectors is the structural domain, which is an area within which the rock properties and fabric are consistent. Typical structural domain boundaries are lithologic contacts and major structures that separate areas of dissimilar fabric.

The second criterion is wall orientation. Since rock is usually anisotropic, different wall orientations within the same structural domain can have significantly different modes of instability and different optimum angles. One way of handling the situation of a change in highwall orientations with the rock properties and fabric remaining constant is to divide the sector into subsectors based on the changing orientation (see Figure 8.3-19).

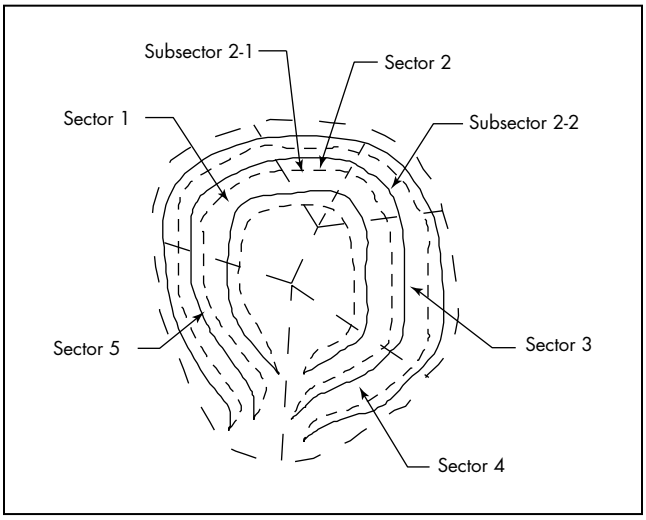


Figure 8.3-19 Pit plan with slope design sectors

A third criterion for defining design sectors is operational considerations. Sectors containing critical structures such as in-pit crushers, conveyors, and haul roads require different stability criteria than the same wall orientation in the same structural domain without the critical structures. That is, the design factor of safety may have to be modified upward based on exposure and risk considerations. For each of the design sectors or subsectors, the rock fabric, major structure orientation data, and planned highwall orientation can be plotted on a stereographic projection. This diagram is used to determine which failure modes may be kinematically possible and to select structure sets for stability analyses.

Catch Bench (Berm) Design

Bench faces are normally mined as steeply as possible so that some bench-scale rockfalls and raveling can be expected. Development of catch benches in mine slopes is necessary in areas of such rockfall since the catch benches, if designed properly, prohibit rocks from rolling from upper portions of the pit slope to the working areas where personnel and equipment are located. The bench geometry defines the steepest interramp slope that can be mined while maintaining adequate catch bench widths. The two primary factors that control bench configuration are the type of mining equipment that is used and the bench face angles that can be achieved. The type of mining equipment determines the safe operating height of the bench. The achievable bench face angles are controlled by rock strength, geologic structure characteristics, and the mining techniques used to construct the slope (e.g., the blasting and digging practices) (Ryan and Pryor 2000).

Bench design is the process of conducting stability analyses to estimate the bench face angles, selecting the bench width, and, to a limited extent, the bench height. The bench height is controlled by the height of the mining levels, but it is possible to increase the height by leaving catch benches on every other level (double benching) or every third level (triple benching).

The sole purpose of a catch bench is to "catch" rocks so that they do not continue unhindered to the toe of the slope or away from the base of the rock cut. The bench is designed with a specific width relative to its height so that rocks will come to rest before falling off the next crest. In addition, a back-break distance is often incorporated into the design as the bench crest will often fail from its (usual) vertical position. The design catch bench width is thus always wider than the bench width required for safety, as shown in Figure 8.3-20 (Zostrich Geotechnical 2009). Design catch bench width, w, can be estimated from the following equation (Bertuzzi 1999; Martin and Piteau 1977; Piteau and Martin 1982) (see Figure 8.3-21):

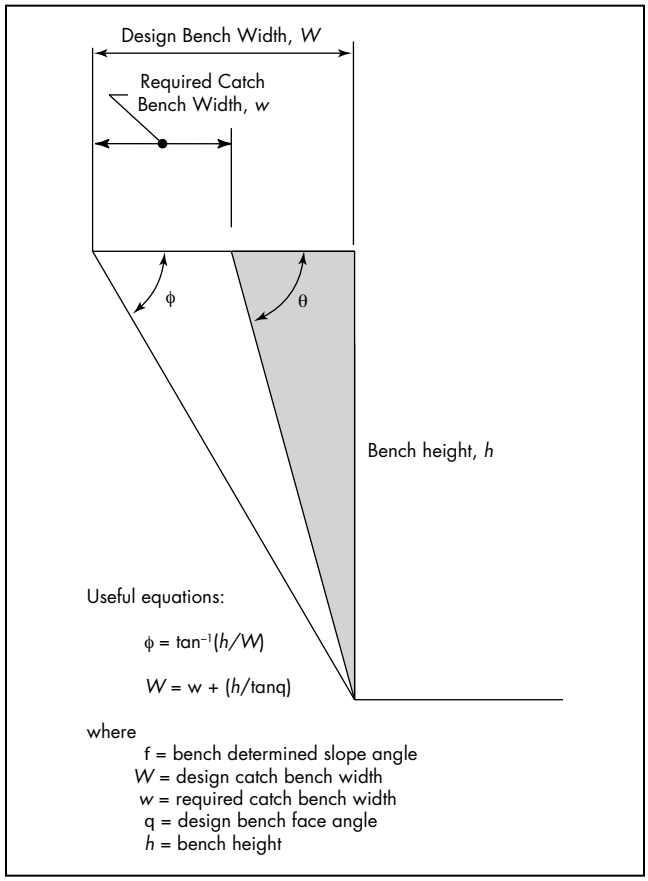


Figure 8.3-20 Required catch bench width versus design bench width

$$w = h\sqrt{x \times z \times \gamma_{bulk}}$$

where

h = bench height

$$x = \frac{1}{\tan\beta} - \frac{1}{\tan\alpha}$$

$$z = \frac{1}{\tan r} - \frac{1}{\tan\beta}$$

γ_{bulk} = 1 + [(material % swell)/100)]

and

β = dip of the discontinuity

α = bench slope angle

r = angle of repose of broken rock; typically about 37°

Where the defect is short and affects only part of the slope (Figure 8.3-22), the catch bench width, w, can be estimated from

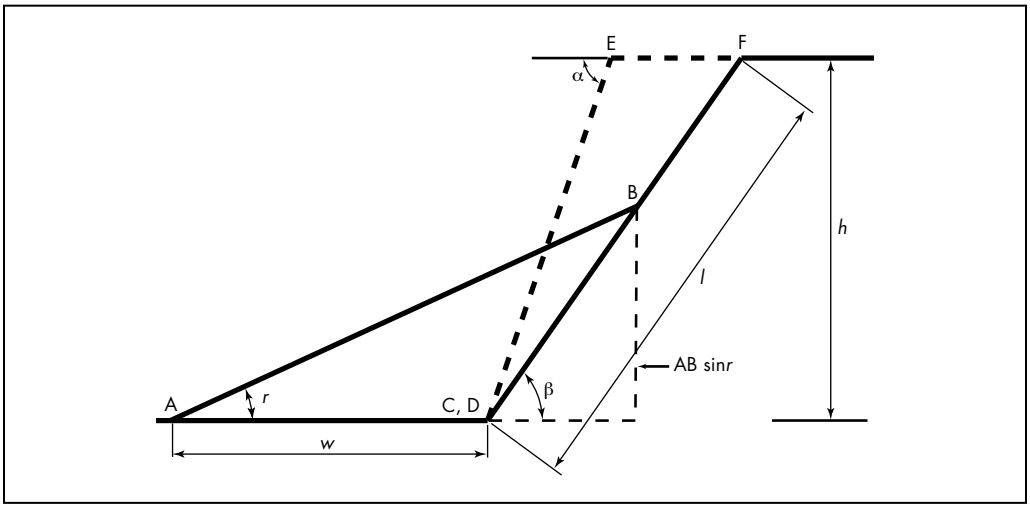


Figure 8.3-21 Geometry for catch bench design when the defect is long and affects the entire slope

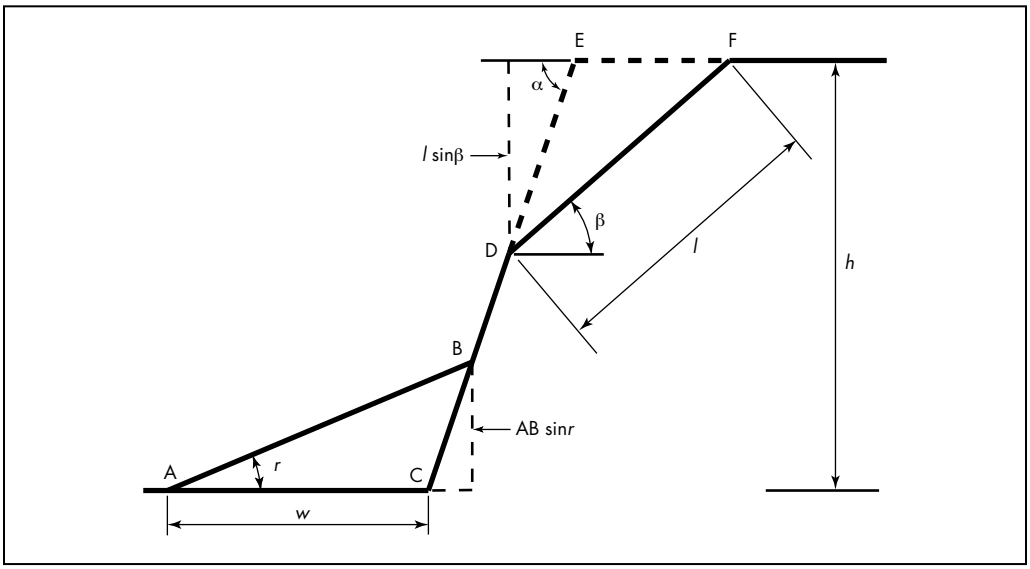


Figure 8.3-22 Geometry for catch bench design when defect is short and affects only part of slope

$$w = l \times \sin\beta \sqrt{x \times y \times \gamma_{\text{bulk}}}$$

where

l = throughgoing discontinuity length

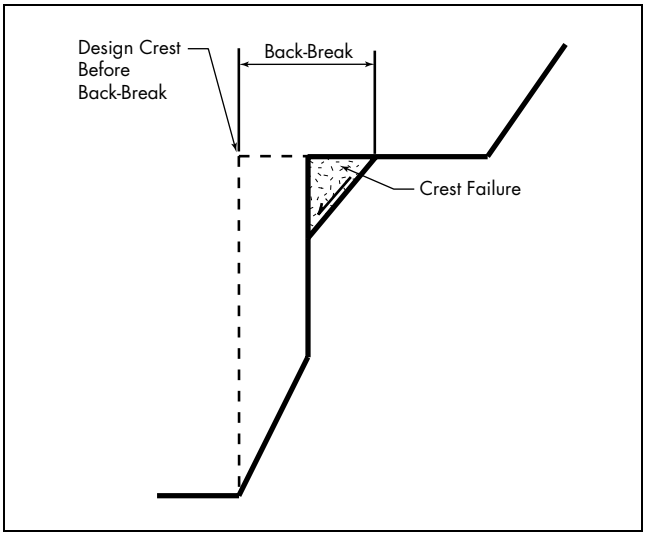
x = defined previously

$y = \frac{1}{\tan r} - \frac{1}{\tan \alpha}$, where r and α are defined previously

Based on an analysis of rockfall mechanics, Ritchie (1963) developed width and depth criteria for a ditch at the toe of a slope to protect highways from rockfalls. Falling rocks impact close to the toe of the slope, but, because of horizontal momentum and spin, can roll considerable distances from the toe. The concept of Ritchie's design was that the rock would impact in the ditch, and the side of the ditch would stop the horizontal roll.

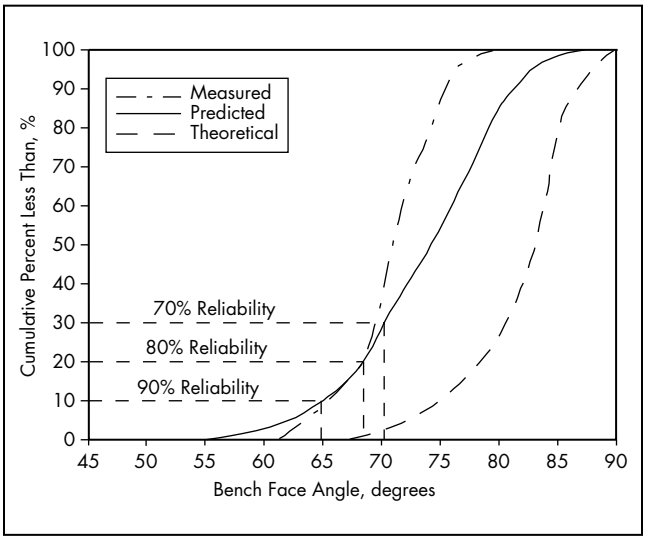
A reliability approach to catch bench design, wherein the analysis is structured to evaluate the percentage of the slope area that meets or exceeds a chosen catch bench width criteria, has been recently developed (Ryan and Prior 2000). The developers have found this technique to be a more useful risk management approach for rockfall containment and for slope management. In this approach, a combination of structural modeling, bench face stability analysis, and the modified Ritchie criteria is used to determine the catch bench reliability, which refers to the percentage of benches having final widths equal to or greater than the modified Ritchie criteria. The developers state that the selection of the proper reliability for maintaining a catch bench of a certain width is dependent on such factors as

- The potential for slope raveling,
- The proximity to large slope failures,
- The decision to contain overbank from a higher pushback on the benches,
- The length of time the benches are expected to be functional,
- The climate,



Source: Ryan and Pryor 2000.

Figure 8.3-23 Crest failures often make up the majority of back break



Source: Ryan and Pryor 2000.

Figure 8.3-24 Effective bench face angle distributions: theoretical versus predicted versus measured

- The type of blast control, and
- The operator's experience.

Because the bench configuration is based on the reliability of the catch bench width, it is the amount of local back break rather than the bench face angle that is of most concern.

The percent reliability, therefore, represents the percentage of the bench along a given level that would be wider than the minimum required bench width to catch rockfalls. The reliability should be selected on the basis of the potential for rockfall and the exposure of personnel and equipment. For example, the catch bench in raveling ground above a haul road requires a greater reliability than catch benches in a stripping area with more competent ground. In practice, reliabilities from 60%–90% have been satisfactory.

Interramp Slope Angle

For a given bench height and corresponding bench width, the upper limit of the interramp angle becomes a function of the bench face angle. The bench face angle, however, is not a unique value, as variability of the rock fabric results in varying amounts of back break. Back break is defined as the distance from the design crest to the as-mined crest (Figure 8.3-23). Because of this variability, it is preferable to use a reliability approach rather than using the mean bench face angle. (Calculating an interramp slope using the minimum bench width and the mean bench face angle results in 50% of the benches being too narrow.) The procedure is to select a percentage reliability and use the cumulative frequency distribution of the bench face angle to find the angle where the percentage greater is equal to the reliability (Figure 8.3-24). This gives the design bench face angle to use, with the minimum bench width and the bench height, to calculate the interramp slope angle.

In an operating property, the actual bench faces can be measured and the measured bench face angle distribution can be used in design. Where existing bench faces are not available, a bench-face angle distribution can be obtained by running a stability analysis for a vertical face. For this analysis, the plane shear, wedge, and step path analyses are run using the fracture data. The height analysis should be incremented in steps up to the bench height, and the resulting back break composited, as short fractures that would not result in full bench failure can still cause crest back break. This bench face angle distribution is referred to as the theoretical bench face distribution, as the effect of blasting and digging is not included. If there is a strong geologic control such as bedding or foliation, the measured and theoretical bench face angles are the same. Where no strong structure exists, the theoretical bench face angles should be reduced to include the effect of blasting. Based on comparisons that have been made between measured and theoretical angles, the reduction should be between 10° and 20°, depending on the controlled blasting to be used.

Interramp Design

The stability of interramp slopes is primarily controlled by intermediate and major structure failure geometry. Where major structures can be specifically located in space, the geometry relative to the slope can be defined and a discrete stability analysis can be conducted. Commonly, however, the number of mapped structures is large and the distance between the mapping sites and the design wall is greater than the length of the structures. In this case, the structural data must be considered a statistical representation of the structures that will occur in the design slope, and a probabilistic analysis is required.

To obtain the input for stability analysis, the wall orientation can be plotted on a lower hemisphere stereographic plot of the poles of the fractures and the major structures. The fractures and major structures are sorted into design sets based on their orientation relative to the orientations for failure modes, and the distribution of orientation, length, and spacing can be computed for the design set. These design sets may not correspond to geologic sectors, although the boundaries of the sets may be adjusted to avoid splitting a geologic sector. An advantage of this approach is that it is based on kinematic tests for viable failure geometry, making it unnecessary to test all the structures for each failure mode.

Major Structures

In the case of throughgoing major structures where the geometry is known, a safety factor can be calculated for specific slope angles and slope heights using analytical models described in the references for the appropriate failure model. For a deterministic design, the slope angle with the desired factor of safety would be selected.

In the reliability method, the probability of sliding can be calculated by Monte Carlo sampling of the shear strength distribution to obtain a distribution of safety factors and computing the area of the safety factor distribution <1. Other techniques can be used, such as the point estimate method (Harr 1984) or calculating the probability that the shear strength is less than the strength required for a safety factor of 1.

Because of the variability of the shear strength, a safety factor >1 is used to reduce the risk of instability to an acceptable level. One problem with this is that a given safety factor will have a different level of risk depending on the dispersion of the input parameters. The advantage of the reliability approach is that it deals directly with the risk.

Failure Volume Estimation

Where the geologic structures compose a statistical population, the probability of failure for the single occurrence of a specified failure mode is a function of the probability that the structures exist and form a viable failure geometry, as well as the probability of sliding (Marek and Savely 1978). The probability of existence is calculated from the orientation, length, and spacing of the structures.

To calculate the expected number of failures and the expected failure volume for input to a cost–benefit analysis, the probability of failure for the possible failure modes must be calculated for a range of heights and angles and then composited.

Cost of Failure

Given the expected number of failures and the expected failure tonnage, the cost of slope failure can be estimated. Failure costs consist of cleaning up failure material, repairing haul roads, repair of facilities, lost production due to disruption of operations, the value of lost ore buried by a failure, and engineering costs. The method used to estimate failure cost is a "what if" mine planning procedure. A failure is postulated for a design sector, a plan for responding to the failure is made, and the cost of the plan is estimated. These exercises are useful whether or not a full cost–benefit optimization is done, as they can lead to modifications of the mine plan that will reduce the impact of slope instability.

Overall Slope

The overall slope is usually flatter than the interramp slope because of catch benches or other step-outs. Thus the overall slope normally will be more stable than the interramp except for stress-induced failure or failure modes not analyzed for the interramp.

Numerical modeling techniques can be used, in addition to deterministic techniques, to analyze the potential for the various modes of failure including block flow, multibench wedge, planar type, and rotational. Numerical techniques have been time-consuming and expensive in the past; however, with the faster computers and better software currently available, it has become very feasible and routine. A quick check can be made for block flow potential using the charts developed by Coates (1981). If the charts do not indicate block flow potential with any regional stress assumption, numerical modeling is not needed unless there is a high contrast in stiffness between adjacent materials in the slope. The charts assume a homogeneous material and would therefore not indicate stress concentrations produced by stiffness contrasts.

Changes in the overall slope angle have relatively little effect on the stress concentration at the toe of the slope, where a greater concentration could produce block flow. Therefore, block flow potential would not be a suitable method for selecting overall slope angles. A more effective design approach would be to design the slope based on other criteria, and to make provision in the mine plan for step-outs, if needed, in the toe area of the pit to reduce the stress concentration produced by the notch effect of the bottom of the slope. The loss of ore from step-outs at the toe would have less economic impact than the amount of stripping required to have the same effect on block flow potential.

Rotational shear analysis should be run for the overall slope, even on rock slopes, to verify that it would not be a critical failure mode. Rotational shear would be a primary method of analysis for both interramp and overall slopes in alluvium and low rock mass-strength slopes such as soft coal measures.

The general surface analysis should be used for the overall slope to evaluate mixed mode failure types where part of the failure is structurally controlled and part is failure of low rock mass strength. Nondaylighted wedge and plane shear failures in which the weak rock at the toe fails are becoming recognized as a more significant failure mode. This is in part because pits are becoming steeper and deeper, and partly because more pits have been designed for the simpler sliding block failure modes.

Slope Support and Stabilization

Slope stabilization techniques can be divided into six general categories:

1. Grading
2. Controlled blasting
3. Mechanical stabilization
4. Structural stabilization
5. Vegetative stabilization
6. Water control

Grading

Grading involves the shaping of the rock slope into a more stable configuration. It may include flattening the slope, leaving benches in the slope face, or sculpting the slope face to a more natural appearance. Catch bench design and application are as discussed previously.

Controlled Blasting

Specialized blasting techniques are used to control overbreak and produce a competent final excavation wall by minimizing damage to the final pit walls from production blasts.

The purpose of overbreak control is to achieve a stable highwall by limiting the damage from production blasting beyond the cut limit. Often, a secondary purpose is to achieve an aesthetically appealing wall. Several drilling and blasting techniques have been developed for overbreak control to accomplish this purpose (McKown 1984; Floyd 1998), including the following.



Source: Kliche 1999.

Figure 8.3-25 Mine highwall where presplit blasting has been utilized to control overbreak and produce a stable final wall

Modified production blasts. In modified production blasting, the energy level is decreased adjacent to the wall to reduce overbreak. This decrease in energy level is often achieved for competent rock simply by reducing the charge weight in the row nearest the slope by about 30%–60% (Floyd 1998). For less-competent rock masses, additional modifications to the blast design may be required to minimize overbreak damage. These modifications can include using decked charges, reducing the burden (i.e., the distance to the nearest free face) and spacing of the last row, minimizing subdrilling, and increasing the delay interval between the last two rows of blastholes. The primary advantage of the modified production blasting technique is that it requires few design changes. The primary disadvantage is that the wall rock is not protected from crack dilation, gas penetration, and block heaving (Floyd 1998). Reducing the number of holes per delay will reduce the peak particle velocity, but for the perimeter row of holes and the buffer row, a production-hole charge is usually too large and must be reduced. To maintain the same powder factor, the hole spacing must be reduced concurrently with the reduction in hole charge. In practice, this method of controlled blasting increases the measured bench face angle by 5° (Savely 1986).

Presplit blasting. Presplitting uses lightly loaded, closely spaced drill holes that are fired before the production blast to form a fracture plane across which the radial cracking from the production blast cannot travel (Konya 1995). As a secondary benefit, the fracture plane formed may be aesthetically appealing. Figure 8.3-25 shows a mine highwall where presplit blasting has been utilized to control overbreak and produce a stable final wall. The presplit blast may be fired a considerable amount of time before the production blast (hours, days, weeks, or longer) or shortly before, as on a prior delay. Delayed blasting techniques are frequently used to separate the detonation times of explosive charges (i.e., individual holes or series of holes, as in a row of holes).

Trim (cushion) blasting. Trim blasting is a control blasting technique that is used to clean up a final wall after the production blast has taken place (Konya 1995). The trim blast may be on a later delay of the production blast or at a much later date, possibly years after blasting. The purpose of the trim blast is twofold: to create an aesthetically appealing final wall and to enhance the stability of the final wall by removing overbreak from the production blasting. Because the trim row is shot after the final production row, the trim blast does little to protect the stability of the final wall from production blasting. It does, however, provide enhanced stability by removing the loose material caused by overbreak from the production blasting. In trim blasting, cost is incurred from extra drilling, longer blasthole-loading time, and some reduced mine production, but these short-term costs are often offset by decreased future costs in terms of stripping and slope failure. Both of these potential future costs—stripping and slope failure—can be incurred as a result of overbreak into the final highwall from the production blasting.

Line drilling. Line drilling is often included as one of the controlled blasting techniques. However, it is not a blasting technique per se. Line drilling uses a single row of unloaded, closely spaced drill holes at the perimeter of the excavation. When a production blast adjacent to a series of line-drilled holes is detonated, the shock wave from the detonating holes will cause a stress concentration around the unloaded, line-drilled holes. If the stress exceeds the rock strength, failure will occur in the form of a crack extending from one line-drilled hole to the adjacent line-drilled hole. Line drilling is an expensive perimeter control technique because of the number of holes required for the technique to work properly.

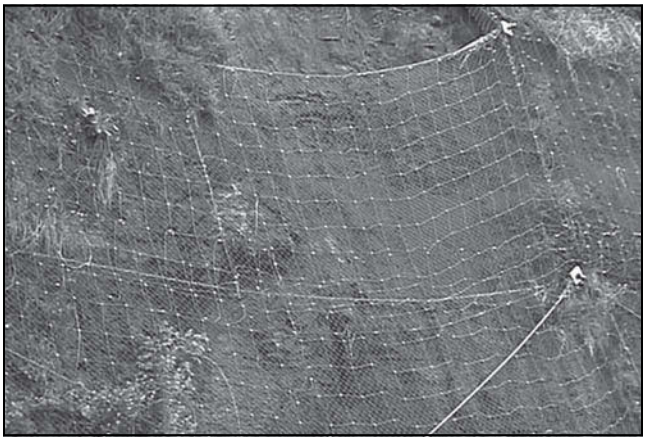
Two aspects of final wall design should be considered in selecting the appropriate controlled blasting technique, or combination of techniques, for a particular job: defining rock damage criteria and developing a procedure to design blasts that will minimize rock damage without seriously affecting production.

Mechanical Stabilization

Mechanical methods of slope stabilization are those that alter or protect the slope face to reduce erosion, prevent rockfall, or to reduce raveling. Common methods include protective blankets, geotextiles, and wire net or mesh.

Protective blankets. Protective blankets made from jute, excelsior, burlap, cotton, or other natural or manufactured materials have been used for many years for erosion control and to prevent or reduce raveling on cut slopes. The blankets are usually pinned to the slopes and combined with seed and fertilizer. The purpose of pinning the blanket is to hold it in place until the vegetation takes root. The blankets are often expected to deteriorate and thus biodegrade over time as the vegetation takes hold.

Geotextiles. A geotextile is defined by ASTM as "any permeable textile material used with foundation, soil, rock, earth, or any other geotechnical engineering-related material, as an integral part of a man-made project, structure, or system" (Christopher and Holtz 1985). Geotextile applications can be divided into four primary functions: separation, drainage, reinforcement, and filtration (Christopher and Holtz 1985). In separation, layers of different sizes of solid particles are separated from one another by the geotextile (e.g., landfill covers). In drainage, the geotextile allows water to pass; in the special case of "drainage transmission," the geotextile itself acts as a drain to transmit water through soils of low permeability (e.g., horizontal drains below heap leach pads). In the case of reinforcement, the geotextile acts as a reinforcing element in the earth through either stress distribution or an increase in soil modulus (e.g., a net against rockfalls). For filtration, the fabric acts in a similar fashion to a two-dimensional sand



Courtesy of Mine Safety and Health Administration.

Figure 8.3-26 Combination of Geobrugg-type rockfall net with chain link fence pinned to the highwall utilizing rock dowels and cable lashing

filter, allowing water to move from the soil while retaining the soil (e.g., silt screens).

Wire net or mesh. Another method of slope stabilization involves draping or pinning wire netting over the slope face to prevent rockfalls from bouncing outward from the toe region. Three types of wire mesh are commonly used for this purpose: (1) welded wire fabric, such as that used in concrete reinforcement; or (2) chain-link mesh, as is commonly used for fencing; and (3) flexible wire rope nets with or without underlaid wire netting (Geobrugg-type netting). A typical welded wire mesh application would be to use mesh with a 100 × 100 mm (4 × 4 in.) or 150 × 150 mm (6 × 6 in.) opening and a wire size from 9 to 4 gauge (Seegmiller 1982). Chain-link fence is often coated with a galvanizing agent and will therefore better withstand adverse environmental conditions. Also, because of the nature of its construction, chain-link fence tends to be more flexible and stronger.

Pinning the net or wire mesh to the face holds the rock in place and reduces rock removal at the toe (see Figure 8.3-26). The pins (typically rock bolts, rock dowels, thread bars, or rebar) must be strong enough and spaced close enough to hold large, loose rocks and prevent them from dislodging and tearing the mesh.

Structural Stabilization

Structural stabilization includes those methods that reinforce the structure of the rock at the slope face or provide a structure that supports the slope. Methods available include the use of gunite or shotcrete, rock bolting, and construction of rock buttresses or retaining walls.

Shotcrete. One common method of mechanical stabilization uses pneumatically applied mortar and concrete (generally known as gunite or shotcrete) sprayed or pumped onto the slope face to seal the face and bind together small fragments on the face. This approach is used primarily to prevent weathering and spalling of a rock surface, as well as to knit together the surface of a slope. Generally, for rock slope stabilization, the material is applied in one 50–75-mm (2–3-in.) layer (Brawner 1994). One disadvantage of shotcrete is its low tensile strength. For this reason, welded wire mesh, anchored to the rock, is often used to reinforce the shotcrete. A problem with using wire mesh as reinforcement for shotcrete is the difficulty of molding the mesh to a rough surface. Where the surface is irregular, large gaps may develop between the mesh and the rock, making bonding of the shotcrete to the rock difficult. Additives can be added to either the wet or dry mix to provide additional strength and durability. Steel fibers, when added to the mix, increase the tensile strength of the shotcrete by providing numerous bonding surfaces within a small area. The fiber reinforcement also reduces the risk that shrinkage cracks will develop during curing. In many cases, the addition of fibers can replace wire mesh as reinforcement, thus reducing the overall cost.

Steel reinforcement members. Steel reinforcement in the form of rock bolts, cable bolts, resin-grouted thread bars, or rock dowels are used to tie together the rock mass so that the stability of a rock cut or slope is maintained. Rock bolts are commonly used to reinforce the surface or near-surface rock of the excavation, and rock anchors are used for supporting deep-seated instability modes in which sliding or separation on a discontinuity is possible.

A rock anchor generally consists of a bar or cable of high-strength steel tensioned inside a borehole to about 60%–70% of its yield strength. Tension in the member is transmitted to the surrounding rock mass by anchorage points at the ends. The length of the rock anchor can be from 3 m (10 ft) to >100 m (330 ft) (Sage 1977).

Resin-grouted, tensioned thread bar and grouted cables provide a means to control large failure blocks. Lengths of the units may be as short as 3 m (10 ft) or as long as 100 m (330 ft), depending on the specific application. Holes for installation of the anchors are normally drilled well past the potential failure plane; then the anchors are inserted and grouted in place with or without tensioning.

Grade 60 (423.7 MPa or 60,000 psi), resin-grouted, tensioned thread bar comes in diameters ranging from 19.1 mm (0.750 in.) to 57.3 mm (2.26 in.) and in nominal lengths of 12.2 m (40 ft). Such thread bar can be cut to lengths shorter than 12.2 m (40 ft) or made longer by coupling two or more units together.

Grouted cables were introduced to mining for reinforcement of the backs of cut-and-fill stopes. Cable bolting, whether tensioned or untensioned, is widely used in mining applications. The cable bolts should be made from high-strength steel (about 1,380 MPa or 200,000 psi yield strength, typically) because the steel will creep in tension; as a result, a gradual decrease in anchor load will occur over time. This loss in strength is approximately the same for all types of steel (Sage 1977). A special prestressing jack (Figure 8.3-27) is required for tensioning the bolt. Also, for the tensioned cable bolt, a special bearing plate, an anchor block, and cable-gripping cones are required.

For a more complete discussion of steel reinforcement members that may be used in slope reinforcement, refer to Chapter 8.8.

Vegetative Stabilization

Vegetative techniques are most frequently used for aesthetic purposes, such as slope reclamation. However, there are many treatment methods that use vegetation to improve the stability of a slope. Generally, these methods are most successful when minor or shallow instability (such as raveling or erosion) is involved, as is usually the case for soil slopes or highly fractured rock slopes (Buss et al. 1995). The establishment of vegetation on steep-soil slopes or loose-rock slopes is often

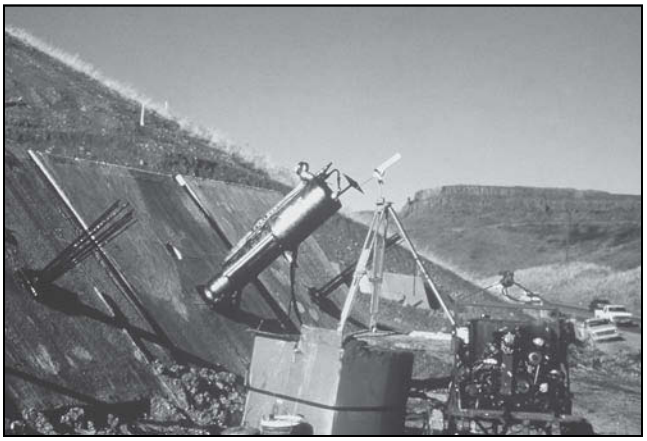


Photo by Don Berger. Courtesy of Homestake Mining Company.
Figure 8.3-27 Hydraulic prestressing jack

enhanced by the construction of benches or stair-step terraces in the slope face. These arrangements act to hold the seed mix in place, to encourage infiltration, and to impede water flow in order to minimize erosion and sedimentation. Natural or manufactured mats, webs, or fabrics can also be used for erosion control and to hold seeds in place, though their cost and effectiveness often limit their use. They require high labor inputs for installation and cost much more than tacked or hydromulched straw; in addition, some are not well adapted to fitting to rough surfaces. They must also be heavy enough or anchored in enough spots to prevent wind whipping.

The planting of trees or other large woody plants on rock slopes is beneficial in "softening" the appearance of the cut slope to make it appear more natural (Buss et al. 1995). In order to make access easier for planting trees or shrubs, benches, berms, or furrows must be constructed in the slope face. If access allows, the use of a tree spade enables the transplanting of large, mature trees. The plants are placed in holes that have been previously excavated on the reclamation site. (A single hole drilled and blasted in rock will provide an excellent site for a transported tree or shrub.) The plants are transplanted with a minimum of root disturbance. Tree spades, however, are expensive to use, and their use should be reserved to transporting hard-to-establish trees or to achieve an objective of immediate stocking with mature trees.

Water Control

The presence of groundwater in a rock slope can have a detrimental effect on stability for the following reasons:

- Water pressure reduces the stability of the slopes by diminishing the shear strength of potential failure surfaces. Water pressure in tension cracks reduces stability by increasing the driving forces.
- Changes in moisture content of some rock, particularly shales, can cause accelerated weathering and decreases in shear strength.
- Freezing of groundwater can cause wedging in water-filled fissures due to temperature-dependent volume changes in the ice. This can result in expansion of an opening, such as a tension crack upon freezing, which does not contract upon thawing of the water.
- Erosion of weathered rock by surface water and of low-strength infillings by groundwater can result in local instability where the toe of the slope is undermined or a block of rock is loosened.
- Excavation costs can be increased when working below the water table.

By far the most important effect of groundwater in a rock mass is the reduction in stability resulting from the water pressures within the discontinuities.

Grading and shaping are major considerations in the control of surface water. Surface water can be controlled through a combination of topographic shaping and runoff control structures (Glover et al. 1978). Surface water allowed to flow down a slope, or to pond on benches of a slope, can infiltrate into the ground along discontinuities and thereby cause an increase in the driving forces on an unstable area through a buildup in pore pressure. Runoff control structures include dikes, waterways, diversion ditches, diversion swales, and chutes or flumes (Glover et al. 1978). The purpose of these structures is to intercept surface water flow before it reaches a critical area and to divert it to a disposal area.

The purpose of subsurface drainage (i.e., groundwater control) is to lower the water table and, therefore, the water pressure to a level below that of the potential failure surfaces. Methods of subsurface drainage include drain holes, pumped wells, and drainage galleries or adits.

Drain holes. Horizontal drain holes drilled into the face of the slope from the toe region offer an effective method of slope drainage. Normally, the holes are 50–150 mm (2–6 in.) in diameter and are drilled at an inclination of +3° to +5° from the horizontal. The length of the holes should extend beyond the critical failure surface. The direction of the drain holes depends on the orientation of the critical discontinuities; the optimum design is to intersect the maximum number of significant discontinuities for each unit length. Spacing of the drain holes can range from about 7 to 30 m (20 to 100 ft), and lengths into the slope should not exceed one-half the slope height, with a minimum length of 15 m (49 ft) and a maximum length of 100–125 m (330–410 ft) (Brawner 1982). For high rock cuts, installation of drain holes at different levels is recommended. Where rock is taken out in several lifts, drain holes should be drilled at the toe of every lift (Brawner 1994).

Dewatering wells. Dewatering wells are designed primarily to lower the groundwater level to a predetermined depth and to maintain that depth until all below-ground activities have been completed. The main purposes for construction dewatering include the following (Driscoll 1986):

- Intercepting seepage that would enter an excavation site and interfere with construction activities
- Improving the stability of slopes, thus preventing sloughing or slope failures
- Preventing the bottoms of excavations from heaving because of excessive hydrostatic pressure
- Improving the compaction characteristics of soils in the bottoms of excavations
- Drying up borrow pits so that excavated materials can be properly compacted in embankments
- Reducing earth pressures on temporary supports and sheeting

Drainage galleries or adits. Drainage adits or galleries driven under a pit—or into a slope or highwall to intercept the groundwater—can provide an effective method of drainage. Where employed, drain holes should be drilled from the adit

upward in a fan pattern to increase drainage effectiveness. An adit can be used not only for drainage, but also as a means of obtaining detailed discontinuity information. This type of data is useful for future slope-stability predictions. Additional information, such as water quality and the variations in permeability along the length of the adit, can also be gathered. The cost of driving an adit is high. However, it need only be large enough to allow efficient excavation and to properly drain the problem area, generally 1.0–1.5 × 2.0–2.2 m (3.3–4.9 × 6.5–7.2 ft).

SLOPE MANAGEMENT

Three general principles of slope mechanics should be kept in mind in cases of slope instability (Call and Savely 1990):

1. Slope failures do not occur spontaneously. One or more of the forces acting on a potentially unstable rock mass must change in order for the mass to become unstable.
2. Most slope failures tend toward equilibrium. A slope fails because it is unstable under the existing conditions. Failure tends to bring the slope to some sort of equilibrium. It normally involves a reduction in the driving forces and/or an increase in the resisting forces of the failed zone.
3. A slope failure does not occur without warning. Prior to failure, measurable movement and/or the development of tension cracks will occur. These indications of failure can develop, indicating imminent slope failure, then subside for a long period of time, indicating apparent stability.

In the evaluation of the necessity and type of stabilization technique to be used, the first issue to be considered is the degree of urgency. If the slope has started to move, immediate remedial actions should be taken. These actions may include evacuating structures, closing roadways, or cordoning off a section of a mine.

After the situation has become less urgent, determining the cause of the instability is necessary by the following means: visual observation based on experience, water level measurements, slope instrumentation, tests on the materials, and a survey of the discontinuity patterns (Golder 1971). Laboratory tests may be required on the rock and discontinuities to determine appropriate strength parameters.

Detection and Monitoring of Instability

When a rock or soil mass is disturbed, either by the actions of people or by natural events, it undergoes a redistribution of stresses, resulting in a change in shape. This readjustment is reflected in displacements, deflections, pressures, loads, stresses, and strains, which can be detected and measured. Many of the same measurement methods and instrumentation techniques can also be used to investigate the mechanical properties of the mass, the interaction between the mass and any associated artificial structures, and the effectiveness of remedial measures proposed to correct defects in either the mass or the structures.

Monitoring is the surveillance of engineering structures either visually or with the aid of instruments (Brown 1993). The objectives of a rock slope monitoring program are as follows (Call 1982):

- To maintain a safe operation for the protection of personnel and equipment
- To provide advance notice of instability, thus allowing for the modification of the excavation plan to minimize the impact of the instability
- To provide geotechnical information in order to analyze the slope failure mechanism, design appropriate remedial measures, and/or conduct a redesign of the rock slope

Monitoring can be done from within the rock mass and/or on the excavation boundary. The techniques available to measure the various components of rock deformation may be placed in two general categories: observational and instrumentation (Windsor 1993). Observational techniques include simple visual observations, photographic recording, and electronic and optical surveying. Instrumentation techniques include the application of mechanical and electronic instruments such as extensometers, inclinometers, strain gauges, and crack gauges.

An overview of both observational and instrumental techniques, as well as the types of instruments for monitoring deformation, is given in Table 8.3-1.

An effective slope monitoring program consists of the systematic detection, measurement, interpretation, and reporting of evidences of slope instability. Measurements are normally made of both surface and subsurface displacement in order to provide an accurate assessment of slope instability.

Surface Displacement

Surface displacement measurement by means of observational techniques (surveying by conventional methods, automatic surveillance, and/or Global Positioning System) and/or instrumentation techniques (movement indicators, extensometers, inclinometers, convergence indicators) is the preferred monitoring method. A combination of the methods and instruments should be used, as no one method would give the entire picture.

Tension crack mapping. Tension cracks are an early, obvious indication of instability. By systematically mapping the cracks, the geometry of a failure can be better defined. All cracks should be mapped regardless of apparent cause. Often cracks that appear to be the result of local bench failure or blasting form a pattern showing an impending larger failure when plotted on a pit map.

The ends of the cracks should be flagged or marked so that on subsequent visits new cracks or extensions of existing cracks can be identified.

Wireline extensometers. Portable automated wireline extensometers can be used to provide monitoring in areas of active instability across tension cracks. These monitors can be quickly positioned and easily moved. The extensometer should be positioned on stable ground behind the last visible tension crack, and the wire should extend out to the unstable area. For warning devices, or for information on deformation within a sliding mass, wire extensometers can be placed at any strategic location. Anyone working in the area can make an immediate check on slope movement by inspecting the instruments.

The automated slope monitoring system has the ability to record data automatically and then transmit the data to a central computer system for analysis. The system is used for real-time displays of slope movement, as well as for long-term analysis of all recorded information (Martin 1996). The automated system is composed of two main components: the slope monitor unit, which is located in the field, and the central computer and radio, which may be located in the mine office. The system may be programmed to transmit alarms if certain conditions occur, such as broken wire, excessive movement or velocity, or communication failures. For instance, if the wire breaks or the slope anchor probe pulls loose, the wire spool falls to the base

Table 8.3-1 Classification of instruments for measuring the various components of rock deformation

Deformation Measurement Technique	Measurement Access	Measurement Method	Measurement Sensitivity
Observational Techniques			
Global Positioning System	Exposure	Manual	Manual
Terrestrial surveying	Exposure	Manual	Manual
Electronic distance meter and automatic surveillance	Exposure	Automatic	Manual
Slope-stability radar	Rock face	Automatic	High
Instrumentation Techniques			
Movement indicators			
Axial	Borehole/Rock face	Observational	Low
Shear	Borehole/Rock face	Observational	Low
Convergence indicators			
Wire/tape	Rock face	Manual	Manual
Rod	Rock face	Manual/Automatic	Manual
Strain meters			
Resistance strain gauges	Borehole	Automatic	High
Vibrating wire strain gauges	Borehole	Automatic	High
Joint meters			
Glass plates	Rock face	Manual	Manual
Pin arrays	Rock face	Manual/Automatic	Manual
Strain gauges	Borehole	Automatic	High
Proximity transducer	Rock face	Automatic	High
Fiber optic	Borehole	Automatic	High
Potentiometers	Borehole/Rock face	Automatic	High
Extensometers			
Fixed extensometer			
Wire/rod	Borehole	Manual/Automatic	High
Reference point sensing	Borehole	Manual/Automatic	High
Strain sensing	Borehole	Automatic	High
Portable extensometer			
Magnetic anchor	Borehole	Manual	High
Magnetostrictive	Borehole	Manual	High
Sliding micrometer	Borehole	Manual	High
Inclinometers			
Fixed inclinometers	Borehole	Automatic	High
Portable inclinometers	Borehole	Manual	High
Deflectometers	Borehole	Automatic	High
Extensometer–inclinometer	Borehole	Manual	High
Extensometer–deflectometer	Borehole	Manual	High

of the mounting tripod; the falling spool pulls a magnet off the electronics box, which immediately radios a warning to the central computer (Martin 1996). The central computer system collects and processes the data and generates screen displays and/or reports on slope movement status. A solar panel can be attached and used to recharge the battery power supply.

Survey monitoring. The most widely used method of monitoring for movement employs the theodolite/EDM (electronic distance meter), total station or programmable robotic theodolite in conjunction with an array of monitoring prism targets. This continues to provide the most detailed movement history in terms of displacement directions and rates in the unstable areas. Locations of the targets should be chosen so that relative movement of the unstable area can be monitored. Additionally, the permanent control point(s) from which the targets are shot (i.e., observed) must be located on stable ground outside the slide area and within view of the targets. The network should be a set of well-conditioned triangles, with each vertex point being visible from two other points and the length of each line of sight within the measuring range of the equipment (Windsor 1993).

The instrument, as well as any backsights used, should be located on stable ground away from the slide area(s). Computerized, automatic theodolites are gaining popularity to monitor movement at 100 or more survey prism stations. The robotic theodolite may be housed in a small shed or dispatch building at a vantage point in direct line of sight with all prism stations. A computer system may also be housed in the robotic slope-monitoring shed. This system uses a specialized software package to control the theodolite, which, after preprogramming, shoots the array of prisms automatically on a programmed cycle. After completion of a cycle, the system may be set to shoot one or more subcycles or may start another cycle of shooting the set of prisms. Readings can be trans-



Courtesy of Travis Nice.
Figure 8.3-28 Slope-stability radar unit

ferred via hard wire, Internet, modem, or radio telemetry link to a base computer located elsewhere for further processing.

Slope-stability radar. The slope-stability radar (Figure 8.3-28) is a state-of-the-art technology used for slope-stability monitoring of open-cut mine walls. It provides continuous, precise, and real-time on-line measurement of rock wall movements across the entire face of a wall. It remotely scans a rock slope to continuously monitor the spatial deformation of the face. Using differential radar interferometry, the system is used to detect deformation movements of a rough wall with submillimeter accuracy, and with high spatial and temporal resolution. The effects of atmospheric variations and spurious signals can be reduced via signal processing means. The advantage of the slope-stability radar over other monitoring techniques is that it provides full area coverage without the need for mounted reflectors or equipment on the wall. In addition, the radar waves adequately penetrate through rain, dust, and smoke to give reliable measurements, in real time and 24 hours a day (GroundProbe 2009).

Subsurface Displacement

Surface displacement measurements do not determine the subsurface extent of instability, although it is possible to make inferences from displacement vectors. There are many situations where measurement of subsurface displacement is needed. These measurements are commonly made using borehole inclinometers or borehole extensometers.

Borehole inclinometers. An inclinometer measures the change in inclination (or tilt) of a borehole and thus allows the distribution of lateral movements to be determined versus depth below the collar of the borehole as a function of time (Wilson and Mikkelsen 1978). Therefore, the application of inclinometers to slope-stability studies is important for the following reasons:

- To locate shear zone(s)
- To determine whether the shear along the zone(s) is planar or rotational
- To measure the movement along the shear zone(s) and determine whether the movement is constant, accelerating, or decelerating

Two types of inclinometer are in common use: the in-place inclinometer and the traversing probe type of inclinometer.

The in-place inclinometer was developed for automated monitoring. It is composed of a string of inclinometer sensors permanently mounted in the casing. The sensors are normally positioned within the casing to span the zone where movement is anticipated. The string of sensors is usually attached to a data acquisition system that is programmed to trigger an alarm if certain boundary conditions are exceeded. The in-place inclinometer is expensive; its use is generally limited to only the most critical applications (Boisen and Monroe 1993).

The traversing probe type of inclinometer was developed to address the problem of expense. It employs a single sensor that can be used to monitor any number of inclinometer casings. Tilt readings are typically obtained at 2-m intervals (2-ft intervals are used with probes that use the U.S. customary system of measure) as the probe is drawn from the bottom to the top of the casing. The main drawback to the system is that it is slow and requires an on-site operator (Boisen and Monroe 1993).

Borehole extensometers. The fixed borehole extensometer measures only axial displacement between a fixed number of reference points on the same measurement axis. When more than two reference points are used, the instruments are referred to as multiple-position or multipoint extensometers. Multipoint extensometer data can reveal the relative movement between anchor points and the distribution of displacement, in addition to the magnitude, rate, and acceleration of displacement (Slope Indicator Company 1994).

The basic components of a fixed borehole extensometer are an anchor, a linkage, and a reference head. The reference head is installed at the borehole collar. The linkage system may be composed of wires or of solid rods; it spans the distance between the reference head and the anchor. A change in this distance indicates that ground movement has occurred. Measurements are taken at the reference head with a depth micrometer or an electronic sensor and are used to determine the displacement.

Precision, Reliability, and Cost

The number of different devices that can be used for monitoring, as well as the precision and sophistication of the devices, are a function of the ingenuity, time, and budget of the engineer in charge of monitoring. Since none of these factors is infinite, hard choices must be made. Some general guidelines for decision making follow.

Measure the obvious things first. Surface displacement is the most direct and most critical aspect of slope instability. Tension cracks can easily be mapped or photographed; the surveyor can set survey points and log movement on a regular basis.

Simpler is better. The reliability of a series system is the product of the reliability of the individual components. A complex electronic or mechanical device with a telemetered output to a computer has significantly less chance of being in operation when needed than do two stakes and a tape measure.

Precision costs money. The cost of a measuring device is often a power function of the level of precision. Measuring to 10 mm (0.4 in.) is inexpensive compared to measuring to 0.001 mm (0.0004 in.).

Redundancy is required. No single device or technique tells the complete story. A single extensometer or survey point will not indicate the area involved in the instability, and, if it is destroyed, the continuity of the record is lost.

Multiple survey prisms should be placed in and around a slide area that is being monitored using optical or robotic surveying techniques.

Timely reporting is essential. Data collection and analysis must be rapid enough to provide information in time to make decisions. Modern computerized data acquisition systems are not always available, therefore persons doing the monitoring and reporting should submit promptly. Any indication of movement acceleration requires attention.

Establish a monitoring schedule. A definite monitoring schedule should be established. The frequency of monitoring is a function of the precision of the system, the rate of movement, and how critical the area is. If there is heavy rain or a large blast in the area, additional measurements should be made.

Cooperation between operations and engineering is important. Equipment operators often have an intuitive feel for ground conditions. Any changes in the condition of an area observed by operators (e.g., an observation of the development of tension cracks) should be reported to engineering for follow-up.

Data reduction and reporting should be conducted. The following measurements or calculations should be made for each survey reading:

- Date of reading, incremental days between readings, and total number of days the survey point has been established
- Coordinates and elevation
- Magnitude and direction of horizontal displacement
- Magnitude and plunge of vertical displacement
- Magnitude, bearing, and plunge of resultant (total) displacements
- Velocity and any change in velocity (acceleration/deceleration) of the movement vector

Both incremental and cumulative displacement values should be determined. Calculating the cumulative displacement from initial values rather than from summing incremental displacements minimizes the effects of occasional survey aberrations.

Slope displacements are best understood and analyzed when the monitoring data are graphically displayed. For engineering purposes, the most useful plots are

- Horizontal position,
- Vertical position (elevation versus change in horizontal position, plotted on a section oriented in the mean direction of horizontal displacement),
- Displacement vectors,
- Cumulative total displacement versus time, and
- Incremental total displacement rate (velocity, usually in feet or meters per day) versus time.

All graphics should be kept up-to-date and should be easily reproducible for ease of distribution. By studying several graphs simultaneously, the movement history of a particular slope can be determined.

Precipitation data should also be recorded in order to evaluate possible correlations with slope displacement. A rain gauge or system of rain gauges located at the mine site can be used to measure occurrences and amounts of precipitation. In addition, measurement of the average daily temperatures will provide some indication of freeze and thaw periods.

The location of mining areas and the number of tons mined should also be recorded on a regular basis, because slope displacements are often associated with specific mining activity. Blasting records, including seismic records, should be kept and referenced to the unstable areas. A histogram can be made of tons mined versus time, and this plot can then be compared to the total displacement graphs.

A formal monthly slope-stability report should be prepared, containing at least the following data:

- Maps (plan view showing the unstable areas and locations of monitoring devices, displacement vectors, blast locations, tension crack map, piezometric surface)
- Plots and graphs (cumulative displacement versus time, displacement rate versus time, precipitation versus time, mining versus time)
- Prediction of time to failure for critical unstable areas
- An "action plan" for each critical unstable area

Time-Dependent Slope Movement Characteristics

All excavations, whether natural or human-made, deform with time in response to excavation. According to Zavodni (2000), the most commonly observed evidence of time-dependent deformation of cut slopes is the development of tension cracks behind the slope crest, formation of cracks on the slope, and toe heave.

Zavodni further states that mining operations can proceed safely with minimum interruption if failure mechanisms are understood and slopes are properly monitored. A serious slope instability condition is usually accompanied by gradual development of one or more tension cracks behind the slope crest. This situation then normally allows for time-displacement monitoring. Surface-displacement measurements employing prism targets are usually adequate for monitoring slope movement. The prisms may be accompanied by extensometers closely monitored to determine slope-movement velocities in order to predict slope behavior. Recent advances in survey equipment—robotic theodolites and real-time transmission of data—allow increasingly precise monitoring of movement accompanied by immediate data display and analysis.

Initial Response

Upon excavation of a rock mass, thereby creating a rock slope, the slope experiences a period of initial response as a result of elastic rebound, relaxation, and/or dilation of the rock mass due to changes in stress induced by the excavation. This initial deformation or rebound normally occurs without the development of a defined failure surface or failure mechanism. The slope rebound and extension can develop by spreading along joints in the rock mass or along low-strength discontinuities.

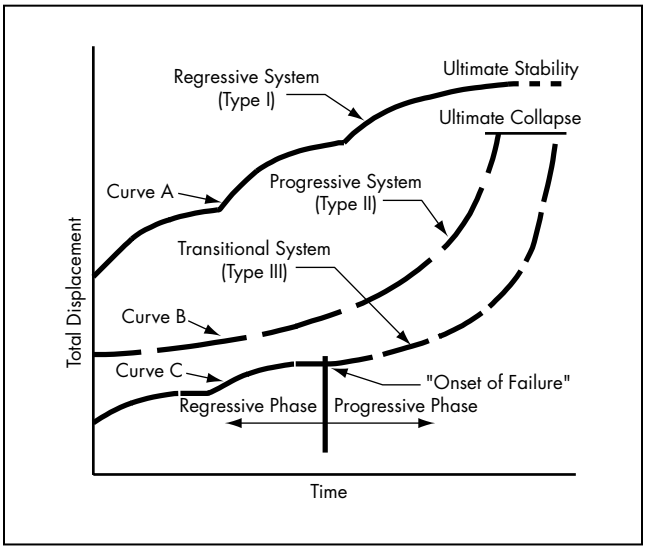
Zavodni (2000) reported that initial response movements ranged from 0.10 to 4 mm/d (0.004 to 0.16 in./d), although much higher instantaneous velocities could be encountered at the exact time of excavation. The rates of movement during initial response periods decreased with time and eventually dropped to zero. Results from numerous case histories indicate this decrease in movement rate can best be fit by a negative exponential relationship of the form:

$$R = A \times e^{-bt}$$

where

R = movement rate

A and b = constants that are a function of the rock mass properties, slope geometry (height and angle), mining rate, external influences, and ultimate failure mechanism. Zavodni's reported values



Source: Zavodni and Broadbent 1982.

Figure 8.3-29 Typical regressive/progressive state displacement curves

of A range from 0.113 to 2.449, while values for b range from 0.0004 to 0.00294.

t = time (days)

Zavodni reported that during a slope steepening project in Nevada (United States), documented rebound was noted 120 m (400 ft) into the slope (near the pit toe) without evidence of surface cracking. Finite element modeling confirmed these deep-seated and near-surface slope dilations.

Regressive and Progressive Movement

If it is to occur, a conventional open-pit "slope failure" begins after the initial response of the excavation and is normally associated with the creation of one or more tension crack(s) at or near the crest of the slope. The development of such cracks is evidence that the slope is at limiting equilibrium and the driving forces (stress) just equals or exceeds the resisting forces (stress). In the case of the slope at limiting equilibrium, stability is decreased by increasing the driving forces (stresses), decreasing the resisting forces (stresses), or by changing both the driving and resisting forces (stresses).

Whether the failure is regressive or progressive depends on whether a potential or active rock slope failure tends to become more stable or less stable. A regressive failure is one that shows short-term decelerating displacement cycles if disturbing events external to the rock are removed from the slope environment. A progressive failure, on the other hand, is one that will displace at an accelerating rate, usually an algebraically predictable rate, to the point of collapse unless active and effective control measures are taken (Zavodni and Broadbent 1982). A third type defined by Zavodni and Broadbent is what is known as the regressive/progressive condition (or "transitional system"). These curves are shown on Figure 8.3-29.

Type I: Regressive condition. The regressive failure type is shown as Curve A on Figure 8.3-29. This failure type is characterized by a series of either accelerating or decelerating displacement trends, as revealed from continuous monitoring programs. The characteristic that qualifies this curve as regressive is the deceleration of each cycle between external stimuli. The cycles are believed to be initiated when the driving force (stress) temporarily exceeds the resisting force (stress), thereby causing the rock slope condition to drop slightly below a safety factor of 1. The velocity of movement will decay if the external disturbance is eliminated. The excess driving force is usually related to an external event such as a mine blast, earthquake, precipitation event, temperature change groundwater pressure change, or excavation of buttressing rock.

Characteristics of regressive type failures are when the

- Ratio of driving stress to resisting stress decreases with displacement, and
- Slope will tend to become more stable with time and show decelerating or stick–slip behavior.

Type II: Progressive condition. The progressive failure demonstrates an increase in the rate of displacement over time until collapse, as shown by Curve B in Figure 8.3-29. Decelerating cycles may be present but would be subtle and nearly indistinguishable from the long-term trend. The time period over which progressive displacement of a large-scale failure takes place is usually short, 4 to 45 days (Zavodni 2000).

Type III: Regressive/progressive condition. As shown by Curve C in Figure 8.3-29, a regressive type failure may transition into a progressive type failure and rapidly lead to collapse. Causes of this change in behavior can include a situation where mining daylights a sliding surface, breakup, or excavation of rock at the toe of a slope, or an increase in water pressure.

Slide Management

Some degree of slope instability can be expected with virtually any slope cut in rock, whether the slope is a mine highwall, is for a road cut, or is a part of some other construction project. When slope-stability investigations indicate that the possibility of slope failure exists, there are a number of response options available (Call and Savely 1990):

- Leave the unstable area alone.
- Continue mining without changing the mine plan.
- Unload the slide through additional stripping.
- Leave a step-out (i.e., a bench or berm of unexcavated rock at the toe of the slope to increase the resisting forces of the slope).
- Conduct a partial cleanup.
- Mine out the failure.
- Support the unstable ground.
- Dewater the unstable area.

The option, or combination of options, that is chosen depends primarily on the nature of the instability and its operational impact. Each situation should be evaluated separately, with safety aspects carefully considered, and with the cost of any remedial action and benefit included.

Contingency Planning

Mine planning should have the flexibility to respond to slope instability. Rather than an after-the-fact crisis response to forced deviation from a rigid mine plan, contingency plans should be prepared in advance so that the response to slope instability is well thought out. A management philosophy of "Do something, even if it's wrong!" will frequently result in more problems than the original event (i.e., slope instability) precipitated. Also, the

lack of adequately trained personnel for the geotechnical group will result in significant scheduling problems and production delays when a slope failure event occurs.

Operational flexibility should be built into the mining plan. For example,

- Adequate ore should be uncovered and available so that production is not dependent on a single location;
- There should be more than one access road into the pit for service and haulage vehicles. The loss of the haul road, when it is the only access into the mine or a portion of the mine, will shut down production for an extended period of time;
- Whenever possible, more than one access to working benches should be maintained; and
- Labor and equipment should be available for slide cleanup when necessary.

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CHAPTER 8.4

Rock Mechanics

Ross Seedsman

INTRODUCTION

Rock mechanics is the applied science of the mechanical behavior of rock and rock masses in a stress field. In mining, knowledge of rock mechanics is combined with empirical reasoning to make predictions of performance of excavations so that safety, environmental impact, and economic value can be optimized.

This chapter focuses on the "applied" part of the definition and particularly how the science can be applied to the practicalities of mining. It provides a ready reference for some of the key definitions and methods in current use. The opportunity is taken to provide some guidelines on the use of various concepts, highlighting where major uncertainties are still present. The coverage of the subject is not detailed or complete but is presented to introduce the subject to nonspecialist mining engineers. Numerous publications provide the detail, the theoretical basis, and recent case studies to which the specialist geotechnical engineer is referred (e.g., Hoek et al. 1998; Brady and Brown 2004).

There continue to be major advances in both mining rock mechanics and mining engineering. Mining practices have evolved in the face of increasing expectations of workplace safety and environmental impact. Zero harm is now the stated goal of most mining companies. Greater reliance is now placed on bulk mining systems, such as sublevel and block caving in metal mines, and draglines and longwalls in the coal sector. The costs of delays in bulk mining systems can be very high. There are now greater demands that the operations perform rigorously to both plan and budget. In this environment, monitoring of mining systems is an essential part of ensuring workplace safety, but it is not adequate to ensure continued production or appropriate environmental impact. Better forecasts of mining conditions are required, and these demand a much better understanding and application of rock mechanics.

In the science of rock mechanics there have been major developments in computer-based stress analysis, such that two- and three-dimensional (2-D and 3-D) stress analyses of openings in rock masses are now routine. The ease at which these can be conducted, together with the associated graphics, may give an unjustifiable level of confidence in the predicted performance of the excavations. Although knowledge of the stresses around a mine opening is essential, it is not sufficient to ensure adequate performance. One key objective of the mining process is the control of displacements into and around the openings. Failure of rock and rock masses is readily induced around mining openings, so the prediction of displacements requires knowledge of the strength of rock masses. Unfortunately, knowledge of rock strength is comparatively poor, with fundamental revisions being made within the last decade.

Much of rock mechanics depends on the appreciation of engineering geology. It is only by understanding the distribution and characteristics of rock types, the associated discontinuities, and the size of the blocks defined by them that rock mechanics can be applied.

Nature of Rock

Rock and rock masses are inherently complex (Brady and Brown 2004). The complexities relate to the following:

- Fractures are developed in a compressive stress field. This limits the ability to transfer knowledge from other branches of engineering where failures are induced in a tensile stress regime.
- Mine openings can vary in scale from the order of meters to hundreds of meters. Over this scale there is a need to consider both intact rock and the rock mass with all its discontinuities.
- As a result of the discontinuities, the tensile strength of a rock mass must be considered to be zero.
- In most cases, mining takes place below the original water table. Water may have an impact in terms of both effective stress and in terms of reducing the strength of certain rocks.
- Weathering of the rock around mine openings can lead to chemical and/or physical alteration of the mineral components and hence rock strength.

The importance of the concept of scale is illustrated in Figure 8.4-1. At the scale of a roadway or a bench, the behavior of the excavation may be controlled by the strength of the intact portions or by the presence of one or two joints or

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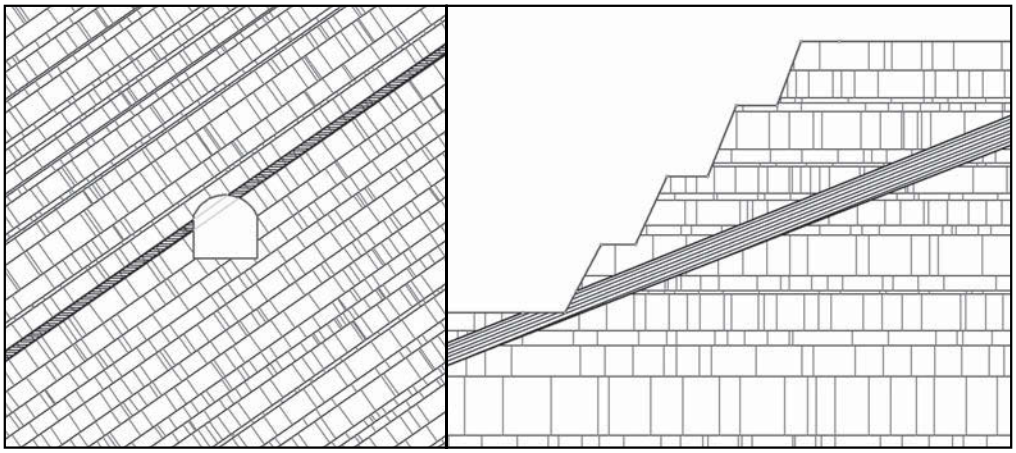


Figure 8.4-1 Different features of the rock mass control its behavior at different scales

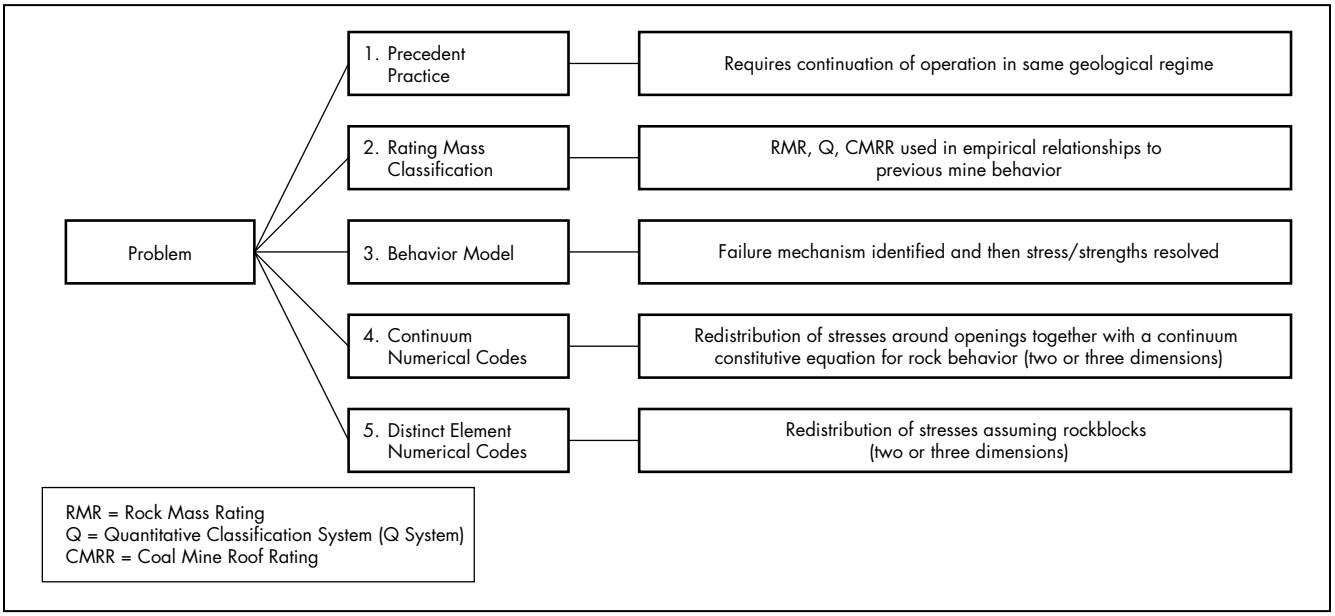


Figure 8.4-2 Different pathways for concept formulation and analysis

beds. At a large scale—for example, the wall of a deep surface mine—the behavior of the rock mass may be controlled by the overall joint and bedding structure. Should the excavation intersect a major fault zone, then the behavior will be controlled by that single feature.

For analytical convenience, rock masses are often considered to be continuous, homogeneous, isotropic, and linear elastic (CHILE) according to the following definitions:

- **Continuous:** Intact rock with no breaks
- **Homogeneous:** All the same rock
- **Isotropic:** Rock properties are the same in all directions.
- **Linear elastic:** An increment or decrement in stress produces the same increment or decrement of strain (there is no permanent deformation and no failure)

In reality, rock masses are discontinuous, inhomogeneous, anisotropic, and nonlinear elastic (DIANE). The assumption of CHILE continuum behavior introduces major risks. Many of the operational mining problems relate to the rock mass not behaving as a continuum. For example, in many applications ground support is best addressed through the consideration of blocky rock, as is bench design in surface mines, though there are notable exceptions.

Rock Mechanics and Mine Design

Rock mechanics design fits within an engineering design methodology (Bieniawski 1993). Steps in the methodology emphasize the need to characterize the site geology as well as the strength of the materials. There is a need to simplify the complex conditions into geotechnical models (concept formulation) that can be subsequently analyzed. There are five pathways through the concept formulation and analysis steps, four of which rely on aspects of the science of rock mechanics (Figure 8.4-2). Precedent practice is the exception—this is still an important pathway, so long as there is confidence that the ground conditions are the same.

Rock mass classification schemes seek to transfer knowledge gained from previous mining experiences to other sites.

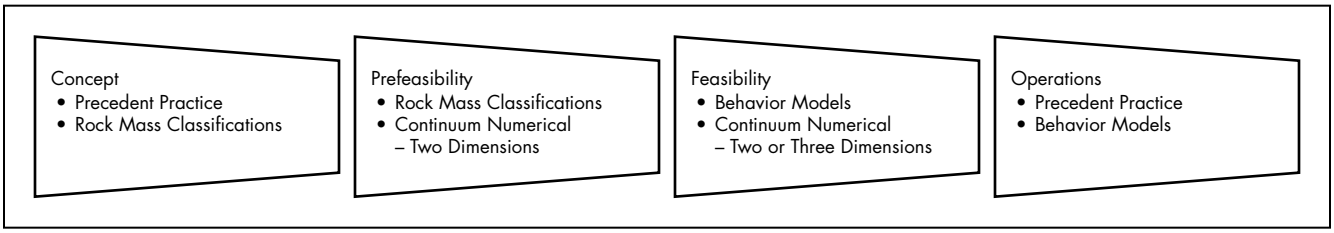


Figure 8.4-3 Suggested minimum application of design pathways for different project stages

The schemes allocate numerical values to features of the rock mass considered likely to influence overall behavior and then combine them into a single rating value. Although useful for translating experience within a single mine site or mining district, caution is required in their use in the absence of analysis of the mechanics of the problem being considered (Brady and Brown 2004). The schemes are useful for planning purposes, but they are not necessarily useful for the prescription of rock support during actual construction (Palmstrom and Broch 2006).

The term *behavior model* is used here to mean the identification of failure or collapse modes separate from the calculation of stresses. Wedge, planar, and toppling failures in the benches of surface mines are examples. The limit equilibrium methods characteristic of soils engineering are also examples of how the failure mechanism is identified prior to the analysis. Kaiser and Kim (2008) argued that by concentrating on the behavior of the excavation boundary, the significance of brittle tensile failure was identified and a new failure criterion was developed. Numerical codes may be used to assist in the calculation of stresses.

In the numerical approaches, the failure zones are identified after the stresses are calculated. Therefore, they depend on the failure criterion input to the model. Field stresses, constitutive equations (stress–strain relationships), and failure criteria are required, and these require assumptions/simplifications of equal or greater magnitude as the formulation of behavior models. Some continuum numerical codes (e.g., Phase 2, FLAC [Fast Lagrangian Analysis of Continua]) determine stresses about openings, assuming the rock mass behaves as a continuum. In some codes, discontinuities can also be modeled. In distinct element codes (e.g., UDEC [Universal Distinct Element Code]), the rock mass is treated as quasi-rigid blocks that interact through movement along the discontinuities. Codes for both 2- and 3-D analyses are now in widespread use.

Which pathway to use depends on the stage of the mining project. It is good practice to use at least two pathways to provide an internal check on the design. Figure 8.4-3 provides general guidelines for each stage; there will be situations where different levels of geotechnical knowledge will require different strategies. Precedent practice is the strongest pathway for operations, but this should be accompanied by a way of confirming that the same geotechnical regime is present. Historically, the latter has been done through the rock mass classification pathway, and the limitations of this were mentioned earlier. The classification schemes were developed before the advent of personal computers that make stress analyses readily available. Since 1990, the focus has been on the numerical code pathways, and there is now a realization that knowledge of the constitutive equations and failure criteria is limited. The behavior model pathway may be more appropriate as it demands good observation of how the rock actually behaves, and this not only identifies immediate hazards but also identifies the need for new scientific knowledge. Figure 8.4-3 suggests that the rock mass classification pathway is particularly suitable at early stages of a mining project when there is little data available. At the prefeasibility level, numerical codes can be used to determine stress redistributions around openings, and they allow for optimization of the overall layout while, at the same time, work is done to characterize the discontinuity fields. As the mine moves to the operational phase, the focus needs to move to the performance of the excavation boundaries (roadway or bench), and the behavior of relatively small blocks of rock may be best addressed by the behavior model pathway.

Sophisticated analytical and prediction tools do not ensure adequate or appropriate predictions. Using concepts from soils engineering, there is a need to be aware that for a given quantum of data there is an appropriate degree of sophistication in the analysis, beyond which the accuracy of the prediction may decline (Lambe 1973). In referring to Lambe, it is not suggested that accurate Class A predictions are essential or even possible in mining ventures; although the better the Class A predictions, the lower the risk to capital. The message is that the mining engineer must "maintain an ongoing awareness of factors that contribute to unsuccessful performance and introduce this awareness into comprehensive risk management tools" (Morgenstern 2000).

Scope

The application of four of the pathways requires knowledge of rock mechanics, either of rock mass classification systems or of the combination of the strengths of rock and rock masses as well as the stresses around excavations. This chapter is a general introduction to the subject of rock mechanics so as to provide a broad understanding of the issues and the associated uncertainties. It is anticipated that the scope will be of value to mine planners seeking to understand the constraints that need to be applied, to mining engineers who are using the readily available stress analysis codes and require an understanding of the input parameters, and to engineering geologists that are looking for an introduction to the engineering concepts.

Specialist geotechnical engineers are referred to many of the standard texts for detailed treatment of the subject.

ROCK PROPERTIES

There are two fundamental sets of rock properties: the discontinuities and the substance itself. This section describes how rocks can be characterized at the laboratory and outcrop scale. The rock properties in the following discussion are used to different degrees in the four analytical pathways discussed earlier, so they must be determined as accurately as possible.

Table 8.4-1 Definitions of terms for spacing and persistence

Spacing Descriptions	Spacing, mm
Extremely close	<20
Very close	20–60
Close	60–200
Moderate	200–600
Wide	600–2,000
Very wide	2,000–6,000
Extremely wide	>6,000
Persistence Descriptions	**Modal Trace Length, m**
Very low	<1
Low	1–3
Medium	3–10
High	10–20
Very high	>20

Source: ISRM 1978.

Discontinuities

The distinctive feature of rock masses is the presence of discontinuities. In this context, discontinuities are defined as features of a rock mass that have zero or negligible tensile strength. This terminology allows a differentiation from textural features within intact rock. From a geological perspective, discontinuities can include joints, bedding, partings, faults, veins, shear zones, cleavage, and schistosity. The term *defect* can also be used interchangeably with discontinuity. A corollary of this definition is that the tensile strength of a rock mass should be assumed to be zero.

The International Society for Rock Mechanics (ISRM) provides the framework for the description of discontinuities.

Spacing and Persistence

Table 8.4-1 presents the ISRM definitions of terms for spacing and persistence. Although this table may be satisfactory for igneous and metamorphic regimes, it does not adequately address the range of conditions that can be encountered in sedimentary sequences, such as found in coal mining. Bedding partings can have persistence values very much greater than 20 m, and there are situations in some of the coal mass-mining regimes (longwalls) where it is necessary to describe bedding parting spacings in the order of 30 to 50 m.

In most rocks, it appears that discontinuity spacing tends to follow a negative exponential function, although in some cases a lognormal distribution better applies (Brady and Brown 2004). In relatively undisturbed sedimentary rocks, a valid first-order assumption is that the mean joint spacing is equal to the mean spacing of bedding partings (Ji et al. 1998); the rock mass is composed of cubes, though this simplification should be applied with caution.

The rock quality designation (RQD) is a simple measure of discontinuity spacing and rock mass quality. The RQD is the percentage of core that reports in lengths greater than 100 mm. The core lengths should be measured along the center line, and only double- or triple-tube drilling should be used.

Reduced discontinuity spacing and greater persistence should be anticipated near faults. Many ground control problems that are reported in the vicinity of faults are related more to the smaller block size defined by the closer-spaced discontinuities than to higher stresses that are often presumed to be present. In fact, the deviatoric stresses near faults are likely to be lower than in the adjacent rock mass.

Table 8.4-2 Typical JRC values as a function of shape and roughness of discontinuities

Description	Planar (continuous without change in orientation)	Stepped (well-defined sharp changes in orientation)	Undulating (wavy shape)
Rough: Like sandpaper; many small irregularities associated with the grain size	2.5	20	14
Smooth: Roughness not detected with finger	1.5	14	11
Polished/slickensided: Reflects light; striated	0.5	11	7

Orientation

The onset of planar, wedge, and toppling failures in surface mines, and also around underground openings, is related to the relative orientation of the excavation to the dominant discontinuities. The determination of the dip and dip direction/strike of the discontinuities is therefore a critical step in any mine design. Mapping of any exposures should be conducted and supplemented with information from drill holes. Modern borehole imaging tools are to be preferred over core orientation to obtain orientation data from drilling programs.

In some situations, knowledge of the regional geology can assist in forecasting orientations. For example, in gently deformed sedimentary regimes it may be possible to infer joint orientations from knowledge of the orientation of fold axes (Fookes et al. 2000).

Surface Conditions—Joint Roughness, Aperture, and Fill

Discontinuities in rock are rarely perfectly smooth and planar. The undulations and roughness of natural discontinuity surfaces have a significant influence on their shear-strength properties, particularly at the low stresses around the immediate excavation boundary (Barton 1973). One set of definitions for surface conditions is presented in Table 8.4-2, along with corresponding joint roughness coefficients (JRCs).

The adjacent rock walls of a discontinuity may be separated by infill such as clay, calcite, and fault gouge. The aperture of a discontinuity is the perpendicular distance between the adjacent rock walls when the intervening space is filled with air or water. In most cases the aperture will be small unless there has been recent movement along irregular surfaces, dilatant stress conditions, or erosion of fill materials. Hydraulic conductivity with laminar flow is proportional to the cube of the aperture.

Shear Strength

If a rough irregular discontinuity is considered between two blocks of rock held together by a stress normal to the surface and exposed to a shear stress, as the shear stress increases, shear displacement will increase. Eventually, a peak shear stress will be reached (peak strength) and continuing shear displacement will require a lesser shear force (residual strength). Repeating this test for different normal loads allows for the construction of two sets of envelopes for the normal stress/shear stress pairs. Shear testing is normally done using a direct shear test (analogous to the left-hand diagram in Figure 8.4-4).

With a restricted number of laboratory tests (e.g., three to five), the peak strength envelope is typically found to be bi-linear. The angle of the second linear segment is called the basic friction angle (φb), and the gradient of the initial

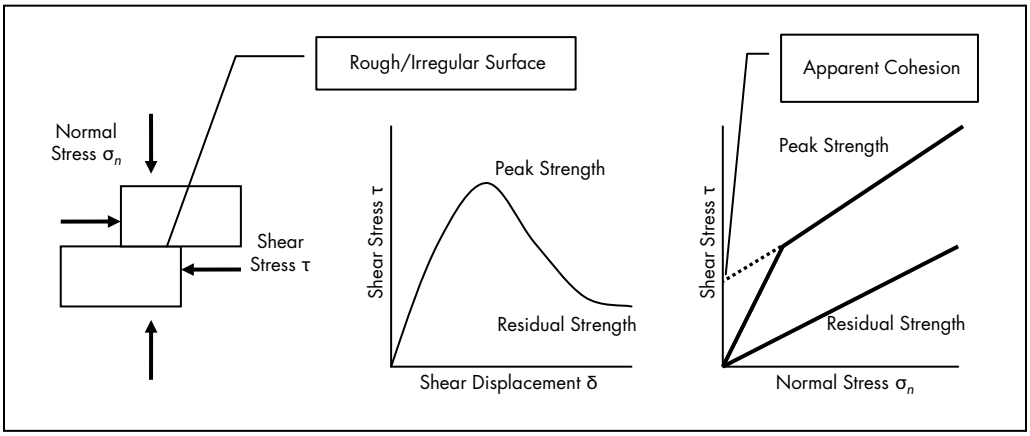


Figure 8.4-4 Terminology for shear-strength testing

Table 8.4-3 Typical basic friction angles, modular ratios, and Hoek–Brown *m* values

Rock Type	Basic Friction Angle, degrees	Modular Ratio	Hoek–Brown *m* Parameter
Amphibolite	32	400–500	26 ± 6
Basalt	35–38	250–450	25 ± 5
Bentonite	9–17	—	—
Chalk	30	>1,000	7 ± 2
Clays	12–19	—	—
Conglomerate	35	300–400	21 ± 3
Dolomite	31–37	350–500	9 ± 3
Granite	31–35	300–550	32 ± 3
Limestone	33–39	400–1,000	8 ± 3
Quartzite	—	—	20 ± 3
Sandstone	26–35	200–350	17 ± 4
Schists	31–4	250–450	10 ± 3
Shale	27–32	150–250	6 ± 2
Siltstone	31–33	350–400	7 ± 2
Slate	25–30	400–600	7 ± 4
Tuff	—	—	13 ± 5

Source: Adapted from Barton 1973, Hoek et al. 1998, and Hoek and Diederichs 2006.

linear segment is related to both the basic friction angle and the roughness of the discontinuity. Typical values for the basic friction angle for different rock types are given in Table 8.4-3.

The shear strength of natural discontinuities (Barton 1973) can be expressed as follows:

$$\tau = \sigma_n \times \tan(\text{JRC} \log10(\text{JCS}/\sigma_n) + \varphi b)$$

where

JRC = joint roughness coefficient (Table 8.4-2)
JCS = joint compressive strength (typically equal to the uniaxial compressive strength unless thick infill)

The JRC for the residual envelope is taken as zero.

At low normal stresses or high joint compressive strengths, the roughness factor results in a high equivalent friction angle and, hence, a high shear resistance. For low values of the JCS/σ_n ratio, the equivalent friction angle is lower and the shear resistance is also lower.

An alternative interpretation of the peak strength could be a single straight line that would give the finite shear strength value at zero normal load (referred to as apparent cohesion). This interpretation would overestimate the shear strength at low normal loads.

In numerical models, there is often a requirement for the joint normal stiffness, *Kn*, and the joint shear stiffness, *Ks*. Values for these two parameters are neither well documented nor easy to measure.

One analytical approach is to estimate *Kn* from the rock mass modulus, the intact rock modulus, and the joint spacing (Brady and Brown 2004). If it is assumed that the deformability of a rock mass is due to the deformability of the intact rock and the deformability of the joints in the rock mass, then

$$\frac{1}{Erm} = \frac{1}{Ei} + \frac{1}{KnL} \text{ or } Kn = \frac{EiErm}{L(Ei - Erm)}$$

where

Erm = rock mass modulus
Ei = intact rock modulus
L = mean joint spacing

The same reasoning can be used to derive the following expression for the joint shear stiffness:

$$Ks = \frac{GiGm}{L(Gi - Gm)}$$

where

Ks = joint shear stiffness
Gi = intact rock shear modulus
Gm = rock mass shear modulus
L = mean joint spacing

Another approach to estimating joint stiffness assumes that a joint has an infill material with known elastic properties. The stiffness of a joint can be estimated from the thickness and modulus of the infilling material by the following equation:

$$Kn = \frac{Eo}{H} \text{ and } Ks = \frac{Go}{H}$$

where

Eo = Young's modulus of infill material
H = joint thickness or aperture
Go = shear modulus of infill material

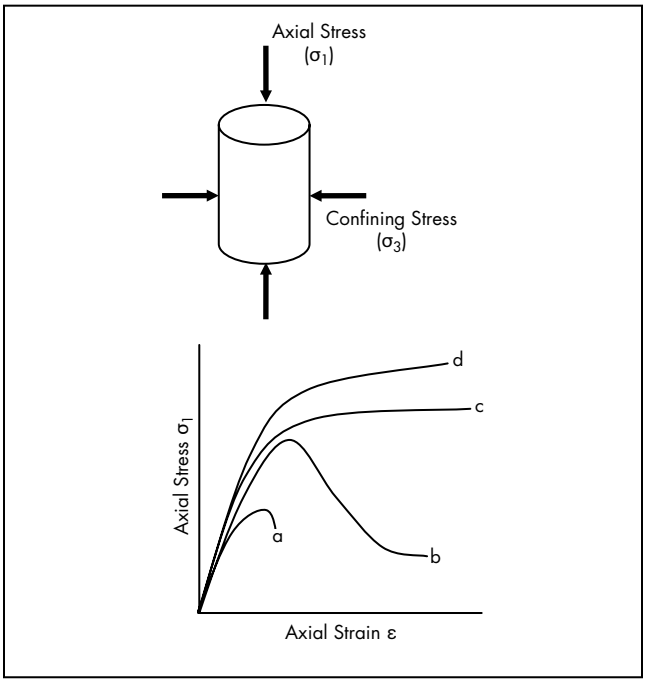


Figure 8.4-5 Terminology used for compressive strength testing

This equation may also be applied to clean joints if the value of H is assumed to be very small (e.g., <1 mm).

Intact Rock

The strength of intact rock is determined in the laboratory by a series of tests on rock cores. The cores are installed in a purpose-designed pressure vessel and exposed to a number of different confining pressures. An axial stress is applied, and as it increases, the rock compresses axially, and it initially may contract slightly laterally. Later in the test, the rock will dilate laterally. The relationship between stress and strain is not linear over the full range of applied stress; sometimes a yield point can be defined where the stress–strain curve deviates significantly from being linear. The peak strength is measured for each confining stress, and this increases with the magnitude of the confinement (Figure 8.4-5). At the same time, the shape of the post-peak part of the curve may change from brittle (Curve a) through strain softening (Curve b) to plastic (Curve c) and strain hardening (Curve d) as the confinement increases, depending on the nature of the rock material.

Uniaxial Testing

Strength. Laboratory unconfined or uniaxial compressive strength (UCS) is the key index to rock strength. The UCS is the strength measured in the laboratory on samples with the length/diameter ratio greater than 2 and typically on 50- to 65-mm-diameter samples. Testing should be conducted on fresh core and at a moisture content as close as possible to the in-situ value. Control of moisture content is important, especially for the lower-strength rocks (e.g., <20 MPa), as it is these that may be easily overstressed in some mining conditions. Testing conducted at lesser aspect ratios will result in higher indicated strengths, and testing of larger samples will result in lower strengths.

Table 8.4-4 Simple ways of estimating UCS

Term	UCS, MPa	Field Attribute
Extremely strong	>250	Chipped with a geological hammer
Very strong	100–250	Many blows with geological hammer to break core
Strong	50–100	More than one blow with geological hammer to break core
Medium strong	25–50	Cannot be scraped or peeled with pocket knife; core can be broken with single blow from hammer
Weak	5–25	Can be peeled with pocket knife; shallow indentation made by firm blow with point of geological hammer
Very weak	1–5	Crumbles under firm blow with point of hammer; can be peeled with a pocket knife
Extremely weak	0.25–1	Indented by fingernail

Source: Adapted from Hoek and Brown 1997.

Field estimates (Table 8.4-4) have been found to be very useful for the lower-strength rocks that can be readily overstressed around openings and yet may be difficult to sample and test in the laboratory. When using this table, one should be careful not to include core breaks along incipient discontinuities or textures.

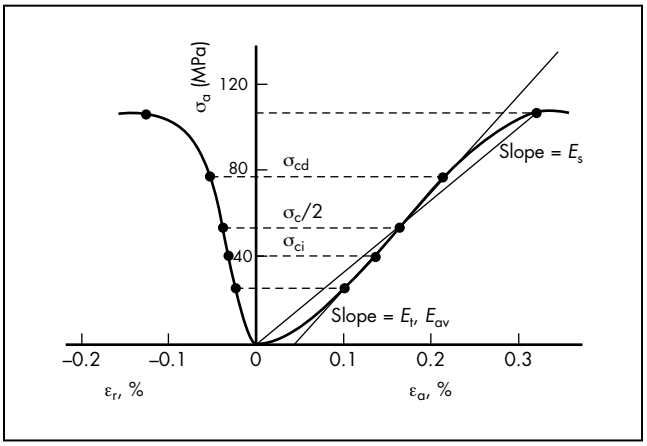
The UCS is a fundamental parameter in most strength criteria and should be measured directly whenever possible. The engineering literature suggests that the UCS can be determined from the point-load strength index (UCS = 24 I_{s50} value, which is the point-load strength corrected to a 50-mm-diameter equivalent). Unfortunately, the constant has been found to vary massively between various rock types, and the default value of 24 should be used with extreme care. The testing standards require a large number of tests—at least twice as many as the UCS. A high degree of scatter in the results is common. Point-load testing should not be relied on for rocks with UCS values of less than about 25 MPa.

In the coal sector, sonic velocity obtained from slim-line borehole tools is often used as an index to UCS. Similar to the point-load test, the relationship is empirically derived for each site. Any extrapolation to other sites needs to be applied carefully, and care needs to be taken to ensure that the lower-strength rocks are adequately tested. An exponential relationship to the inverse of the sonic velocity has been found to fit the data reasonably well.

$$\text{UCS} = 5{,}800e^{-17{,}400/vel}$$

where vel is the sonic velocity in m/s (between 2,000 and 4,500 m/s). The indirect tensile strength of rock core can be measured by the so-called Brazilian test on core discs. The tensile strength is typically 1/10 to 1/20 of the UCS.

Deformation parameters. In mining rock mechanics, a common assumption is made that, prior to failure, rock can be considered to be linear elastic. In elasticity theory, there are 21 constants in the constitutive equations that relate deformation to stress. With increasing material symmetry, the number of elastic constants is reduced. An orthotropic material has three material axes and nine independent elastic constants. A gneiss or granite may be orthotropic. Many sedimentary rocks show pronounced differences in properties of samples parallel and perpendicular to the bedding and can be considered



Source: Brady and Brown 2004, reprinted with permission from Springer Science and Business Media.

Figure 8.4-6 Typical stress–strain curves during uniaxial loading

to be transversely isotropic. Five elastic constants are needed for the description of transversely isotropic rock. Only two independent elastic constants are needed for the description of isotropic rock that, by definition, lacks directional mechanical properties.

The prefailure shape of typical stress–strain curves for rock is not perfectly linear (Figure 8.4-6). Assuming linear-elastic and isotropic behavior, the elastic deformation parameters of Young's modulus and Poisson's ratio can be calculated from laboratory data. The Young's modulus is calculated as the ratio of change in axial stress to the corresponding change in axial strain, and the Poisson's ratio is the ratio of the change in the radial strain to axial strain. Typically, tangent values are determined at 50% of the peak strength, and average values are determined to be between one-third and two-thirds of peak strength. Young's modulus and Poisson's ratio are assumed not to vary with confining stress. Secant values to peak strength may also be quoted.

The interrelationships between other elastic parameters are

$$G = E/2(1 + \nu)$$
$$\lambda = 2\nu G/(1 - 2\nu)$$

where

G = shear modulus (also known as the modulus of rigidity)
E = Young's modulus
ν = Poisson's ratio
λ = Lame's constant

It has been determined empirically that the ratio of the Young's modulus to UCS (the modular ratio) varies within relatively narrow limits depending on rock type (Table 8.4-3).

The layering in sedimentary rocks introduces a degree of transverse isotropy. Implementation of transverse isotropy requires two deformation moduli and two Poisson's ratios as well as an independent shear modulus. There is very little guidance on the choice of values of the independent shear modulus. For continuum analyses of layered rock masses, the author has found that values in the order of 100 to 250 MPa may be necessary to account for both textural impacts and bedding partings.

Confined Strength

In the last 20 years, and especially since 2000, there have been major changes in the models for the confined strength of rock. As of 2009, there is no single failure criterion for the full range of stresses, and separate analyses are needed for the excavation boundary and the far field.

The envelope of confining stress/axial stress points for peak strength is nonlinear. The more sample points that are collected, the more the nonlinearity of the relationship becomes apparent. The three common approaches to fitting a relationship are

- **Straight line:** The Mohr–Coulomb criterion that is often used in software codes
- **Power:** Hoek–Brown criterion (Hoek and Brown 1980)
- **Multilinear:** The brittle criterion (Kaiser et al. 2000)

The testing cells used in standard laboratory tests mean that one confining stress is applied. The test is conventionally referred to as a triaxial test, with the understanding that the magnitudes of two of the stress axes are identical. A consequence of this is that the failure criteria refer to only two stresses, σ_1 and σ_3, because $\sigma_2 = \sigma_3$ (discussed later). A true triaxial failure criterion is the subject of ongoing research.

Mohr–Coulomb criterion. The linear fit has the following relationship:

$$\sigma_1 = \sigma_c + \sigma_3 \tan\psi$$

where

σ_c = intercept for zero confining stress and should be the same value as the UCS
ψ = slope in σ_1/σ_3 space

These parameters can be manipulated to give the cohesion, c, and friction angle, Φ, such that the following conditions are met:

$$\tan\psi = (1 + \sin\varphi)/(1 - \sin\varphi)$$
$$\sigma_c = 2c\cos\varphi/(1 - \sin\varphi)$$
$$\sin\varphi = (\tan\psi - 1)/(\tan\psi + 1)$$
$$c = \sigma_c(1 - \sin\varphi)/2\cos\varphi$$

The value of σ_c may not be the same as the UCS value. This is because the confined samples typically fail by shear, whereas the unconfined samples may fail by axial splitting.

The tensile intercept, σ_T, equals $2c\cos\phi/(1 + \sin\phi)$. When measured in the laboratory, the tensile strength is often lower than this, and it is common practice to specify a tensile cutoff T_0. Recognizing the presence of discontinuities, it is prudent to set T_0 to zero for rock masses.

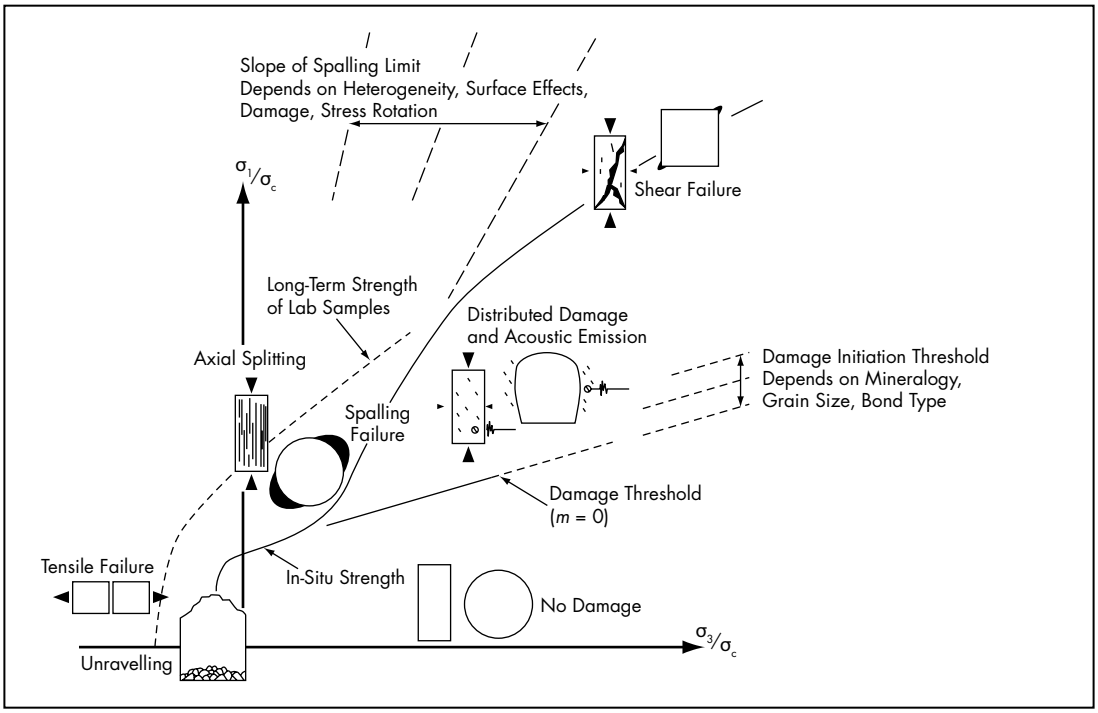
The empirical basis for a straight-line fit is not strong, but there is a legacy of widespread use because of the simplicity of the parameters and the ability to make an analogy with soil strengths. The Mohr–Coulomb criterion is often implemented by a line of best fit over an appropriate stress range—this is a valid response to the nonlinearity, provided the stress range is adequately assessed.

Hoek–Brown criterion. The empirical Hoek–Brown failure criterion was developed in response to observations that the failure envelope of rock is not linear. It states that

$$\sigma_1 = \sigma_3 + (m\sigma_c\sigma_1 + s\sigma_c^2)^a$$

where

m = a value that reflects lithology (Table 8.4-3)
s = 1.0



Source: Diederichs et al. 2004, with permission from Elsevier.

Figure 8.4-7 A failure criterion incorporating brittle failure, spalling limits, and Hoek–Brown components

$a = 0.5$ for intact rock

This criterion has been subsequently extended to rock masses.

For intact rock, the implication is that $\sigma_T = \sigma_c/m$. There is typically no provision in computer software to override this with T_0 = zero.

This criterion has been widely adopted, perhaps in part because of the absence of an alternative. It has been refined several times and now forms the basis of the generalized Hoek–Brown criterion for rock masses (discussed later). Recent work suggests that this criterion should not be applied in the immediate vicinity of excavations.

Brittle failure criterion. The onset of damage around an excavation is poorly predicted with the standard Hoek–Brown criterion, and it is better modeled with a set of revised parameters such as $m = 0.0$ and $s = 0.11$ (Martin et al. 1991). This is equivalent to a cohesion value of UCS/6 to UCS/4 and a friction angle of zero degrees if the Mohr–Coulomb criterion is used.

Subsequently, the following tri-linear envelope (Figure 8.4-7) has been proposed (Kaiser et al. 2000):

- $\sigma_1 - \sigma_3$ = deviatoric stress = $\sqrt{s}\ \sigma_c$, where s is in the range of 0.11 to 0.25
- σ_1/σ_3 = spalling limit = 20:1 to less than 3.4:1
- Hoek–Brown line at higher stresses

This failure criterion has been found to predict the onset and depth of failure in the vicinity of excavation boundaries in metaliferous mines, tunnels, and underground coal mines. One advantage of the criterion is that the onset of failure is a function of the ratio of one strength parameter (the UCS) to the major principal stress, and this allows ready normalization.

The full criterion is not yet implemented in numerical codes, so the current advice is to run two sets of analyses (Kaiser and Kim 2008)—one for the excavation boundary using the $m = 0$ relationship and a second for the far field (greater than, for instance, 10 m from the boundary) using the standard Hoek–Brown rock-mass criterion (discussed later). Particularly in coal measures, there is a need to run a third analysis using the spalling limit portion of the criterion, and it may be that spalling limits as low as 3.4 need to be assessed. The $m = 0$, $s = 0.11$ form of the criterion infers artificially large tensile strengths. Until the full implementation of the criterion is available in numerical codes, it may be preferable to use the Mohr–Coulomb form, as software codes usually allow a tensile strength cutoff that can be set to zero.

Post-Peak or Yielding Behavior

Referring to Figure 8.4-7, a stress state cannot exist in a rock or rock mass above any of the failure criteria. In elastic models, the separation of the stress field from the failure criterion is expressed as some sort of strength factor. Alternatively, the post-peak or post-failure behavior can be considered in nonlinear models. This is an advanced area of rock mechanics and outside the scope of this chapter. The term *yield* is often used in this context—this is not strictly correct unless the assumption of perfect linear-elastic behavior is valid, but it is a convenient and common usage. Some very general guidance is available.

Figure 8.4-5 illustrates the following four post-peak behaviors: brittle, strain softening, plastic, and strain hardening.

Implementation in numerical codes requires failure parameters, a dilation angle, and strength parameters in the post-peak range, and there is very little experimental data to support the selection of the latter two parameters (Table 8.4-5). There are precedents in using the elastic-plastic assumption. Post-peak behavior can only be implemented in continuum numerical codes if the rock is assumed to be isotropic. Post-peak behavior is explicitly considered if slip along joints is implemented in a numerical model. Calibration of any

Table 8.4-5 Possible post-peak parameters

	Brittle	Strain Softening	Elastic Plastic
Dilation angle	φ/4	φ/8	0
Cohesive strength	0	UCS/2	UCS
Deformation modulus	E/4	E/2	E

Source: Adapted from Hoek and Brown 1997.

models that consider post-peak behavior is strongly advised. Application of any models outside conditions against which they have been calibrated should be done with great caution.

ROCK MASSES

Some mining problems are at such a scale that a large volume of the rock mass must be considered. There are two approaches to assess the performance of rock masses. One approach is the classical rock mass classification route that bypasses the need for any analysis of modes of behavior. A more recent approach has been to assume that the rock mass is so impacted by discontinuities that it can be considered to be a continuum and to then use a rock mass classification to generate equivalent strength and deformation parameters.

RMR and Q System

Rock mass classification systems seek to apply a single numerical value to describe the condition of rock masses based on a consideration of what are believed to be key parameters controlling behavior. Two of the most popular systems are the rock mass rating (RMR) of Bieniawski (1973, 1976, 1989), which has been revised several times, and the Q system (Barton et al. 1974). In both systems, the UCS of the rock plays a relatively minor part in the characterization reflecting the origin of the systems in what have been referred to as "hard" rock. Commentaries on the application of both systems often state that they are suitable for hard rock—unfortunately, *hard* is not defined. It is quite possible that *hard* may refer to intact rock strengths in excess of about 50 to 75 MPa. The rock mass classification approach is ideally suited for situations where the failure process is controlled by sliding and rotation of intact rock pieces; this approach is less reliable for squeezing, swelling, spalling, or slabbing ground or for rock bursting under very high stresses (Marinos et al. 2005). The RMR adds together scores for UCS, RQD, joint spacing and condition, and groundwater to get a value out of 100 (Table 8.4-6). The precision of the system is in the order of ±10 units, as reflected in the definition of five grades of rock. The most common application is in the identification of stand-up times. A related system has been developed for coal measures (Molinda and Mark 1994). There is also an application of RMR to slope stability (Romana 1993).

The Q system is based on multiplying three ratios that are considered to reflect block size, inter-block frictional strength, and the active stress.

$$Q = \left(\frac{\text{RQD}}{J_n}\right)\left(\frac{J_r}{J_a}\right)\left(\frac{J_w}{\text{SRF}}\right)$$

where

RQD = percentage of the core of a rock mass unit in lengths greater than 100 mm
J_n = joint set number (varies between 0.5 and 20)
J_r = joint roughness number (varies between 0.5 for slickensided and 5 for nonpersistent widely spaced)
J_a = joint alteration number (varies between 0.75 and 20)
J_w = joint water reduction factor (varies between 0.05 and 1.0)
SRF = stress reduction factor (varies from 0.5 for high stress and 20 for heavy squeezing)

The quantification of these parameters is listed in Table 8.4-7.

The resulting Q score can vary from 0.001 to 1,000, and with each of the three ratios varying over two to three orders of magnitude. The Q system is in common use for the specification of ground support in tunnels, assessments of the stability of raise-bored shafts, and in stope designs in underground metal mines. It has not been used extensively in underground coal mines.

Given the simplicity of the methods, both classifications should be used separately. The two methods are fundamentally different in that RMR considers only strength, whereas Q seeks to classify performance as it includes a stress term. In some applications the J_w/SRF term is set to unity, so as to reflect only rock mass strength—the resulting parameter is referred to as Q'. There have been numerous papers that seek to relate RMR values to Q values, but this is considered poor practice.

Any use of rock classification schemes should note the following warning issued by Brady and Brown (2004): "Although the use of this approach is superficially attractive, it has a number of serious shortcomings and must be used only with extreme care. The classification scheme does not always fully evaluate important aspects of a problem, so that if blindly applied without supporting analysis of the mechanics of the problem, it can lead to disastrous results."

Rock mass classification schemes are a valuable aid for assessing the likely impact of rock behavior on other aspects of a mining venture—they are particularly useful for initial scoping studies. They should not be relied on in the operational phase and can be misleading in the feasibility stage if not used in combination with other analytical techniques. As implied by Brady and Brown (2004) and many other workers, the key to good implementation is to first obtain site-specific data on the nature and behavior of the particular rock mass as soon as possible and then start analyzing the candidate behavior mechanisms.

Geological Strength Index

A relatively recent use of the rock mass classification systems, particularly the RMR, has been to use it to quantify the parameters required by a CHILE assumption for a rock mass (Hoek and Brown 1997). Since its introduction, this approach has become very popular and tends to be used without adequate consideration of its limitations. First, the scale of the excavation must be such that the consideration is either of the intact rock itself or of a very large volume with sufficient discontinuities such that an equivalent continuum is present. The implication of the intact rock scale is that one is dealing with a situation near the excavation boundary, where the brittle parameters ($m = 0.0$, $s = 0.11$) are more valid. At the other end of the scale, there is a need to ensure that throughgoing geological structures (Figure 8.4-1) are not present so as not to dominate behavior and make the continuum assumption invalid.

The method of determining the geological strength index (GSI) has been summarized in a number of cross plots (e.g., Figure 8.4-8) where consideration of the discontinuities is

Table 8.4-6 RMR classification system*

A. Classification Parameters and Their Ratings

	Parameter		Range of Values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4–10 MPa	2–4 MPa	1–2 MPa	For this low range, uniaxial compressive tests are preferred		
		Uniaxial compressive strength	>250 MPa	100–250 MPa	50–100 MPa	25–50 MPa	5–25 MPa	1–5 MPa	<1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core quality RQD		90%–100%	75%–90%	50%–75%	25%–50%	<25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		>2 m	0.6–2 m	200–600 mm	60–200 mm	<60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (see E)		Very rough surfaces; not continuous, no separation; unweathered wall rock	Slightly rough surfaces; separation <1 mm; slightly weathered walls	Slightly rough surfaces; separation <1 mm, highly weathered walls	Slickensided surfaces or gouge <5 mm, thick or separation 1–5 mm; continuous	Soft gouge >5 mm thick or separation >5 mm; continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10-m tunnel length, L/min	None	<10	10–25	25–125	>125		
		(joint water press)/ (major principle σ)	0	<0.1	0.1–0.2	0.2–0.5	>0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. Rating Adjustment for Discontinuity Orientations (See F)

Strike and dip orientations		Very favorable	Favorable	Fair	Unfavorable	Very unfavorable
Ratings	Tunnels and mines	0	–2	–5	–10	–12
	Foundations	0	–2	–7	–15	–25
	Slopes	0	–5	–25	–50	–

C. Rock Mass Classes Determined from Total Ratings

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	<21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. Meaning of Rock Classes

Class number	I	II	III	IV	V
Average stand-up time	20 years for 15-m span	1 year for 10-m span	1 week for 5-m span	10 hours for 2.5-m span	30 minutes for 1-m span
Cohesion of rock mass, kPa	>400	300–400	200–300	100–200	<100
Friction angle of rock mass, degrees	>45	35–45	25–35	15–25	<15

E. Guidelines for Classification of Discontinuity Conditions

Discontinuity length (persistence) Rating	<1 m 6	1–3 m 4	3–10 m 2	10–20 m 1	>20 m 0
Separation (aperture) Rating	None 6	<0.1 mm 5	0.1–1.0 mm 4	1–5 mm 1	>5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling <5 mm 4	Hard filling >5 mm 2	Soft filling <5mm 2	Soft filling >5 mm 0
Weathering Rating	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

F. Effect of Discontinuity Strike and Dip Orientation In Tunneling

Strike Perpendicular to Tunnel Axis		Strike Parallel to Tunnel Axis	
Drive with dip: Dip 45°–90°	Drive with dip: Dip 20°–45°	Dip 45°–90°	Dip 20°–45°
Very favorable	Favorable	Very favorable	Fair
Drive against dip: Dip 45°–90°	Drive against dip: Dip 20°–45°	Dip 0°–20°, irrespective of strike	
Fair	Unfavorable	Fair	

Source: Hoek et al. 1998.

*Some conditions are mutually exclusive. For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. In such cases, use A.4 directly.

Table 8.4-7 Q-system parameter values

Description	Value	ϕr degrees (approx.)	Notes
1. Rock Quality Designation	***RQD***		
A. Very poor	0–25		1. Where RQD is reported or measured as ≤10 (including 0), a nominal value of 10 is used to evaluate Q.
B. Poor	25–50		2. RQD intervals of 5 (i.e., 100, 95, 90, etc.) are sufficiently accurate.
C. Fair	50–75		
D. Good	75–90		
E. Excellent	90–100		
2. Joint Set Number	J_n		
A. Massive, no or few joints	0.5–1.0		1. For intersections, use ($3.0 \times J_n$).
B. One joint set	2		2. For portals, use ($2.0 \times J_n$).
C. One joint set plus random	3		
D. Two joint sets	4		
E. Two joint sets plus random	6		
F. Three joint sets	9		
G. Three joint sets plus random	12		
H. Four or more joint sets, random, heavily jointed, "sugar cube," etc.	15		
I. Crushed rock, earthlike	20		
3. Joint Roughness Number	J_r		
a. Rock wall contact			1. Add 1.0 if the mean spacing of the relevant joint set is greater than 3 m.
b. Rock wall contact before 10-cm shear			2. $J_r = 0.5$ can be used for planar, slickensided joints having lineations, provided that the lineations are oriented for minimum strength.
A. Discontinuous joints	4		
B. Rough and irregular, undulating	3		
C. Smooth undulating	2		
D. Slickensided undulating	1.5		
E. Rough or irregular, planar	1.5		
F. Smooth, planar	1		
G. Slickensided, planar	0.5		
c. No rock wall contact when sheared			
H. Zones containing clay minerals thick enough to prevent rock wall contact	1.0 (nominal)		
I. Sandy, gravely, or crushed zone thick enough to prevent rock wall contact	1.0 (nominal)		
4. Joint Alteration Number	J_a	ϕr degrees (approx.)	
a. Rock wall contact			1. Values of ϕr, the residual friction angle, are not intended as an approximate guide to the mineralogical properties of the alteration products, if present.
A. Tightly healed, hard, nonsoftening, impermeable filling	0.75		
B. Unaltered joint walls, surface staining only	1.0	25–35	
C. Slightly altered joint walls, nonsoftening mineral coatings, sandy particles, clay-free disintegrated rock, etc.	2.0	25–30	
D. Silty- or sandy-clay coatings, small clay fraction (nonsoftening)	3.0	20–25	
E. Softening or low-friction clay mineral coatings (i.e., kaolinite, mica). Also chlorite, talc, gypsum, graphite, etc., and small quantities of swelling clays (discontinuous coatings, 1–2 mm or less).	4.0	8–16	
b. Rock wall contact before 10-cm shear			
F. Sandy particles, clay-free, disintegrating rock, etc.	4.0	25–30	
G. Strongly over-consolidated, nonsoftening clay mineral fillings (continuous <5 mm thick)	6.0	16–24	
H. Medium or low over-consolidation, softening clay mineral fillings (continuous <5 mm thick)	8.0	12–16	
I. Swelling clay fillings (i.e., montmorillonite; continuous <5 mm thick). Values of J_a depend on percentage of swelling clay-size particles and access to water.	8.0–12.0	6–12	

(continues)

Table 8.4-7 Q-system parameter values (continued)

Description	Value		Notes
4. Joint Alteration Number (continued)	J_a	ϕr degrees (approx.)	
c. No rock wall contact when sheared			
J. Strongly over-consolidated, nonsoftening clay mineral fillings (continuous <5 mm thick)	6.0		
K. Medium or low over-consolidation, softening clay mineral fillings (continuous <5 mm thick)	8.0		
L. Swelling clay fillings (i.e., montmorillonite; continuous <5 mm thick). Values of J_a depend on percentage of swelling clay-size particles and access to water.	8.0–12.0	6–24	
M. Zones or bands of silty- or sandy-clay, small clay fraction, nonsoftening	5.0		
N. Thick continuous zones or bands of clay	10.0–13.0		
O. and P. (see G, H, and I for clay conditions)	6.0–24.0		
5. Joint Water Reduction	J_W	**Approx. Water Pressure (kgf/cm²)**	
A. Dry excavation or minor inflow (i.e., <5 L/m locally)	1.0	<1.0	1. Factors C to F are crude estimates: increase JW if drainage installed.
B. Medium inflow or pressure, occasional outwash of joint fillings	0.66	1.0–2.5	2. Special problems caused by ice formation are not considered.
C. Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5–10.0	
D. Large inflow or high pressure	0.33	2.5–10.0	
E. Exceptionally high inflow or pressure at blasting, decaying with time	0.2–0.1	>10	
F. Exceptionally high inflow or pressure	0.1–0.05	>10	
6. Stress Reduction Factor	***SRF***		
a. Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated			1. Reduce these values of SRF by 25%–50%, but only if the relevant shear zone influences do not intersect the excavation.
A. Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth)	10.0		2. For strongly anisotropic virgin stress field (if measured): when $5 \le \sigma_1/\sigma_3 \le 10$, reduce σ_c to 0.8 σ_c and σ_t to 0.8 σ_t. When $\sigma_1/\sigma_3 > 10$, reduce σ_c and σ_t to 0.6 σ_c and 0.6 σ_t, where σ_c = unconfined compressive strength, and σ_t = tensile strength (point load), and σ_1 and σ_3 are the major and minor principal stresses.
B. Single weakness zones containing clay, or chemically disintegrated rock (excavation depth <50 m)	5.0		
C. Single weakness zones containing clay, or chemically disintegrated rock (excavation depth >50 m)	2.5		3. Few case records available where depth of crown below surface is less than span width. Suggest SRF increase from 2.5 to 5 for such cases (see H).
D. Multiple shear zones in competent rock (clay free) loose surrounding rock (any depth)	7.5		
E. Single shear zone in competent rock (clay free) (depth of excavation <50 m)	5.0		
F. Single shear zone in competent rock (clay free) (depth of excavation >50 m)	2.5		
G. Loose open joints, heavily jointed or "sugar cube" (any depth)	5.0		
b. Competent rock, rock stress problems	σ_c/σ_1	σ_t/σ_1	***SRF***
H. Low stress, near surface	>200	>13	2.5
I. Medium stress	200–10	13–0.66	1.0
J. High stress, very tight structure (usually favorable to stability, may be unfavorable to wall stability)	10–5	0.66–0.33	0.5–2
K. Mild rock burst (massive rock)	5–2.5	0.33–0.16	5–10
L. Heavy rock burst (massive rock)	<2.5	<0.16	10–20
c. Squeezing rock, plastic flow of incompetent rock under influence of high rock pressure			
M. Mild squeezing rock pressure			5–10
N. Heavy squeezing rock pressure			10–20

(continues)

Table 8.4-7 Q-system parameter values (continued)

Description	Value	Notes
d. Swelling rock, chemical swelling activity depending on presence of water	***SRF***	
O. Mild swelling rock pressure	5–10	
P. Heavy swelling rock pressure	10–15	

Source: Hoek et al. 1998.

Notes: When making estimates of the rock mass quality (Q), the following guidelines should be followed in addition to the notes listed in the tables:

1. When borehole core is unavailable, RQD can be estimated from the number of joints per unit volume, in which the number of joints per meter for each joint set are added. A simple relationship can be used to convert this number to RQD for the case of clay-free rock masses: RQD = 115 – 3.3J_V (approx.), where J_V = total number of joints per cubic meter (0 < RQD < 100 for 35 > J_V > 4.5).
2. The parameter J_n, representing the number of joint sets, will often be affected by foliation, schistosity, slaty cleavage or bedding, etc. If strongly developed, these parallel "joints" should obviously be counted as a complete joint set. However, if there are few "joints" visible, or if only occasional breaks in the core are due to these features, then it will be more appropriate to count them as "random" joints when evaluating J_n.
3. The parameters J_r and J_a (representing shear strength) should be relevant to the weakest significant joint set or clay-filled discontinuity in the given zone. However, if the joint set or discontinuity with the minimum value of J_r/J_a is favorably oriented for stability, then a second, less favorably oriented joint set or discontinuity may sometimes be more significant, and its higher value of J_r/J_a should be used when evaluating Q. The value of J_r/J_a should in fact relate to the surface most likely to allow failure to initiate.
4. When a rock mass contains clay, the factor SRF appropriate to loosening loads should be evaluated. In such cases the strength of the intact rock is of little interest. However, when jointing is minimal and clay is completely absent, the strength of the intact rock may become the weakest link, and the stability will then depend on the ratio rock-stress/rock-strength. A strongly anisotropic stress field is unfavorable for stability and is roughly accounted for as in the second note in the table for stress reduction factor evaluation.
5. The compressive and tensile strengths (σ_c and σ_t) of the intact rock should be evaluated in the saturated condition if this is appropriate to the present and future in-situ conditions. A very conservative estimate of the strength should be made for those rocks that deteriorate when exposed to moist or saturated conditions.

reduced to one factor referred to as the surface condition. The various plots suggest that a precision of ±2.5 points may be the best possible (this is assessed to be highly optimistic), a precision of ±5 is likely to be the best, and a precision of ±10 is a more typical result. For RMR values in excess of 18 and 23, respectively, the GSI is numerically equal to the RMR value if the latter is calculated with the 1976 version or equal to the 1989 version less 5 points. When lower values of the GSI are indicated, the GSI can be estimated from Q' as follows:

$$\text{GSI} = 9\ \text{Ln}Q' + 44$$

The GSI is input to the generalized Hoek–Brown criterion, whereby the values of the m, s, and a parameters are given by the following equations, respectively:

$$m_b = m_i e^{\left(\text{GSI} - \frac{100}{28} - 14D\right)}$$

$$s = e^{\left(\text{GSI} - \frac{100}{9} - 3D\right)}$$

$$a = 0.5 + \left(e^{-\frac{\text{GSI}}{15}} - e^{-\frac{20}{3}}\right)/6$$

where

m_b = rock mass
m_i = intact laboratory-scale rock
D = disturbance factor, ranging from 0.0 for mechanical excavation to 1.0 for large-scale surface blasts

An indication of the sensitivity of the key strength and deformation parameters to the GSI value is given in Figure 8.4-9. The cohesion and tensile and unconfined compressive strengths drop rapidly with GSIs between 100 and 70. Depending on the GSI value, the precision of the estimates of the various parameters is in the range of 10% to 50%.

A number of cautions apply (Marinos et al. 2005). Firstly, there is the question of scale and the applicability of the mass criterion as discussed earlier. Secondly, the method gives a non-zero tensile strength that may not be justifiable in the case of discontinuous rock masses. Thirdly, the surface condition is independent of the UCS of the rock, which is counter to the JRC concepts for shear strength. It is possible that this is implicit in some of the other cross plots, for example, the one for flysch that is included in RocLab (a software program from Rocscience, Inc., for determining rock mass strength parameters based on the generalized Hoek–Brown failure criteria). Flysch is a sedimentary rock deposited in the marine environment; it is possibly a "soft" rock. It may be necessary to reduce the GSI by about 15 units if it is being applied to soft rock. And finally, there are concerns about the disturbance factor. The depth of blasting damage may be in the order of a few meters for most blasts. At this scale, which is that of a roadway or a bench, the Hoek–Brown criterion probably should not be applied.

Deformation Parameters

A number of analyses require values for the deformation modulus of rock masses as well as their strength.

Before the GSI system, the rock mass modulus (Erm) was determined directly from the RMR without reference to values measured in the laboratory. Such relationships could produce a modulus value that exceeds that measured in the laboratory for rocks with UCS values less than around 100 MPa. This problem is perhaps another reflection of the RMR system being applicable to hard rocks and not necessarily to soft rocks.

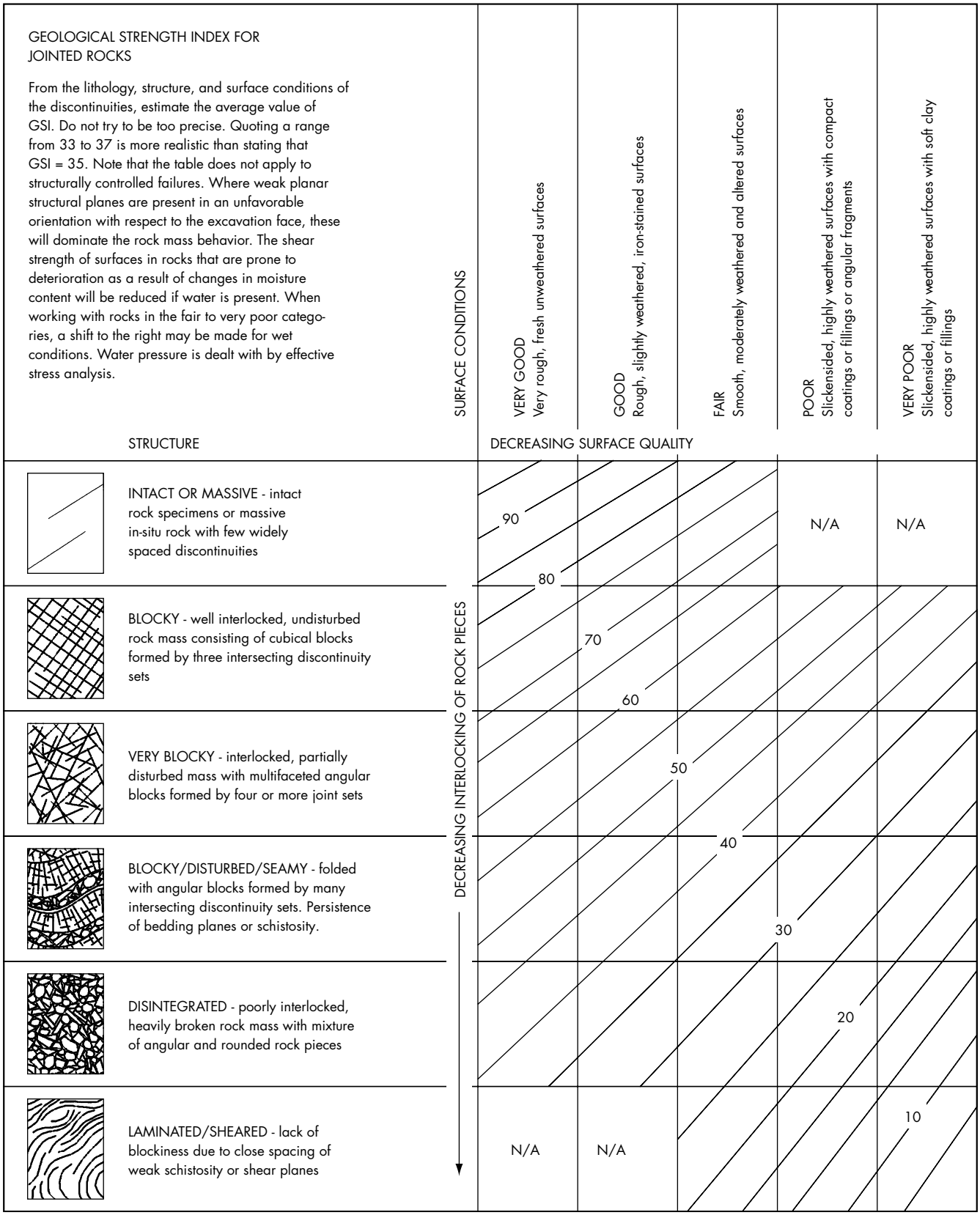
The GSI provides a system for estimating the rock mass modulus that is based on factoring down the laboratory value (Hoek and Diederichs 2006), as in the following equation:

$$\text{Erm} = \text{Ei}\left(0.02 + \frac{1 - D/2}{1 + \text{e}^{((60 + 15D - GSI)/11)}}\right)$$

There is no guidance on the Poisson's ratio for rock masses.

Interaction with Water

There are both chemical and physical deteriorations of rock mass strength with the presence of water. The presence of water, especially when combined with oxygen introduced by



Source: Brady and Brown 2004, reprinted with permission from Springer Science and Business Media.

Figure 8.4-8 Estimating the geological strength index

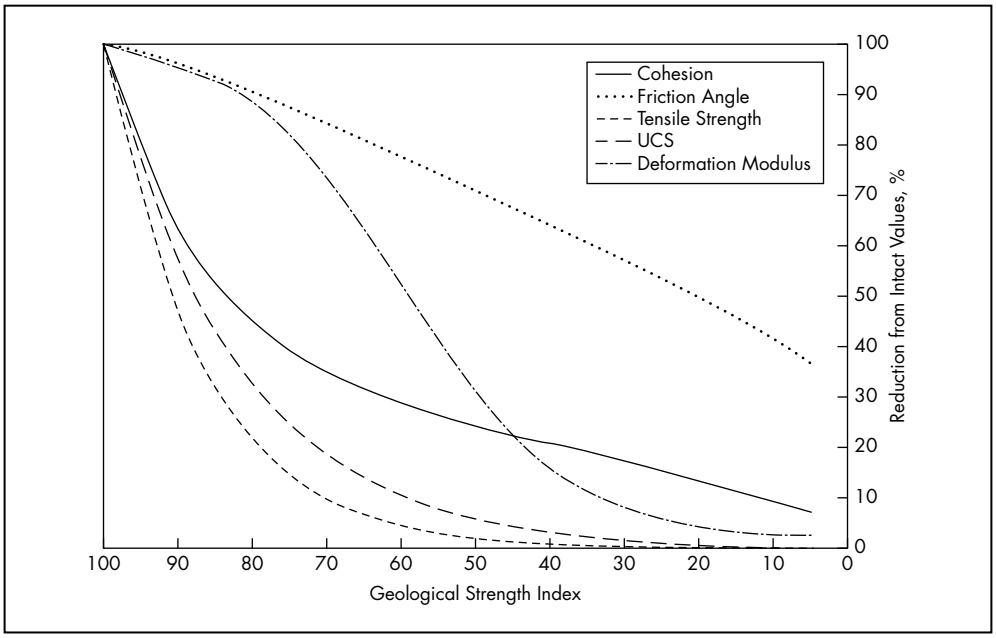


Figure 8.4-9 Sensitivity of rock mass parameters to changes in the GSI value (m_i = 10, D = 0.0)

exposure to the atmosphere, can lead to major changes in rock strength—and usually strength reductions.

Water pressures in the immediate vicinity of mine excavations are often (but not always) reduced to zero, either because free-flowing water is a nuisance underground or because drainage is critical to slope stability in the surface mines. Because of this, it is conventional in mining rock mechanics to consider stresses in terms of total (applied) stresses.

There are some applications where the principle of effective stress needs to be considered. Effective stress equals the total stress minus the pore pressure. In the context of shear along discontinuities, the pore pressure can reduce the normal stress, and hence, it can reduce the shear resistance; this is important behavior in rock slope stability. In addition, low modulus rocks that are also aquifers may undergo significant deformations during depressurization, such that the stress field is altered immediately ahead of the excavation compared to that of the far-field condition. This effect has been identified in porous sandstones and possibly in coal (Seedsman 2004).

STRESSES

Rock and rock masses deform and collapse in response to self-weight and applied stresses. Self-weight is the stress driver for planar and wedge failures and is ultimately the driver for most collapse modes. There are some closed-form solutions for the redistribution of applied stress around simple excavations. Since the 1980s and building on the availability of personal computers, there have been huge advances in the scope and capability of computational methods to study the distribution of stresses around excavations in both two and three dimensions, with the important proviso that the material can be considered to be linear elastic. Boundary element, finite element, distinct element, and finite difference are merely different computational methods to solve the same complex equations, and each has strengths and weaknesses.

In contrast to civil excavations, mining ventures seek to operate at the limit of elastic behavior, if not beyond, so long as the workplace is safe and the business risks are acceptable. Because of the inability to adequately characterize the post-peak behavior of rock, many mining problems are beyond the capacity of even the most complex numerical codes.

Computational methods require knowledge of the stress field that was acting prior to mining. There are now several tools to measure or estimate this stress field. The stress field that is measured is itself a function of the properties of the rock mass in the immediate vicinity to which it is measured. It is unlikely that the stress field can be adequately characterized prior to mining. Close observations of early excavations are still required, and it is essential that the interpretation of the stress field does not become distorted by the models that are applied. Most observations are of deformations, and this means that any calibration of the stress models includes assumptions not only on the deformation properties of the rock but also the stress redistribution about the excavation.

Nature of Stresses and Stress Equations

The rock in which mining is conducted is stressed by forces associated with the gravitational weight of the overlying rock, tectonic stresses, and residual stresses related to earlier changes in depth of burial and rock temperature. In mining rock mechanics it is conventional practice to assume that pore pressures are zero close to the excavation. Soil mechanics is the field of engineering that concentrates on the implications of non-zero pore pressures, and while this subdivision is convenient, there are still many mining circumstances, such as with very weak and extremely weak rock, where the concepts of effective stress need to be invoked.

Any stress field is 3-D, and it is conventional to resolve the field into the three principal planes and their associated principal stresses. Principal planes are those planes on which there are no shear stresses acting, and by definition the principal stresses act normal to these planes. The faces of excavations and the earth's surface are free of shear stresses, and hence they are locally principal planes.

The terminology for stresses includes the following:

- σ_1, σ_2, and σ_3 are the major, intermediate, and minor principal stresses, respectively.
- Compressive stresses are taken as positive.
- σ_{xx}, σ_{yy}, and σ_{zz} are the magnitude of the stresses acting parallel to a local coordinate system.
- σ_p is the stress acting normal to a plane at an angle α to the coordinate system.
- σ_{ll} and σ_{mm} are the stresses acting in a plane.

In two dimensions, the following relationships can be of use:

$$\sigma_{ll} = 0.5(\sigma_{xx} + \sigma_{yy}) + 0.5(\sigma_{xx} - \sigma_{yy})\cos 2\alpha + \sigma_{xy}\sin 2\alpha$$
$$\sigma_{mm} = 0.5(\sigma_{xx} + \sigma_{yy}) - 0.5(\sigma_{xx} - \sigma_{yy})\cos 2\alpha - \sigma_{xy}\sin 2\alpha$$
$$\sigma_{lm} = \sigma_{xy}\cos 2\alpha - 0.5(\sigma_{xx} - \sigma_{yy})\sin 2\alpha$$
$$\alpha = \tan^{-1}\left(\frac{\sigma_p - \sigma_{xx}}{\sigma_{xy}}\right)$$

When considering the performance of rock, the concepts of mean stress, σ_m, and deviatoric stress are also useful. These are defined by the following equations:

$$\text{mean stress} = (\sigma_1 + \sigma_3)/2$$
$$\text{deviatoric stress} = \sigma_1 - \sigma_3$$

In-Situ Stresses

The World Stress Map (Heidbach et al. 2008) can be used to obtain an initial assessment of the direction of the major principal stress in a horizontal direction. A better estimate may be obtained if the structural geology of the deposit is known—the local stress field is highly likely to be aligned with one of the axes normal to any dominant throughgoing discontinuity. In fact with knowledge of the fault orientations and the associated fiction angles, it is possible to constrain the estimate of the stress field. In layered materials close to the surface (e.g., within 200 m), one of the horizontal stress axes must be parallel to the strike of the layers. If it develops, borehole breakout is a reliable indicator of stress directions that can now be readily determined using downhole tools.

Many researchers have made compilations of stress magnitudes with depth. Until more site-specific information is available, it is reasonable to assume as a first approximation that the vertical stress can be calculated from the depth of cover, and the major principal stress has a magnitude of between 0.3 and 2.5 times this vertical stress to a depth of 1,000 m and a magnitude of between 0.3 and 1.0 times this vertical stress below 1,000 m. It is emphasized that the stress magnitudes are not the result of the so-called Poisson's ratio effect. If stresses were considered to be applied in plane strain, then higher stress magnitudes would be found in the stiffer units. The stress field in coal is different to the stress field in the adjacent rock—part of this may be due to the stiffness contrast, but there is also a component that may be related to stress redistributions associated with volume changes as the coal is depressurized by mining. Stress magnitudes near faults will be less than at distant locations.

It is apparent that predicting the stress field before mining is difficult. There are several techniques for measuring the stresses prior to mining, but they are relatively expensive; and because of the same geological controls that make the stress field variable, it is possible that they will not be representative of the whole mine. Until in-mine observations are possible, the approach should be to combine any site data with inference of stress directions from an interpretation of the structural geology and to consider a range of stress ratios between 0.3 and 2.5. It is important not to ignore the possibility of low ratios, such as, for example, in coal mining.

Elastic Redistributions

The ready availability of numerical codes makes the calculation of stresses around excavations in two and three dimensions a relatively trivial exercise, providing that the material can be considered linear elastic. As the graphics are improved, the interpretation and understanding of the stress field in three dimensions is improving.

For metaliferous mines and tunnels excavated by drill-and-blast or by roadheaders, the mining methods tend to produce rounded corners, and the simplifying assumptions of circular and ellipsoidal holes are valuable simplifications.

For circular and elliptical holes, the stresses on the boundary are as follows:

$$\sigma_{\text{side}} = p(1 - K + 2q)$$
$$\sigma_{\text{crown}} = p(K - 1 + 2K/q)$$

where

K = ratio of stresses
$q = W/H$ (Figure 8.4-10)

Importantly, in linear-elastic theory, the redistribution of stresses is a function of the stress ratio and the aspect ratio and independent of the deformation properties, and this applies to all excavation shapes. Large excavations are less stable as a result of the greater likelihood of encountering discontinuities.

For circular openings, K values less than 0.33 induce tensile stresses in the crown, and for a W/H value of 2, a K value of 0.5 induces tensile stress in the crown. Recalling that a rock mass with discontinuities should be assumed to have zero tensile strength, the importance of induced tensile stresses cannot be overestimated. The onset of tensile stress should be considered to be the precursor to possible collapse that may develop rapidly and without warning.

The two equations suggest that there is an ability to modify the induced stresses by modifying the aspect ratio of an excavation—the maximum boundary stress can be reduced by having an aspect ratio similar to the stress ratio. It is unlikely that a mining operation will have such flexibility.

In coal mines, continuous miners result in rectangular roadways and if the seam is flat lying, two of the stress axes will be co-planar with the bedding. Typical elastic stress patterns in isotropic materials are shown in Figure 8.4-11 for a 6- × 1.5-m rectangular opening and for a circular opening for a 2:1 stress ratio. The following three stress components are of interest:

1. **Vertical stress, σ_{yy}:** Elevated stress can cause failure in the rib.
2. **Horizontal stress, σ_{xx}:** Low values could allow mobilization of vertical joints.
3. **Deviatoric stress, $\sigma_1 - \sigma_3$:** High values could cause compressive failure of the rock.

For the case where $K = 2.0$, the vertical stress at the upper corner of the excavation is increased by about 14%, and at the midpoint of the sides the vertical stress is reduced to about 60% of the initial value. The horizontal stress is increased by about 60% at the corner and reduced to 80% at the centerline of the roof. The deviatoric stress increases fourfold at

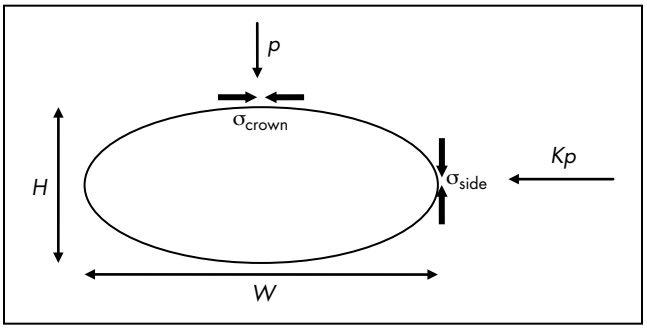


Figure 8.4-10 Definition of symbols used for the equations for stress around an elliptical hole

the corners and doubles at the centerline of the roof. Another characteristic is the tendency for the deviatoric stresses to arch over rectangular excavations, whereas they concentrate in the crown of a circular excavation.

Figure 8.4-11A and 8.4-11B shows the location of three points that are located 250 mm in from the boundary at the spring line and the crown and also 250 mm in the roof directly above the rib line. The relationships between the stress magnitudes at these three points with variations in the stress ratio and the aspect ratio of the roadway are shown in Figure 8.4-12. The following key observations can be made:

- The horizontal stresses in the roof become tensile at *K* values of about 0.6.
- The vertical stresses are higher and the horizontal stresses are lower for the excavation with the flatter aspect ratio.
- The deviatoric stresses for the rectangular roadways are between 2 to 3.5 times the vertical stress.
- The horizontal stresses at the crown of the circle are greater than at the roof centerline of the rectangular excavations.

For layered materials, the isotropic assumption can result in the anticipation of lower stresses than actually develop. For a condition where the value of independent shear modulus (G) is 1/100 of the isotropic value, the induced deviatoric stress within 0.5 m of the excavation boundary may be higher by approximately 50% for a rectangular excavation and 100% for a circular excavation.

The magnitude of the induced stresses decrease with the square of the distance from the excavation, so that at two diameters away the stress change is less than 5%. These simple elastic models readily demonstrate that large excavations can induce major changes in the stress field around adjacent smaller excavations. This is well recognized in metaliferous mining and is also a feature of longwall and pillar extraction in coal mines. The extraction of a cut-and-fill stope can result in temporary increases in stresses as the excavation approaches the level of access drives and then a relaxation of stresses as the extraction level rises (Figure 8.4-13). In a longwall, both the horizontal and vertical stresses increase at the face corner against unmined coal; in the tailgate adjacent to previously extracted coal, the horizontal stresses decrease and the vertical stresses increase.

An example of parametric analyses of the deviatoric stresses that can develop at the base of surface excavations is summarized in Figure 8.4-14. In these analyses, the stresses are generated by self-weight only, and the figure presents the

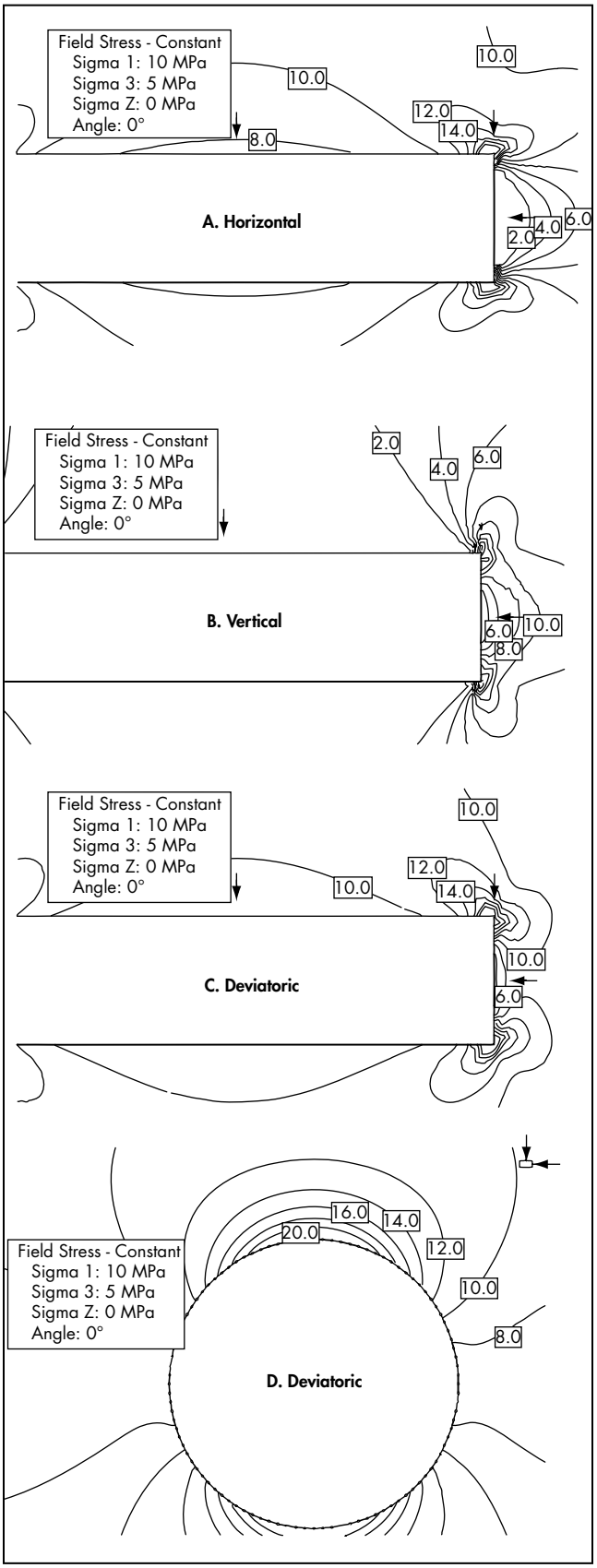


Figure 8.4-11 Distribution of stresses with rectangular roadways (A, B, and C) and with a circular excavation (D)

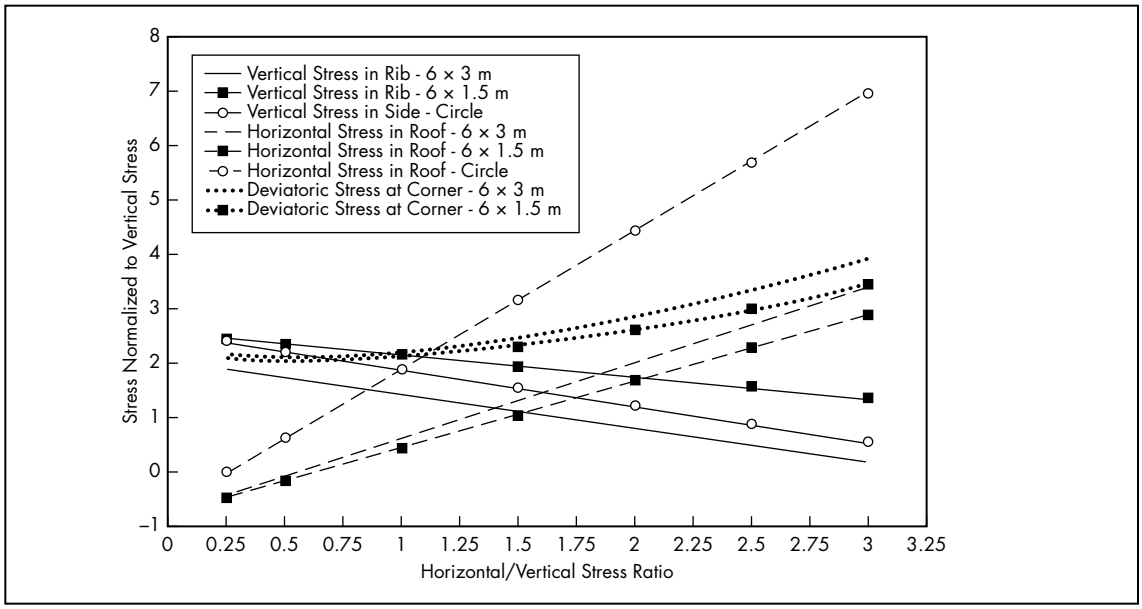


Figure 8.4-12 Variation in stress concentration factors with *K* ratio and aspect ratio of rectangular roadways

normalized stress magnitude along a line drawn vertically down at the toe. The normalization is based on the deviatoric stress that was present at the toe horizon prior to excavation. Depending on the face angle, the deviatoric stress can be concentrated two to three times, and the zone of influence is restricted to about 30% of the excavation height.

Nonelastic Behavior Around Excavations

Yielding

Yielding can result from either the peak strength of the rock being exceeded or shear along discontinuities being induced. Either way, movement is induced. In most mining situations, the applied stresses can be considered to be "stiff," such that they are redistributed elsewhere if the rock about the excavation moves beyond the elastic limit (Figure 8.4-15). The limit of elastic deformation is in the order of 10 mm. The implication is that after initial roadway formation, the stresses in the immediate roof should be assumed to be very low. This behavior, and the limitation of many computational methods to adequately manage it, is particularly important when roadways experience changes in stress conditions during their lifetime.

Stress relaxation can also occur in the roof if there is yielding or failure in the sides or floor of an excavation (Figure 8.4-15). In this case, the relaxation may be sufficient to induce the onset of tensile horizontal stresses and the mobilization of steep or vertically oriented joints. The amount of deformation that is required for relaxation of roof will be partly a function of the relaxation that has already developed in the roof. The hazard should be anticipated if the roadway defines a pillar that is designed to yield or if there is a low-strength floor horizon that may compress or fail.

Body Stresses

Recalling that rock masses are assemblages of blocks, if the blocks begin to rotate, they can interact and generate areas of high compressive stress, while in immediately adjacent areas there can be an opening of joints in response to localized tensile stress. An example of this is the voussoir beam mechanism (Sofianos and Kapenis 1998) that develops in a roof with no applied compressive stresses and yet failure can be via compression (Figure 8.4-16). Cantilevering behavior can also be anticipated in some layered materials. Both these two behaviors may generate compressive failure in the roof/rib corner at the same location as stress concentrations develop in linear-elastic codes. An implication of this is the need for an underground observer to be careful about inferring the nature of the applied stress field from simple observations—an analysis of likely stress magnitudes and rock strength is needed before the stress field can be inferred.

PRACTICAL IMPLEMENTATION OF ROCK MECHANICS

In many respects, this chapter highlights the many uncertainties that are inherent in the application of rock mechanics to mining. The general nature of these uncertainties is no different to those that face any engineering venture in soil or rock. Absolute guarantees of acceptable performance are not possible. The rock mechanics engineering profession has developed a number of strategies to manage the hazards and risks that are generated by the uncertainties of rock.

Observations and monitoring are recognized as an essential part of developing and applying empirical reasoning to any mine design. Recognizing the uncertainties of soil and rock, the observational method (Peck 1969) was introduced as a formalized process to manage contractual risks and to avoid latent condition claims. It requires full characterization of rock mass and prior assessment of likely and extreme conditions that may be encountered, such that pre-agreed responses are available. It is stressed that monitoring is not an alternative to fundamental design.

Because it requires detailed knowledge of conditions, the observational method should only be considered for specific well-constrained projects (both in time and geographically). It is obvious that the method cannot be applied to decisions of major capital expenditures prior to the operational phase. Bulk mining systems require very large daily production, and any unscheduled delays can have major cost impacts. It is possible

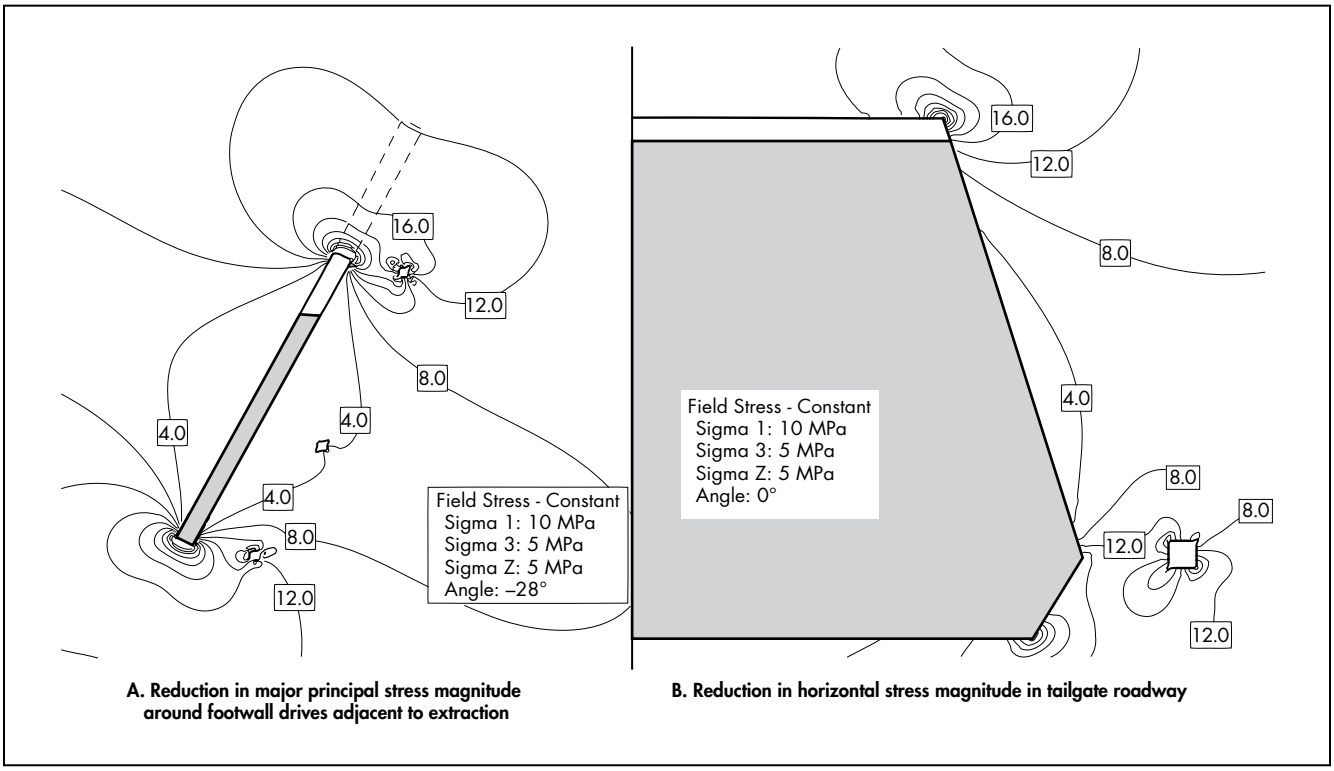


Figure 8.4-13 Large excavations result in large changes in stress regime at adjacent smaller excavation

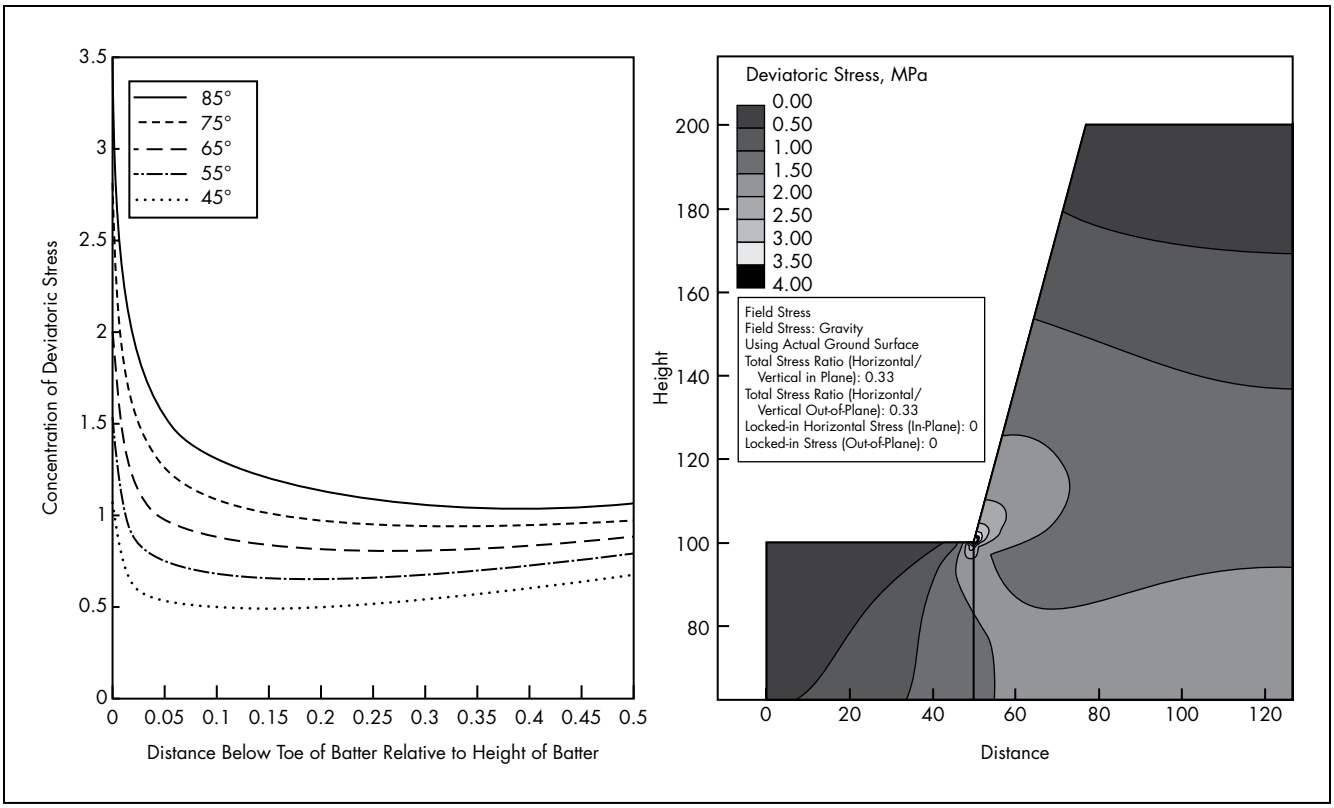
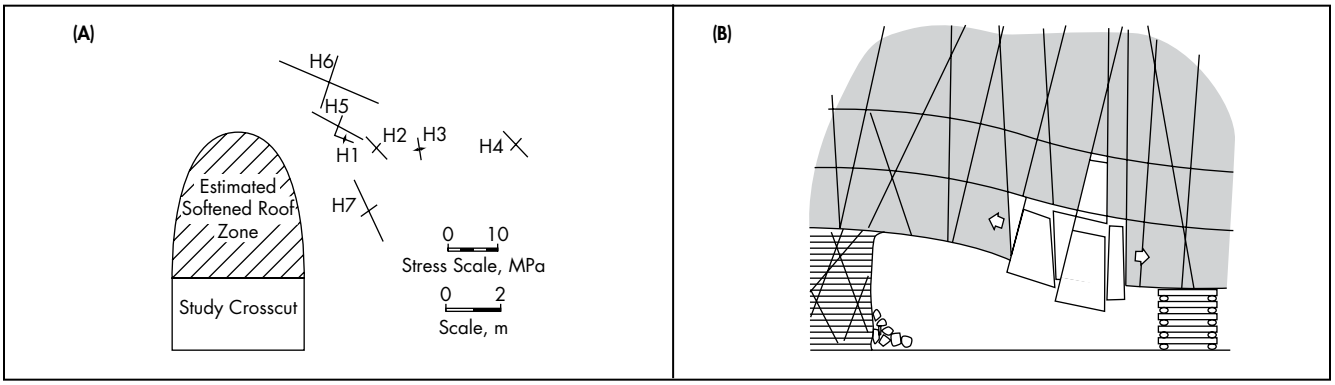


Figure 8.4-14 Elastic stresses around a surface excavation



Source: (A) Mark et al. 2007; (B) Diederichs and Kaiser 1999.

Figure 8.4-15 Redistribution of stresses if yield develops in roof or sides of roadway

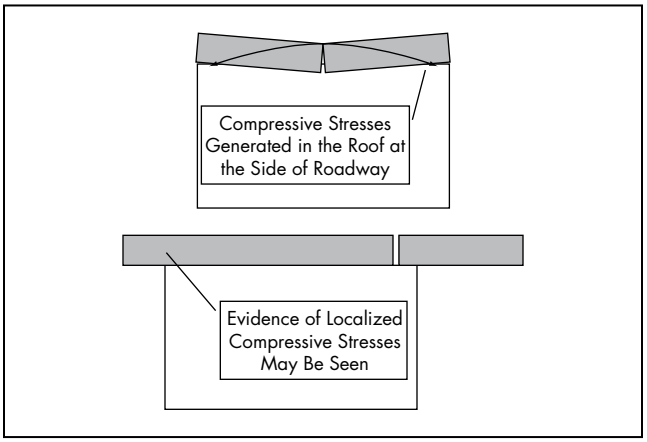


Figure 8.4-16 Roof deformations produce localized areas of compressive stress

that bulk-mining systems should not consider the observational method as they can afford/demand greater geotechnical certainty that can be delivered by a worst-case design.

Monitoring to calibrate a complex numerical model may mean that in the initial stage of operations there is no "acceptable" design in place. Careful consideration of the safety and business implications of this is needed. Monitoring may also be required for triggered action response plans (TARPs), but these plans are principally about personnel safety with a secondary application of stepping between agreed support strategies as in the observational method. TARPs are an essential part of modern mining as they provide the rationale to evacuate a place, but they are not sufficient to ensure an adequate mining outcome if there is inadequate time to stabilize the ground.

An overreliance on the observational method and/or in-mine monitoring to validate designs can lead to poor business performance. What is needed is a rigorous design process for rock (Bieniawski 1993). The following are key aspects of the design methodology (Figure 8.4-17):

- Safety of the work force is the key requirement.
- There may be practical constraints to the ideal solution.
- As much geological data as possible should be collected and interrogated.
- Numerous pathways are available (Figure 8.4-2), and the choice of pathway must be justified.
- Uncertainties in the geological input requires sensitivity studies on the outcomes.
- A number of options with different levels of business risk may be identified.
- The final recommendations may carry residual business risk, which needs to be highlighted.
- Monitoring to confirm design assumptions may be additional and to separate from that required for TARPs.

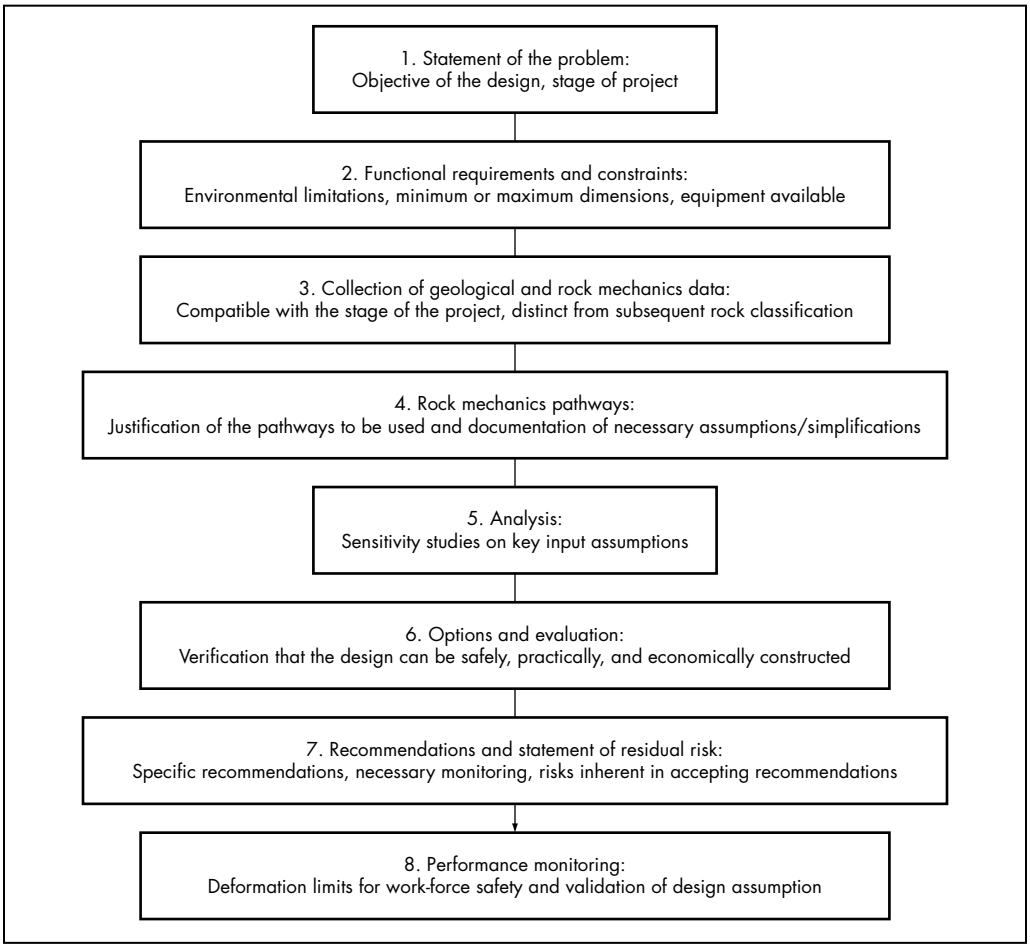
The general methodology can be applied at all stages of a mining project with different pathways more applicable at different stages (Figure 8.4-3). Even within a single pathway there may be differences in how the analyses are conducted and what behavior models are used. The underlying principle is to maintain flexibility and be prepared to apply judgment if the analyses do not appear sensible—the state of the art is still not perfect.

Overall Mine Design

Because modern mining requires large capital investments, it is not valid to advocate the observational method for overall mine design. There must be a high level of confidence that the capital can be recovered within the indicated time line. A more conservative mine plan may be required initially until the uncertainties are understood; once the capital is recovered, the focus can then move to maximizing long-term revenue.

There are adequate tools to design the general size and shape of excavations in a mine. Computational methods in two and three dimensions using equivalent continuum assumptions and the generalized Hoek–Brown criterion are appropriate for underground metal mines. The simpler block geometry of many underground coal mines and bench-scale designs in surface mines allow for the application of simple behavior models and limit equilibrium analyses. Specific analyses of major throughgoing geological structures need to be considered as early as possible.

The stress-field assumptions that may be necessary prior to mining need to recognize the uncertainties in their measurement and how well they represent the entire deposit. A check of the stress field against static equilibrium on pervasive weakness planes should be conducted. If possible, decisions on the orientation of the excavations should be deferred as long as possible so that in-mine knowledge can be obtained.



Source: Adapted from Bieniawski 1993.

Figure 8.4-17 Design methodology for rock engineering

Failure and Collapse Modes Around Excavations

The serviceability of individual benches in surface mines and access roadways in underground mines is just as important as the performance of the larger extraction volumes. This serviceability can be adversely impacted by failure of a component of the overall mine design (such as a pillar) or by the localized collapse of the side or roof of the roadway or an individual bench. A logical framework for the design of specific excavations (Step 5 in Figure 8.4-17) is presented in Figure 8.4-18 and is equally applicable to surface and underground excavations with slight changes in emphasis. The rock mass classification pathway using the RMR or Q systems is an alternative to the use of this flow chart.

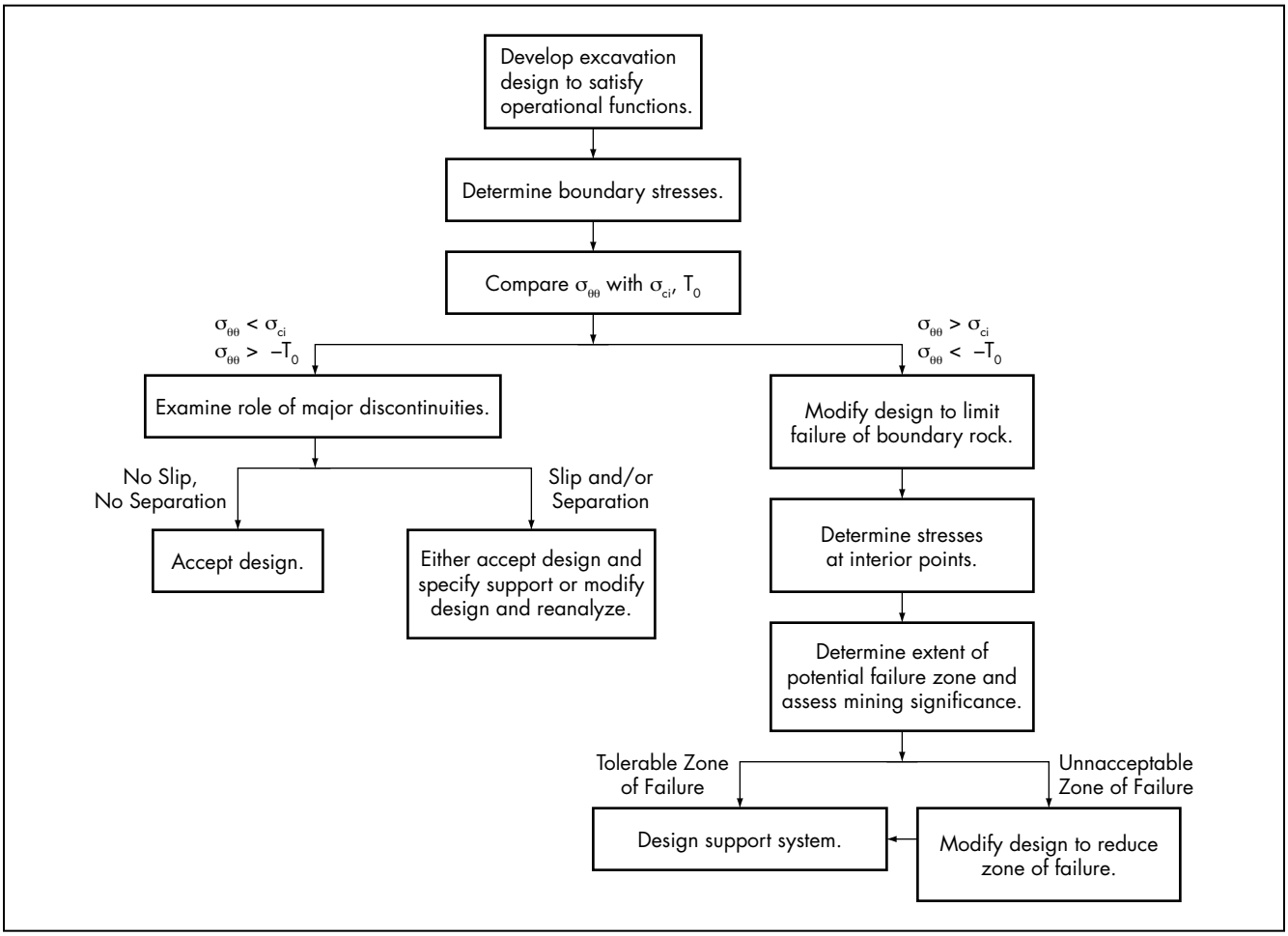
In underground excavations, the first step is to determine the boundary stresses. This is now a relatively trivial task with a range of software programs available for CHILE materials. If the lithologies are layered, there may be a need to consider the sensitivity of the stresses to the assumption of isotropic behavior. Determining the stresses at the excavation boundary induced by subsequent stress changes is still a challenge.

For surface mines, the assumption of body forces only is valid for routine analyses. Any consideration of the applied stresses in a surface mining application is best left to specialist geotechnical analyses in which the progressive excavation is modeled.

Roof

When selecting the discontinuity branch in Figure 8.4-18, there are readily available limit equilibrium methods to assess the stability of joint-bounded blocks and to specify support strategies. Accepting the concept that bedding partings define beams, there are also a number of simple elastic beam and voussoir beam models that can be applied. The proximity of the roof bolting to the mining face will determine if the determination of the forces of slip and separation need to be estimated with 2- or 3-D stress models. If bolting is conducted close to the advancing mining face (within a distance equal to the span), a 3-D model will give a better estimate of the forces developed after the support is installed.

There are many situations where the elastic boundary stress may be tensile. It is good practice to assume that the rock mass will have a tensile strength of zero, so the possibility of tensile elastic boundary stresses should be carefully reviewed as failure is indicated. Depending on the shape of the excavation, tensile conditions will be encountered if the K ratio is less than about 0.5 (i.e., when the vertical stresses are significantly higher than the horizontal stresses). Low K ratios should be anticipated in roadways close to the sides of valleys or surface excavations, when large excavations are in close proximity, and in coal roofs. Because there is no new rock breakage (just relaxation across joints), tensile failures



Source: Brady and Brown 2004, reprinted with permission from Springer Science and Business Media.
Figure 8.4-18 Logical framework for excavation design

in rock and coal masses can occur without audible or visual warning.

The onset of compressive failure can be readily assessed if the brittle failure criteria are considered, as the procedure is to compare the deviatoric stress with one strength parameter (UCS). A particularly useful step is to normalize the UCS to the estimated in-situ vertical stress based on the depth of cover and the average density. Depending on the *K* ratio and the shape of the excavation, values less than about 4 can be interpreted to indicate a greater likelihood of compressive failure at the excavation boundary.

Sides—Pit Walls and Underground

On the bench- and access-roadway scale, the discontinuity branch in Figure 8.4-18 leads to the analysis of planar slides, wedges, and topples. Rotational slips should also be considered. Numerous solutions and associated software are available.

The brittle-failure criterion is also applicable to the sides of underground excavations; in fact, the identification of this failure criterion was based on extensive work on the spalling of the sides of roadways in massive rock.

Floors

Adverse behavior of the floor should also be considered. Assuming the same deformation properties, the elastic deformation of the floor should be of a similar magnitude as for the roof—in the order of 10 mm. Because of the highly likely presence of water in mine floor, there may be some swelling of the contained minerals and some additional floor movements. Buckling of thin floor beams may also need to be considered. Floor heave is a relatively common occurrence, even in hard-rock mines under conditions of high stress.

In the underground coal sector, the possibility of a bearing capacity failure along thin layers of low-strength clayey materials needs to be considered. This is better considered in the context of soil mechanics.

CONCLUSIONS

Rock mechanics knowledge continues to evolve rapidly. The basic science is well established, and advances in computer analyses provide much greater insight into the complex stresses that develop around and between mining openings. Rock masses are complex, and their reaction to changing stress conditions is controlled by the inherent strength of the rock material, movement along large- or small-scale discontinuities, and the presence of water. The application of rock mechanics continues to drive improvements in safety and mine productivity, and even greater success will be achieved as the ability to characterize and quantify rock mass strength improves.

Currently, a wide range of design tools is available, ranging from rock mass classification schemes and closed-form solutions to 2- and 3-D stress codes. The practicing mining engineer should always carefully consider the design problem at hand and should particularly consider the scale of the opening relative to the spacing of the discontinuities in the rock mass. All the design tools have strengths and weaknesses; it is good practice to use multiple approaches. The most important part of the design problem is conceptualizing how the rock mass is likely to behave, as this will determine how it can be best analyzed. Completely different design approaches with different sets of assumptions may be needed to deal with a borehole, an excavation boundary, the interaction between drives or roadways, and the interaction between the mine and the surrounding rock mass.

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CHAPTER 8.5

Geotechnical Instrumentation

Erik Eberhardt and Doug Stead

INTRODUCTION

Geotechnical instrumentation is a fundamental component of surface and underground mining engineering. Its use extends from prefeasibility studies to mine closure. Its purpose is multifold, serving both investigative and monitoring functions that are in part a necessity to ensure the economic feasibility of the mine operations and in part due diligence to ensure safe operations.

Investigative functions include

- Providing an understanding of the ground conditions for prefeasibility and design purposes,
- Providing input values for design calculations, and
- Checking for changing ground conditions as the operations expand or as workings progress to greater depths.

Monitoring functions include

- Assessing and verifying the performance of the design;
- Calibrating models and constraining design calculations; and
- Providing a warning of a change in ground behavior, thus, enabling intervention to improve safety or to limit damage through a design change or remediation measure.

The required versatility in how instruments can be deployed (on surface, from boreholes, etc.) and what they are meant to measure (rock properties, ground movements, water pressures, etc.) has led to the development of a wide variety of devices. Instrument selection, however, is only one aspect of a comprehensive step-by-step engineering process that begins with defining the objectives of their use and ends with implementation of the data (Dunnicliff 1993). It is therefore important to ask the following series of questions prior to undertaking any mine instrumentation project: What are the objectives of the monitoring project? (If the objectives are not known, then the project should not proceed.) What parameters need to be measured, and how will these aid mine excavation, ground support measures, and assessing design performance? How might these parameters vary spatially? What are the risks due to variable or poor ground conditions? What are the magnitudes of expected movement or stress increase? What are the optimal locations for instrument installation?

Only after such a reasoning exercise has been undertaken should the project proceed. The following advice of Dunnicliff and Powderham (2001) is pertinent: "The purpose of geotechnical instrumentation is to assist with answering specific questions about ground/structure interaction. If there are no questions, there should be no instrumentation."

When choosing instruments for a particular project, the engineer must consider and balance the job-related requirements of the following:

- **Range:** Range is the maximum distance over which the measurement can be performed, with greater range usually being obtained at the expense of resolution.
- **Resolution:** The resolution is the smallest numerical change an instrument can measure.
- **Accuracy:** The degree of correctness with respect to the true value is the accuracy, and it is usually expressed as a plus-or-minus number or as a percentage.
- **Precision:** Precision is the repeatability of similar measurements with respect to a mean, usually reflected in the number of significant figures quoted for a value.
- **Conformance:** Conformance is whether the presence of the instrument affects the value being measured.
- **Robustness:** This is the ability of an instrument to function properly under harsh conditions to ensure that data accuracy and continuity are maintained.
- **Reliability:** Reliability is synonymous with confidence in the data; poor quality or inaccurate data can be misleading and is worse than no data.

Dunnicliff (1993) is an excellent text on geotechnical instrumentation for monitoring field performance. The text provides a very useful discussion on these issues. Sellers (2005) eloquently discusses the concept of accuracy and puts it into perspective in relation to instrument resolution, linearity, precision, and most importantly real-world issues such as economics, reliability, and the uncertainty and natural

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Table 8.5-1 Common borehole core orientation tools

Technique	Advantages	Disadvantages
Physical Techniques		
Weighted core barrel (clay imprint, spears, etc.)	Low cost; simple to use	Impression may require interpretation; unsuitable in boreholes inclined at shallow angles (<30°)
Ballmark	Simple to use; minimal drilling delays	Triggering mechanism may not operate in weak and fractured rock
Scribe system	Continuous scribing of core referenced to borehole orientation	Moderate to highly complex to use; difficult to interpret in weak and fractured rock
Ezy-Mark	Works in soft and broken rock; operates in most borehole orientations (uphole and downhole)	Requires an inclined borehole
Digital Techniques		
Reflex ACT electronic tool	Eliminates reliance on mechanical movement; highly accurate and consistent core orientations	Requires training in operation
Acoustic televiewer	Provides a continuous record of borehole wall (three-dimensional [3-D] virtual core); provides high accuracy and confidence in data; can be used in highly fractured rock	Requires a stable borehole; requires water or mud in borehole to operate
Optical televiewer	Provides a continuous record of borehole wall (3-D virtual core); provides high accuracy and confidence in data; can be used in highly fractured rock	Requires a stable borehole; requires air or clear water to operate

Source: Adapted from Read et al. 2009.

variability in most geotechnical projects. An important, but rarely discussed issue, is the true cost of geotechnical instrumentation, which according to McKenna (2006) should be considered with regard to the "life-cycle of the instrument—from its initial geotechnical design and procurement through drilling and installation, including reading and maintenance and data management and geotechnical analysis back in the office." This is in addition to decommissioning.

In addition to providing typical guideline costs associated with instrumentation, McKenna (2006) outlines good practices and emphasizes the care required in "all aspects of design, procurement, installation, reading, maintenance, data management, and quality assurance/quality control and analysis."

INSTRUMENTATION FOR INVESTIGATION

Geotechnical instrumentation plays an important role in the investigation of mine-site geology, geological structures (faults, jointing, etc.), rock mass properties, groundwater conditions, and in-situ stress fields. These are necessary inputs for carrying out prefeasibility studies and mine design, optimizing existing operations, and mitigating uncertainty in the mine design (but unfortunately not eliminating it). It must be recognized that rock and soil are natural earth materials, the products of many geological processes and complex interactions, and as such they are inherently variable.

Data should therefore be measured and recorded in systematic ways using standardized procedures. Furthermore, specific data may be required to make a decision on a particular aspect of mine design. Much time and effort can be wasted by collecting data that may be irrelevant or inadequate. Accordingly, the type and quality of the geotechnical data required will vary as a mine project matures. Parallels can be drawn with the Joint Ore Reserves Committee's (JORC's) code for reporting ore reserves—*inferred, probable, and proven* reserves (AusIMM 2004).

As a mining project moves from prefeasibility through to detailed mine design, the amount of data collected will increase as efforts are made to minimize uncertainty and reduce risk. The nature of the data will also become more specialized and measurement will transition from surface boreholes to the development and production levels.

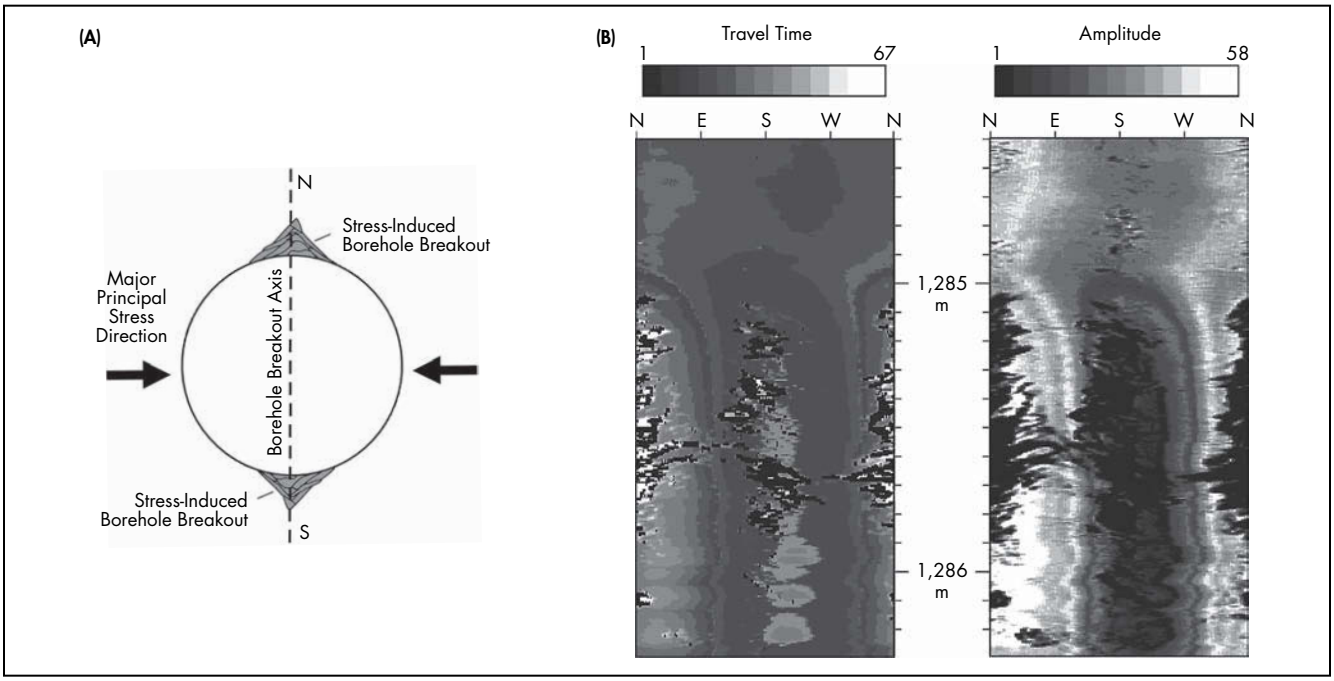
Ground Characterization: Borehole Techniques

Prefeasibility assessments and subsequent mine designs are often required to be completed prior to underground development. As a result, borehole information from drill cores and downhole geotechnical instrumentation often provide the only direct observation of the rock mass that hosts the mine in design. Core drilling is typically the most expensive phase of a mine geotechnical investigation, and therefore, every effort should be made to maximize the quality and quantity of the data collected.

Core Orientation

Discontinuities represent planes of weakness along which failure may occur. Their orientations and inclinations are key factors influencing mine design, as they may intersect each other to form potentially unstable wedges or blocks. In open-pit design, kinematic analyses using discontinuity data often controls how steep the bench, inter-ramp, and overall slope angles can be, all of which have a significant impact on the mining economics. Logging of oriented cores and borehole surveys provide a means to determine the dip and the dip direction of discontinuities intersecting the borehole. Read et al. (2009) summarizes several common core orientation tools. These may be divided into physical marking and digital imaging techniques, as shown in Table 8.5-1.

Physical marking typically involves using an orienting device during drilling to identify the top or bottom edge of the core before it is removed from the borehole. This is done by lowering the orientation device down the borehole between drill runs until it reaches the bottom, producing a mark on the top of the next piece of rock to be cored. This process is repeated so that the marks on the core from each drill run can be aligned using a reference line drawn from mark to mark. Techniques such as the downhole spear and Ezy-Mark (Table 8.5-1) are often favored for being inexpensive, quick,



Source: Adapted from Maybee et al. 2002.

Figure 8.5-1 (A) Stress-induced borehole breakout and its orientation relative to the direction of the major principal stress; (B) breakout observed in an ATV log at INCO's Totten mine. The north–south breakout is persistent for approximately 1 m along the length of the borehole.

simple, and robust. Alternatively, scribe systems (Table 8.5-1) use tungsten carbide knives differentially spaced around the core barrel to continuously mark the core as it enters the barrel. Scribe systems help to insure orientation of the core in formations with extensive fracturing.

Digital imaging techniques (e.g., acoustic and optical borehole televiewers) provide a direct, oriented, permanent, and unwrapped 360° continuous record of the borehole wall. They are used principally to map the orientation of discontinuities intersecting a borehole, detect thin beds, characterize lithology, and inspect well casings. They can also be used to provide indirect information on in-situ stress orientations through borehole breakout mapping (Figure 8.5-1A). This enhanced data output has meant that borehole televiewers are increasingly replacing conventional methods of core orientation.

Borehole Televiewers

Acoustic televiewers (ATVs) work by emitting an ultrasonic pulse-echo and measuring the return time and amplitude of the acoustic signal that is reflected back from the lithological and structural features present in the borehole wall. Optical televiewers (OTVs) illuminate the borehole wall and use a charge coupled device camera to record a direct image. They can operate in a wide range of borehole diameters and produce very high-resolution 24-bit RGB (red-green-blue) images with a vertical resolution of the order of 1 mm and a commercially available horizontal/circular resolution of up to 1,440 pixels. Both ATVs and OTVs are oriented using axial magnetometers and accelerometers contained within the instrument.

Some of the added advantages of ATVs and OTVs include the availability of sophisticated software for interpreting and displaying image logs. Manual or automatic highlighting of fractures and the determination of dip and azimuth from the logs can be undertaken. Data can be presented on image logs using tadpole and stick plots, stereographic projections, and frequency plots. Commercially available software allows for powerful interpretation of the data from multiple logging sources. Three-dimensional (3-D) virtual cores can be produced that are very useful in comparing logs with the actual core and investigating breakouts and borehole deformation. Figure 8.5-1B shows an example of a borehole breakout in a televiewer log from INCO's Totten mine in Canada, as reported by Maybee et al. (2002). The breakout is identified by the dark vertical bands located on opposite sides of the borehole image, indicating that the major principal stress is approximately east–west (i.e., 90° to the north–south breakout direction).

Borehole Geophysics

Nonintrusive and nondestructive, borehole geophysics provide valuable subsurface information for developing mine geological and geotechnical models. Surveys may be conducted such that both the source (seismic, radar, electric, etc.) and receiver are located together on a downhole probe or spaced through different configurations of borehole and surface source and receiver groupings (Figure 8.5-2). For effective application, it is important to ensure that all aspects of a geophysical survey are properly implemented. The International Society for Rock Mechanics (ISRM) has published suggested methods for borehole geophysics (Takahashi et al. 2006).

Geophysical logs. Wireline tools involve a variety of different sensor types, including mechanical, acoustic, electric and electromagnetic, and nuclear (Table 8.5-2). Some involve passive sensors (without excitation), and others involve active sources of excitation combined with detection sensors. Each

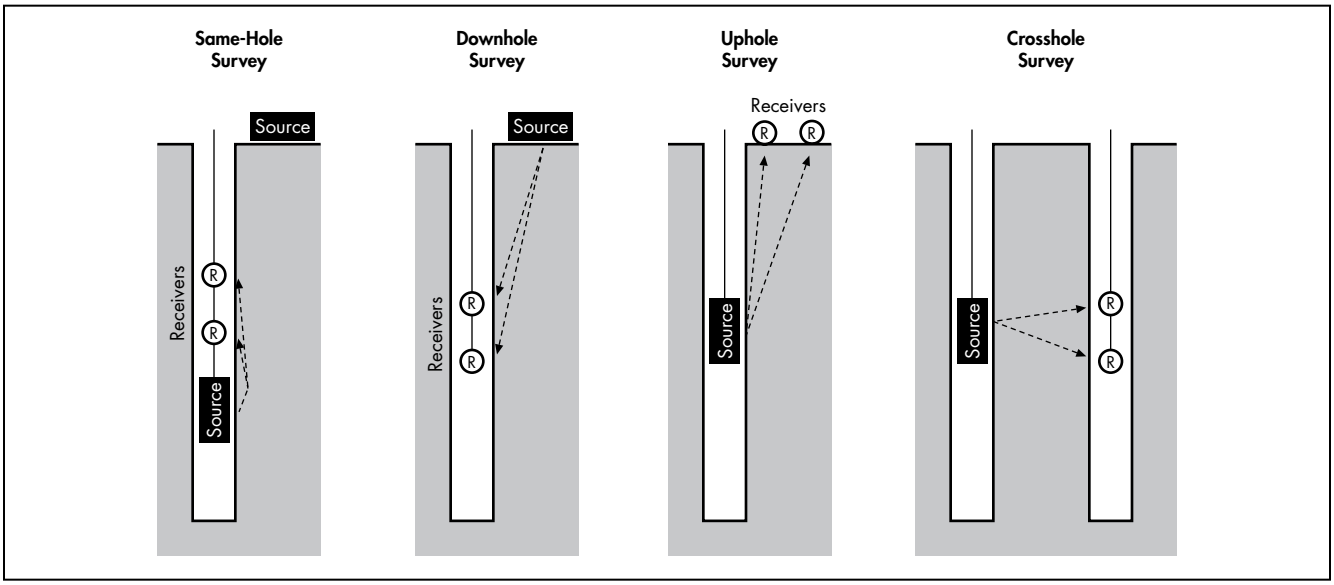


Figure 8.5-2 Example borehole and surface source and receiver configurations

Table 8.5-2 Common borehole geophysical tools used for geotechnical investigations

Source	Examples	Operation Principles and Applications
Geometry	Caliper	Measures variations in borehole diameter using spring-loaded mechanical arms that conform to borehole walls
Acoustic	Velocity; full waveform sonic; dipole shear imager	Measures P-wave or P- and S-wave velocity profiles along borehole; full waveform can be used to generate an attenuation log; provides information on rock composition, degree of fracturing/ weathering, and rock properties
Electric and electromagnetic	Resistivity; spontaneous potential (SP); induction; full bore formation micro-imager	Profiles the resistivity of geological formations/fluids along a borehole; SP uses passive electric logging to measure natural electric potentials along a borehole; induction logging utilizes electromagnetic induction to profile resistivity; provides information on geological contacts, lithology, permeable formations, groundwater level, degree of weathering, and variations in clay
Nuclear	Natural gamma; spectral gamma; gamma gamma; neutron	Includes active methods (gamma gamma, neutron), which measure the formation's response to induced radiation; includes passive methods (natural gamma) that measure the natural radiation of the formation; provides information on geological contacts, lithology, clay content, structural features, density, porosity, and water content

sensor type and tool has its own advantages for identifying rock lithologies or measuring certain rock or fluid properties, either directly or indirectly through empirical relationships. For example, full wave sonic tools measure P- and S-wave velocities, for which there are empirical relationships to calculate porosity, quartz content, and clay content. When combined with density data, Poisson's ratio, shear, bulk, and Young's modulus values can be calculated. The use of multiple tools and composite plots allow comparison and validation (e.g., Figure 8.5-3).

Vertical profiling. Borehole geophysics can be combined with surface surveys to derive more information regarding the subsurface conditions. These can then be used to resolve geological features, or they can be combined with density logs to calculate the various elastic moduli along the borehole profile. There are several variations in the measurement methods, depending on source and receiver configurations split between the borehole and surface (see Figure 8.5-2). Variations include downhole, uphole, and crosshole techniques. Downhole surveys involve seismic, radar, or electric sources placed on the surface with signals measured at regular intervals down the borehole using a string of receivers or a single receiver moved incrementally up and down the hole. Uphole surveys place the receivers on the surface and place the source in the borehole. In the case of a vertical seismic profile, the P-wave arrival times for each receiver location are combined to produce travel time versus depth curves for the complete hole.

Crosshole tomography. Borehole tomography involves the measurement of seismic/radar signals between two or more boreholes to derive an image of velocity/resistivity in the intervening ground. Data are collected using one borehole for the source and additional boreholes for a string of receivers. This results in a network of overlapping ray paths that can then be used to model the velocity/resistivity profile. The resulting image is termed a *tomogram* and enables identification of anomalous velocity/resistivity zones between boreholes, as well as imaging individual velocity/resistivity layers.

Ground Characterization: Remote Sensing

The use of digital remote sensing techniques for characterizing rock mass structure in underground and surface mine environments has increased significantly in recent years. A

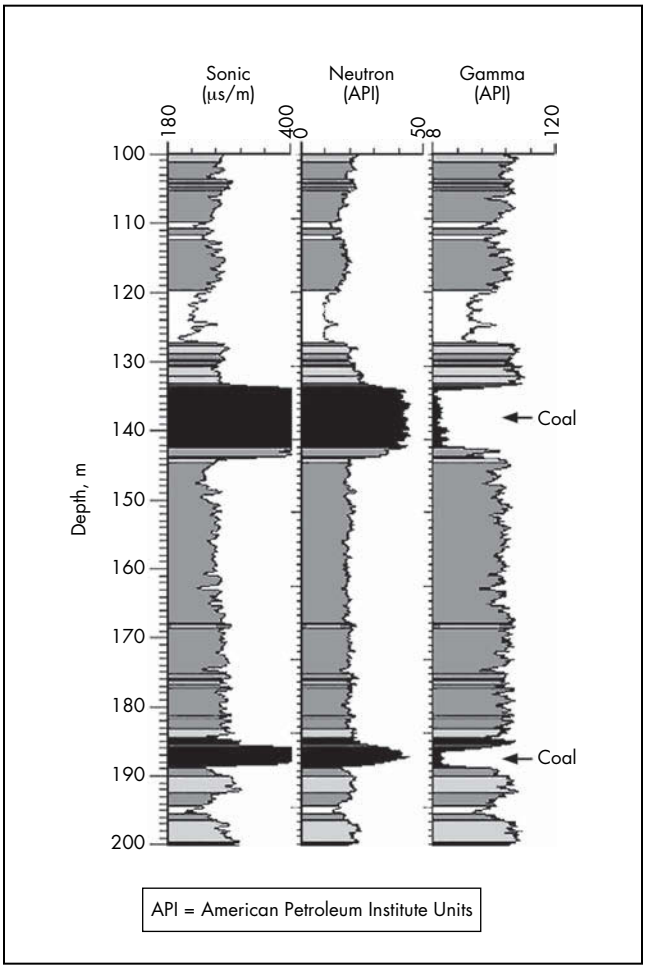


Figure 8.5-3 Example of composite geophysics log plot used to identify coal seams

major advantage of these methods is that they are able to provide data for remote and inaccessible areas, where safety concerns often preclude conventional mapping. Notwithstanding, Sturzenegger and Stead (2009a) and Lato et al. (2010) emphasize that these techniques suffer measurement bias (e.g., orientation, truncation, and censoring), which must be fully considered during processing, analysis, and interpretation of the data.

LiDAR

Light detection and ranging (LiDAR) is an optical remote sensing technique that measures properties of scattered light to determine the range of a distant target. Details on the theory of LiDAR measurement techniques are presented in Lichti et al. (2002) and in Lichti (2004). The product of a ground-based LiDAR laser scan is a 3-D point-cloud image of a pit slope or mine excavation wall, each point having x, y, and z coordinates and a pixel intensity value. The intensity may be in gray scale (Figure 8.5-4) or in color and may be draped on a photograph of the outcrop. A permanent 3-D digital image of the rock mass allows sections to be easily constructed wherever appropriate and imported into mine design and numerical modeling computer codes.

Using commercially available software (e.g., SplitFX, PolyWorks), orientation measurements of discontinuities may

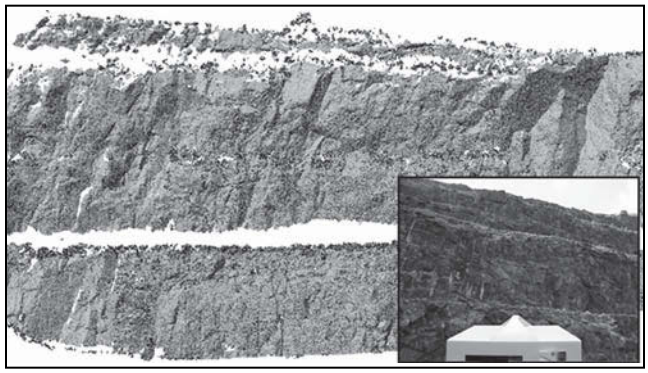


Figure 8.5-4 Three-dimensional point cloud from a LiDAR scan of open-pit benches at the Palabora mine in South Africa

be made from surfaces and traces in the point cloud, allowing for definition of discontinuity sets for future engineering analysis. The point cloud may also be used to provide data on discontinuity spatial location, persistence (trace length), and spacing (intensity). LiDAR has been used to characterize large-scale roughness and waviness; small-scale roughness still requires the use of conventional mapping surveys, as does the assessment of other parameters such as aperture and discontinuity infill. Comprehensive works on the use of LiDAR in rock mass characterization include Tonon and Kottenstette (2007), Kemeny and Turner (2008), Stead et al. (2009), and Sturzenegger and Stead (2009a). The use of LiDAR underground is less common than that on the surface, although excellent results can be obtained, as illustrated by Warneke et al. (2007).

Digital Photogrammetry

Ground-based digital photogrammetry is now routinely undertaken at large open-pit and underground mines. The use of this technique was pioneered by CSIRO Australia though the development of the Sirovision and Sirojoint software programs. Today, these codes, in addition to 3DM Analyst and 3G software programs, have found widespread use in the characterization of rock masses in open pits and underground mines (e.g., Birch 2006, 2008; Poropat and Elmouttie 2006). Using off-the-shelf digital cameras, stereo images of rock slopes and underground mine walls can be constructed and used to determine the orientation of discontinuities and derive joints sets in addition to obtaining similar geometric information as was discussed for ground-based LiDAR.

The advantages of stereo photogrammetry over LiDAR include the relative low cost for the hardware (a digital camera as opposed to a laser scanner) and the current ability to obtain 3-D stereo models from greater distances when using telephoto camera lenses (e.g., Sturzenegger et al. 2009; Sturzenegger and Stead 2009b). It is possible to form 3-D photo models of an open-pit mine at varying scales using a range of focal length lenses from 20 to 400 mm. This allows rock mass structures to be investigated at different image resolutions.

Figure 8.5-5 shows a 3-D photo model of the Palabora open pit using a 20-mm focal length lens and a model of pit wall benches using a 400-mm lens taken at a distance of more than 1.5 km. Figure 8.5-6 shows a photo model of a pillar using a 20-mm lens and an AdamTech underground lighting system. Both photogrammetry and LiDAR techniques complement each other and provide very similar data in terms of a point cloud and the derived rock mass parameters.

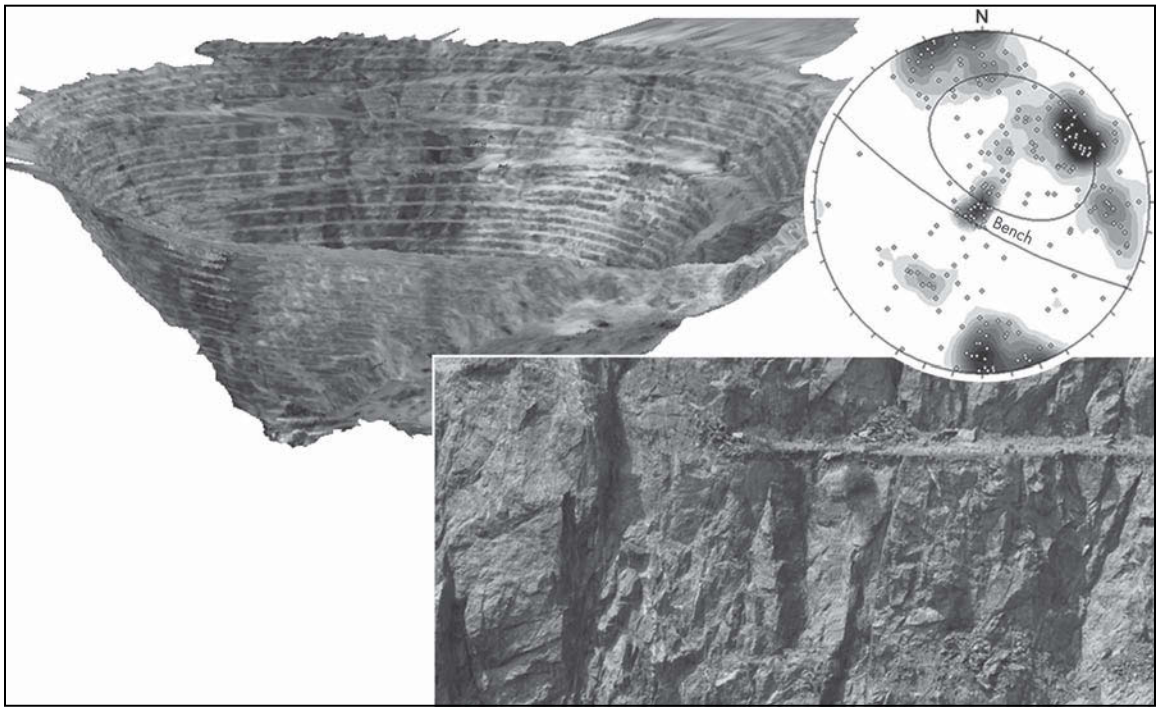


Figure 8.5-5 Three-dimensional photomodel of the Palabora open-pit mine in South Africa (20-mm lens), superimposed with a bench-scale photomodel (400-mm lens) and derived stereonet of discontinuity orientations

Surface Geophysics

Three-dimensional surveys (seismic, radar, etc.) can provide detailed information on geological contacts, faults, and other geological features important to mine design. Advantages over conventional two-dimensional (2-D) surveys include high-density data coverage, improved spatial delimitation of geologic and stratigraphic features, and the ability to visualize geologic, stratigraphic, and structural features in ways not possible with 2-D profiles.

The strength of 3-D surveys lies in their ability to establish continuity across subtle geological features. For example, Larroque et al. (2002) discuss how high-resolution 3-D seismic data were used to detect thin-layered platinum ore bodies at the Karee mine in South Africa, together with faults and other geological features that disrupted their continuity. These disturbances were of prime importance for implementing galleries, optimizing reef extraction, and positioning the shafts. Eso et al. (2006) demonstrated the application of a 3-D electrical resistivity survey to image a water-infiltrated void ahead of a mine drift in an underground potash mine. The detection of such voids is of prime importance in these mines to avoid flooding.

Ground-penetrating radar (GPR) is another surface geophysics technique that has been successfully applied in the mining industry (Pittman et al. 1982). The technique involves pulling a high-frequency radio transmitting and receiving antenna over the ground surface and recording any variations in the reflected return signal. GPR is used to map near-surface geologic conditions, including faulting and old workings that pose a hazard to miners, locating water tables, and detecting contaminant plumes. GPR can also be applied underground. White et al. (1999) provide several examples of routine use of GPR in South African mining, including ore-body delineation, mapping of faults and intrusives, and delineation of roof discontinuities.

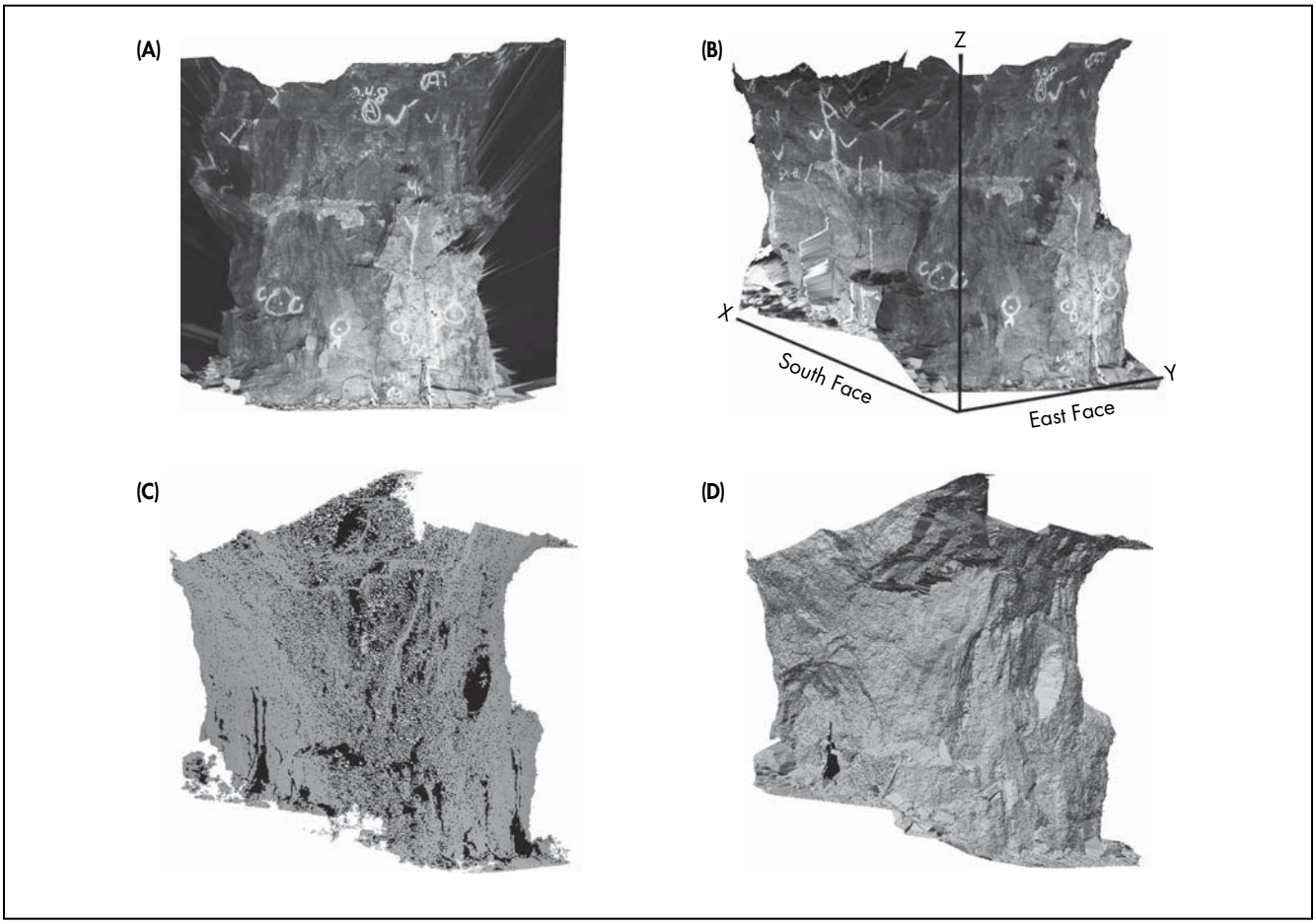
Groundwater Characterization

The determination of the water level and water pressures is an extremely important input for mine design, particularly for open-pit slopes. Pore water pressures can critically influence the stability of rock, waste rock, and soil/fill slopes. Measurement of pore water pressures is also important for monitoring the effectiveness of slope dewatering/depressurization schemes and the investigation of seepage and groundwater movements.

Piezometers

Piezometers are devices used to monitor pore and joint water pressures in boreholes. The most commonly used device is an electrical water level sensing probe that is used in combination with uncased boreholes (observation wells) to determine the depth to the water table by means of lowering the probe down the borehole. When the probe comes in contact with the water, an electrical circuit is completed, and the device makes an audible noise. Although this technique is quick and inexpensive and provides useful data in the initial stages of a project, it may be unreliable, especially in the presence of perched water tables, vertical groundwater gradients, and artesian conditions.

Open standpipe piezometers (Casagrande piezometers) involve cased boreholes, perforated at the depth of interest, in combination with a polyvinyl chloride (PVC) standpipe with a sealed-off porous filter element attached at the end. A sand filter zone is placed in the annulus around the filter tip up to the top of the filter zone with the remaining borehole backfilled with a bentonite grout to prevent any flow of water into the filter from other horizons. An electrical sensing probe can then be used to measure the water level corresponding to the groundwater pressure for the monitored interval. Combined standpipe/inclinometer installations are possible and are commonly used at surface mining operations.



Adapted from Styles et al. 2010.

Figure 8.5-6 (A, B) Three-dimensional photomodel of a pillar using photos of the four faces, and the corresponding (C) triangular mesh and (D) point-cloud representations developed from the photomodel

Although more complex, piezometers are also commercially available that provide a direct, reliable, and accurate reading of the pore water pressure at a specific depth in a borehole. They have rapid response times and the ability to be automatically logged. In rock masses, the correct location of these devices with respect to permeable fracture zones is very important. The principal differences between the various types of piezometers include the following:

- Single-point or multipoint measurements
- Vibrating wire, pneumatic, fiber-optic, strain gauge, or micro-electromechanical system- (MEMS-) based sensors (see the "Instrumentation for Monitoring" section of this chapter for a general description of instrument sensor types)
- Conventional installation or push-in types for soft ground

There has recently been considerable discussion as to the preferred method of piezometer installation. Many authors recommend fully grouted installations that have excellent zone isolation and rapid response to pore pressure changes. The reader is referred to Dunnicliff (1993), McKenna (1995), Contreras et al. (2008), Weber (2009), and Read and Stacey (2009) for further information on piezometer completions.

Multipoint piezometers offer the advantage of monitoring pore water pressures at selected intervals along the borehole. Several commercial varieties exist that vary principally on the method of measurement and isolation. Strings of piezometers can be connected on a single cable and fully grouted within the borehole. Installations include those for conventional vertical boreholes as well as horizontal boreholes in open-pit slope walls as part of a horizontal drain drilling program (Read et al. 2009).

Another popular multipoint system is the Westbay MP system, which comprises a modular casing system that allows for a large number of monitoring zones to be established in a single borehole. Access to individual zones is provided through port couplings with a filter on the outside and a spring-loaded check valve on the inside. Seals between monitoring zones are provided by either grout backfill or hydraulically inflated packers. Wireline tools and instruments are then used to operate the monitoring system by accessing the ports. Measurement ports not only allow pore water pressures to be monitored at any number of monitoring zones, but they also provide locations for fluid sampling. Pumping ports also provide the capability to undertake hydraulic-conductivity testing.

Table 8.5-3 In-situ stress measurement methods and key issues related to their applicability

Method	Advantages	Limitations	Suitability
Overcoring	Most developed technique in both theory and practice; three-dimensional (3-D)	Scattering due to small rock volume tested; requires drill rig	Measurement depths to 1,000 m
Doorstopper	Works in jointed and high-stressed rocks	Only two-dimensional (2-D); requires drill rig	For weak or high stressed rocks
Undercoring	Simple measurements; low cost; can utilize existing underground excavation	Requires measured local stresses to be related to far-field in-situ stresses; rock may be disturbed	During excavation
Hydraulic fracturing	Can utilize existing boreholes; tests large rock volume; low scattering in the results; quick	Only 2-D; theoretical limitations in the evaluation of maximum horizontal stress (σ_H)	Shallow to deep measurements
Hydraulic testing of preexisting fractures	Can utilize existing boreholes; 3-D; can be applied when high stresses exist and overcoring and hydraulic fracturing fail	Time-consuming; requires existing fractures in the hole with varying strikes and dips	Where both overcoring and hydraulic fracturing fail
ASR, DSCA, and RACOS*	Usable for great depths	Complicated measurements on the micro-scale; sensitive to several factors	Estimation of stress state at great depth
Acoustic emissions (Kaiser effect)	Simple measurements	Relatively low reliability; requires further research	Rough estimations
Focal mechanisms	For great depths; existing information from earthquake occurrence	Information only from great depths	Seismically active areas
Core discing	Existing information obtained from borehole drilling	Only qualitative estimation	Estimation of stress at early stage
Borehole breakouts	Existing information obtained at an early stage; relatively quick	Restricted to information on orientation; theory needs to be further developed to infer stress magnitudes	Deep boreholes or around deep excavations
Back analysis	High certainty due to large rock volume	Theoretically, not a unique solution	During excavation
Geological indicators	Low cost; 2-D/3-D	Very rough estimation; low reliability	At early stage of project

Source: Adapted from Ljunggren et al. 2003.
*ASR = anelastic strain recovery; DSCA = differential strain curve analysis; RACOS = rock anisotropy characterization on samples.

Groundwater Flow Testing

The collection of data on groundwater flow is an important area of geotechnical instrumentation, particularly in large open-pit design where slope depressurization may be critical to slope stability. Beale (2009) and Read and Stacey (2009) provide detailed descriptions of the importance of developing an adequate hydrogeological model and the requirements of slope depressurization programs. The characterization of groundwater response in fractured rock masses involves the use of drill-stem injection testing, falling or rising head tests (slug tests), packer testing, and pumping tests. Read and Stacey (2009, Appendix 1) provide an excellent summary.

In-Situ Stress Measurement

In-situ stress measurement provides an important boundary condition and input for mine design, including the selection of mining method, assessment of short- and long-term performance of underground openings, design of rock support and ground improvement (e.g., grouting), and assessment of rock burst potential. Hudson et al. (2003) discuss strategies for rock stress estimation, and Amadei and Stephansson (1997) and Ljunggren et al. (2003) provide comprehensive reviews of in-situ stress measurement methods. These may be classified according to methods that involve direct measurements, for example, the rock mass's response to hydraulic fracturing or induced strains from overcoring and those that involve indirect observations (indicator methods). The second category includes borehole breakouts, core disking, acoustic emissions (Kaiser effect), strain recovery methods, earthquake focal mechanisms, geological observational methods, and statistical treatment of databases. The advantages and limitations of these techniques are summarized in Table 8.5-3.

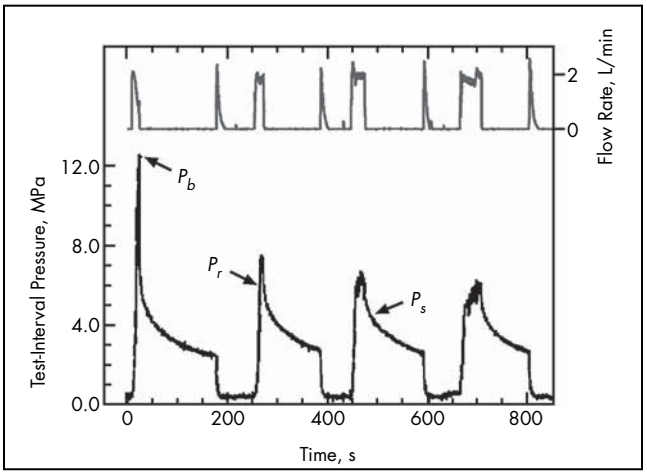
Overcoring

Overcoring is a stress-relief method that involves isolating a rock sample from the stress field that surrounds it and monitoring the strain response. As such, the measured stresses are not related to applied pressures but are inferred from strains generated by the unloading process.

Suggested methods for overcoring stress measurements are provided by Sjöberg et al. (2003). The method involves first drilling a large-diameter borehole followed by a smaller pilot hole in which a strain measuring device is inserted and fastened. The large-diameter hole is then resumed, relieving stresses and strains in the hollow rock cylinder that is formed. Changes in strain are recorded with the instrumented device as the overcoring proceeds past the plane of measurement. The in-situ stresses are calculated from the measured strains with knowledge of the elastic properties of the rock. Hence, overcoring requires the assumption of continuous, homogeneous, isotropic, and linear-elastic rock behavior. Sjöberg et al. (2003) noted that errors are introduced because these conditions are seldom encountered in rock masses. Even when seemingly ideal conditions apply, some scattering of the results always occurs.

Hydraulic Fracturing

Haimson and Cornet (2003) describe in detail the two main hydraulic methods used for rock stress estimation: hydraulic fracturing (HF) and hydraulic testing of preexisting fractures (HTPFs). For the HF method, a borehole interval devoid of natural fractures is sealed off and pressurized with water pumped under a constant flow rate until a fracture initiates in the rock. The following pressure measurements are then made: the water pressure at which the fracture occurred (the breakdown pressure), the subsequent pressure after pumping



Source: Adapted from Haimson and Cornet 2003.

Figure 8.5-7 Example of a hydraulic fracturing test showing the breakdown pressure (P_b), shut-in pressure (P_s), and reopening pressure (P_r)

is stopped and the fracture closes (the shut-in pressure), and the pressure required to later reopen the same fracture (the reopening pressure). These cycles may be repeated to provide redundant readings. A typical HF record is shown in Figure 8.5-7.

In a vertical borehole, the shut-in pressure, P_s, is assumed to be equal to the minimum horizontal stress, σ_h. The direction of σ_h is obtained directly from the orientation of the hydraulic fracture; vertical HFs propagate perpendicular to σ_h. The maximum horizontal stress, σ_H, is calculated based on the breakdown pressure, P_b. In this calculation, the breakdown pressure has to overcome the minimum horizontal principal stress (concentrated three times by the presence of the borehole) and the in-situ tensile strength of the rock, such that in the absence of pore fluid pressure the maximum horizontal stress, σ_H, is as follows:

$$\sigma_H = T + 3\sigma_h - P_b$$

The orientation of σ_H is taken to be perpendicular to the σ_h direction (i.e., the direction of fracture propagation). Other assumptions include that of homogeneous, isotropic, linear elastic behavior of the rock surrounding the borehole, and impermeability of the host rock so that pumped water has not significantly penetrated the rock and affected the stress distribution. Also, classical interpretation of an HF test is possible only if the borehole axis is parallel to one of the principal stresses and is contained in the induced fracture plane (Haimson and Cornet 2003).

The HTPF method follows a similar procedure of sealing off a borehole interval, but it involves reopening an existing fracture of known orientation. By using a low fluid-injection rate, the fluid pressure, which balances exactly the normal stress across the fracture, is measured. The method is then repeated for other nonparallel fractures of known orientation. By determining the normal stresses acting across several nonparallel fractures and knowing their orientation, a system of equations can be created to determine the six in-situ stress components without making any assumptions with regard to the orientation of the principal stresses.

Table 8.5-4 Systematic approach to planning a monitoring program

Step	Description
1	Define the project conditions.
2	Predict mechanisms that control behavior.
3	Define the geotechnical questions that need to be answered.
4	Define the purpose of the instrumentation.
5	Select the parameters to be monitored.
6	Predict the magnitudes of change.
7	Devise remedial action.
8	Assign tasks for design, construction, and operation phases.
9	Select the instruments.
10	Select instrument locations.
11	Plan recording of factors that may influence measured data.
12	Establish procedures for ensuring reading correctness.
13	List the specific purpose of each instrument.
14	Prepare the budget.
15	Write the instrument procurement specifications.
16	Plan the installation.
17	Plan regular calibration and maintenance.
18	Plan data collection, processing, presentation, interpretation, reporting, and implementation.
19	Write contractual arrangements for field instrumentation services.
20	Update the budget.

Source: Adapted from Dunnicliff 1993.

INSTRUMENTATION FOR MONITORING

Predictive monitoring systems are best implemented after a period of investigative monitoring. Monitoring information must be assessed in the context of the physical setting and the conclusions of the investigation phase. Planning of a monitoring program should be logical and comprehensive because the measurement problem may require a number of different instrument types collecting information across a range of varying scales. Furthermore, because of physical limitations and economic constraints, all parameters cannot be measured with equal ease and success. To assist with these challenges, Dunnicliff (1993) proposed a detailed systematic approach to planning a geotechnical instrumentation program, which is outlined in Table 8.5-4.

The monitoring system can be viewed as connecting a transducer to a data acquisition system via a communication link. The transducer serves to convert a physical change in the parameter being measured into a corresponding output signal, which can be read manually or automatically. Reliability of the instrument is paramount; this therefore requires some understanding of the transducer housed within the instrument and its sensitivity to the surrounding environment (e.g., temperature extremes, presence of water or high humidity, dust and dirt, exposure to shock or vibrations, and erratic power supplies). Several transducer types are common to the various instrument types described in this chapter. These are summarized in Table 8.5-5 for reference.

Surface Displacements

Geodetic

Geodetic monitoring provides a means for measuring the magnitude and rate of horizontal and vertical ground movements. Methods are well established and are often entirely adequate

Table 8.5-5 Transducer types commonly used in geotechnical instruments

Sensor Type	Operation Principles
Linear variable differential transformer (LVDT)	An LVDT consists of a movable magnetic core passing through one primary and two secondary coils. An excitation voltage is applied and a voltage is induced in each secondary coil. When the core moves off center, the output voltage increases linearly in magnitude. LVDTs are commonly used in instruments to measure displacements.
Vibrating wire	This involves a high tensile strength steel wire fixed at both ends and tensioned so that it is free to vibrate at its natural frequency. The wire is magnetically plucked by an electrical coil, and its frequency is measured. When one end moves relative to the other, the tension in the wire, and therefore the measured frequency, changes. Vibrating wire transducers are commonly used in pressure cells, piezometers, and deformation gauges.
Accelerometer	This consists of a damped mass suspended in a magnetic field; under the influence of external accelerations (or motion), the mass deflects from its neutral position and the deflection is measured. Accelerometers are commonly used in tiltmeters and inclinometers.
Fiber optics	Light is emitted into and confined to a glass fiber core and propagates along the length of the fiber. Any disturbance of the fiber alters the guided light, which can then be related to the magnitude of the disturbing influence. Fiber optics is finding increased use in piezometers and deformation monitoring instruments.
Micro-electro-mechanical system (MEMS)	A MEMS is a small, integrated device that combines electrical and mechanical components on a submicrometer to submillimeter scale. This allows for transducers (e.g., accelerometers) that are much smaller, more functional, lighter, more reliable, and produced for a fraction of the cost of conventional transducers.

for performance monitoring. Measurement accuracy (and reliability) is controlled by the characteristics of reference datums and monitoring points. Checks are required to make sure these datums are located on stable ground. Geodetic survey methods represent the most commonly relied on slope and displacement monitoring system at surface mine sites. The reader is referred to Dunnicliff (1993) for a detailed coverage of survey methods for monitoring displacement.

Recent advances that are of particular importance are the incorporation of robotic total stations (RTSs) into mine monitoring networks. Cook (2006) describes an RTS as a survey instrument combining a theodolite (with automatic target recognition) and an electronic distance measurement device that can be operated remotely. The RTS monitors point by sighting prisms and tracking them as movement occurs. After the RTS has "learned" the location of the prism, it returns to redetermine the location of that prism during successive monitoring cycles; angular and distance measurements are made and the new prism locations are calculated. Many hundreds of targets can be included in the survey network. In addition to installation and operational issues and the factors controlling the accuracy of the RTS network, Cook (2006) provides a useful practical list of the advantages and disadvantages of an RTS as compared to conventional survey methods. Under optimal conditions, an accuracy of ±0.6 mm at 60 m is reported as achievable.

Little (2006) documented automatic prism monitoring at the Potgietersrus Platinum's open-pit operation in South Africa, where prisms were located on a highwall at 50 m horizontal and 45 m vertical spacing. At this mine, Leica Geosystems GeoMoS system was used and the automatic prism-monitoring network was integrated with the use of laser scanners and slope-stability radar. Cahill and Lee (2006) describe the use of an automated Leica system using more than 400 prisms at the Harmony pit at the Leinster Nickel mine in Australia. The continuity of the prism data allowed for the long-term assessment of wall movements. The geo-referenced nature of the data allowed it to be used for assessing vector movements of the pit walls and failure mechanism determination.

Brown et al. (2007) describe combining Global Navigation Satellite System (GNSS) receivers with RTS instruments to provide a stable reference frame for total stations sited in an unstable environment. Kim et al. (2003) describe a similar example for an RTS system georeferenced using GNSS/GPS at the Highland Valley copper mine in Canada. Such integration provides a fully automated, accurate, efficient, and cost-effective means for monitoring points in a large open pit where no stable location can be found to place the instrument and control points.

Global Positioning Systems

GPS surveys utilize the U.S. network of space-based global navigation satellites to provide reliable positioning of a GPS receiver. The GPS receiver calculates its position by precisely timing the signals sent by multiple satellites, computing the distances to each satellite to determine its own position (through trilateration). When paired with an antenna, automatic monitoring of pit wall movements is possible with millimeter accuracy.

GPS offers several advantages over traditional geodetic surveying techniques. In general, GPS is more efficient, highly automatic, less labor intensive, and line-of-sight is not required between stations. However, in an open pit a limiting factor for large-scale use of GPS is that, for automatic monitoring of deformations, each monitored point needs to be equipped with a receiver and antenna. This makes GPS an expensive option for slope monitoring. In addition, issues of data communication, power supply, and system control are difficult to resolve when GPS instruments are distributed at isolated points.

Another limitation of GPS involves its use with steep pit slopes. Stewart et al. (2000) describe the use of GPS to monitor more than 60 points within the Palabora pit in South Africa, generally to an accuracy of 5 mm. However, the accuracy of monitoring points deeper in the pit is less because of the limited satellite window available (i.e., satellite visibility is obstructed). This limitation can be overcome by linking a GPS receiver with multiple antennas mounted at several monitoring points, or by using pseudolites, small transceivers that transmit a local, ground-based GPS-like signal (e.g., Bond et al. 2007).

Extensometers

This group also includes crackmeters, jointmeters, strainmeters, crack gauges, convergence gauges, distometers, and sliding micrometers. Extensometers are devices used to measure the changing distance between two points. Measurement points may be located on the surface to measure ground

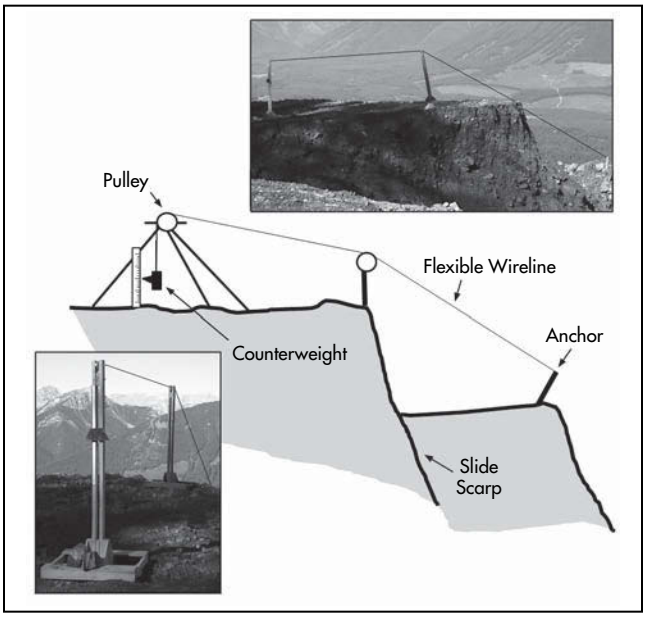


Figure 8.5-8 Schematic diagram (with inset photos) showing the use of a wireline extensometer to monitor waste dump movements at a British Columbia, Canada, coal mine

movements (e.g., spanning a tension crack to monitor its opening rate), or they may be located in a borehole to measure differential displacements along the borehole. Extensometers vary in type between those that involve manual measurements and those that are automated using vibrating wire electronics, differential transducers, or more recently fiber optics. Measurement accuracy and repeatability depend on the type of sensing device and the distance between the monitoring points. Typical accuracies range from submillimeter to millimeter over distances of less than a meter when using stiff sliding rods fixed between the monitoring points, and typical accuracies range from millimeter to centimeter over distances of meters or several tens of meters when extending a flexible tape or wire between the monitoring points.

In surface mining operations, wireline extensometers are frequently used to monitor pit slope and waste rock dump movements. These devices consist of a wire anchored in the unstable ground and tensioned across a pulley located on the stable ground behind the last tension crack using a counterweight (Figure 8.5-8). As the unstable portion of the ground moves away from the stable ground, the weight will move and the displacements can be recorded. These devices can be quickly positioned and easily moved, but care must be taken to minimize sag or thermal expansion/contraction in the wire, which can produce measurement errors. Rose and Hungr (2007) describe the use of wireline extensometer data at several mines in the United States to forecast the time of pit wall failure using an inverse velocity technique to interpret the data.

Tiltmeters

This group also includes clinometers and tilt sensors. Tiltmeters are devices used to monitor the change in inclination of a ground surface point. A detailed description is found in Dunnicliff (1993). The device consists of a gravity sensing transducer (servo-accelerometer, electrolytic tilt sensor, pendulum-actuated vibrating wire, MEMS) capable of measuring changes in inclination as small as one arc second. Tiltmeters may be uniaxial or biaxial, allowing measurement of tilt in two orthogonal directions. They are used to monitor slope movements where the landslide failure mode is expected to contain a rotational component. Advantages of using tiltmeters are their light weight, simple operation, and relatively low cost. Tiltmeters can be read manually or automated by connecting them to a data logger. They may be combined with inclinometers and extensometers in what have been termed as *integrated pit slope monitoring systems*.

Tiltmeters can be used to monitor pit-slope movements and ground subsidence above underground mines, particularly longwall and block cave operations. O'Connor et al. (2001) provides an example of a tiltmeter array used to measure the subsidence profile over an advancing longwall coal mine face. The array consisted of a series of tiltmeters connected to a central data acquisition system for automated real-time monitoring, with alarm thresholds set to warn of excessive ground deformations where the longwall panels crossed under an interstate highway.

Borehole Displacements

Probe Inclinometers

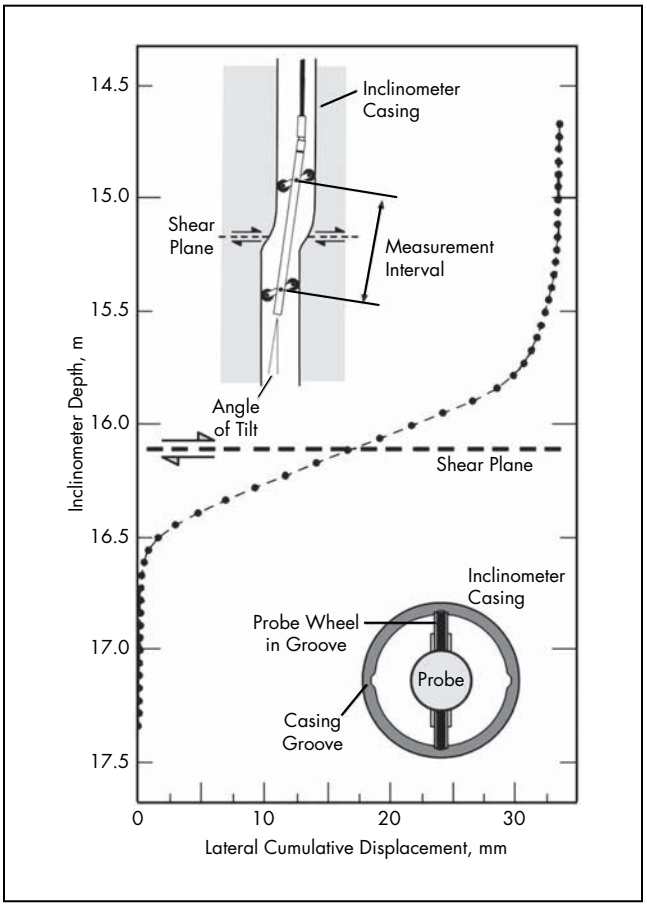
Probe inclinometers also include transverse deformation gauges and slope indicators. Inclinometers are devices used to monitor subsurface movements through a probe transducer (accelerometer, MEMS) designed to measure inclination with respect to vertical. A detailed description is found in Dunnicliff (1993). Operation involves lowering the probe down a borehole with inclinometer casing, which has two pairs of orthogonal grooves in which the probe wheels run. One pair of inclinometer casing grooves is usually oriented in the dip direction of the pit slope or in the anticipated direction of ground movement. Inclination is then measured at a number of fixed points as the probe is pulled back to the surface (Figure 8.5-9). Comparison of repeat periodic surveys provides an indication of differential displacements at depth as a function of time. Displacements are usually summed, resulting in profiles of cumulative displacement along the borehole (Figure 8.5-9).

When installing inclinometer casing, it is important to select the appropriate diameter. Large-diameter casing is better suited to shear zones, multiple shear zones, and slope failures. Moderate- to small-diameter casing can be used for short-term installations or slopes where smaller displacements distributed along the borehole are anticipated. Correct installation of the casing is important; and in deep holes, particularly the influence of helical deformation must be considered. A digital spiral sensor probe can be used to check the spiraling of the casing.

Inclinometer monitoring is widely used at the Syncrude oil sands mine in Canada to monitor highwall performance while draglines operate above it. McKenna et al. (1994) reported that approximately 20% of the highwalls have significant potential for slope failure, requiring intensive monitoring to ensure safe operation of the draglines. From 300 to 800 inclinometers were being installed each year to monitor shear movements and the development of slide surfaces at discrete depths (Figure 8.5-9).

In-Place Inclinometers

Where the depth of localized subsurface displacement is known or can be anticipated, an in-place inclinometer system



Source: Adapted from McKenna et al. 1994.

Figure 8.5-9 Inclinometer data of shear displacements at depth recorded for a highwall at the Syncrude oil sands mine in Canada, showing basic operation principles of a probe inclinometer

may be used to monitor continuous movements across known active shear surfaces. This system may also incorporate MEMS technology. In-place inclinometers consist of one (or several) inclinometer sensors housed within a stainless-steel enclosure. Sensors are separated from one another by stainless-steel rods and wheel assemblies. Sensors are installed in zones of expected displacement allowing remote automatic monitoring of pit slope movements.

Borehole Extensometers

Another variant from those previously discussed involves the use of mechanical or electrical probes (i.e., probe extensometers) to monitor changing distances between fixed points along a borehole as determined by the probe position. Typical accuracies for probe extensometers range from submillimeter to millimeter over distances of less than a meter.

A wide variety of borehole probe extensometers are commercially available, which differ according to the following:

- Single-point or multipoint measurement
- Measurement transducer/principle (dial gauge, resistance, magnetic, sonic, fiber optics, vibrating wire, and MEMS technology)

Multipoint borehole extensometers monitor displacements in a borehole at various depths within a slope or excavation wall/floor/roof. They usually comprise up to eight rods per drill hole and have manual readout or vibrating wire transducers with a measurement range of between 25 to 100 mm and a sensitivity of between 0.01 and 0.1 mm. Magnetic extensometers consist of spider magnetic anchors positioned along a PVC tube with a stable anchor at the base of the borehole (Figure 8.5-10A). The vertical movement of the anchors can be monitored manually with a probe.

Other versions of magnetic extensometers allow for automatic monitoring. The Increx system is comprised of brass rings located at fixed intervals along the outside of the PVC borehole casing (Figure 8.5-10B). A probe is used to measure the distance between successive rings, and the result is compared to an initial survey. Surveys taken at specific time intervals provide indications of whether compressive or extensile displacements have occurred. Changes as small as 0.001 mm can be measured with an accuracy of ±0.01 mm/m. Applications of the Increx system include the monitoring of vertical and horizontal deformations around underground and surface excavations.

A variety of sonic probe extensometers are also commercially available. The Sondex system installation consists of regularly spaced steel sensing rings and a corrugated Sondex pipe installed over inclinometer casing (Figure 8.5-10C). The annulus between the borehole wall and the Sondex corrugated pipe is filled with soft grout. This couples the pipe to the surrounding ground, so that the pipe and rings move with settlement or heave. As the probe passes a ring, an audible sound emits and the depth reading is taken. Settlement and heave are calculated by comparing the current depth of each ring to the previous readings of depth. Sonic extensometers have found considerable application in monitoring roof, wall, and floor deformations at specific depths. In stratified rocks, this allows for the estimation of dilation and bed separation around the opening.

Resistance wire extensometers are comprised of a pretensioned electrical resistance wire element fixed within a 1- to 2-m-long plastic tube and a strain gauge measuring readout. The wire is attached, under mild tension, to each end of the tube. The extensometer is installed by grouting into a borehole; a number of resistance wire extensometers can be installed end to end. Significant tensile strain can be measured with high sensitivity. Resistance wire extensometers are used for the monitoring of tensile strains around underground excavations and slopes. Applications have included pillar, roof, and stope wall displacement.

Convergence Monitoring Systems

Bock (2000a, 2000b) provides a useful summary of geotechnical instrumentation used in tunneling with particular focus on performance monitoring to assist with construction control. Measurements normally required include tunnel convergence, for which there are several instruments. Tunnel-profile monitoring systems comprise a series of linked rods fixed to the tunnel wall to monitor displacement; each rod has a high-accuracy displacement meter and a tiltmeter that record changes in displacement and tilt. The Bassett convergence system uses a system of articulated arms fitted with tilt sensors to monitor the movement of reference points that are mounted

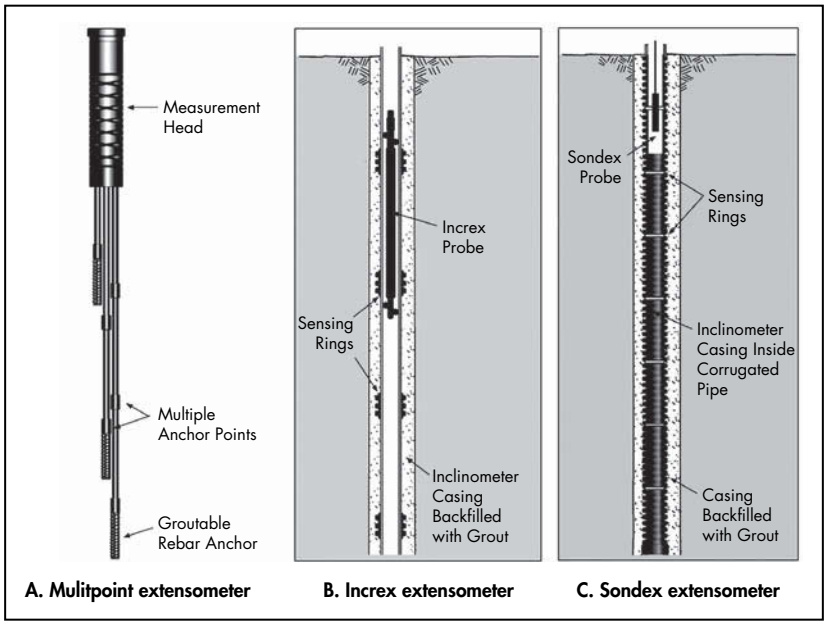


Figure 8.5-10 Types of borehole extensometers

on the tunnel wall. Metje et al. (2006) describe a new fiber-optic-based Smart Rod tunnel monitoring system to measure structural displacements and deformations. The Smart Rod system was designed as a series of short rod sections, joined at fixing positions around the circumference of the tunnel.

Shaped Accelerometers

A new, wireless, MEMS-based system recently developed for real-time deformation monitoring is described by Abdoun and Bennett (2008). This system is based on triaxial MEMS accelerometer measurements of angles relative to gravity. Three accelerometers are contained in each 30-cm-long rigid segment for measuring x, y, and z components of tilt and vibration. The segments are connected by composite joints that prevent torsion but allow flexibility in two degrees of freedom. These rigid segments and flexible joints are combined to form a sensor called a ShapeAccelArray, which is capable of measuring 3-D ground deformation from within a borehole, for example, at 30-cm intervals to a depth of 100 m (Figure 8.5-11). The arrays consist of subarrays of eight segments connected end to end. Microprocessors, one per array, collect data from the groups of sensors.

Time Domain Reflectometry

Time domain reflectometry (TDR) is a standard electrical method for locating faults in cables. TDR is finding increasing use in mining geotechnical applications. The essential principle of TDR involves the effect of cable deformation on the passage of a voltage pulse along a two-conductor coaxial metallic cable, with the pulse being partially reflected by the deformation. TDR instrumentation allows for the location of the cable deformation using the travel time of the voltage pulse and the propagation velocity of the signal in the cable. Shearing of a TDR cable grouted in a borehole in rock or soil allows the location of the shear zone through changes in geometry and impedance between the inner and outer coaxial cable conductors. The magnitude of the soil and rock deformation over time can be determined, as it is proportional to the amount of cable deformation and hence changes in amplitude of the reflected voltage signal.

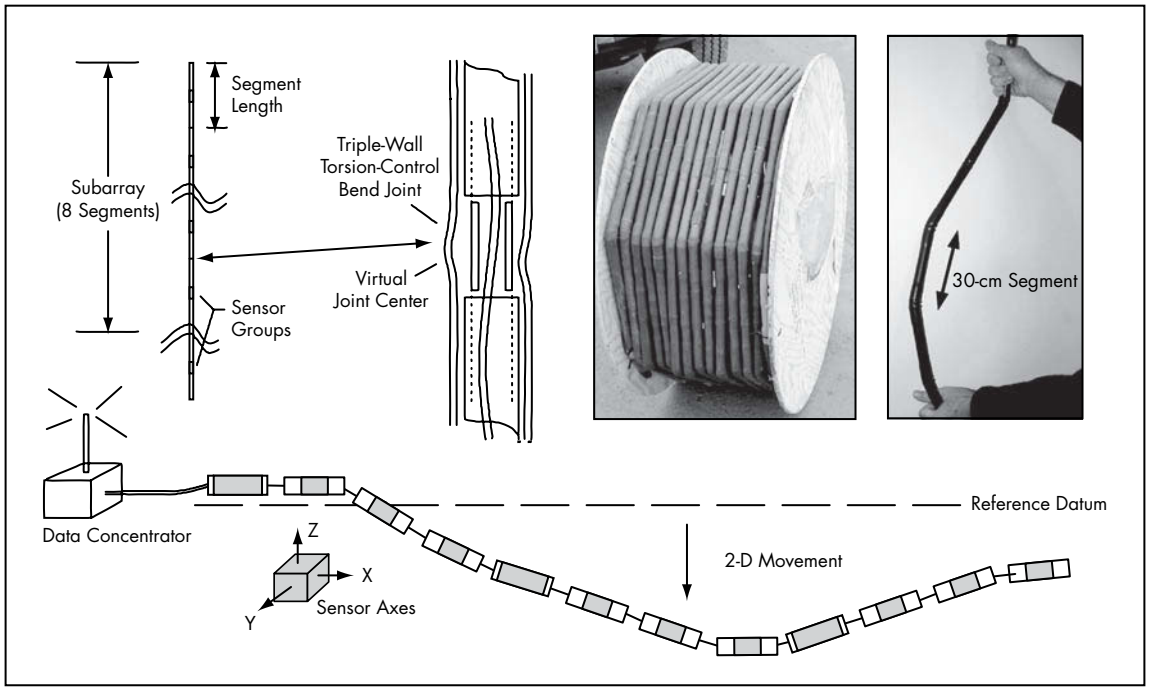
Dowding et al. (2003a, 2003b) provide a synopsis of the concept of TDR, its use in geotechnical engineering, and valuable practical experience gained in its use. Of particular interest are correlations between TDR magnitude and inclinometer displacements, the use of long (>300 m) horizontal TDR sensor cables above underground mines and parallel to both roads and slopes, monitoring at great depth (>500 m), and the use of TDR in geotechnical alarm systems (O'Connor 2008).

TDR has found application in surface mining, underground mining, and subsidence (Kniesley and Haramy 1992; Dowding and Huang 1994; Allison and de Beer 2008; Carlson and Golden 2008). Carlson and Golden (2008) describe successful use of TDR in remote monitoring of cave initiation at the Henderson mine (Colorado, United States). Allison and de Beer (2008) describe the monitoring system used to monitor the cave at the Northparkes mine (Australia), Lift 2. Displacement monitoring including TDR cables, convergence, and multipoint borehole extensometers are associated with damage mapping, borehole video, and microseismic monitoring. These authors emphasized the need to minimize the time between installation of TDR cables and the commencement of monitoring in order to reduce the possibility of cable damage.

Szwedzicki et al. (2004) clearly showed the applications and success in TDR monitoring at PT Freeport's Deep Ore Zone block cave mine (Indonesia). Results at PT Freeport from TDR cables monitored over a period of 2 years provided information on vertical progression of the cave and zones of dilation. Horizontal progression of the cave was monitored using cables installed from the undercut level. TDR at the Deep Ore Zone mine also provided information on the cave ratio (cave back height to height of draw) and the rate of progressive caving.

Fiber Optics

Fiber-optic technology (Table 8.5-5) is being increasingly used in the development of geotecthnical instrumentation.



Source: Adapted from Abdoun and Bennett 2008.

Figure 8.5-11 ShapeAccelArray subarray assembly

Commercially available fiber-optic-based instrumentation includes displacement transducers, piezometers, strain gauges, and temperature gauges. The following four main types of fiber-optic sensors exist:

1. Point sensors using Fabry-Pérot interferometric sensors
2. Multiplexed sensors using fiber Bragg grating sensors
3. Long-base sensors using interferometric SOFO sensors
4. Distributed sensors using either distributed Brillouin scattering sensors or distributed Raman scattering sensors

These sensors provide exciting new development potential for geotechnical instrumentation (e.g., Inaudi and Glisic 2007a). Of particular interest is the future potential of distributed sensors as reported by Bennett (2008) and Inaudi and Glisic (2007b). Distributed fiber-optic sensors can use a single optical fiber with a length of tens of kilometers to obtain dense information (every meter) on strain distributions across geotechnical structures or on the surface above underground mine excavations.

Remote Sensing of Ground Deformation

Satellite InSAR

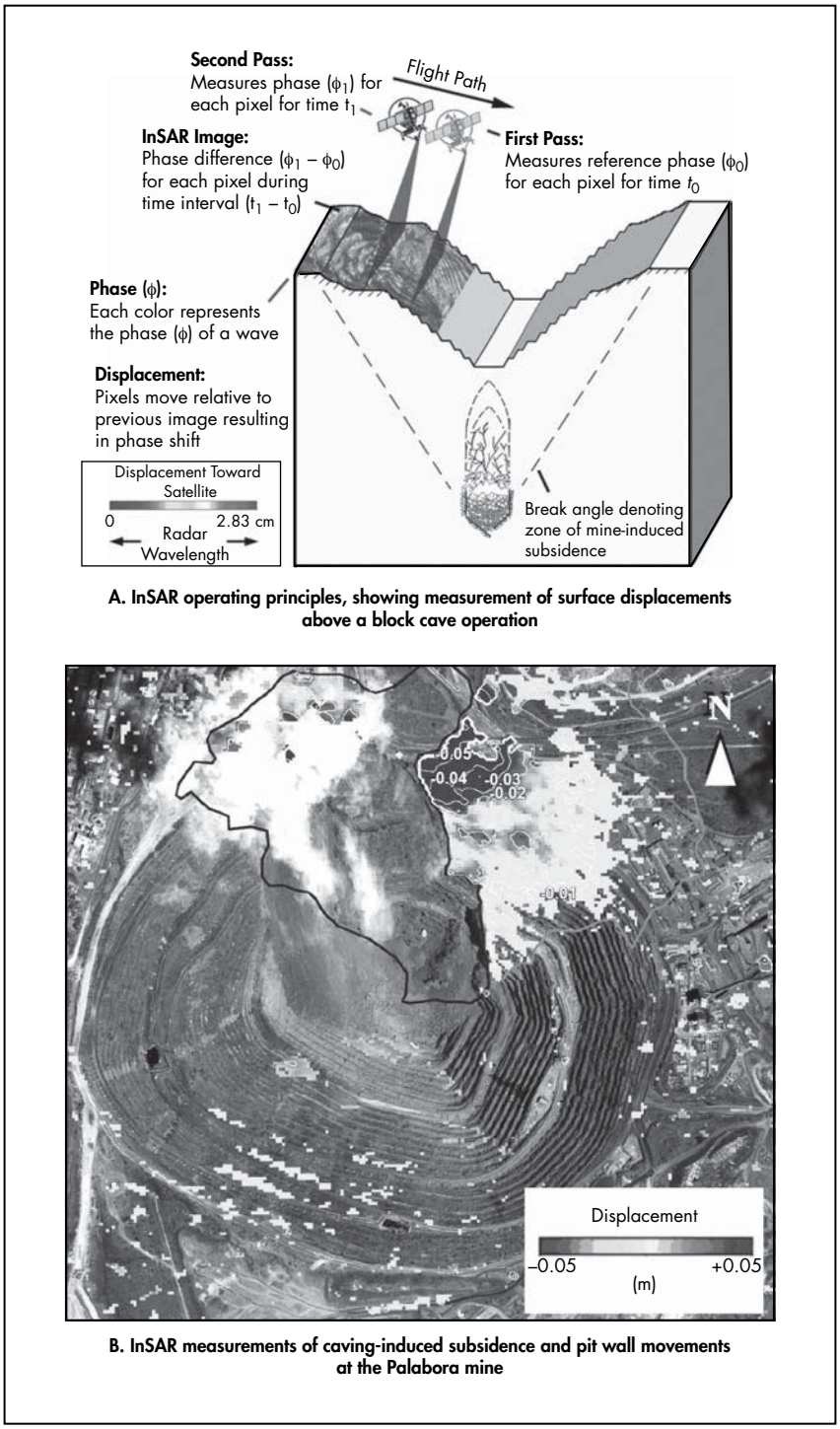
Space-borne interferometric synthetic aperture radar (InSAR) involves the use of satellite-based microwave radar to remotely monitor ground deformations. With repeated orbits and image capture (referred to as stacks), interferometric techniques can be used to resolve 3-D information of surface deformations by analyzing differences in the phase between waves being transmitted and received by the satellite (Figure 8.5-12A). Ground deformations on the scale of centimeters to millimeters can be detected for a surface area resolution of several square meters using these techniques.

Jarosz and Wanke (2003) describe the feasibility testing of InSAR for two mine sites in Western Australia. Results are provided for the Leinster Nickel mine, a sublevel caving operation beneath an open pit, for which InSAR was used to detect the extent of subsidence within the pit and active mining area. Kosar et al. (2003) used InSAR at the Island Copper Mine on Vancouver Island in western Canada to test its ability to provide adequate warning of potential failures during flooding of the pit during its decommissioning. Small ground movements along the steep pit slopes were successfully detected. This lead Kosar et al. (2003) to point to the continuous spatial coverage provided by InSAR compared to the large number of survey or GPS monuments that would have been required to cover the same area. Kosar et al. (2003) also pointed out the ability to remotely obtain data from sections of the pit that were otherwise inaccessible due to safety concerns.

Rabus et al. (2009) describe the use of InSAR to identify and map spatial movements within and around the Palabora open-pit mine due to block-cave mining beneath the finished pit (Figure 8.5-12B). This ability is important for protecting key mine infrastructure located near the pit rim.

Surface Radar

Since about 2000, ground-based radar has become an increasingly efficient method of monitoring open-pit slope movements (e.g., N. Harries et al. 2006; de Beer 2007). These systems are able to provide accurate displacement measurements along the line of sight of a high number of targets (natural or artificial reflectors) with submillimeter precision. The slope-stability radar (SSR) system, described by Harries and Roberts (2007), uses a real aperture on a stationary platform positioned 30 to 1,400 m away from the slope. Extended-range versions are now able to obtain a maximum range of 2,800 m.

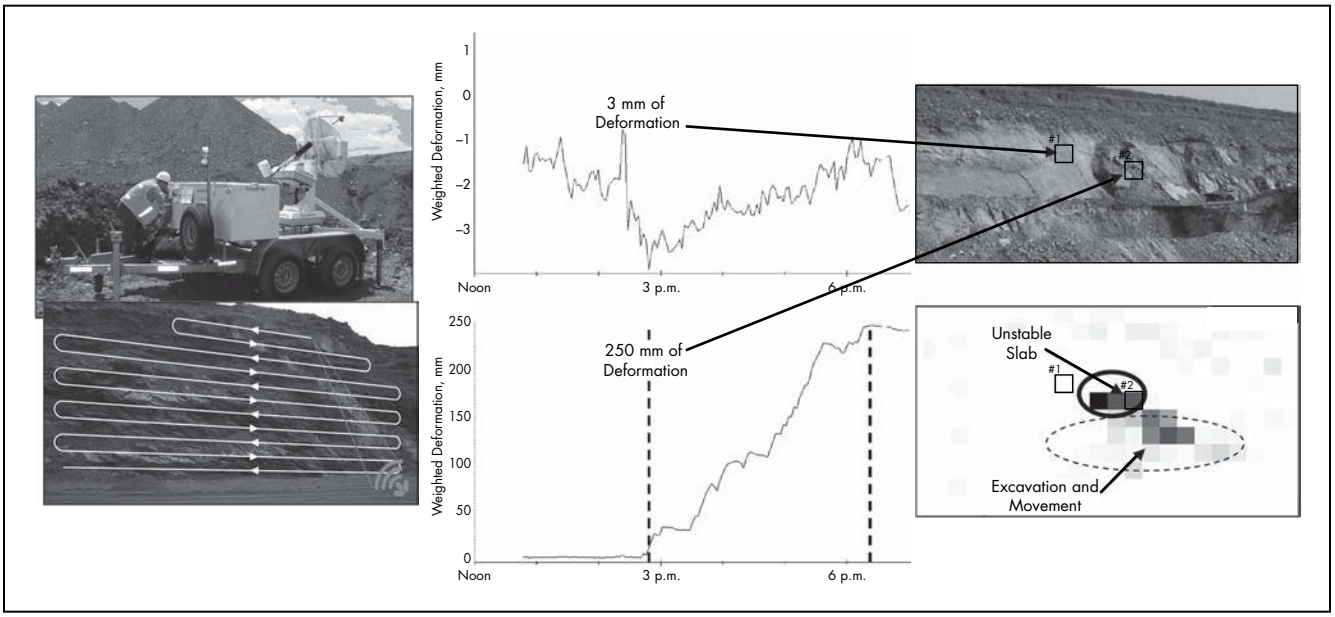


Source: (A) Adapted from Rabus et al. 2009; (B) Adapted from AMEC 2006.

Figure 8.5-12 InSAR measurements

Figure 8.5-13 shows the SSR equipment, the generated data that scans a region of the pit wall, and comparisions of the phase measurement in each footprint (pixel) with a reference scan to determine the amount of movement of the slope.

Slope radar technology has revolutionized surface mine monitoring, providing full coverage of a rock slope and offering submillimeter measurements of wall movements. Adverse affects due to rain, dust, and smoke are minimized, although reduced precision occurs in pixels due to low coherence between scans, for example, due to vegetation. Harries et al. (2006, 2009) describe the application of the technology at numerous open-pit mines where it has been successfully used to monitor and provide impending warning of pit slope failures just tens of minutes to hours before failure. Cahill and Lee (2006), Joost and Cawood (2006), and Little (2006) provide well-documented examples of the benefits of this



Source: Adapted from Harries et al. 2006.

Figure 8.5-13 SSR system showing the continuous monitoring of millimeter-scale movements across the face of an unstable open-pit mine slope

technology in managing the risks due to slope instability at major open-pit mines.

LiDAR and Photogrammetry

In addition to pit slope and rock mass characterization, terrestrial LiDAR and photogrammetry can also be used for pit-slope displacement monitoring. Early use of terrestrial LiDAR monitoring focused on surface mine operations, in particular blast design and control. Coggan et al. (2001) illustrated the potential use of LiDAR in monitoring the retrogression of a mine slope failure in a china clay quarry pit. The use of ground-based LiDAR in an integrated surface mine monitoring program is described at the Potgietersrus Platinum mine (South Africa) by Little (2006), where two permanently mounted LiDAR scanners were used to scan a pit wall and help demarcate areas of slope deformation. The ground-based LiDAR at this mine was used in combination with prism surveying and also with slope-stability radar.

Terrestrial photogrammetry has an even longer history in the monitoring of surface mine slopes. Digital photogrammetry forms an excellent record of slope performance and rockfall activity. Tunnel scanners are used for profile scanning (e.g., determination of overbreak, to verify shotcrete thickness, tunnel face change detection with time, and tunnel surface deformation). Systems may be either LiDAR or photogrammetrically based. Wilson and Talu (2004) describe the use of a tunnel scanner at the Finsch mine (South Africa) for providing data on deviation of tunnel profiles from planned and actual, tunnel shape, damage, and alignment.

Photogrammetric systems may use either two digital cameras mounted on a portable fixed bar or one camera with a specialized tripod head allowing controlled repeatable multi-imaging of the tunnel. All systems provide a digital 3-D stereo image of the tunnel. Birch (2008) and Wimmer et al. (2008a, 2008b) describe the use of photogrammetry in underground blasting and fragmentation studies.

Stress Change and Pore Pressures

The change in stress associated with various stages in mining is of significant importance. This can range from monitoring pressures within pillars as adjacent rooms are excavated to the monitoring of pressures in the roof of excavations. Often the associated instruments are used in association with convergence and borehole extensometers to provide data for optimizing future mine design using numerical models and ensuring safety.

Pressure Cells

Borehole pressure cells typically have a measurement range of 0 to 70 MPa. They may be a flat jack (two steel plates welded together with hydraulic oil in between) configured to detect changes in stress perpendicular to the cell, or they may be cylindrical in design measuring the average change in pressure in the plane perpendicular to the borehole. Push-in (or spade) pressure cells are particularly useful for applications such as measuring total pressures in earthfills. These cells can be fitted with integral piezometers to allow measurement of pore water pressures and derivation of effective stresses. Push-in cells have standard ranges of operation up to 5 MPa. Shotcrete stress cells generally consist of two rectangular steel plates welded together with de-aired fluid in between. Changes in pressure in the shotcrete lining are recorded by a change in pressure in the fluid within the cell; electrical resistance or vibrating wire technology is used to record this change in pressure. Standard measurement ranges from 2 to 35 MPa are common.

Direct measurement of stresses in tunnel linings can also be undertaken using the slot-relief or flat-jack compensation method. This involves locating measurement points positioned adjacent to a future diamond saw cut, cutting a narrow slot and measuring the convergence across it due to stress relief between the measurement points, and inserting a flat jack and inflating it until the convergence of the points is fully

reversed. This value is termed the *compensation pressure* and approximates the value of stress in the shotcrete.

Variations in pore water pressure during the lifetime of a mining project are likewise an important component of geotechnical instrumentation and design. Piezometric instrumentation was described previously in the "Groundwater Characterization" section. Ongoing monitoring of pore water pressures can be compared with deformation measurements used to provide an indication of groundwater conditions ahead of the mining front, used to provide information for remediation measures, and used as an input for numerical modeling.

Microseismicity

Microseismic monitoring provides mining and rock mechanics engineers with information on the stress conditions in the rock mass and how the ground is responding to induced stresses due to changing mine excavation geometries. The location of seismic events and their characteristics provides valuable information, both in terms of improved mine safety, and optimization of mine design and sequencing. Commercial microseismic monitoring systems have been in use since the 1970s and were originally used in underground rock-burst-prone mines (see Blake and Hedley 2004). The systems are now increasingly being used in both underground and surface mines.

Numerous companies provide state-of-the-art 24-bit digital seismic recorder systems that integrate into local area networks (LANs) or wireless networks. Hudyma and Brummer (2007) address the key questions in the design of a seismic monitoring system, mainly the optimal number, type, and location of the sensors. Seismic sensor arrays are usually composed of uniaxial and triaxial sensors. Triaxial sensors can provide seismic source parameters (energy, seismic moment, and magnitude), whereas uniaxial sensors primarily provide accurate seismic locations.

The sensitivity of a seismic array is directly proportional to the number of sensors used. The source location accuracy is also proportional to the number of sensors in the seismic array. Hudyma and Brummer (2007) present a guideline that the system source location should be approximately 5% to 10% of the intersensor spacing. Where possible, the seismic array should surround the rock mass of interest, but if this is not practical, the array should be spread geometrically in three dimensions. If there are an insufficient number of triaxial sensors, the seismic parameters may be influenced by attenuation or by seismic-event energy radiation patterns.

The reader is encouraged to consult Hudyma and Brummer (2007) for further useful discussion of the practical aspects of seismic array design in underground mines, including design for future mining stages, sensor installation, automatic source location reliability, system calibration and maintenance, seismic data analysis, and ensuring optimal performance from seismic systems. These authors emphasize that microseismic monitoring in underground mining can be optimized through good system design, frequent data analysis, and routine auditing of the system. Delgado and Mercer (2006) provide an interesting description of one of the largest mine microseismic systems in the world at the Campbell mine in Red Lake, northern Ontario, Canada. Also discussed are some of the issues faced.

Recent years have seen an increase in the successful use of microseismic systems in open-pit mines, spurred on by the ever-increasing depths of large open pits and the presence of underground mines beneath open-pit slopes. Lynch and Malovichko (2006) describe how microseismic monitoring in open-pit slopes has been routinely practiced since 2002 at mines in Namibia, South Africa, and Australia. They report that monitoring had been conducted for more than 25 open-pit slopes, all of which showed signs of brittle fracturing, in one case at a slope height of only 80 m. They emphasize that for reliable event locations, the seismic sensors array should surround the volume of rock being monitored, which in an open-pit mine means that they must be located near to the surface as well as at the bottom of the monitored volume.

In practice, potentially unstable slopes are monitored rather than the entire pit. Typical sensor separations are in the order of 100 to 200 m. The system described by Lynch and Malovichko (2006) involved near-surface sensors installed in short (i.e., 10 m) vertical boreholes using 4.5-Hz geophones, and in long inclined holes (i.e., 100 to 300 m) using 14-Hz omnidirectional geophones. These authors show correlations between microseismic activity and mining at the base of the slope, removal of broken rock, and the location of seismically active structures behind the pit wall.

Wesseloo and Sweby (2008) emphasize the increasing role that microseismic monitoring will be required to play as open pits increase in depth with a consequent increase in stress and a greater uncertainty in the pit slope deformation mechanisms. These authors provide an overview of microseismicity in rock slopes and in mine slopes in particular. They also provide an excellent account of microseisimic response to mining, event size (energy), and S-wave to P-wave ratios. An informative case study of microseismicity at an Australian open pit is presented by Wesseloo and Sweby demonstrating the future potential of this monitoring technique in open-pit environments.

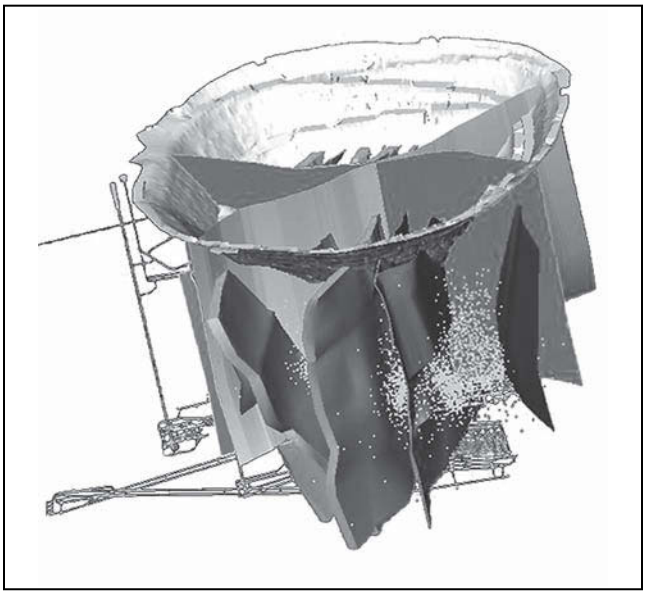
DATA AQUISITION AND PRESENTATION

Data reliability is of primary importance, requiring mine personnel to have confidence in the performance of an instrument. This can be gained, in part, through the performance of routine calibration checks, instrument inspections, and maintenance. Data integration and data management are also key issues. Important new elements such as Web geographic information system (GIS) services can be integrated into the operational resources of decision makers. These services are linked to early warning systems through wireless data acquisition and transmission technologies, which enable real-time data from multiple remote monitoring sites to be accessed and viewed by mine geotechnical staff, both on- and off-site, by means of the Internet. This is proving to be a highly valuable resource where an unstable pit slope threatens production and/or worker safety. Spatially and temporally distributed measurements should be combined with a knowledge engine and an evolving rule base to form the hub of a decision-support system (Hutchinson et al. 2007).

Wireless Data Transmission

Wireless technology enables continuous real-time monitoring of production activities and rock mass response to mining throughout an operation. Although automatic data-acquisition systems cannot replace engineering judgment, when combined with wireless data transmission, the following advantages can be gained (Dunnicliff 1993):

- More frequent readings
- Retrieval of data from remote/inaccessible locations



Source: Adapted from Woo et al. 2010.

Figure 8.5-14 Integrated mine-data model combining surface and underground mine plans, geology, and microseismic data

- Instantaneous transmittal of data over long distances
- Measurement of rapid changes/fluctuations in monitored parameter
- Increased reading sensitivity
- Reduction of measurement and recording errors
- Increased flexibility in selecting, managing, and storing data

From a system's operation perspective, automated wireless systems increase safety and lower costs. The fundamentals of a communication system include a transmitter, a receiver, and a surge arrestor, which allows reliable communication over distances of a few kilometers. The advantages of these systems over satellite or cell phone communication include lower costs and ease of use. The disadvantages include the requirement to use repeaters if a line of sight over long distances is not possible. It is vital that the network be reliable and always available, even in difficult terrain and harsh weather conditions.

Data Immersion and Visualization

The acquisition of geotechnical mine data can lead to the generation of massive volumes of data from in-situ surveys and mine operations, making managing, storing, and utilizing the data difficult. Improved computer performance and new software developments are changing this situation. Easy-to-use integrated geotechnical data-management systems with 3-D visualization and data immersion can be envisaged, linking monitoring, analysis, prediction, and remediation.

These attempts at data "fusion" are moving toward the adoption of virtual reality technology, where the identification of hidden relationships, the discovery and explanation of complex data interdependencies and the means to compare and resolve differing interpretations, can be facilitated (Kaiser et al. 2002). Spatial databases can be developed to integrate the different data sets being used for mine design and geotechnical analyses (geological, geotechnical, operational, etc.) into an interactive 3-D visualization and virtual reality environment (Figure 8.5-14).

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CHAPTER 8.6

Hard-Rock Ground Control with Steel Mesh and Shotcrete

John Hadjigeorgiou and Yves Potvin

Ground control practices in underground mines are constantly evolving, always with the goal of improving mine safety and economics. These changes are often influenced by the mining methods used, the level of mechanization and types of equipment used, and the labor cost and skill set. Ground control practices are further defined by rules and regulations linked to risk tolerance and to the mining cultures of different jurisdictions or countries.

In this chapter, a distinction is made between reinforcement and surface support:

- *Reinforcement* is a technique in which elements such as rock bolts are applied internally to the rock mass. Chapter 8.8 provides a thorough review of reinforcement techniques.
- *Surface support* is a technique in which elements such as shotcrete or steel mesh (the term *screen* is also commonly used in North America) are applied to excavation surfaces externally to the rock mass. This chapter examines surface support as it is used for underground metal mining. A comprehensive collection of surface support applications in mines is presented in Potvin et al. (2004).

The traditional approach to ground control in underground metal mines was to install reinforcement to stabilize potential wedges and blocks that might "daylight" at the surface of excavations. This required good ground-awareness skills on the part of the mine workers, who used their experience to read the ground and install reinforcement at appropriate locations to stabilize the rock mass around excavations. Small blocks or "scats" that formed at the rock surface around the excavations due to stress changes and blast vibrations were controlled by frequent scaling. When ground conditions were less competent, the rock surface was controlled with mesh. The main role of surface support was to support the dead weight of smaller rock blocks between the rock bolts.

Since about 2000, a concerted effort toward minimizing the risk of rockfall injuries has accelerated the development of improved ground control practices. When it was recognized that small rock falls cause many injuries, the systematic use of surface support became common practice in most mechanized mines (Nedin and Potvin 2005). Several mines changed their ground support strategy, designed originally to suit local conditions, to the systematic application of pattern bolting and surface support. Current ground support standards are well documented and reflect the current industry's low-risk-tolerance approach to rock falls.

In this new ground support strategy, reinforcement elements hold or pin larger blocks, and surface support elements catch smaller blocks. However, this approach is less applicable in ground conditions where the rock mass is already in a postfailure or yielding state due to excessive static or dynamic loading. In these cases, the excavation often experiences large displacements and the surrounding rock mass crumbles. To maintain the integrity of the excavation surface (wall or roof), surface support should be capable of containing the bulk of the broken rock mass and later movement resulting from stress changes such as stoping. The surface support must deform with the rock mass while the load transfers from the surface support, through the plate or surface fixture arrangements, to the reinforcement. The function of reinforcement is to retain the surface and limit displacement or convergence. Thus, reinforcement and surface support are part of an integrated system that transfers and shares load until the excavation surface is stabilized or until the support system fails at its weakest link.

SURFACE SUPPORT ELEMENTS

Although a wide variety of rock bolts are available, the choice of surface support elements is limited to steel mesh, straps, and shotcrete. Steel mesh is the most popular in metal mines, but shotcrete has been gaining in popularity since its introduction in the late 1980s. Straps, also considered to be surface support even though they do not completely cover the rock surface, are used to contain the rock mass between reinforcement elements and distribute load between the reinforcement elements used to install the straps.

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Yves Potvin, Director of the Australian Centre for Geomechanics, University of Western Australia, Perth, Australia

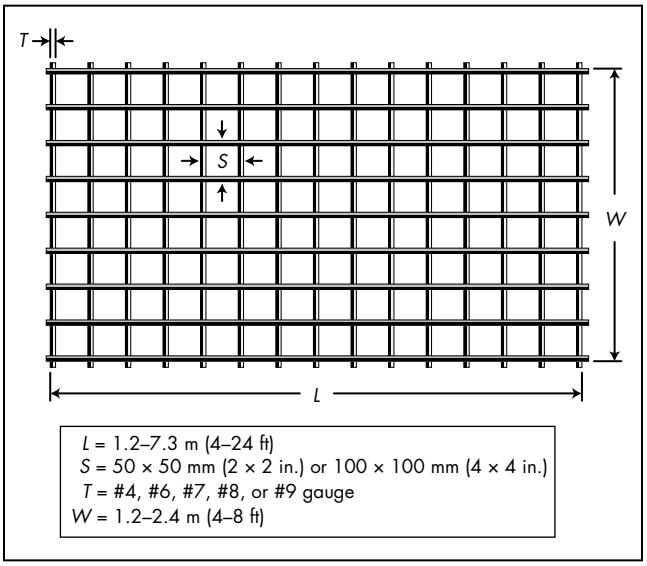


Figure 8.6-1 Welded-wire mesh (typical dimensions)



Courtesy of MacLean Engineering and Marketing Co. Ltd.

Figure 8.6-2 Installation of bolts and mesh by a MacLean bolter



Courtesy of the Australian Centre for Geomechanics.

Figure 8.6-3 Installation of mesh using a pressure plate

Steel Mesh

Two main types of steel mesh are used in the mining industry: welded-wire mesh (weld mesh) and chain-link mesh, also called diamond mesh. Expanded metal mesh is also available but is not widely used. Mines that use a mechanized approach to mesh installation use predominantly precut sheets of welded-wire mesh.

Welded-Wire Mesh

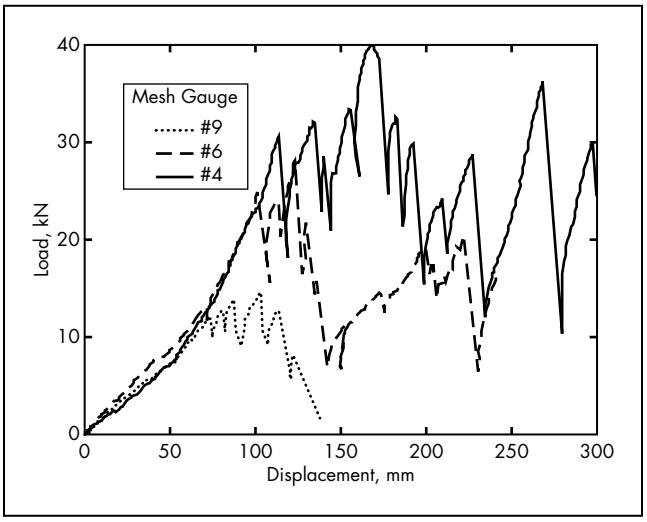
Sheets of welded-wire mesh are precut to customized dimensions. In Australia, sheets are generally large (2.4 × 3.0 m [8 × 10 ft]) to promote productivity; accounting for overlap, a row of six sheets is commonly used to cover 40 m^2 (~130 ft^2). In North America, installation involves more manual handling, and sheets tend to be smaller (1.2–1.5 × 3.0 m [4–5 × 10 ft]) and thus lighter in weight. Typical dimensions for welded-wire mesh are shown in Figure 8.6-1.

Notwithstanding quality control issues and assuming the use of standard-aperture welded-wire mesh, wire thickness (*T* in Figure 8.6-1) is the main influence on wire performance. In North America, wire thickness is expressed in terms of gauge; #4, #6, and #9 gauge (5.8-, 4.9-, and 3.7–3.8-mm diameters, respectively) are most commonly used. In Australia, a diameter of 5.6 mm is most commonly used, but sometimes diameters as small as 4.95 mm are also used.

Selection of mesh aperture (*S* in Figure 8.6-1) is in theory based on the smallest block that the mesh must contain. In North America, mesh aperture is often standardized at 100 × 100 mm (4 × 4 in.). A smaller aperture (such as 50 × 50 mm or 75 × 75 mm [2 × 2 in. or 3 × 3 in.]) is used where extra load-bearing capacity is required. A larger aperture (such as 150 × 150 mm [6 × 6 in.]) can be used when shotcrete is to be sprayed over the mesh; the larger aperture allows for good penetration of the shotcrete through the mesh. When the mesh is expected to provide support for a long time or in corrosive environments, galvanized coating is used to protect against corrosion.

Welded-wire mesh is relatively inexpensive and can be installed quickly because it is easily attached to existing reinforcement by means of plates and rock bolts. It can support small blocks and its energy absorption properties enable it to provide some control during rock bursts. It can also be used with shotcrete. No. 9 gauge mesh is susceptible to damage from fly rock. No. 6 gauge mesh can be a good substitute but is less flexible and more difficult to install around corners. Mesh is fabricated using longitudinal wire to which cross wires are spot-welded. When it is to be installed near an advancing face, optimally the cross wire should be laid against the rock surface and the longitudinal wire laid on the outside (facing the opening) and along the drift axis. This procedure mitigates blast damage by flying rocks, which are less likely to catch into cross wires.

In North America, it is popular to install ground support systems using dedicated bolting machines. The MacLean bolter features a scissor platform with a rock-bolting boom mounted at the tail end and operator controls mounted at the front end (Figure 8.6-2). Hooks to hold sheets of mesh are



Source: Tannant 2004.

Figure 8.6-4 Load vs. displacement for three gauges of welded-wire mesh

located on the side of the platform. Installation of bolts and mesh in a single pass requires some manual handling of the mesh during bolting. Mesh can also be installed in a second pass, after bolting is completed, by means of pressure plates fitted over installed threaded bars (Figure 8.6-3).

Welded-wire mesh is commonly installed manually by means of an air leg, which is often the only option in countries where mining mechanization is low. Manual installation is performed with a handheld drill in smaller excavations, or from scissor platform trucks or integrated tool-carrier baskets in larger drifts. In practice it is difficult, especially with large-diameter wire or stiff welded-wire mesh, to ensure that the mesh sheets are installed tightly against the rock mass surface, and manual installation increases the risk of minor injuries.

A number of other quality control issues are common to all welded-wire-mesh installations. In particular, poor overlap between sheets is highly undesirable, because it may later expose mine personnel to rockfall hazards. Standard practice is to overlap three squares of mesh, but in difficult ground conditions, drift profiles are often erratic and proper overlap can be difficult to obtain.

Villaescusa (2004) identified three modes of mesh failure:

1. Shear failure at the weld point, presumably due to the technology used for welding and to quality control issues such as dirty electrodes or dirty wire
2. Heat-affected zone failure, due to excessive weld head pressure and temperature
3. Tensile failure of the wire, due to excessive tensile load

The load and displacement capacity of mesh depend on the type of metal and size of the wire and dictate the load-bearing capacity of the wire and weld. It follows that weld capacity should be comparable to wire strength.

Tannant (2004) and Thompson (2004) developed testing procedures to better define the capacity of mesh and published pull-test results for a variety of mesh and bolt arrangements. On tests of standard (5.6-mm diameter) Australian welded-wire mesh, Thompson found load capacity to be ~20–40 kN with displacements of 100–350 mm [4–14 in.]. The variability of results arises from variations in simulated bolt patterns and test procedures. In general, a smaller bolt pattern (1 × 1 m [3.3 × 3.3 ft]) supports higher loads but produces a stiffer response than does a larger bolt pattern (1.5 × 1.5 m [5 × 5 ft]). Tannant tested load as a function of displacement for the three common mesh gauges used in Canada in a simulated bolt pattern of 1.2 × 1.2 m (4 × 4 ft) (Figure 8.6-4).

Chain-Link Mesh

Chain-link mesh is used less than welded-wire mesh. Its main advantage is its energy absorption capacity, which is associated with its greater displacement capacity. Consequently, one application of chain-link mesh is part of a rock burst support system. Its main disadvantages are that it tends to unravel when damaged and that it is difficult to apply shotcrete through it. Typical chain-link mesh dimensions are shown in Figure 8.6-5.

Chain-link mesh differs from welded-wire mesh in its ease of installation and its deformation capacity. It has very low rigidity, which makes handling and installation awkward and difficult. It comes packaged in rolls and needs to be gradually and manually unrolled during installation to its final position against the rock surface. Installation is labor-intensive and difficult to mechanize. As a result, it is not widely used in highly mechanized underground mines but remains popular in the mines of, for example, South Africa and South America.

Mechanization of chain-link-mesh installation is currently being addressed by development of the Rock Mesha mechanized mesh handler, which can be mounted on one of the booms of a twin-boom jumbo drill (Figure 8.6-6) (Coates et al. 2009). One boom unrolls the mesh against the rock surface while the other boom installs bolts. A prototype of this equipment is being tested in Australia.

Interestingly, the low rigidity of chain-link mesh can be an advantage in terms of its deformation capacity. Tannant (2004) pull-tested three types of mesh: chain-link, welded-wire, and expanded-metal (Figure 8.6-7 and Table 8.6-1). Expanded-metal mesh is seldom used in underground mines and is not discussed here. Chain-link mesh demonstrated a displacement capacity of ~400 mm (16 in.) at peak load with good post-peak behavior, retaining >50% of its load capacity at 800-mm (32-in.) displacement. The load capacity of chain-link mesh, like that of welded-wire mesh, is largely a function of wire size and the type of steel used. In Canada, #9 gauge (3.7-mm) wires are commonly used; in South Africa, wires of 3.2–4 mm are standard.

High-tensile, light steel chain-link mesh is also available for use when ground conditions require extra load-bearing capacity, whether the situation is static or dynamic (Coates et al. 2009). For example, high-tensile Tecco mesh, which uses wire 4 mm in diameter and weighs only 2.6 kg/m^2 (0.53 lb/ft^2), was tested at 110 kN using the test procedure described in Player et al. (2008).

Load Transfer Between Mesh and Reinforcement

The load-bearing capacity of mesh is well suited for containing small blocks that can detach between reinforcement elements. Standard reinforcement patterns used in mines generally call for elements to be spaced 1–1.5 m (3.3–5 ft) apart in both directions, so that the maximum block weight that can fall from between the bolts is ~2 t, or metric tons (~20 kN), well within the capacity of most mesh.

The detached blocks exert a pulling force, which is transmitted from the mesh to the bolts via the bolt plate. The role of the bolt plate is to transfer force to the mesh, maintain tension

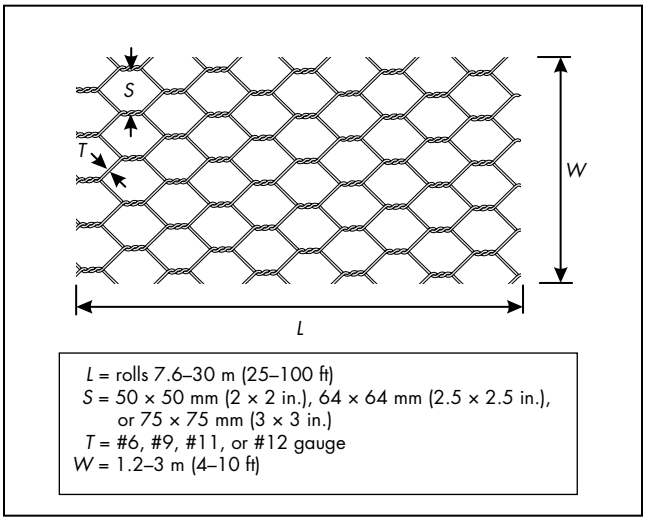
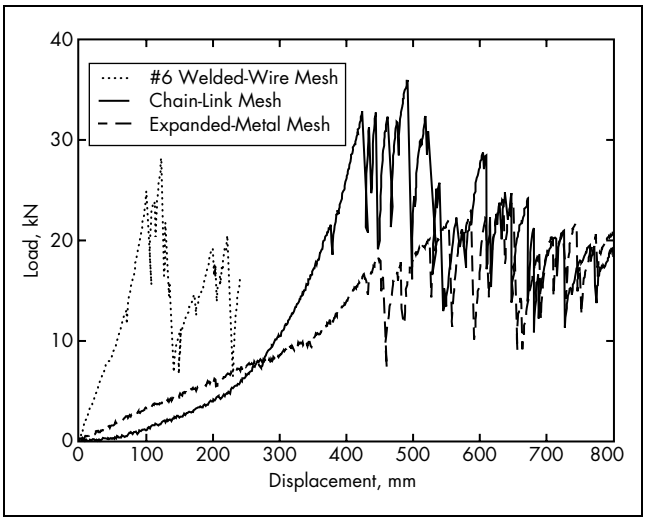


Figure 8.6-5 Chain-link mesh (typical dimensions)



Courtesy of Rock Australia Mining and Civil Pty Ltd.

Figure 8.6-6 Mechanized installation of chain-link mesh rolls



Source: Tannant 2004.

Figure 8.6-7 Load vs. displacement for three types of steel mesh

Table 8.6-1 Test results for three types of steel mesh, bolted in a 1.2-m (4-ft) diamond pattern

Mesh Type	Gauge	Number of Tests	Measurement*, mean (standard deviation) K, kN/mm	Do, mm	Dp, mm	Lp, kN
Welded-wire	#9	5	0.15	5	105	14.8
	#6	6	0.27	23	114	24.3
	#4	4	0.38	33	152	38.2
Chain-link	#9	4	0.24	273	417	34.4
Expanded-metal	N/A	5	0.06	129	595	24.6

Source: Adapted from Tannant 2004.

*Do = displacement offset; Dp = displacement at peak load; K = loading stiffness; Lp = peak load; N/A = not applicable.

in the bolts, keep the mesh and straps in place, and maintain the confinement of rock blocks beneath the collar.

When bolt plates are small, sharp, and square, the total force may be transmitted entirely through only a few wires, and the sharp edge of a plate can cut the wire. This situation can be remedied by using larger plates to distribute load to more wires or by using shaped plates to minimize cutting edges. Circular plates are available, as are other specially designed plates such as butterfly plates, which are large, thin, and deformable. The more popular bolt plates are shown in Figure 8.6-8 and their dimensions are listed in Table 8.6-2. It is generally accepted that domed plates provide the best contact for irregular rock surfaces. If the rock surface is very irregular, more flexible plates (6 mm [0.25 in.]) should be used.

The weakest link of the total support system can be the mesh, the plate, the connection between the plate and the mesh (e.g., due to plates cutting wires), or the rock bolt. Rock bolts and plates are designed to fail at comparable loads so as to avoid a weak link. However, this is true only under ideal tensile conditions. When the load has a strong shearing component, or a point load is applied to one component because of incorrect installation, early failure of the bolt heads or the plates can occur. The heads (ring) of friction bolts are particularly prone to early failure when bolts are installed at an angle to the rock surface or when they are hammered in too hard.

In squeezing ground or in rock burst conditions where the rock mass experiences large deformation, both the load capacity and the deformation capacity of the support system are tested. Comparison of the load and deformation capacities of mesh and bolts shows a clear mismatch in their properties. Grouted bolts are stiff and allow displacements of only a few millimeters before failure, but they have a load capacity in the range 150–200 kN. In contrast, mesh allows large displacements of several hundreds of millimeters but has a load capacity of only ~20–40 kN. As the ground deforms, mesh deforms with it but fails when either the local load exceeds 20–40 kN or the displacement exceeds 100–350 mm (4–14 in.). In these cases, mesh is incapable of transmitting loads that exceed its own load or displacement capacity to the bolts. A tighter bolting pattern in such cases divides the excavation surface into smaller areas between the bolts, resulting in lower local loads for the mesh to contain. Where ground displacement is considerable, bleeding the mesh and rehabilitating the ground support is a recurring issue.

Straps

Straps are generally used as surface support in special applications rather than as a part of a systematic support system in

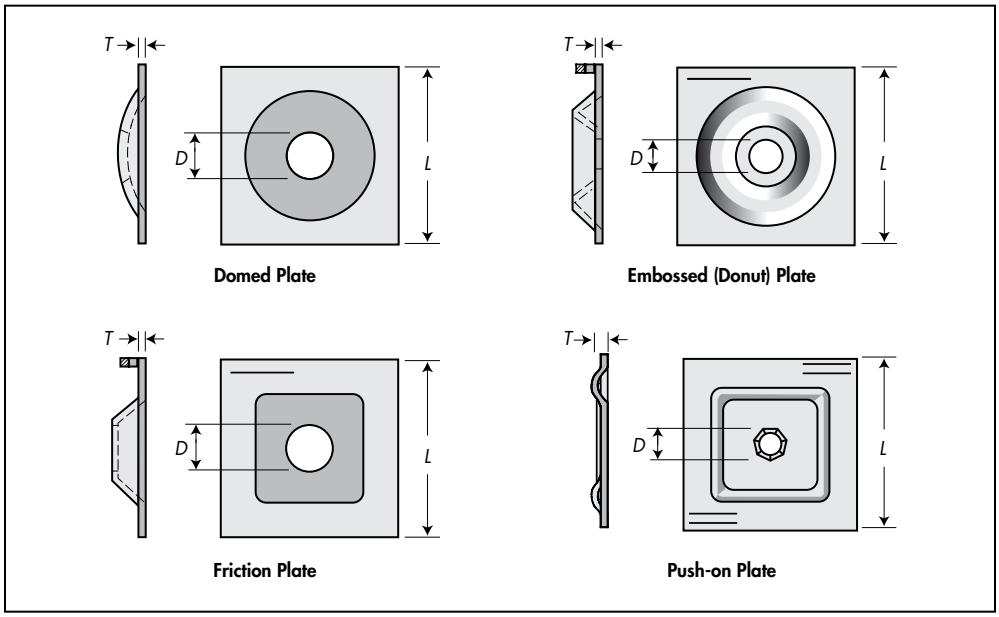


Figure 8.6-8 Common types of bolt plates for installing mesh

Table 8.6-2 Bolt plate dimensions

Plate Type	*L*, mm (in.)	*T*, mm (in.)	*D*, mm (in.)
Domed	125, 150, or 300 (5, 6, or 12)	6.4 or 9.5 (¼ or ⅜)	Variable
Embossed (donut)	150 or 200 (6 or 8)	21 to 29 (13⁄16 to 1⅛)	21 to 29 (13⁄16 to c⅛)
Friction	150 (6)	33-mm bolts: 4.8 (3⁄16)	33-mm bolts: 36 (1 7⁄16)
		39-mm bolts: 4.8 (3⁄16)	39-mm bolts: 41 (1⅝)
		46-mm bolts: 6.4 (¼)	46-mm bolts: 48 (1⅞)
Push-on	125 (5)	1.7 (0.068)	16, 19, or 22 (⅝, ¾, or ⅞)

mines. They can be installed either on their own against the rock mass surface by means of rock bolts or over mesh to provide extra strength and stiffness to the surface support. When used alone, they can support key blocks between rock bolts. However, if key blocks behind the straps loosen and fall, the strap no longer has good contact with the rock surface and may become ineffective in this area. Best results are obtained when the straps are installed tightly across the most prominent structures. Common practice is to wrap straps around pillar "noses" to prevent progressive deterioration of the pillars, particularly those submitted to vertical load. The three types of strap used in mining operations are steel straps, mesh straps, and Osro straps.

Steel Straps

Steel rock bolt straps are long rectangular bands of steel with preperforated holes through which rock bolts are inserted for installation purposes. Two types of steel straps are available: the plain steel strap (Figure 8.6-9) and the W steel strap (Figure 8.6-10). The plain steel strap is often thicker (up to 6 mm [0.25 in.]) and narrower (100 mm [4 in.]), and is variable in length (0.9, 1.2, 1.4, 1.8, or 2.4 m [3, 4, 4.6, 6, or 8 ft]) to accommodate different situations. It has elongated holes to facilitate installation. The W steel strap is thinner (up to 4 mm [0.15 in.]), has circular holes often spaced every half meter, and has shaped edges to increase rigidity and resistance to bending. It is available in variable widths (~125–330 mm [5–13 in.]) and lengths (~1.3–6 m [4–20 ft]).

It is sometimes difficult to achieve a quality installation with steel straps because the preperforated holes do not always line up with areas where rock bolt holes can be easily collared and drilled. As a result, the straps cannot always be installed very tightly against the rock mass and can therefore be less effective in containing key blocks.

Mesh Straps

Mesh straps are bands of heavy #0 gauge welded mesh, available in precut lengths of ~1.2–4 m (4–13 ft) (Figure 8.6-11). They are easy to install because bolts can be fitted at convenient locations through the mesh, and are therefore used increasingly in underground mines.

Mesh straps are particularly popular in difficult ground conditions, such as areas of squeezing ground or areas prone to rock bursts, where they are installed over welded-wire mesh to provide extra load-bearing capacity, limit bulging of the mesh, and facilitate load-sharing between bolts (Figure 8.6-12).

Osro (Oslo) Straps

The Osro (also known as Oslo) strap was developed for deep South African rock-burst-prone gold mines. It is strong and flexible for high-energy absorption. It is made of a series of (usually five) longitudinal, parallel, round, smooth bars, typically of 6–10 mm in diameter although larger diameters can be used. The longitudinal bars are attached together with strapping crossbars of 5.5 mm in diameter (Figure 8.6-13). The crossbars are not welded to the longitudinal bars but rather are loosely twisted around them so as to allow free rotational movement of the longitudinal bars. This adds to the ductility

and therefore the energy absorption capacity of the strap when it is submitted to large deformation or dynamic events. Strap widths vary from 250 mm to 1 m (~10 to 40 in.); lengths vary from 1 to 6 m (~3 to 20 ft).

Cable Lacing

A technique known as cable or rope lacing is commonly used in deep South African gold mines as a surface support in rock-burst-prone areas. It is also sometimes used to support drives in highly deformable kimberlite in diamond block caving mines in South Africa. The technique consists of installing destranded, degreased hoisting cables to form a cable web on the surface of an excavation, generally over wire mesh or alternatively over shotcrete (Figure 8.6-14). It allows for large deformation and provides high load-bearing capacity. Tests performed by Stacey and Ortlepp (2007) show it to have the highest dynamic energy absorption capacity of all surface support techniques, with excellent load-transfer capability to rock reinforcement.

In the late 1980s, some Canadian mines used cable lacing in specific rock-burst-prone areas. Although it was very effective for controlling rock burst damage, the technique was seen as being too labor-intensive and costly, and is seldom used in countries that practice mechanized mining.

SHOTCRETE

Shotcrete is concrete that is applied to the rock surface by pneumatic shooting at high velocity using specialized equipment. It is composed of 15%–20% cement, 30%–40% coarse aggregates, 40%–50% fine aggregates, and 2%–5% additives. Water is added to the mix to hydrate the cement and initiate the chemical reaction whereby the concrete gains strength. The water must be clean and free of chemical components that are potentially detrimental to the reaction. The water/cement ratio (expressed in units of liters per kilogram) influences cement hydration. The final product should be in the range 0.3–0.5 L/kg. Often steel or synthetic fibers are added to the mix to improve the post-peak behavior of the concrete; the resulting product is commonly called fiber-reinforced shotcrete (FRS) or Fibercrete.

The use of shotcrete as a surface support, as well as for construction applications in mines, has increased rapidly since the 1980s. The flexibility of the equipment, the choice of mixing methods, and the availability of shotcrete has contributed to its growing popularity as the preferred surface support method in underground mines. Shotcrete is often perceived as the surface support method of choice when the rock mass is heavily fractured. Its initial adhesion properties lend it a glue-like effect on heavily broken rock mass, which helps prevent the rock mass from crumbling. Contrary to the case for mesh, which contains blocks of rock only after they detach from the surface and therefore begins its action after the rock mass has failed, shotcrete provides immediate reactive support action at very low displacement, as soon as the rock mass starts to move and long before it reaches a yielding state. As a result, shotcrete is much more efficient at preserving the confinement within the rock mass surrounding an excavation.

In recent years, systematic application of shotcrete after each blast, also known as "in-cycle shotcreting," has gained popularity as an alternative to the so-called "campaign shotcreting" of an entire section of tunnel. In-cycle shotcreting has a number of advantages and disadvantages. Its advantages are as follows: (1) its rapid application minimizes time-dependent

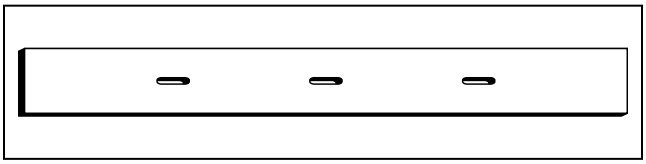
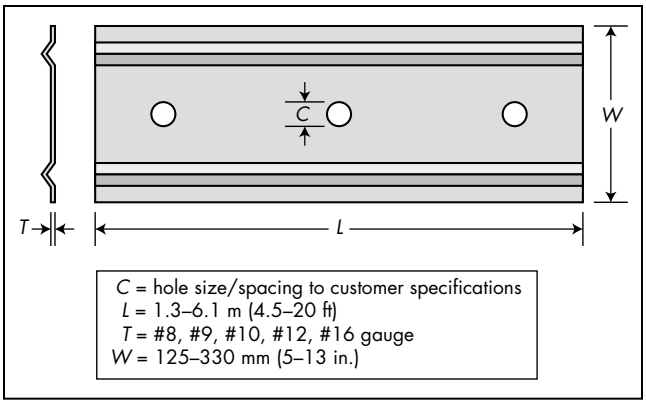


Figure 8.6-9 Plain steel strap



Source: Dywidag Systems International 2005.

Figure 8.6-10 W steel strap

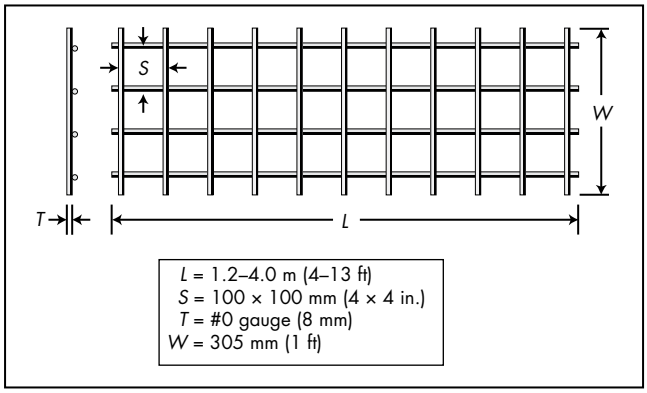
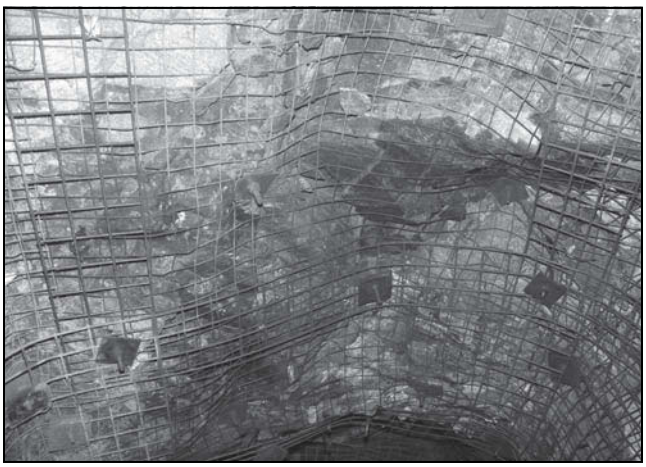
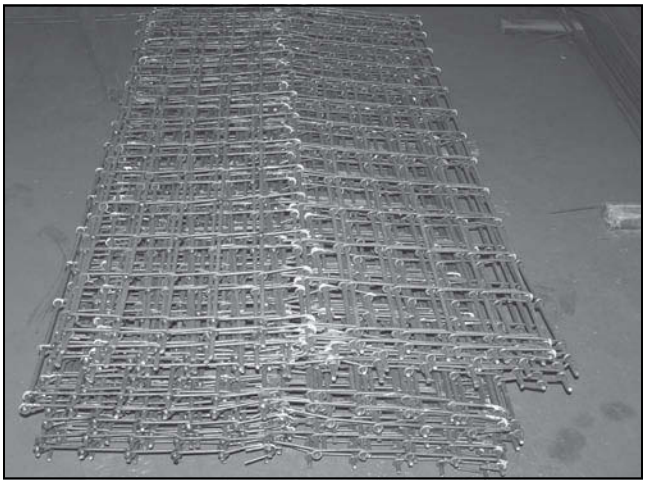


Figure 8.6-11 Mesh strap (typical dimensions)



Courtesy of the Australian Centre for Geomechanics.

Figure 8.6-12 Mesh straps installed over mesh



Courtesy of ATM Components Republic of South Africa.
Figure 8.6-13 Osro straps



Courtesy of the Australian Centre for Geomechanics.
Figure 8.6-14 Cable lacing in a South African mine

rock mass deterioration; (2) it allows for reinforcement to be installed after the shotcrete, by means of bolts that pin the shotcrete to the wall; and (3) it favors good load transfer between the surface support and the reinforcement. A disadvantage is that the rock mass is covered immediately after being exposed, leaving little opportunity for geological and geotechnical data to be collected. New data-collection techniques such as digital stereophotogrammetry are being used to overcome this problem.

Two main techniques exist for applying shotcrete in mines: dry-mix and wet-mix. In the dry-mix technique, water is added to the mix only at the nozzle. Advantages and disadvantages of the technique are as follows (based on ACI Committee 506 [2005] and the authors' experience).

- Advantages of the dry-mix technique:
  - Maximum flexibility in delivery in terms of time, because curing begins only when the shotcrete is applied
  - Flexibility in terms of transportation, because shotcrete ingredients can be carried even long distances to the workplace in premixed bulk bags, and the necessary equipment (including mixers, pumps, hoses, and more) is compact and mobile; this flexibility makes this method very popular in deep mines that are accessible only by shaft
  - Instantaneous control over mixing water and consistency of the mix at the nozzle to meet variable field conditions
  - High suitability for placing mixtures containing lightweight aggregates or refractory materials
- Disadvantages of the dry-mix technique:
  - Large percentage of rebound
  - Dusty environment created
  - Low application productivity
  - Low volume per hose size

In the wet-mix technique, shotcrete mix is prepared at a custom-built concrete plant often located on the surface but sometimes underground, or from a local supplier. Shotcrete is transported by agitator truck from the plant to the point of application. Alternatively, a slick-line can be installed to alleviate some of the transportation difficulties of bringing underground shotcrete and other concrete used for construction. Advantages and disadvantages of the technique are as follows (based on ACI Committee 506 [2005] and the authors' experience).

- Advantages of the wet-mix technique:
  - Generally minimal quality control issues, because mixing is done in a controlled plant environment
  - Relatively low sensitivity to the skills of the nozzle operator
  - High delivery rates
  - Low percentage of rebound
  - Minimally dusty environment
  - Ease of mechanization
  - Control of mixing water at the delivery equipment, where it can be accurately measured
  - Good assurance that mixing water mixes thoroughly with other ingredients
  - Minimal dust and cementitious materials lost during the shooting operation
  - Normally low rebound, resulting in less waste
  - High volume per hose size
- Disadvantages of the wet-mix technique:
  - Transport and delivery logistics are difficult in deep mines having shaft access only
  - Equipment is more expensive
  - Not suitable for small-volume applications

A prerequisite for the successful application of shotcrete is the use of properly operated and maintained equipment by qualified personnel. In general, dry-mix equipment consists of either single- or double-chamber guns (batch) or rotary continuous-feed guns. With batch guns, a charge of material is placed in the chamber and the chamber is closed and pressurized, feeding the material into a delivery pipe or hose. With double-chamber guns, the upper chamber serves as an airlock during the charging cycle to allow for continuous operation. Rotary guns use the rotating airlock principle to supply a continuous feeding action.

Wet-mix applications make use of pneumatic feed guns. Slugs of material are introduced into the delivery hose and compressed air is added at the discharge sump at the nozzle to increase the velocity of the mixture. Positive displacement

guns can also be used. A solid column of shotcrete is forced by mechanical, air, or hydraulic pressure through a hose in a continuous stream to the nozzle. Air is injected at the nozzle to break up the stream and increase exit velocity.

Basic Components of Shotcrete

The main components of shotcrete—cement, aggregates, water, admixtures, and potential cementitious additions—are described in Vlietstra (2009) and ACI Committee 506 (2005).

Cement

The binding agent in shotcrete is cement, quite often an ordinary portland cement. Portland cement contains lime, silica, alumina, and iron oxide. Lime is the main component; the other three are included to balance the chemical composition of the cement. To control the initial reaction of the cement with water (the hydration process), gypsum is also added. The chemical reaction of hydration is complex and beyond the scope of this chapter. It is important to note, however, that the process is time-dependent, being slower during the first few hours. It is also temperature-dependent, being faster at higher temperatures. The other factor affecting the hydration process is grind size, which is often measured in terms of surface area and quantity of 45-μm residue. Finer grind facilitates more rapid hydration and earlier strength gains.

Aggregates

Aggregates form the matrix of the shotcrete and are bound together by a cementitious paste to become concrete. Both coarse and fine aggregates are used. The combined aggregate should generally comply with one of the gradations listed in Table 8.6-3. Maximum aggregate size is dictated by pumping limitations and the need to minimize rebound. Coarse particles can block the nozzle and subsequent cleaning can be time-consuming. Larger particles tend to rebound when sprayed on a hard surface. They can also penetrate already placed shotcrete and produce craters that are difficult to fill (Melbye et al. 2001). A guideline to avoid segregation and clogging is to ensure that ≤30% passes through a 0.3-mm sieve. If the proportion of fine material exceeds this, more water is required for hydration.

Aggregates should be clean—that is, free of dirt, chemical impurities, and organic material. The mineralogical composition of the aggregate should be inert under normal mining conditions, and free of sulfides and other reactive minerals. Equidimensional aggregate particles generally give better results than do those with elongated or platy shapes, and a rough surface assists cement paste bonding. Aggregates must be at least as strong as the specified concrete final product.

Water

Mixing and curing water should be clean and free of material that can be detrimental to concrete. If potable water is not available, the water must be tested to ensure that the strength of mortar cubes made with it is ≥90% of the strength of mortar tubes made with distilled water.

Admixtures

Admixtures are chemicals that are put into concrete to affect the mixing, placing, and curing processes. Typical admixtures are accelerators, air-entraining agents, and pozzolans.

For wet-mix shotcrete, a consistency corresponding to a target slump of ~150 mm (~6 in.) facilitates mixing, pumping, and shooting. In a slump test, a concrete sample is placed in a metal slump cone of height 300-mm (12-in.), bottom diameter of 200 mm (8 in.), and upper diameter of 100 mm (4 in.). The concrete is tapped down following ASTM C143/C143M 2009 procedures and the cone is lifted vertically to remove it without disturbing the sample. The sample is allowed to slump until it stabilizes. The slump height is the distance that the sample settles below its original 300-mm (12-in.) height.

Table 8.6-3 Gradation limits for combined aggregates

Sieve Size (U.S. Standard square mesh)	% by Weight Passing Individual Sieves	
	Gradation #1 (fine-aggregate grading)	Gradation #2 (coarse-aggregate grading)
19 mm (¾ in.)	—	—
12 mm (½ in.)	—	100
10 mm (⅜ in.)	100	90–100
4.75 mm (No. 4)	95–100	70–85
2.4 mm (No. 8)	80–98	50–70
1.2 mm (No. 16)	50–85	35–55
600 μm (No. 30)	25–60	20–35
300 μm (No. 50)	10–30	8–20
150 μm (No. 100)	2–10	2–10

Source: ACI Committee 506 (2005).

Because consistency and slump are controlled mainly by the water/cement ratio, which in turn influences the final strength of the shotcrete, water-reducer admixtures (plasticizers and superplasticizers) are often used to achieve the desired consistency for shotcrete application. Hydration controllers are also often required to delay hydration and accommodate the time required for the wet mix to be transported from the plant to the working face. Without admixtures, shotcrete shelf life would be only 1 to 2 hours, depending on ambient temperature. Hydration controllers can extend shelf life up to 3 days without affecting the short-term or long-term strength of the shotcrete.

An accelerator can be added at the nozzle to cancel the retarding effect of the hydration controller. For both dry-mix and wet-mix techniques, accelerators are required to obtain early adhesion to the rock surface and induce rapid gains in shotcrete strength. Accelerators are also used to apply thick layers of shotcrete in a single pass. Unfortunately, however, accelerators tend to reduce the final strength of shotcrete. Accelerator dose is generally expressed as a percentage of the total weight of cementitious content. Doses are typically 2%–5% for dry mix and 3%–10% for wet mix. A number of accelerators are available and performance testing is strongly recommended prior to product selection because local cement can respond differently to accelerators. Noncaustic alkali-free accelerators are gaining in popularity because they are relatively environmentally friendly and induce rapid gains in shotcrete strength with minimal long-term loss of strength.

Cementitious Additions

It can be advantageous to replace a portion of the cement with pozzolanic material such as microsilicia (silica fume) or fly ash. If these materials can be obtained at reasonable cost, they can accrue economic benefits and improve some of the short-term and long-term properties of shotcrete.

Silica fume is an ultrafine material (particle diameter of 0.1–0.2 μm) whose spherical particles have high pozzolanic

properties. In the short term, it has a lubricating effect on the mix, increasing flowability and reducing equipment wear. It also minimizes rebound and aids adhesion to rock. In the long term, it improves shotcrete strength and durability.

Fly ash is a coarser material (particle diameter of <50 μm). Compared to silica fume, it is generally less expensive but also less effective in improving shotcrete properties.

Reinforced Shotcrete

Concrete (including shotcrete) is a brittle material with high compressive strength but low tensile and flexural strength. The failure mechanism for shotcrete is discussed in the "Shotcrete Stabilization Mechanisms" section. The way it is applied in underground mining causes it often to be subject to bending and flexural forces. Therefore, some type of reinforcement is required in most mining applications. Two common types of reinforced shotcrete are used in mines: mesh-reinforced shotcrete and FRS. Both are widely used and have a number of variations.

Mesh-Reinforced Shotcrete

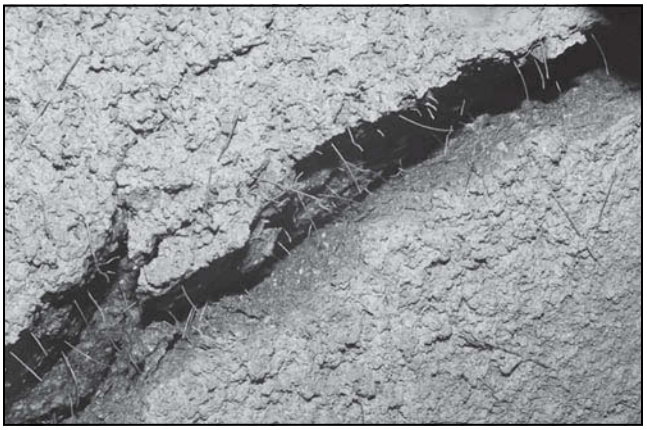
In the early days of shotcrete application in Canadian mines in the 1980s, it was common practice to apply shotcrete over mesh as a reactive action, particularly when ground conditions were deteriorating. The results were often very good and acceptance of the technique was rapid. The technique then became proactively used as a high-capacity surface support system, applied in areas where difficult conditions were anticipated. Mesh-reinforced shotcrete remains widely used in Canada and South Africa.

In extreme conditions, mine operators have tried to embed welded-wire mesh inside shotcrete by placing a layer of shotcrete first, installing mesh over this first layer, and then placing a second layer of shotcrete. The result is a thick shotcrete liner that can maintain a good proportion of its load-deformation capacity after the shotcrete cracks. However, when deformation reaches 10–20 mm, the wires in the mesh, which are point-loaded by the shotcrete crack, tend to fail in tension.

An alternative approach that takes advantage of the ductility of the mesh is to install it as the final layer over the shotcrete—preferably FRS. Then, when the shotcrete cracks, it is contained by the mesh, which is then free to deform to its full capacity of 100–350 mm (4–14 in.). Installing mesh over FRS tends to be the preferred option in Australian mines facing difficult ground conditions.

Fiber-Reinforced Shotcrete

Fiber-reinforced shotcrete is also called FRS or Fibercrete. The addition of steel or synthetic fibers improves the postfailure behavior of shotcrete when it is submitted to bending or flexural forces. Fibers have no effect until the ultimate strength of the shotcrete is reached and cracks start to propagate through the shotcrete layer. The fibers then bridge and prevent further opening of the cracks, so that a significant fraction of the load-displacement capacity of the yielding shotcrete is maintained (Figure 8.6-15).



Courtesy of the Australian Centre for Geomechanics.

Figure 8.6-15 Fiber-reinforced shotcrete, showing fibers bridging cracks

To fulfill their role, shotcrete fibers must have both high tensile strength and a strong bond with the shotcrete. A strong bond is achieved with hooks or cavities.

Two factors particularly affect the bridging performance of fiber. The first factor is the length/diameter aspect ratio of the fiber. In calculating this ratio, the diameter can be taken as the equivalent diameter of a circle whose area is equal to the cross-sectional area of the fiber. A high aspect ratio contributes to better bridging. The second factor is the fiber dosage. Greater numbers of fibers that can bridge cracks contribute to better bridging.

Several types of steel and synthetic fiber are available (Figures 8.6-16 and 8.6-17). Synthetic fibers are usually made of organic polymers that are small enough to permit even distribution in the sprayed mixture during mixing and spraying. Fibers with equivalent diameters of >0.3 mm are called *macrofibers*; fibers with equivalent diameters of <0.3 mm are called *microfibers*. Typical macrofiber aspect ratios are in the range 40–65 for common fiber lengths of 19 to 50 mm (0.75 to 2 in.). Steel fiber lengths are generally <38 mm (1.5 in.); synthetic microfiber lengths vary from 6 to 50 mm (0.25 to 2 in.) (ACI Committee 506 [2008]).

Originally, steel fiber was used in FRS applications for both civil tunneling and mining. Recently, synthetic fiber has been gaining in popularity in civil tunneling and in Australian underground mines. Among the relative advantages of synthetic over steel fiber is its greater resistance to corrosion, lower abrasiveness, and significantly lighter weight. Lighter weight allows for a significantly higher dosage of fibers. Similar performances are obtained with one-fourth the weight of synthetic fiber (which is also twice the volume) as compared with steel fiber. The elastic modulus of synthetic fiber (3,400–4,800 MPa [~500,000–700,000 psi]) can be ~50 times less than that of steel fiber. With more and stiffer fibers per unit volume, synthetic fiber locks cracks earlier and transfers load to the concrete better, promoting preferential formation of multiple local cracks rather than a single crack opening progressively. The main disadvantage of synthetic fiber is that it floats and can plug the pumps of underground sumps or cause problems in flotation cells.

Bernard (2004) investigated the creep behavior of both steel and synthetic fibers and concluded that they have similar resistance to load deformation. Creep is an issue when shotcrete cracks. With steel-fiber-reinforced shotcrete, if a crack develops, there is potential for corrosion of the fibers and loss of ductility. Load energy continues to be absorbed but there is a risk of sudden rupture. In contrast, with synthetic-fiber-reinforced shotcrete, development of cracks is not associated with corrosion. If the load continues, synthetic fibers stretch rather than suddenly fail, and continue to absorb load energy.

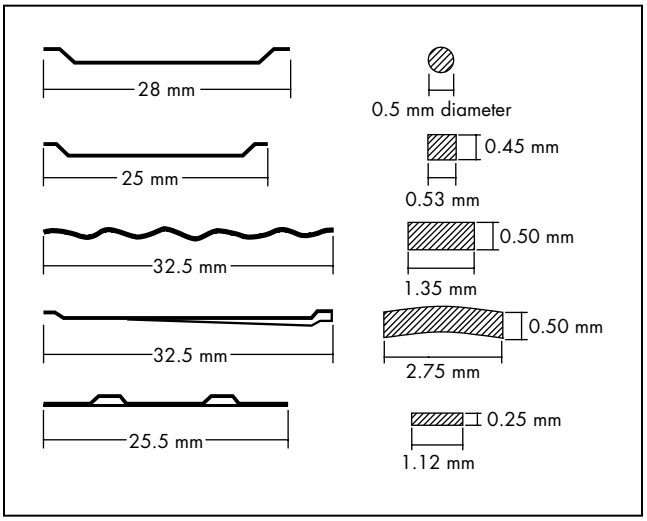
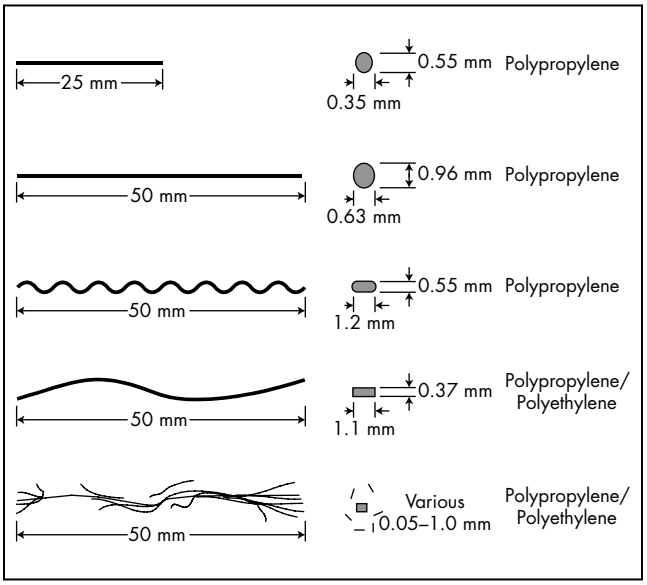


Figure 8.6-16 Types of steel fiber available in North America



Source: Trottier and Mahoney 2004.

Figure 8.6-17 Typical synthetic fibers

This provides an opportunity to take remedial action before the capability of the reinforced shotcrete is surpassed. Table 8.6-4 lists typical recipes for dry and wet mixes.

Often, a certain fiber dosage and a certain compressive strength are specified for a shotcrete application. However, it is more appropriate to specify the required shotcrete performance under flexural load, which is a measure of toughness or energy absorption capacity. This capacity is expressed as the surface under the load-displacement curve, measured in joules, during flexural tests. The EFNARC test (EFNARC 1996) and the round determinate panel (RDP) test (Bernard and Pircher 2001) are accepted methods for measuring shotcrete toughness. The RDP test has become the more accepted of the two tests in recent years because it generates more repeatable results. In the RDP test, a panel is supported at three points spaced 120° apart. A point load applied at the center of the panel is measured as a function of deflection.

Table 8.6-4 Typical steel-fiber-reinforced silica-fume shotcrete mixes

	Wet Mix		Dry Mix	
Component	kg/m³	% wet materials	kg/m³	% dry materials
Portland cement	420	18.1	420	19.0
Silica fume additive	40	1.7	50	2.2
Aggregate	1,600	68.9	1,670	75.5
Steel fibers	60	2.6	60	2.7
Accelerator	13	0.6	13	0.6
Superplasticizer	6 L	0.3	—	—
Water reducer	2 L	0.1	—	—
Air-entraining admixture	If required		—	—
Water	180	7.7	Controlled at nozzle	
Total	2,321	100	2,213	100

Source: Wood 1992.

The geometry of the test is such that failure always occurs between the points of support, and hence enables test repeatability (Figure 8.6-18).

The standard procedure is to consider the energy absorbed at a maximum deflection of 40 mm (1.6 in.) for a specimen of standard size (800-mm [31-in.] diameter, 75-mm [3-in.] thickness; Bernard and Pircher 2001). For a specimen of thickness other than 75 mm, the following series of correction equations is used to scale the load and energy capacity of the panel so as to be equivalent to those of a panel of standard dimensions.

$$W = W'\left(\frac{t_o}{t}\right)^{\alpha}\left(\frac{d_o}{d}\right) \tag{42-1}$$

$$\alpha = 2.0 - \frac{\delta}{80} \tag{42-2}$$

where (all dimensions are in millimeters)

W = "corrected" energy absorption
W' = energy absorption of the specimen of nonstandard dimensions
t_o = thickness of the standard specimen
t = thickness of the nonstandard specimen
d_o = diameter of the standard specimen
d = diameter of the nonstandard specimen
δ = central deflection of a panel, taken to be 0 at cracking

Surface Preparation, Profile, and Shotcrete Thickness

Good rock-surface preparation before shotcrete application involves removing or scaling down loose rocks and cleaning the rock surface of dirt, grease, residues from blasting and diesel exhaust, and more. Kuchta et al. (2004) recommend the use of high-pressure water jets for scaling down loose rocks in mine openings in order to maintain mine safety. Good preparation improves the adhesion strength of the shotcrete, usually defined as the strength of the interface between two materials—in this context between rock and shotcrete.

Malmgren et al. (2005) reported the results of an extensive failure-mapping investigation in 7 km (4.4 mi) of shotcreted drifts at the Kiirunavaara mine in Sweden. Shotcrete was found most likely to fail where the shotcrete layer is thin (<20 mm) and adhesion to the rock is poor to nonexistent. The main factors that contribute to shotcrete failure are rock deformation and dynamic loads due to blasting.



Courtesy of the Australian Centre for Geomechanics.

Figure 8.6-18 Round determinate panel test for measuring shotcrete performance under flexural load

Table 8.6-5 Determination of the roughness factor based on ground condition and drill-and-blast process

Ground Condition		Drill-and-Blast Process			
		Perfect (half barrels throughout)	Good (half barrels across back)	Fair (moderate overbreak)	Poor (considerable overbreak)
	Factor	1.05	1.10	1.15	1.25
Excellent (designed surface area)	1.00	1.05	1.10	1.15	1.25
Good (slightly rough surface)	1.20	1.26	1.32	1.38	1.50
Fair (broken surface)	1.40	1.47	1.54	1.61	1.75
Poor (very blocky ground)	1.60	1.68	1.76	1.84	2.00

Source: Adapted from Wood 1999.

Windsor and Thompson (1999) argued that traditional shotcrete layers of uniform thickness are suitable only for regular and smooth or irregular and evenly coated excavations. They suggested that, depending on the excavation profile, different cover techniques can be used. Wood (1999) built upon this work, developing a methodology for estimating the volume of shotcrete required for any particular project. Such estimates are of particular interest for operations in which shotcreting is contracted out and it is necessary to determine the volume of shotcrete based on the design profiles of excavations.

The estimated volume is based on roughness and rebound. The roughness factor is determined by multiplying the rating that represents the irregularity of the surface of the ground and the rating that represents the quality of the drill-and-blast process (Table 8.6-5). In excellent ground conditions, the final perimeter of the excavation is according to the design specifications. Good ground conditions are characterized by the presence of joints and slightly rough surfaces. Fair ground conditions are characterized by blocky ground and smooth surfaces. Finally, in poor ground conditions with very blocky ground, polished surfaces, possible fault zones, and wet ground, the rock mass has a significant influence on the perimeter of the excavation and is assigned a higher ground-condition factor. The quality of the drill-and-blast process is characterized by the degree of overbreak. A good drill-and-blast process, defined by half barrels across the back and some additional overbreak beyond design, would be assigned a drill-and-blast rating of 1.10. If this was observed in good ground conditions with a ground-condition rating of 1.20, the resulting roughness factor would be 1.32.

The roughness factor and the rebound factor can be used to calculate a specific volume factor for any particular mining project (Table 8.6-6). Rebound varies during the course of shotcrete shooting; it is high at the beginning of the shoot and decreases with increased shotcrete thickness. A site-specific chart can be developed to quantify cumulative rebound with respect to average thickness of shotcrete placed based on shotcrete trials. This can be used to establish a rebound factor. Wood (1999) presents an example in which the design calls for a minimum shotcrete thickness of 25 mm (1 in.), so an average of 50 mm (2 in.) of shotcrete is probably needed to provide the desired cover, associated with a minimum of 15% rebound. Similarly, a minimum design thickness of 50 mm (2 in.) requires an average thickness of 75 mm (3 in.) and the rebound is lower, resulting in a lower rebound factor. The rebound factor can then be multiplied by the roughness factor obtained from Table 8.6-5 to generate a site-specific volume factor.

Gaps in the table indicate situations that are unlikely to occur in practice. For example, it is unlikely that a rock mass with poor roughness would require only 50 mm (2 in.) of shotcrete. Similarly, there is little justification for 100 mm (4 in.) of shotcrete in a high-quality, well-excavated rock mass.

A practical consideration in rapid-development headings is the time required after shotcrete application before it is safe to reenter under newly secured ground. Clements (2004) reviewed the following tests to determine in situ early strength of shotcrete: soil penetrometer test, Meyco needle penetrometer test, sprayed-beam compression test, Hilti nail-gun pull-out test, and standard cylinder uniaxial compressive strength test. Although the sprayed-beam compression test is the most reliable technique for determining in-situ strength at the face, the tool is difficult to handle and unlikely to be used for daily quality control. Clements suggested that the Meyco needle penetrometer may be the more useful, albeit conservative, tool.

Bernard (2008) argued that traditional empirical design methods for FRS are based on term-of-life performance and are not necessarily applicable for early-age shotcrete. Early-age shotcrete fails in shear in response to applied loads from individual loose rocks. As shotcrete strengthens, its failure mode transitions from shear to delamination from the substrate and flexural failure. Thus the relationship between shear and compressive strength over the first few days of strength gain differs from that for mature shotcrete. Bond strength between FRS and the underlying rock surface increases to a maximum

Table 8.6-6 Determination of the volume factor based on roughness and thickness rebound factors

Roughness Factor*		Average Thickness Rebound Factor		
		50 mm (2 in.)	75 mm (3 in.)	100 mm (4 in.)
	Factor	1.15	1.10	1.08
Excellent	1.05	1.2	1.5	—
Good	1.32	1.5	1.8	—
Fair	1.61	—	—	1.8
Poor	2.00	—	—	2.2

Source: Adapted from Wood 1999.
*Obtained from Table 8.6-5.

of 0.2 MPa (2 bar or 29 psi) at 24 hours. These factors must be considered in order to establish safe reentry times.

Dynamic Testing of Surface Support

The EFNARC and RDP tests are quasi-static tests, where load is applied to the sample at a relatively slow rate. Because deep underground mines often encounter seismically active conditions where loads impact the support system via dynamic shockwaves, it is important to understand the response of surface support under dynamic loading.

Stacey and Ortlepp (2007) summarized results from dynamic testing programs performed in South Africa using a drop-weight testing apparatus. In this test, the dynamic-energy impact is delivered through a pyramid of bricks to distribute load and simulate the broken rock mass surrounding deep South Africans mines. Table 8.6-7 shows the average dynamic-energy impact and displacement values at failure for different surface support systems. It is important to realize that a significant amount of energy dissipates in the brick pyramid and only a fraction (~50%, according to Human 2004) is absorbed by the support system. An advantage of this test is that it examines the complete support system, including interaction of the reinforcement (1 × 1 m bolting pattern using both stiff and yielding bolts) with the surface support.

Player et al. (2008) conducted a series of dynamic tests on 1.3 × 1.3 m (50 × 50 in.) panels of two types of mesh: galvanized wire mesh (5.6-mm diameter) welded into a 100-mm (4-in.) square grid pattern and high-strength steel-wire chain-link mesh (4-mm diameter). The average dynamic rupture displacement for welded-wire mesh is 200 mm (8 in.) and for chain-link mesh is 300 mm (12 in.). The average dynamic rupture force for welded-wire mesh is 55 kN and for chain-link mesh is 185 kN. The average dynamic rupture energy for welded-wire mesh is 2 kJ and for chain-link mesh is 9 kJ. Comparisons among different testing programs are difficult because of the different testing methods.

Shotcrete Stabilization Mechanisms

In its original tunneling applications, shotcrete was generally designed to work as a compressive arch, taking advantage of the high compressive strength of concrete. Tunnels were generally either circular or horseshoe-shaped, promoting strong arching action of the shotcrete layer. The surface support layer was often applied floor to floor or all around the tunnel, allowing for good load transfer from the roof and wall to the floor (Figure 8.6-19).

In mining applications, however, because of cost and time constraints, the short-term nature of excavations, and the fact that the general public is not exposed, the application

Table 8.6-7 Energy-absorbing capacities and deformation limits of surface support lining components and systems

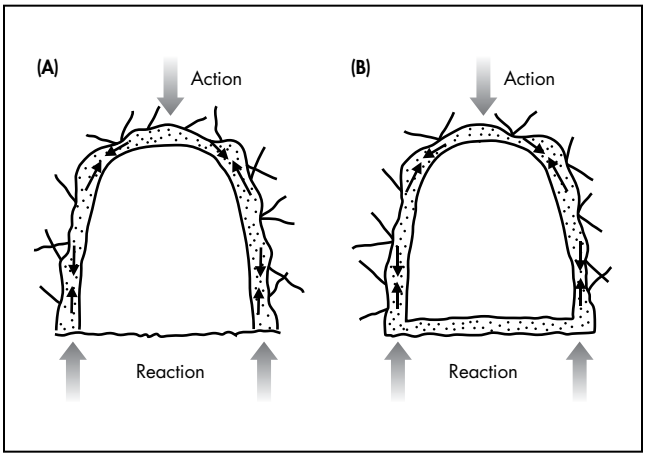
Membrane Component/System	Input Energy Capacity, kJ	Estimated Capacity with Yielding Rock Bolts, kJ	Characteristic Deformation Limit, mm
Unreinforced shotcrete	6	6	50
Weld mesh	10	10	100
Diamond mesh	15	15	220
Special wire mesh*	~35	~45	~400
Weld-mesh-reinforced shotcrete	15	15	150
Dramix-fiber-reinforced shotcrete	20	20	100
Monofilament-polypropylene-fiber-reinforced shotcrete	15	15	120
Weld mesh and wire rope lacing†	50	70	400
Diamond mesh and wire rope lacing†	35	45	300
Special mesh and wire rope lacing†	50	70	450
Fiber-reinforced shotcrete and wire rope lacing†	35	45	170

Source: Stacey and Ortlepp 2007.
*Mesh with yield capability available commercially or in which yield capability has been introduced.
†Wire rope lacing has some inherent yield capacity.

of shotcrete differs considerably from the case for tunneling applications. The floor is very rarely supported. Surface support sometimes does not extend all the way to the floor; or when it does, the layer can be very thin. As a result, the compressive strength of shotcrete is not really brought into play (Figure 8.6-20). Surface support is provided through the adhesion and/or shear strength between rock and shotcrete, the flexural strength of the shotcrete, and load transfer to the rock reinforcement.

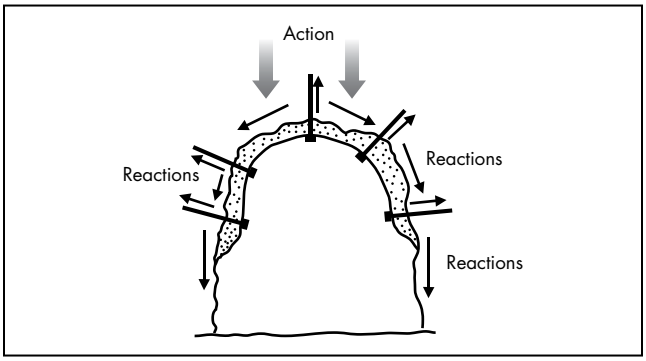
Stacey (2001) has identified the various support mechanisms provided by shotcrete. These mechanisms can occur individually and in combination:

- **Promotion of block interlock.** The immediate reactive support generated by the bond between the liner and the rock mass promotes block interlocking and prevents loosening of the rock mass. A number of submechanisms take place in this process:
  - Interlock due to bonding of shotcrete to the rock and the tensile strength of the shotcrete (Figure 8.6-21A). Shear resistance between the rock mass and the shotcrete is produced by the bonding, restricting blocks from moving, rotating, or loosening.
  - Development of shear strength on the interface between shotcrete and rock (Figure 8.6-21B). This is the case in poorly bonded shotcrete.
  - Penetration of shotcrete into joints and cracks (Figure 8.6-21C). Penetration of shotcrete into surface cracks inhibits the movement of blocks.
  - Applying a thin layer of liner to contain stress-induced spalling (Figure 8.6-21D). A layer of liner applied at an early stage of fracturing inhibits the fracturing process. This is particularly relevant in high-stress



Source: Clements 2009.

Figure 8.6-19 Shotcrete in compression after application either (A) to the tunnel floor or (B) all around the tunnel



Source: Clements 2009.

Figure 8.6-20 Shotcrete in compression after application that terminates above the tunnel floor as in a typical mining operation

situations where some loosening of the blocks would have already occurred.
  - Applying a thick shotcrete layer (Figure 8.6-21E). The internal shear strength of the shotcrete prevents larger failure.
  - Applying a thin shotcrete layer (Figure 8.6-21F). Adhesion between the shotcrete layer and the rock fails locally, causing the tensile or flexural strength of the shotcrete to be solicited to resist movement and loosening.
- **Basket mechanism** (Figure 8.6-21G). When the rock mass behind the shotcrete is crushed and is in a yielding state, the action of the shotcrete layer becomes similar to that of mesh and is required to contain the broken rock mass, somewhat like a basket. In this situation, the toughness of the shotcrete becomes critical to resist deflection of the broken rock mass, which can be exacerbated by stress changes due to stoping nearby. Because the tensile strength of concrete is low, the liner is expected to crack early and therefore must rely heavily on reinforcement material such as mesh or fibers to contain the deflection.
- **Structural arch** (Figure 8.6-21H). The arched shape solicits the compressive strength of the shotcrete to resist movement. Also contributing to the resisting forces is the shear strength between the shotcrete and the rock surface. If only the crown of the arch is supported, as is often the case in mining, the support mechanism relies on load transfer between the shotcrete and the reinforcement, as well as the adhesion, shear strength, and flexural strength of the shotcrete.
- **Slab enhancement** (Figure 8.6-21I). Laminated rock masses, particularly when strong foliation or bedding runs parallel to one of the excavation surfaces, may fail due to buckling. A shotcrete layer effectively thickens the slab, decreasing its slenderness and increasing its resistance to buckling.
- **Extended faceplate** (Figure 8.6-21J). The application of shotcrete extends the area of influence of rock bolt faceplates. It also enhances load transfer from the excavation surface to the reinforcement. Installing bolts after the shotcrete is also highly desirable because if the layer cracks, the bolt can then pin the broken shotcrete pieces to the rock mass and prevent them from falling off and becoming a hazard. The effect of an extended faceplate is lost where shotcrete is applied over mesh or FRS is applied after bolting.
- **Surface protection.** Certain rock mass may be prone to deterioration when subjected to wetting or drying. Shotcrete can act as a protective layer against the environment or mechanical deterioration from blast vibration, equipment collision, and so forth.

Shotcrete Failure Mechanisms

From the stabilization mechanisms just described, Stacey (2001) derived the corresponding failure mechanisms commonly observed in shotcrete (Figure 8.6-22).

Wedge and Block Loading

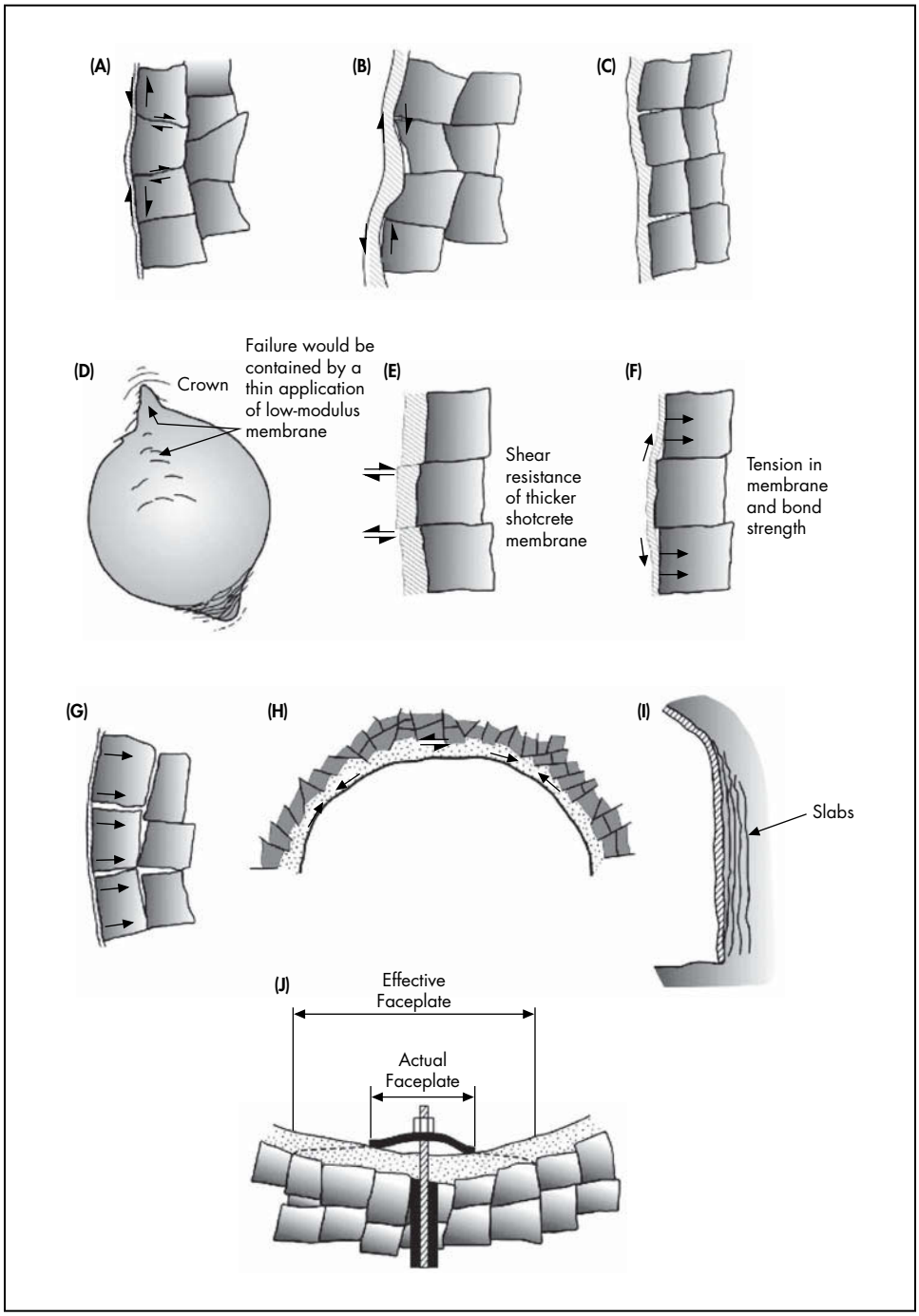
Discrete blocks or wedges defined by local discontinuities can move because of stress and/or gravity and load the shotcrete layer at specific locations (Figure 8.6-22A). As noted previously, if the shear strength and adhesion between the rock mass and the shotcrete holds, failure is likely to occur by shearing of the shotcrete layer. If the shotcrete layer debonds, flexural failure of the shotcrete is the likely mechanism.

Distributed Surface Loading

Distributed loading of the surface can be due to a number of rock-mass-yielding mechanisms, such as gravity acting on failed rock, squeezing ground under high stress, or dynamic loading from seismic events. This mechanism is related to the basket mechanism (Figure 8.6-22B). The surface support must work with the rock reinforcement as a system and, unless a good arch is formed by the liner, failure is likely to be by flexural bending.

Stress-Induced Loading

If a strong bond is achieved between the shotcrete liner and the rock mass, the two deform as a unit. Because concrete is brittle, a number of possible stress conditions can develop within the shotcrete layer, resulting in different modes of failure, including the following: shear failure (Figure 8.6-22C), bending failure in the roof (Figure 8.6-22D), and buckling failure in the walls as a result of floor heaving (Figure 8.6-22E). Complex failure



Source: Stacey 2001.

Figure 8.6-21 Surface support stabilization mechanisms

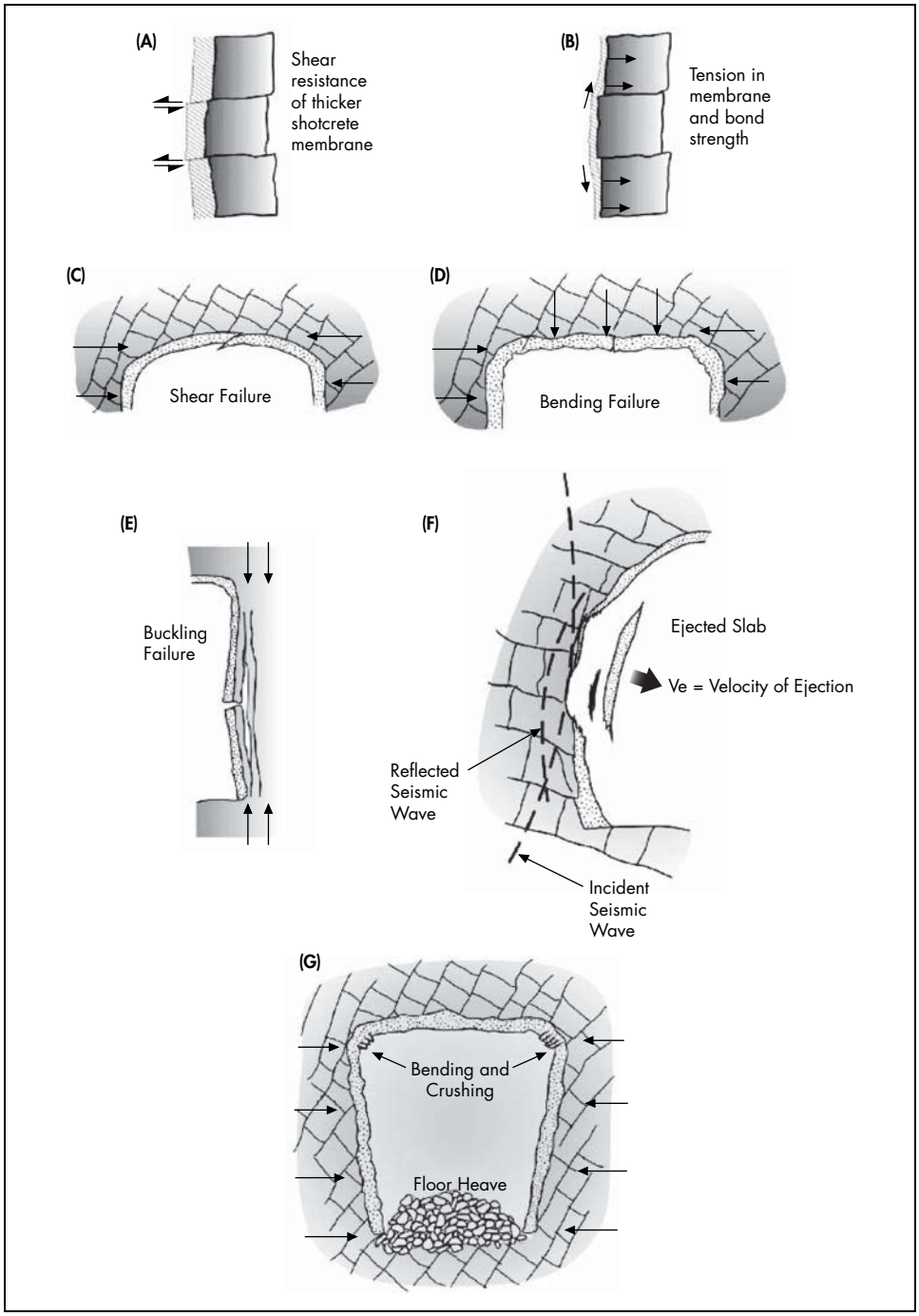
modes involving combinations of the above are also possible. In rock-bursting conditions, a common failure mechanism for shotcrete is slab ejection (Figure 8.6-22F). If a shotcrete shell is not properly drained in areas where groundwater is flowing, water pressure can build up and cause flexural failure. In high horizontal stress conditions where significant floor heaving and wall convergence are observed, the shotcrete shell will bend with the inward wall displacement, resulting in the crushing of the shotcrete at the corners of the excavation (Figure 35-22G).

Design of Shotcrete Liners

Load and failure mechanisms involving liner support can obviously be quite complex. Shotcrete liners can be designed either empirically or deterministically.

Empirical Design Approaches

A reasonable starting point to an empirical approach is to consider the common practice of applying reinforced shotcrete of thickness 50 mm (2 in.) (or 100 mm [4 in.] if the excavation



Source: Stacey 2001.

Figure 8.6-22 Surface support failure mechanisms

span is large) in short-life mining excavations. Hoek (2007) lists recommended shotcrete applications in underground mining for different rock mass conditions.

Several mines rely on rock mass classification systems to define rock mass quality and select what they consider appropriate shotcrete support to complement their reinforcement strategy. One such system, the Q system proposed by Barton et al. (1974), makes specific recommendations for the use of shotcrete and mesh-reinforced shotcrete in combination with rock bolts. Grimstad and Barton (1993) revised the Q system to encompass the use of FRS, which was an unknown product when the system was first published in 1974 (see Figure 8.8-8 in Chapter 8.8). When FRS is used, the spacing of the rock reinforcement (rock bolts) can be expanded by ~20%–40% compared to the case for bolting only.

Grimstad and Barton (1993) proposed an approximate relationship between FRS thickness as a function of rock mass quality Q and excavation size (Figure 8.6-23). Melbye et al. (2001) proposed that when high-performance steel-fiber-reinforced shotcrete is used as a wet mix, the required thickness

proposed by Grimstad and Barton can be further reduced (Figure 8.6-24).

Grant et al. (2001) argued that the recommendations of Grimstad and Barton (1993) were inadequate because they did not consider the impact of the toughness of FRS. Papworth (2002) expanded this work by including reference to the RDP test. Figure 8.6-25 provides further guidelines on the recommended FRS toughness for different rock mass qualities as defined by the Q system. In areas of anticipated significant deformation, seismicity, or potential strain burst, a minimum energy absorption capacity of 400 J should be used based on RDP_{40mm}. In extreme cases, this should be increased to 540 J. Papworth further suggests that, wherever possible, bolting should be done through the FRS.

Deterministic Design Approaches

Fernandez-Delgado et al. (1981) proposed a deterministic model to calculate the shotcrete thickness t required to support the dead weight of a wedge based on arching theory and the compressive strength of shotcrete.

For the rock-reinforcement scenario shown in Figure 8.6-26, Barret and McCreath (1995) proposed deterministic design models to analyze the worst-case scenario of discrete blocks that could develop in between the rock reinforcement. The models consider four failure mechanisms: adhesive, direct shear, flexural, and punching shear (Figure 8.6-27). Model calculations are shown in Table 8.6-8.

Shotcrete Support for Rock Wedges

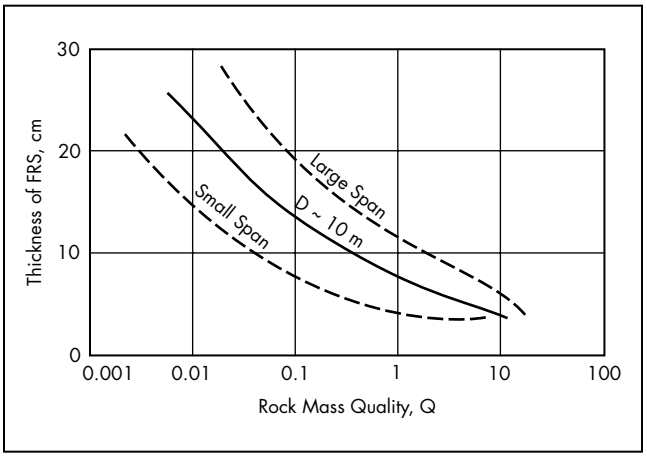
Software is available to conduct kinematic and limit equilibrium wedge analysis for underground excavations. An example of this type of software is Unwedge (Rocscience), which investigates the influence of both rock reinforcement (bolts) and surface support such as shotcrete. A basic assumption is that shotcrete fails in direct shear. A structurally defined wedge shears through an area of shotcrete that is given by the product of the perimeter of the exposed face and the thickness of the shotcrete. The shotcrete acts in a direction normal to the excavation wall that it supports.

Numerical Modeling Approaches

Several numerical models can model shotcrete, either individually or as part of a rock bolt support system. These models are more often used to investigate the impact of different shotcrete characteristics and properties on the stability of excavations rather than for design purposes.

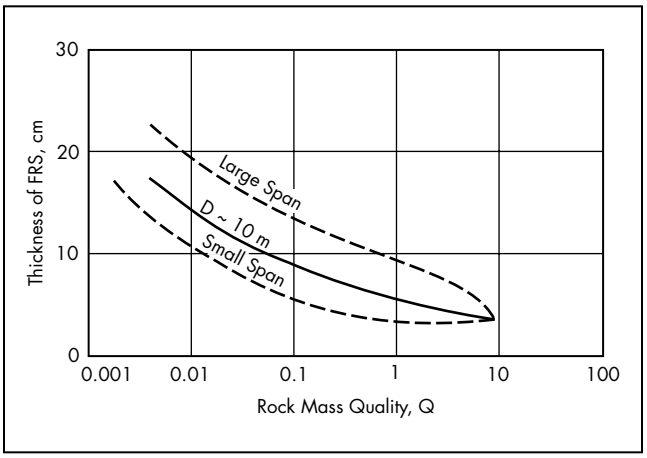
Kuchta et al. (2004) used a distinct element approach to construct a simple drift model containing a triangular wedge in the crown. The model evaluates the relationship between shotcrete thickness and interface strength, and demonstrates that tensile bond strength is more important than cohesive bond strength in determining interface strength.

Malmgren and Nordlund (2008) performed a comprehensive numerical study on the interaction of shotcrete and rock bolts related to conditions at the LKAB underground mine in Kiruna (northern Sweden). Their models demonstrate that rock strength and surface roughness have a significant impact on the number of failures at the rock–shotcrete interface and in the shotcrete lining. They also show that shotcrete lining is sensitive to small amplitudes of surface roughness. The implication is that shotcrete performance depends on the method of excavation (drilling and blasting) and the quality of ground preparation. In all cases, high interface strength is favorable.



Source: Grimstad and Barton 1993.

Figure 8.6-23 Permanent-support recommendation for FRS as a function of tunnel span and Q value



Source: Melbye et al. 2001.

Figure 8.6-24 Permanent-support recommendation for high-performance steel-fiber-reinforced shotcrete as a function of tunnel span and Q value

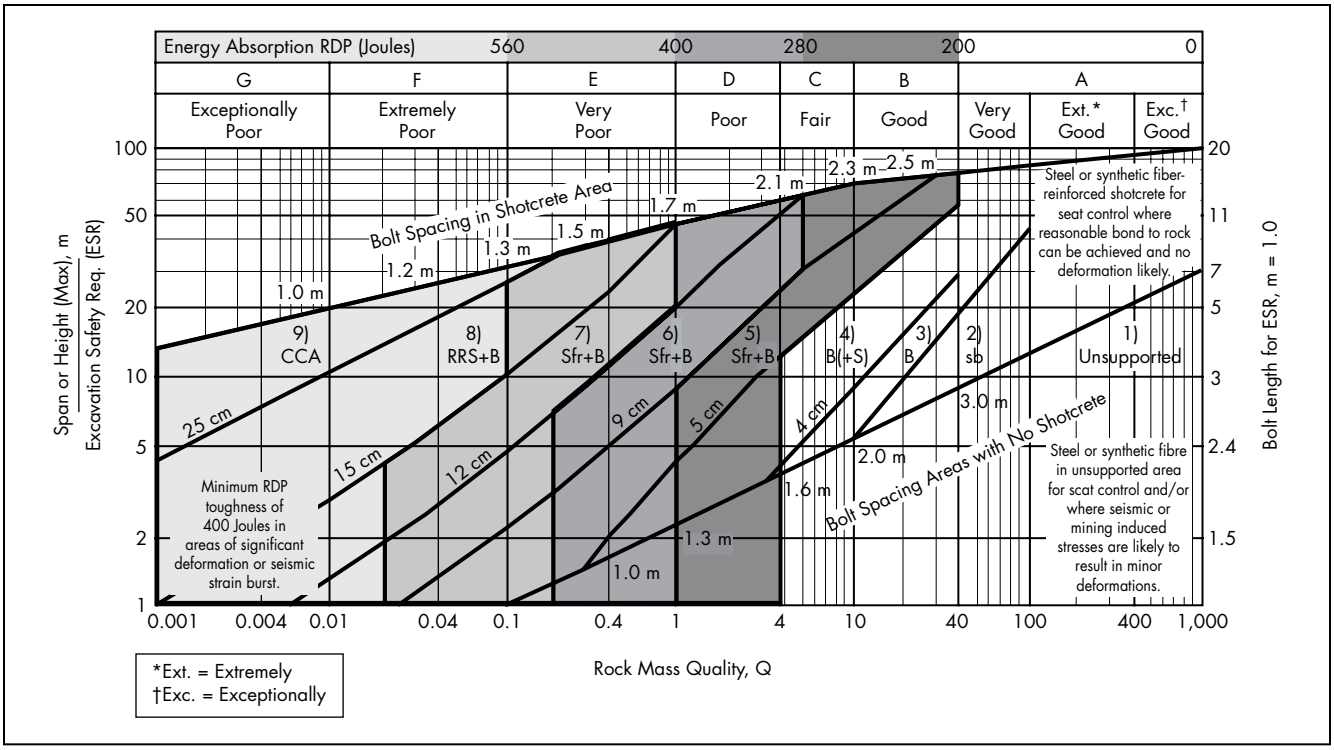
Because the performance of thick shotcrete lining depends on bond strength, there are limited benefits to using thicker shotcrete lining.

Shotcrete Performance

The performance of ground support systems in underground mines is generally intimately related to the quality of their installation, and shotcrete is no exception. Some of the guidelines relevant to quality control of shotcrete are briefly discussed in the following sections.

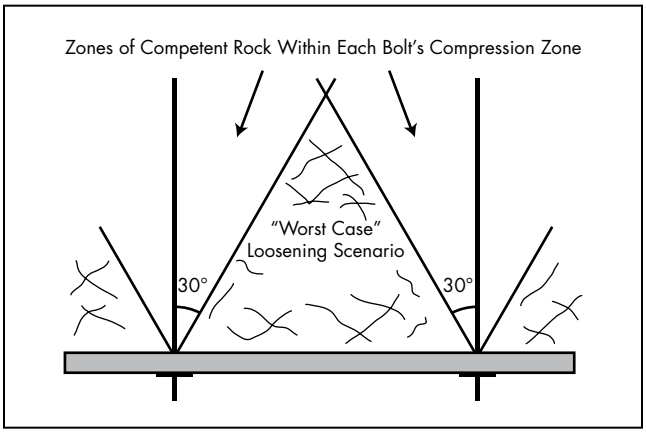
Quality Control

Shotcrete performance in underground mines is influenced by the implementation of appropriate quality control procedures concerning design, materials, application equipment, craftsmanship, and installation techniques. Spearing (2001) identifies the following requirements for a quality control program: it must be reliable, meaningful, timely, simple, and relatively inexpensive. For shotcrete designs derived either empirically or analytically, quality control ensures that shotcrete is applied as designed.



Source: Papworth 2002, based on Grimstad and Barton 1993.

Figure 8.6-25 Permanent-support recommendation for FRS as a function of tunnel span and Q value, with further guidelines on energy absorption



Source: Barret and McCreath 1995.

Figure 8.6-26 Rock-reinforcement scenario for estimating shotcrete support requirements, showing the volume of loose ground that applies load to the shotcrete between the bolts

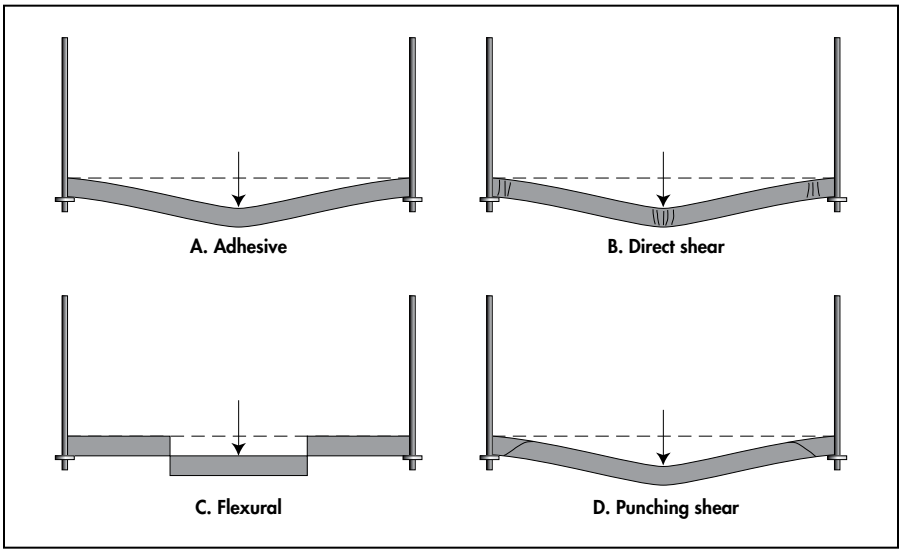
The *Guide to Shotcrete* from the American Concrete Institute (ACI Committee 506 [2005]) specifies the requirements for a comprehensive quality-control program, including material properties and mix design. Shotcrete equipment (compressors, shooting equipment, mixers and batchers, and hoses) should be properly maintained, cleaned, calibrated, and checked regularly for proper function. During a spraying shift, air volume and pressure should be checked regularly. Spearing (2001) reports the following typical requirements:

- For dry shotcreting at 5 m^3/h (3 ft^3/min), air consumption should be ~15 m^3/h (9 ft^3/min) at a pressure in the range of 300,000 to 600,000 Pa (44–87 psi).
- For wet shotcreting at 15 m^3/h (9 ft^3/min), air consumption should be ~12 m^3/h (7 ft^3/min) at a pressure of ~600,000 Pa (87 psi).

An important aspect of quality control is to test the shotcrete before, during, and after application. Quality control tests include slump tests and panel tests. In the latter, variously sized boxes are sprayed with concrete. After hardening, samples are taken with a core saw and tested for uniaxial compressive strength. Normally testing is done at 7 and 28 days, although this can vary. Beaupré et al. (2005) recommend the use of a Schmidt hammer for measuring the compressive strength of underground shotcrete. This method is inexpensive and fast, and the hammer is easy to use, easy to carry, and relatively maintenance free. The precision of the hammer is acceptable, provided that several readings are recorded at a given location.

A good indicator of shotcrete quality is the degree to which the shotcrete adheres to the rock surface. Drill cores can be taken from the wall or back of an excavation and used to expose the layers of shotcrete and the bond between the rock surface and the shotcrete. A simple test is to hit the shotcrete surface with a hammer and judge adhesion quality from the resulting sound; a clear ring indicates good adhesion, a hollow sound indicates poor adhesion, and a "drummy" sound indicates very poor adhesion (NORCAT 1998).

Several simple techniques exist to verify shotcrete thickness. One technique is to install double-headed nails on the



Source: Barret and McCreath 1995.

Figure 8.6-27 Shotcrete failure modes

Table 8.6-8 Deterministic design models for shotcrete

Adhesive Failure

The capacity of shotcrete lining to resist debonding, c_a, is

$$c_a = 4\sigma_a sa$$

where

σ_a = adhesive strength of the shotcrete
s = bolt spacing
a = adhesive bond length, defined as the distance from the panel perimeter

For standard 1.2 × 1.2 m bolt spacing, shotcrete lining with an adhesive strength of 1.5 MPa at 28 days and a bond length of 50 mm can support a dead load of 36 t in adhesion.

Direct Shear Failure

The capacity of the shotcrete to resist direct shear failure is

$$c_{ds} = \tau_{ds}\, st$$

where

c_{ds} = shear strength
s = spacing of the bolts
t = shotcrete thickness

This analysis ignores the stabilizing effect of any axial thrust in the lining developed through arching.

Flexural Failure

The capacity of unreinforced shotcrete lining to resist these moments, c_{flex}, for a slab with a rectangular cross section is

$$c_{flex} = \sigma_{flex}\left(\frac{t^2}{6}\right)\left(\frac{s}{2}\right)$$

where

σ_{flex} = flexural strength of the shotcrete
t = shotcrete thickness
s = bolt spacing

Even if the shotcrete completely debonds, the shotcrete can still be effective in preventing the rock mass from loosening by acting as a slab in bending, provided that it is reinforced by rock bolts.

Punching Shear Failure

In punching shear mode, failure occurs close to the supports where shear forces are at a maximum. Failure occurs not along a vertical plane but rather along planes inclined at approximately 45° to the horizontal, perpendicular to the diagonal tensile stresses in the slab. Thus, the slab actually fails in tension rather than in shear, although normally it is the shear load required to induce diagonal tensile failure that is determined, rather than the diagonal tensile stress itself.

Thus, for a square faceplate, the perimeter of the failure surface is 4 ($c + d$), and the capacity of the slab to resist punching shear failure is

$$c_{ps} = \sigma_{ps} 4(c + d)t$$

Failure is understood to occur along an equivalent vertical plane located a distance $d/2$ from the edge of the faceplate, where d is

$$d = \sqrt{\left(\frac{c}{4}\right)^2 + \left(\frac{V}{4\sigma_{ds}}\right)} - \left(\frac{c^2}{4}\right)$$

where

c = faceplate width
s = bolt spacing

The tributary area for the critical shear load V is determined by the dimensions of the shotcrete panels surrounding a particular bolt and where

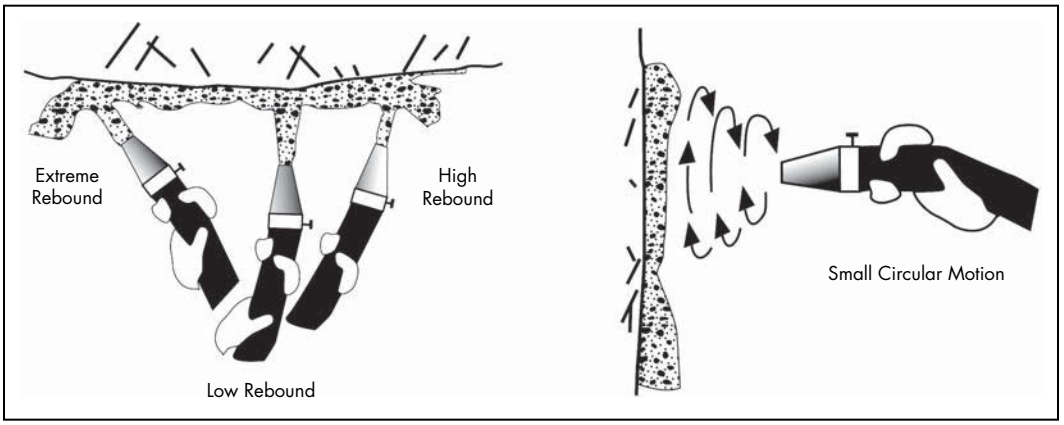
$$V = w(s^2 - c^2)$$

where w is the uniformly distributed load across the tributary area.

Source: Adapted from Barret and McCreath 1995.

excavation walls before shotcrete application. If the nails remain exposed after application, then the shotcrete thickness can be assumed to be insufficient. Another technique is to install nails after shotcrete application using a Hilti nail gun. Eventually it is expected that automated shotcrete spraying machines will be used to scan surfaces before and after application.

Finally, it is important that shotcrete operators be sufficiently trained in the proper application technique. They should manipulate the nozzle so that shotcrete emerges in a steady, uninterrupted flow, ideally perpendicular but never more than 45° to the ground surface, and they should rotate the nozzle steadily in a series of small oval or circular patterns



Source: ACI Committee 506 (2005).

Figure 8.6-28 Proper shotcrete application technique for low rebound and smooth application

(Figure 8.6-28). Failure to apply this technique correctly can result in an uneven, wavy surface that traps significant amounts of rebound and overspray.

Shotcrete Damage Classification

Shotcrete in underground mines, even when damaged, can still maintain some support capacity. Espley et al. (2004), from a series of comprehensive field trials at several Inco mines, identified three levels of damage and required reconditioning (Table 8.6-9).

Where the shotcrete liner is damaged, the area can be rehabilitated with another thin coating of shotcrete. Where excessive damage is accompanied by cracking and shear failures, additional support should be applied using rock bolts and mesh.

OTHER SURFACE SUPPORT TECHNIQUES

A number of specialized techniques have been applied as surface support in underground metal mines, albeit to a lesser extent than mesh, straps, and shotcrete. Three such techniques are briefly discussed: thin spray-on liners, shotcrete pillars, and reinforced shotcrete arches.

Thin Spray-on Liners

Thin spray-on liners (TSLs) are multicomponent polymer materials that can be applied to the rock mass surface as a sealant or as surface support. After more than 20 years of research, development, and trials in mines, the use of TSLs in underground metal mines is still limited to special applications rather than to general application as a systematic surface support such as mesh and shotcrete. Nevertheless, their popularity in certain coal mines and in some less mechanized South African mines has been growing in recent years. The evolution of TSLs until 2004 is covered in Potvin et al. (2004), which includes product reviews, testing and quality control procedures, and case studies.

A main advantage of TSLs is their rapid and easy application, since only a thin layer (~5 mm) is required. The equipment used for application is small and versatile.

A variety of TSL products exist with different chemical components and properties. They can be divided into two categories: those that contain isocyanates and those that do not. The first category includes polyurethanes and polyresins; the second category includes water-, cement-, or latex-based methacrylates and acrylates.

Table 8.6-9 Shotcrete damage classification and reconditioning requirements

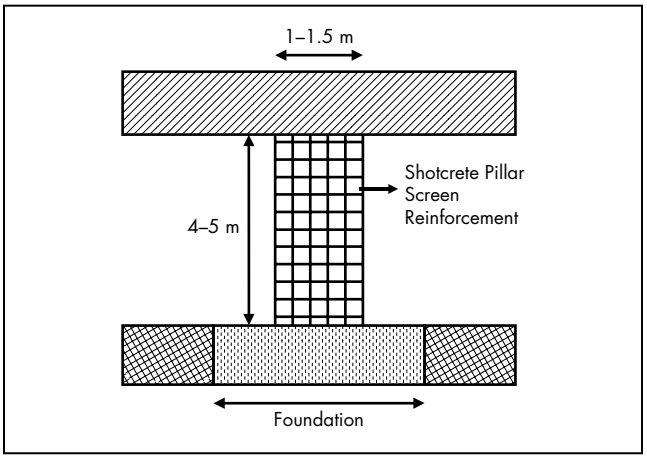
Level of Damage	Reconditioning Required?	Examples
Minor	No	• Fine cracks <3 mm thick, <1.5 m (5 ft) long, spaced at >1.5 m (5 ft) • Drumminess or peeling of shotcrete liner covering an area of <0.25 m^2 (10 $in.^2$)
Moderate	Sometimes	• Intersecting cracks >3 mm and <10 mm thick, >1.5 m (5 ft) long, spaced at <1.5 m (5 ft) • Drummy area of >0.25 m^2 (10 $in.^2$) and <3 m^2 (10 ft^2)
Major	Always	• Shear failure • Drummy area of >3 m^2 (10 ft^2) and intersecting cracks spaced at <1.5 m (5 ft) to complete failure of shotcrete liner or fall of ground

Source: Adapted from Espley et al. 2004.

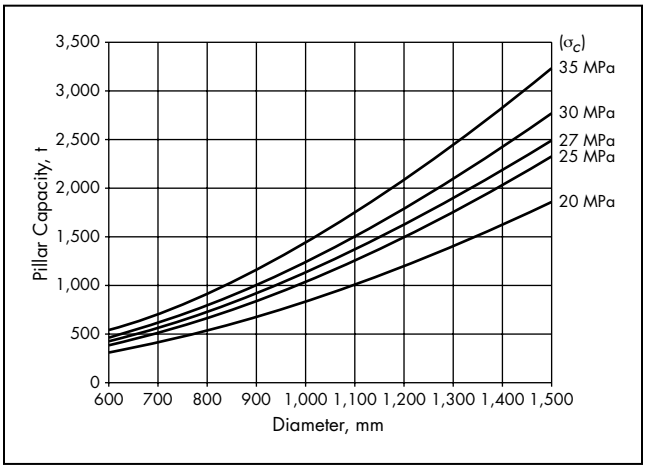
With regard to structural properties, TSLs that contain isocyanates are generally superior to those that do not. Unfortunately, however, isocyanate products have serious toxicity issues, because they contain methyl disphenyl isocyanate (MDI), which in part of the population can induce a severe allergic reaction and even death. Therefore, rigorous health and safety procedures must be followed if these products are to be used.

TSLs have been used successfully for specialized applications, including sealants in South African mines and over shotcrete in Canadian mines to further safeguard against cracking in critical excavations.

A review of the literature indicates that TSL performance assessments are not consistent. In particular, the critical properties and the laboratory and field tests required to estimate these properties are still the object of discussion among scientists and practitioners. The main suppliers of TSL products currently include Minova (Tekflex), BASF (Masterseal), SA Mining & Engineering Supplies (Tunnel Guard), 3M (3M Mining Liner), and Spray On Plastics Ltd. (RockWeb). Most of these products are still under study and field trial and no reports yet exist about TSLs being systematically applied as surface support in mines. Therefore, the technology should be deemed not yet mature and still under development.



Source: Hadjigeorgiou et al. 1999.
Figure 8.6-29 Shotcrete pillar



Source: Hadjigeorgiou et al. 1999.
Figure 8.6-30 Shotcrete pillar capacity as a function of diameter and compressive strength

Shotcrete Pillars

Shotcrete pillars, used mainly in Canadian underground metal mines, are a specialized form of surface support that does not cover the entire rock mass surface. Rather, it is external to the rock mass and stabilizes the roof of an excavation. Use of this method of support depends on factors such as excavation access, potential rock mass failure mechanism, support capacity required, and economics.

Pillar construction is very simple. On solid rock, a pillar is built as a column of shotcrete with a diameter (including final cover of shotcrete) in the range of 1.3 to 1.5 m (4.25 to 5 ft) (Figure 8.6-29). Casing, when used, is made of a #9 mesh cylinder of slightly smaller diameter. Contacts at the floor and back are slightly expanded to give it an hourglass shape.

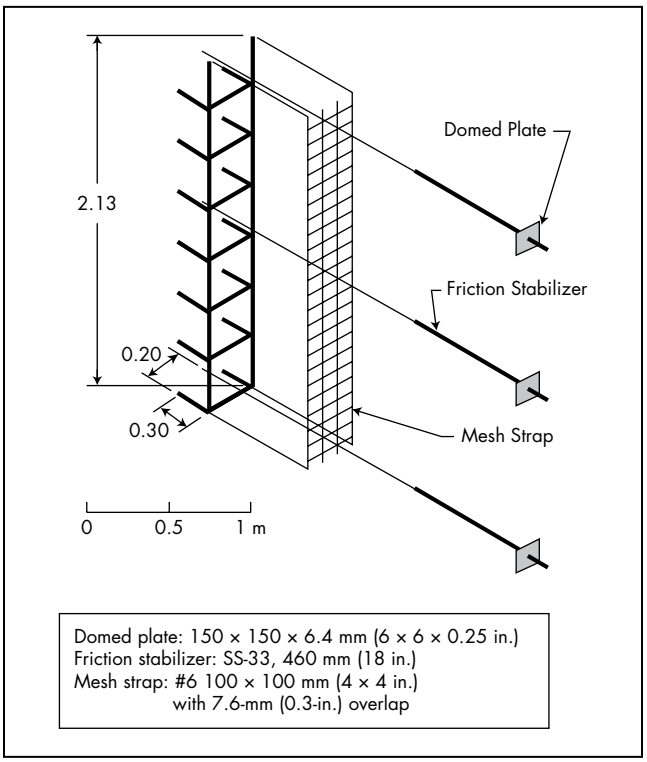
Methods for calculating critical design parameters such as pillar dimension and pillar reinforcement are presented in Hadjigeorgiou et al. (1999). Figure 8.6-30 shows a graphic tool for estimating the capacity of mesh-reinforced shotcrete pillars as a function of pillar diameter and shotcrete compressive strength.

Reinforced Shotcrete Arches

Reinforced shotcrete arches provide stiff, high-capacity surface support. They can be installed quickly and efficiently in an underground mine. Gaudreau et al. (2004) examined shotcrete arches at the Brunswick (Canada) mining and smelting operation, including their method of construction. They are assembled by juxtaposing prefabricated steel frames made of #3 reinforcing bars and #0 mesh straps (Figure 8.6-31). Reinforcing frames are secured to the rock mass surface by means of friction bolts or grouted bars and plates to form a complete arch from floor to floor. The arch is then filled with shotcrete to a thickness of ~50 mm (2 in.), starting from the base, until the entire frame is covered.

Double or triple arches can be built side by side to increase capacity and improve coverage of the supported area when necessary, such as for large spans or very difficult ground conditions. Table 8.6-10 lists the general rules used at the Brunswick mining and smelting operation to select single, double, or triple arches according to the span of the excavation.

When multiple arches are used, they are typically installed every 3 m (10 ft), but this distance can vary when



Source: Gaudreau et al. 2004.
Figure 8.6-31 Reinforced shotcrete arch

Table 8.6-10 Type of arch needed to support a particular tunnel width

Tunnel Width (Drift Span) in Section, m (ft)	Type of Arch Along Tunnel Axis
<5.5 (18)	Single arch
5.5 to 7.5 (18 to 24)	Double arch
>7.5 (24)	Triple arch

Source: Adapted from Gaudreau et al. 2004.

mining in extremely poor rock mass or when rehabilitating collapsed areas. Arches can also be installed in combination with shotcrete pillars, offering versatile solutions to difficult ground conditions.

SUMMARY

Reinforcement and surface support interact to provide an integrated ground support system. Their roles are dictated by the mode of interaction with the rock mass: whereas reinforcement elements preserve the integrity of the rock mass, surface support links reinforcement elements together to resist surface deformation or to contain rock fragments from falling or ejecting.

The main surface support elements used in the metal mining industry are welded-wire mesh, shotcrete, and, to a lesser extent, steel straps. Important properties of surface support include stiffness, load capacity, and deformation capacity.

Welded-wire mesh is soft with high displacement and relatively low load-bearing capacity; its support action generally starts only when the rock mass has failed. Shotcrete is stiff with low deformation capacity and relatively low load-bearing capacity in mining applications because the concrete is often not loaded in compression; its support action begins as soon as the rock mass starts to deform. Straps do not cover the entire surface and are often used over mesh or in special applications.

Innovative support techniques such as shotcrete pillars and reinforced shotcrete arches provide more ground control options. Thin spray-on liners, still under development, are used primarily for special applications in metal mines, although they are increasingly used in coal mines and in deep underground South African mines.

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CHAPTER 8.7

Soft-Rock Ground Control

Rod Stace

Ground control is "the science that studies the behavior of the rock mass in transition from one state of equilibrium to another. It provides a basis for the design of support systems to prevent or control the collapse or failure of roof floor and ribs both safely and economically" (Peng 1978).

Around the world, many mined minerals occur as soft rocks themselves or are found within a rock mass comprised largely of soft rocks. The mining of these resources presents many unique problems. Significant among these is the need to provide supports that are both temporary and permanent, their installation being part of the overall mining system. This chapter deals with this key issue of support by examining the types of mining system that have become widely used around the world to extract these important minerals. It covers changes in stress distribution and rock displacement brought about by mineral extraction and the types of support system that have been used successfully in this complex environment. It concludes by examining planning and design methodologies for support in soft rocks.

In the interests of brevity and efficiency, this chapter concentrates on the working of commonly found sedimentary deposits such as coal but with occasional reference to the working of evaporite deposits.

WHAT IS SOFT ROCK?

Rock is an engineering material having a unique set of engineering parameters. These parameters most commonly include not only uniaxial compressive strength but also tensile strength, stiffness (Young's modulus), the ratio of lateral to vertical strain under load (Poisson's ratio), and behavior when confined, which is defined by the angle of friction and its cohesion.

However, these properties are not the only factors that define the behavior of this material when subjected to stress changes brought about by mining. Material properties measured from small samples in a laboratory give only guidance to a wider understanding of that rock type's behavior within the rock mass. The pattern and frequency of jointing within a rock mass, together with the presence of geological anomalies such as faulting, folding, and weathering, greatly affect the way a rock type might behave in a certain mining application. These factors have been quantified within rock mass classification systems, the most significant of which are the rock mass rating (RMR) system developed by Bieniawski over many years and the Norwegian Q system developed by Barton. The presence of water has a particularly large effect on rock stability, as many soft-rock types are negatively affected by exposure to water.

Another element influencing mining support in soft-rock conditions is time dependency. Excavations in soft rocks become subject to closure with time, due to characteristics of the rock and of the ground mass. The rock types themselves exhibit creep properties—that is, the propensity for strain to increase under a constant load. This is most notable in rock-salt deposits. Additionally, variations in the stress fields within the ground mass and subsequent development of fracturing due to the mining process also result in deformation of the excavation. The support applied to excavations in soft rock must take this movement into account.

The general term *soft rock* is often used to describe a group of rocks with the following specific properties, but it is also a comparative term used to contrast with the other widely used term, *hard rock*. Soft rocks have uniaxial compressive strengths, determined by laboratory testing, in the range <50 to >0.6 MPa (below which the material is termed *very stiff soil*). This range can be further subdivided as follows:

- Moderately weak rock: 10–50 MPa
- Weak rock: 1.5–10 MPa
- Very weak rock: 0.6–1.5 MPa

In the RMR system within the soft-rock range, rocks with an RMR in the range 20–40 are described as *poor rock* and those with an RMR of <20 are described as *very poor rock*.

From the point of view of support, the yield characteristic of a soft rock is more nearly plastic as opposed to brittle. The latter tends to be more characteristic of hard rock.

The terms *soft rock* and *hard rock* are often used to determine the difference between two groups of mines and their mining methods:

Rod Stace, Associate Professor, Department of Civil Engineering, University of Nottingham, Nottingham, UK

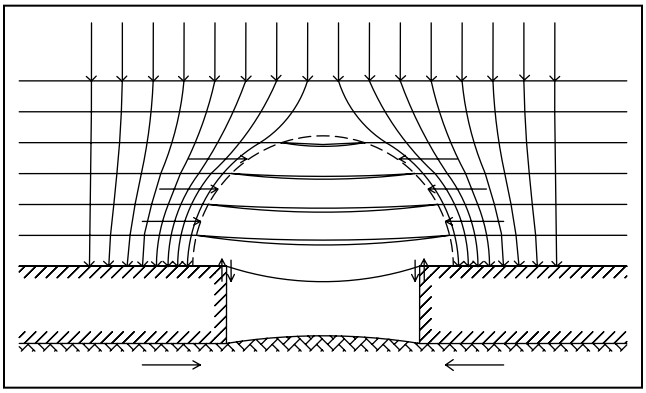


Figure 8.7-1 Forces around an excavated mine opening



Figure 8.7-2 Typical floor lift in a mine roadway

- Soft-rock mines include all coal mines plus, potentially, mines using typical coal-mining methods such as longwall and room-and-pillar. These might include, as examples, mines of gypsum, rock salt, limestone, and potash. The strata are sedimentary in origin.
- Hard-rock mines include those using nearly all the metalliferous mining methods, such as stoping in its many forms. The rock types are almost exclusively igneous or metamorphic.

These are very general classifications, and although there are exceptions, the general rule normally applies. Many examples exist around the world of coal being worked in a rock environment in which a substantial proportion of the rock types exceeds the arbitrary values listed above. However, the problems of mining support in soft rock are significant, and it is the solutions to these that form the basis of this chapter.

THEORETICAL BACKGROUND

This section examines the stress changes around an excavation caused by mining and the influence of these changes on roadway closure. The study of stress change and strata displacement around soft-rock excavations is very mature, much of the work having commenced up to 80 years ago and being developed and refined in the period after the Second World War and up to the 1990s. As a result, although the references quoted and images used may seem rather elderly to the younger mining engineer, their relevance has not diminished with time nor have the findings been superseded.

Stress and Displacement Around an Excavation

In soft rocks found in coal measures or evaporite deposits, stresses as redistributed into the rock mass often exceed the elastic bounds of each rock type. In these circumstances, support plays a significant role in the stability of the excavation.

Soft-rock mechanics is often called strata control. In the longwall method of mining, there is substantial controlled rock-mass movement from the excavation level through to the ground surface. In-built yield is a key design element of many support types used in the soft-rock domain. This process of controlling rock-mass movement is particularly evident when considering choices of support at a longwall face and in the roadways serving the longwall.

Early researchers in the United Kingdom characterized the redistribution of vertical stresses around a coal-mining excavation as a pressure arch (National Coal Board 1950). Although this theory was later modified, the fundamental redistribution of these stresses around an excavation to form high-stress concentrations, known as abutments, around the perimeter of the excavation is widely recognized. An understanding of the nature and role of the components of horizontal stress is also important, although horizontal stress did not really become widely understood in the European coal-mining community until the major move to rock-bolted support at the end of the 1980s. This understanding was based primarily on the work of Gale (1991) in Australia.

In the worst cases, high-stress concentrations at the sides (or ribs) of an opening caused rib-side failures, resulting in effective widening of the excavation. Horizontal stresses that become concentrated in roof and floor measures cause buckling of those rocks above and below the excavation, as evidenced by roof buckling and floor heave (Figures 8.7-1 and 8.7-2). The degree of closure or rock failure depends on the stress values and the strength of the rocks.

Horizontal stresses are often characterized by directionality, the component of stress in one direction having a significantly higher value than the component in the direction at right angles. This can dictate the choice of mine layout, as there are advantages to driving the most important roadways parallel to the maximum component direction to minimize closure. Also, in some shallow deposits, the magnitude of the maximum horizontal stress can be greater than thatt of the vertical stress, which depends largely on depth.

The role of the support member in soft-rock excavation is to accommodate strata displacement in the roof and sides of the excavation in a controlled manner rather than to prevent it entirely. The support action of the rock bolt is an exception to this rule. Support of the floor has always been viewed as technically difficult because any system adopted will interfere with mining activities and infrastructure within the excavation. This support is also difficult to replace or renew should failure occur. Support placed to the sides and roof of a working area or access roadway is normally easier to supplement or replace.

Interaction

Interaction is a mining phenomenon in which the working layout at one horizon influences the stability of workings at another horizon in the mine. This phenomenon is normally experienced only in the mining of stratified deposits where

work is conducted at more than one horizon in the deposit. It is a common feature of coal mining where several seams are worked in the vertical succession.

Planning for interaction has been a feature of coal-mine design in the United Kingdom and Europe for many years. It is increasingly recognized as a constraint on the development and working of coal reserves in the United States, South Africa, and Australia where second-seam working is now practiced more widely.

Interaction can be subdivided in a number of ways. It has been described as being split into "static project" and "dynamic project" activities (Auriol 1972). A static project is one in which an existing working passes into a zone in which the field stress has been modified by previous workings, often long since abandoned; these can be old workings above or below the present working level. A dynamic project is one in which existing excavations—working areas or main roadways—can be influenced in some way by current working activity at another horizon as a result of stress redistribution, ground movement, or both.

Interaction can take a number of forms. It can be solely a redistribution or modification of the stress field being encountered. It can also be induced fracturing caused by previous workings. In addition, in a dynamic project, if the affecting workings are below a horizon, it can be ground displacement caused by the onset of subsidence.

Ground displacement can also lead to induced ground fracturing in the form of tension cracks formed around excavation edges and possible mobilization of fault planes allowing for pathways to be formed for water or gas flow into existing workings.

In early observations of interaction, researchers noted its effects and attempted to ascribe theories to what they had experienced underground. The empirical studies of Walker and associates in the North East England coalfield of the early 1950s were typical of this, as espoused in their pressure arch theory mentioned previously (National Coal Board 1950–1951). Observations of damage to mining excavations suggested that the pressure arch extends both above and below the working horizon, and any subsequent working in another seam that intersects the pressure arch experiences difficult mining conditions (National Coal Board 1953–1954).

The extent of the pressure arch above and below workings was given as twice its width. As an example, for workings at a depth of 610 m, the pressure arch is said to extend 220 m above and below the seam. The magnitude of the stress levels in the pressure-arch legs is related to depth, varying from 1.5 times the cover load at shallow depths and stress concentrations to over 3 times the cover load at depths of ~600 m (the maximum extent of the field data obtained by the investigators).

Subsequent researchers, most notably Whittaker, took this theme further, publishing a set of values showing increased stress at levels above a single rib edge. He states the following: "The mirror image of this pressure profile is valid for the state of stress underneath the solid rib-side except that the magnitudes of the pressures are slightly increased owing to the greater depth" (Whittaker 1972).

King and Whittaker (1970) published a diagram that demonstrates the development of stress contours below a remnant pillar (Figure 8.7-3).

The large body of subsequent published work, especially that appearing in the yearly proceedings of the Annual Conference on Ground Control in Mining (which is hosted by the University of West Virginia [United States] and the U.S. Bureau of Mines and its successor bodies, and is regarded by many as the premier world forum on soft-rock mechanics research), reviews the effects of interaction and its influence on working layouts and excavation stability. This work is dominated by reviews of experience in U.S. mines with occasional Australian, South African, and European examples.

Other research associated with multiseam mining interaction can also be found in the final reports of some of the research projects funded by the European Coal and Steel Community up to 1992, thereafter the Research Fund for Coal and Steel (RFCS).

Stress prediction is important but still must be related to its effect on basic roadway stability in the affected zone; that is, one can mine through a high-stress zone and stability may not be affected or the reverse might apply. Until the late 1980s in U.K. coal mines, roadways were normally formed to an arch profile and supported by steel supports. Roof closure experienced in advancing gate roads of longwall coal faces frequently amounted to more than 50% of the extracted seam section. Floor lift was also commonplace. Closure therefore was expected and a meter of convergence was not uncommon. An additional increment of closure of a few centimeters due to interaction stresses was often not noticeable unless measurements were taken.

When rock bolts became the principal support in rectangular-section retreat roadways and measurement of closure and adjacent ground movement became routine, the effects of interaction began to be recognized. With ground-movement action levels at which additional support is to be set now quoted in millimeters, any additional roadway closure due to interaction effects is identified and significant. The influence of interaction is therefore related to the fundamental stability of the affected roadway.

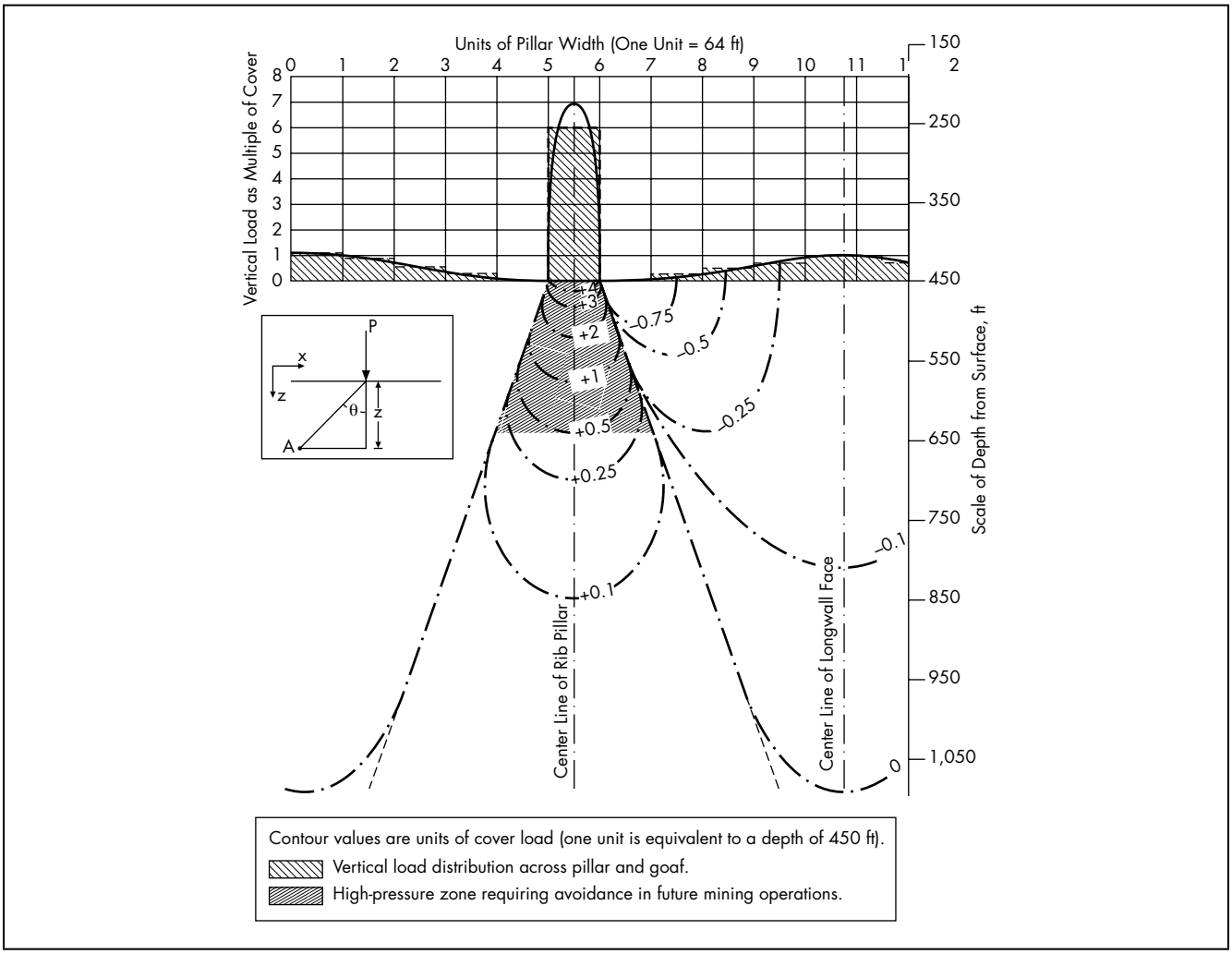
MINING METHODS

Soft-rock mining methods can be subdivided into longwall and derivatives of rock and pillar.

Longwall Mining

Throughout the world, the longwall method of mining produces the highest levels of bulk tonnage of mineral, particularly but not exclusively in coal mining. Coal was worked for many years by mining systems that relied on manual effort for coal extraction and transport. These systems included the extraction-operation elements that form the origins of today's longwall face. From the invention of the armored flexible conveyor, power loader, and powered self-advancing support in the late 1940s and early 1950s, the system of longwall operations that we know today emerged. Single longwall faces between 100 and 150 m long, served by two gate roads in the advancing mode, became the European standard. The gate roads were formed behind the face line at each extremity of the longwall, usually by the removal of seam roof to give additional height for access and ventilation. Working height at the coal face was limited not only by seam thickness but also, in thicker seams, by support designs available for use on the longwall face.

Support of the gate roads was provided by steel standing supports, usually in an arch configuration. On one or both rib sides, where coal had been extracted, packs were built from stone, timber, or other imported materials.



Source: King and Whittaker 1970, reproduced with permission from The Institution of Mining Engineers, now The Institute of Materials, Minerals and Mining.

Figure 8.7-3 Contours of vertical stress below a pillar

Fundamentally little has changed in longwall mining except for the equipment used. Higher-powered, more robust, flexible conveyors now allow face lengths of more than 300 m. More powerful power loaders have increased production by virtue of their speed, the width of web they cut, and the height that they can extract from the seam in a single pass. Powered roof supports now have yield capacities of >1,000 t per unit and are capable of supporting working heights of >5 m at the face.

Additionally, the major underground coal producers of the world have all chosen to practice retreat rather than advance longwall mining. In a retreat system, gate roadways are driven to the extremity of the reserve prior to the commencement of coal extraction (distances of >5 km are not uncommon). Longwall faces are then formed in the seam to work back between them, after which they are abandoned in the longwall waste.

Roadways for retreat faces can be single drivages formed at or near seam height with a rectangular cross section or multiples of two, three, or four parallel roadways connected at frequent intervals by crosscuts. Single-gate roadways are commonplace in the deeper workings of Europe (Figure 8.7-4), whereas multiple-gate roadways serving the longwall are commonplace (and legally required) in mines where the coal seams are substantially shallower, as in the United States and Australia (Figure 8.7-5).

The versions of longwall mining described above were appropriate for coal seams <5 m thick, laid down as near-horizontal deposits. Systems have been developed to work exceptionally thick seams by the longwall method using derivatives of the sublevel caving idea in which caving waste, which is comprised largely of coal, is accessed through the rear shield of the powered support and loaded onto the face conveyor.

Many seams around the world, most notably in northern Spain, are highly inclined. Versions of the longwall method are used with individual gate roadways maintained on the strike of the seam with the production face at or near full dip (Figure 8.7-6). In the United Kingdom, there was always considered to be a limiting seam angle of inclination, in the range 45°–50°, beyond which relatively standard longwall techniques and equipment could not be used. Above this angle, specialist systems and equipment were required.

As production from relatively thick coal seams that lie in near-horizontal deposits can be obtained in substantial

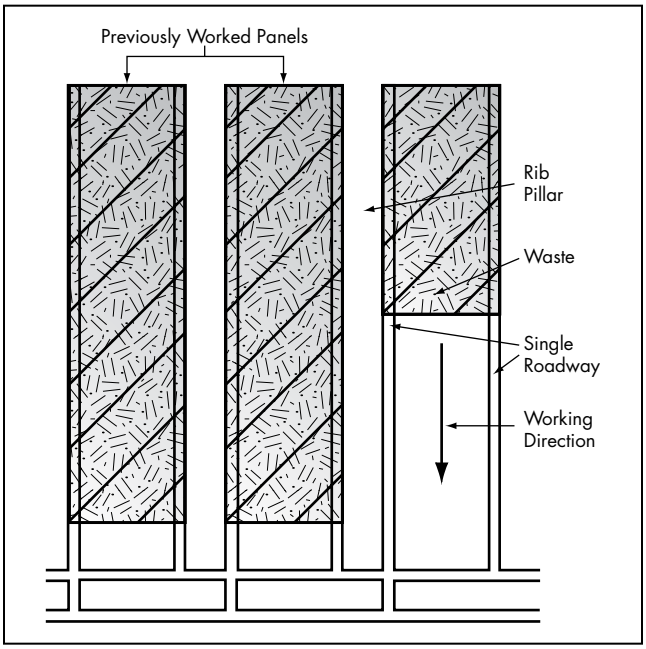


Figure 8.7-4 Typical single-gate roadway layout for longwall retreat

quantities and at very low cost by longwall methods, extraction of coal from thin seams or from steeply inclined or geologically disturbed deposits that require special techniques has waned in importance. Thus this chapter concentrates on standard support techniques required for higher-performing longwalls.

Room-and-Pillar Mining

The alternative to longwall soft-rock mining methods comes under the generic heading *room and pillar* (*R&P*). R&P methods require the driving of multiple parallel roadways in the mineral deposit, joined by crosscuts to form a matrix of unworked pillars of mineral that are stable enough to provide overall long-term support for the mining section (Figure 8.7-7). This is fundamentally an advancing mining system. When mining operations have extended to the planned or natural boundary, the choice remains to either abandon the section or operate a secondary mining system that removes some or all of the pillars upon retreat. Secondary mining requires specialized support systems in which waste is formed by extraction of the support pillars. The extent of this waste must be controlled by temporary supports that separate the remaining pillars and accessible roadways from those where caving is occurring. In recent times, mobile powered roof supports have been used for this purpose with great success.

Many alternative R&P versions exist. Some involve a herringbone-pattern layout; some have pillars that are deliberately smaller than would normally be regarded as large enough for long-term stability; and some have short headings driven into the flank of the panel to maximize extraction. Support requirements for some of these systems are covered in this chapter, but individual systems are too many and varied to be covered here in detail.

R&P methods rarely match the production capacity of longwall methods, but they do offer a low-cost alternative where capital investment is limited or some restrictions exist

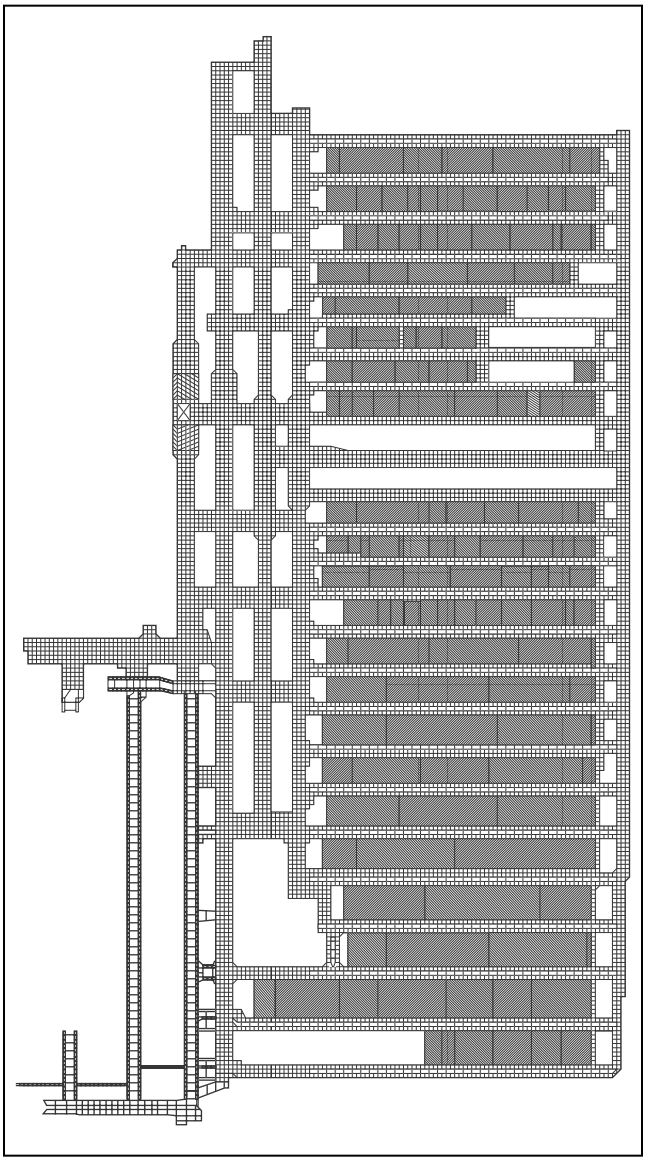
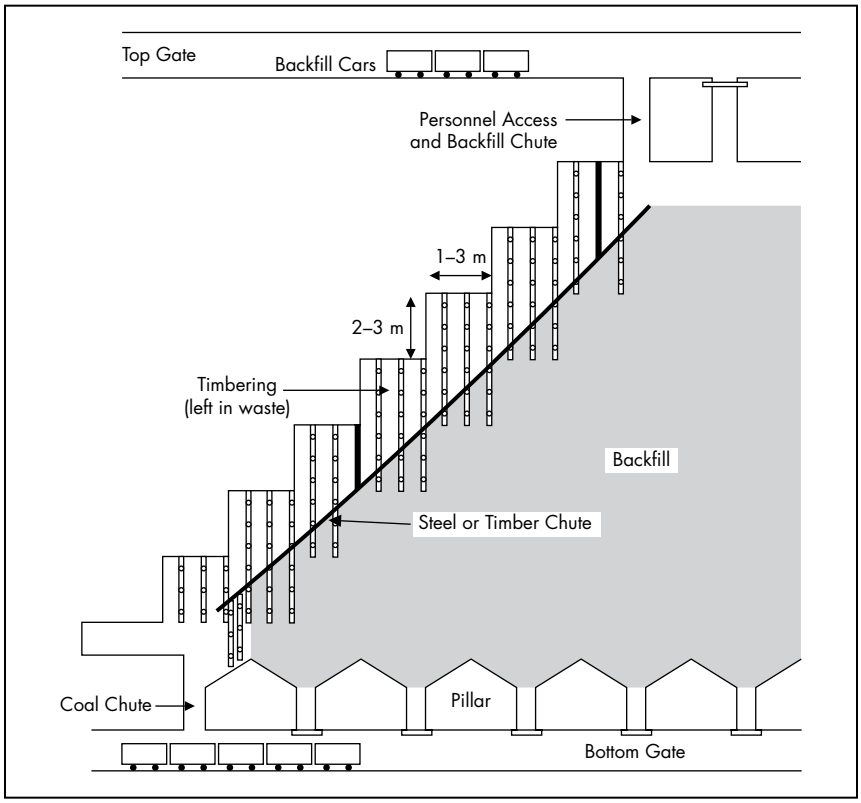


Figure 8.7-5 Typical multiple-gate roadway layout for longwall retreat

on the choice of mining method because of the need to preserve surface features from the effects of subsidence. R&P methods are also used where the roof is so competent that it would not be expected to cave in a regular and controlled manner if longwall were used.

The major restriction on the use of R&P methods is the depth at which they can be efficiently and safely used. It is unusual to find R&P methods used in coal mining at depths greater than ~500 m from the surface. Below this depth, the pillar sizes required to support the district become so large that percentage extraction becomes too low and distances between each advancing heading become too high for efficient operation of the mobile plant normally found in these systems. Additionally, the cost of support required to maintain safe conditions in all the roadways mined can become significant.

Interestingly, some evaporite deposits, most notably potash, are being worked at much greater depths (1,000–1,300 m)



Source: Roberts 1994.

Figure 8.7-6 Typical roadway layout for working steeply inclined seams

by R&P versions in which pillar sizes are small enough to yield with time (Figure 8.7-8).

CHOOSING A TYPE OF SUPPORT: A BRIEF INTRODUCTION

The support that is selected for mining excavation has a number of roles to satisfy. It must apply constraint to the yield zone of failed rock immediately next to the excavation to assist in the natural strengthening of that zone and enable the excavation to remain stable for its designed life (whether 3 months or 50 years). It must provide sufficient working space for mining equipment and operations. It must prevent additional small pieces of loose material from the surface of the excavation from falling and striking people and machinery. Finally, it must provide people who travel through or work in the excavation with the confidence that they can do so safely.

A wide range of support options are available for soft-rock mining. Perhaps the only option that rarely arises is that of no support being needed. Examples of available support types are discussed in the following paragraphs.

Rock bolts reinforce the rock mass, giving it additional strength to withstand imposed stress concentrations. They are designed primarily to resist rather than accommodate strata yield. Rock-bolt systems are highly flexible patterns being altered to react to localized changes in support needs. They are relatively cheap and easy to transport, but require skill and supervision to install. There is also some opportunity for numerical design. Rock bolts are frequently used in conjunction with steel-mesh panels or lengths of geogrid that provide protection in the working area from spalling roof and sides of the excavation.

Steel standing supports (steel ribs) are commonplace in soft-rock excavations. They are erected from sections bolted together on-site and can be easily adapted to variations in roadway size or drivage direction. Although in earlier times they could be found in district development and gate roads in most European coal mines, the wider application of rock bolts has replaced them in a number of shorter-term roadways. Steel standing supports can incorporate yield mechanisms to accommodate rock movement around an excavation and are normally constructed from steel grades that fail in a ductile rather than brittle manner. These support types are relatively inexpensive and operate most effectively with backfill or in smoothly profiled excavations that give multiple points of contact with the rock. Design for these support types tends to be empirically based.

Injection of resins and grouts into fracture planes to improve strata strength has achieved much success in specific locations such as the area around the end of the retreat longwall face. The success of the technique depends on the presence of existing rock fractures. Thus it can be used not to prevent initial development of fractures but only to prevent fractured rock from becoming more fractured and threatening the stability of the mining excavation. It is normally used in conjunction with rock-bolt systems and frequently requires specialist operators and equipment to achieve successful results. Concerns exist about the sensitizing properties and flammability of many of the more successful injection products, especially those based on polyurethane. Targeting of injection in the required areas is difficult. The cost of the chemicals and specialist assistance makes injection relatively expensive.

Hydraulics revolutionized the support of longwall face roofs. A longwall requires temporary, moveable, and yielding

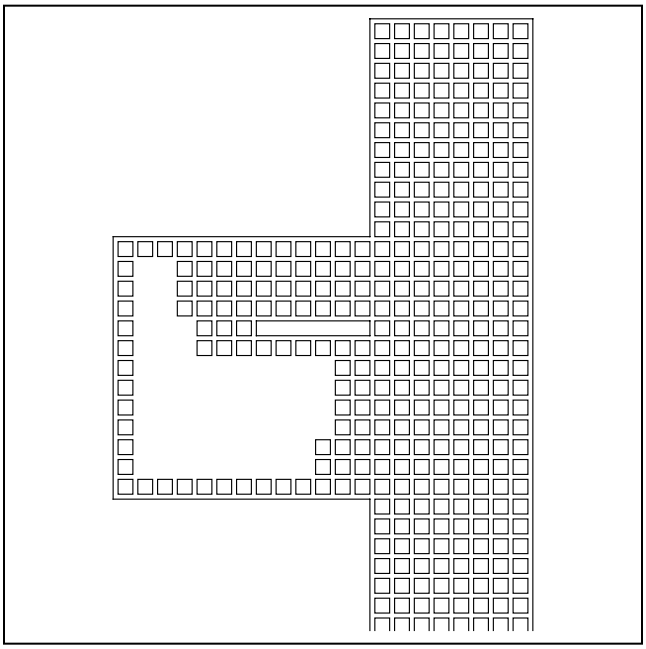


Figure 8.7-7 Room-and-pillar layout with pillar recovery in progress

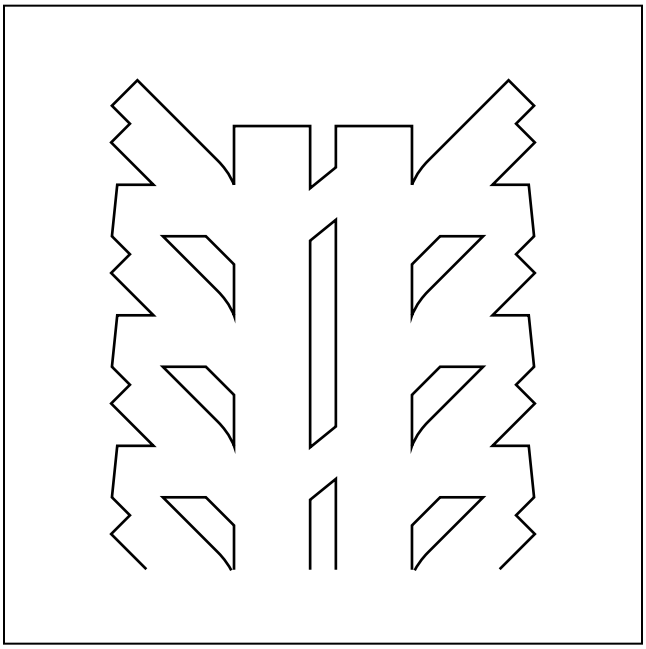


Figure 8.7-8 Room-and-pillar layout modified for deep evaporite extraction

support, and supports based on hydraulic cylinders meet all of these criteria. Single hydraulic props were replaced in a 50-year evolutionary process by powered roof supports that support the longwall working area, advance the face conveyor, and then advance themselves as extraction continues. Versions of powered supports have been used as mobile temporary supports for extracting pillars from R&P systems.

Sprayed concrete (shotcrete) can be applied as an initial temporary support measure, normally in a major drivage, to provide instant support until secondary, more permanent systems are erected. It plays a key role in large excavations such as tunnels, large roadway junctions, or shaft insets where fast-acting, good-quality temporary support is needed. This system requires skilled operators and specialist equipment and products. A potential drawback is that there is little scope for yield when ground eventually moves, and the fractured shotcrete liner itself becomes a safety hazard.

Concrete or cast steel segments are the ultimate in high-strength permanent lining, and are used mainly in shafts, major long-life underground drivages, and places where water inflow might threaten the mine, such as when driving through major faulting or driving shafts through water-bearing rock. These linings are expensive. They are specially designed for a specific excavation and normally require backfill or back-grouting to prevent point-contact loads from destroying their integrity. They are inflexible in that they are difficult to install in other than straight-line excavations and it is difficult to vary the dimensions of the opening. They require major logistical backup to bring them to where they are to be erected and special erection arrangements must be provided.

Packs constructed from stone, timber, concrete blocks, or pumped cementitious material have been used in many parts of a mine as supplementary support. Stone packs were originally a key part of longwall support when built as strips in the waste before the advent of powered supports. They were also used as additional roadside support in roadways of advancing longwall faces. In modern times, packs are frequently used in places such as retreat-face ends where significant additional support is required to retain access ahead of or ventilation flow behind the face line. Their key properties are their flexibility and their significant support-load-carrying capability, but they are labor-intensive to build and require import of materials.

The following sections examine in more detail some of these key support types.

Rock Bolts

Rock bolts are widely used in soft-rock mining, both as principal support and as supplementary additional support or temporary rock reinforcement. Their primary function is to reinforce near-seam rock so that it can experience loading without excessive deformation and, hence, deterioration to the extent that the mine opening is damaged.

Peng (1978) describes three main methods of support action:

1. Suspension of a soft-rock beam from a more competent upper bed
2. Formation of a composite beam by rock reinforcement
3. Increase in the frictional resistance of jointed rock across failure planes to help promote natural arching

Rock bolts were widely and successfully used in coal mines in the United States from the mid-1940s. Importantly, they complemented the normal R&P system of mining. They were also appropriate to strata strengths and stress conditions in U.S. mines. Rock bolts were initially introduced in the United Kingdom in the 1960s as mechanical anchorages, for which use they ultimately failed. They were reintroduced in the mid-1980s because competitive pressures on U.K. coal producers were driving the industry to rely on retreat longwall mining to deliver the required volumes of output at prices that the market could stand. The industry determined that the Australian experience with rock-bolted support in coal mines was appropriate

for U.K. conditions. These conditions dictated that rock-bolt systems must resist rock movement, whereas existing standing supports accommodated movement. There is very little scope for yield in a rock-bolt system and their success was judged on the degree of stiffness in the system.

Only rock bolts that are full-column-anchored with resin have been used successfully in U.K. coal mines. Their performance as a support in high-stress, soft-rock conditions is superior to that of alternative forms of rock anchorage. Much debate has ensued as to whether tensioning of a full-column-anchored rock bolt after installation contributes to its ultimate capability to support the rock strata. Research using numerical modeling (Saleh et al. 2004) has shown that the effectiveness of a pre-tension load varies with both rock-mass quality and in-situ stress level. The findings can be summarized as follows:

- The compression zone generated by tensioning a full-column-anchored bolt is very limited in extent.
- Pre-tensioning offers some advantages in a closely spaced bolting pattern in a weak jointed rock mass (RMR = 30).
- Pre-tensioning also offers advantages in a low in-situ stress field.

Mixed systems of support involving standing steel supports supplemented by rock bolts have been used in the United Kingdom. The support actions of each system differ and therefore do not necessarily complement one another. Because of these differences, the total support generated is not simply the sum of the contributions of each of the two systems. In the early stages of rock movement around a roadway, most resistance is provided by the rock bolts. When the roof has moved sufficiently to load the standing support, the bolts have often ceased to be effective. Only a mixed system, incorporating adequate support density to accommodate each of the components separately, can be expected to deal with a very heavily loaded roadway.

Other rock reinforcement methods used in U.K. coal mines have included the following:

- Reinforcement of geological fault zones encountered by the longwall (that have led to areas of broken roof and slabbing of the coal face) by means of 2-m-long hardwood dowels, full-column-anchored into the affected area. This method provides some cohesion to the rock mass so that mining can continue. However, its use has largely ceased as powered support designs have improved.
- Injection using resins and grouts. This method has found considerable success as a means of artificially bonding strata in the immediate vicinity of the face end to enable control of strata movement during mining. A bonding agent, based on resin or grout, is pumped under pressure into boreholes that are drilled into the roof and rib sides of the retreat roadway. This method is often used to best effect in conjunction with steel or cuttable rock bolts. It has been extended in extreme cases to the recovery of falls on longwall faces.
- Support of not only the roadway's roof but also its rib sides in high-stress, soft-rock conditions. Such support is important because high vertical stresses can create a fractured zone deep into the rib side. On the longwall-face side of the retreat gate road, support is achieved by means of cuttable bolts made of glass-reinforced plastic, full-column-anchored into the rib side during roadway formation. The axial strength of these bolts is comparable to that of steel bolts used in the roof. However, they can be easily cut by the shearer when it reaches the roadway.
- Cable bolts and flexible bolts. The length of rock bolts used in roadways is normally limited by the height of the driven roadway, unless some form of coupling is used to allow installation. Their use is appropriate when a combination of high strata stresses plus weak rock surrounding the excavation exists and tendons longer than 2.5 m are required. These longer tendons have been used to supplement steel rock bolts to provide principal support in heavily loaded roadways.
  - Cable bolts can provide support at depths of ~8 m. Full-column anchorage can be provided by pumping grout to the back of the hole after installation of the tendon. The flexibility of the cable allows the tendon to be bent during installation.
  - Flexible bolts tend to be shorter steel tendons (~4 m long) based on steel rope. They can be spun into resin capsules inserted into the hole, injected after installation, or a combination of both.

Steel Standing Supports

Development of steel standing supports has been led by the coal industries of the United Kingdom and Germany. However, since the 1940s, both countries have differed radically in the choice of standing support type used in roadways. In the United Kingdom, although advancing longwall mining was the predominant mining method, it was normal to support gate roadways with a two-piece or three-piece D-shaped steel rib known as an *arch*. The German industry also used a form of steel rib, but whereas the United Kingdom used rolled steel joists (RSJs) of H-shaped cross section (Figure 8.7-9) based on those specified for structural steelwork in civil-engineering applications, the German industry favored joists of V-shaped cross section with an inbuilt yielding mechanism at the joints. This latter has become known on both sides of the North Sea as the T-H yielding arch, after the original supplier, Toussaint–Heintzmann (Figure 8.7-10).

After the move to retreat mining began in earnest in the United Kingdom in the mid-1980s and in-seam rectangular-section roadways became an operational necessity, two major changes were effected. One was the introduction of rock bolts as principal support in roadways. The other, altogether less obvious and less heralded, was the move to flat-top or cambered-top steel arches (Figure 8.7-11).

Despite the increased use of rock bolting since the 1990s, many gate roads continue to be supported solely with arches or with a combination of rock bolts and arches (a so-called "mixed" support system). In the United Kingdom, arches also continue as the system of choice in many main trunk roadways, all cross-measure drivages, all drivages through goaf (known as scours), all roadways formed by blasting, and all advancing-face gate roads.

The British Colliery Arch

The colliery arch was based on the RSJ specified in British Standard BS 4 (1980). H-section joists (or I-section joists, as they are known in the civil engineering world) were designed to have the highest possible moment of inertia in the plane of bending. This requires that as much of the steel in the section as possible be away from the neutral axis. In civil engineering applications, there is normally only one plane of loading. Thus

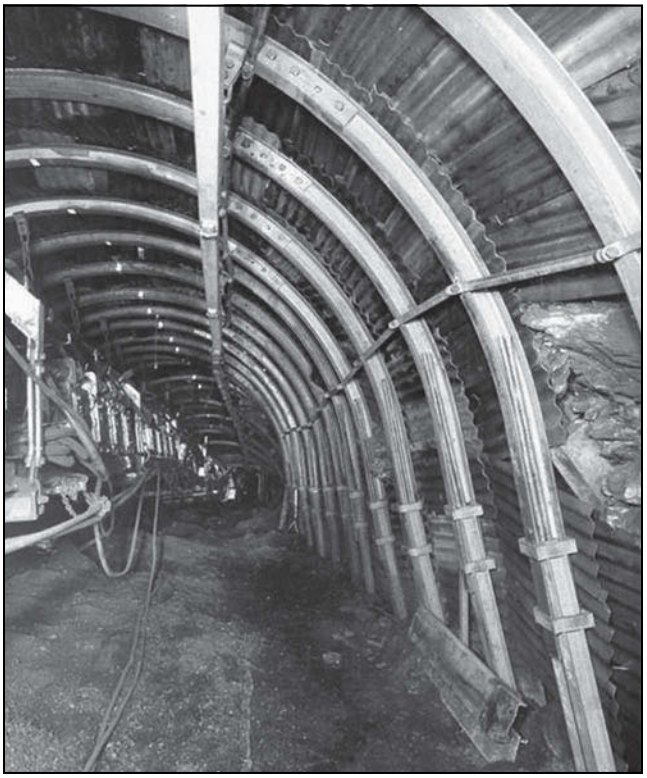


Figure 8.7-9 H-section steel arches



Figure 8.7-10 T-H yielding arches



Courtesy of British Coal Corporation.

Figure 8.7-11 Cambered-top steel arches

most of the steel is in the flanges, in long thin webs, as far as possible from the neutral axis.

In mining applications, loading usually occurs in more than one plane and is often unevenly spread and of a magnitude sufficient to cause the joist to yield. Therefore, mining section rollings always have most of the steel concentrated in the web and tend to have overall dimensions for flanges and webs set at values near one another (e.g., 127 × 114 mm, 152 × 127 mm).

A significant property of any standing support steel is its capability to fail in a totally ductile manner with no sign of brittle fracture. Therefore, most colliery arches, whether D-shaped or flat-topped, are constructed of grade Fe 430 B steel per BS EN 10025.

Arch supports are made by cold-bending steel rollings into roadway supports. This involves forming fishplate and strut holes in the web and sawing the arch into appropriate lengths for ease of handling and assembly. Supports are normally supplied into the mine in arch packs—that is, collections of arch segments packed together with fishplates, bolts, struts, stilts (if used), lagging, and, where appropriate, rock bolts and resin—that can support a given cycle of advance in the drivage of a roadway.

Arch Joints

Arches are formed from two, three, or even four sections of joist, generally connected by means of fishplated joints. However, these joints can become points of weakness affecting the overall strength and operational performance of the support unit.

In the early days of D-section arch supports, all fishplates were of an in-web rolled-steel design. The "wrap-around fishplate," introduced in the United Kingdom in the mid-1970s, gave additional strength to the fishplated joint, especially in the plane parallel to the roadway axis. It also satisfied the requirement for a fishplate that can be loosely assembled, enabling an arch to be erected in one piece, a technique known as whole-arch setting (Figure 8.7-12).

In the early 1980s, the pressed-steel in-web fishplate was introduced. Stronger than other existing fishplates, this one was also lighter and cheaper, a combination of qualities achieved by use of alloy steel containing a percentage of boron. Joint strength in both planes was greater than that of the original sections. Pressed steel plates were also used to form the chevron joints of camber-top and flat-top supports.

Other Arch Accessories

Arch supports are separated by spacers known as struts. The number of struts used between each setting normally depends on the length of the arch perimeter. Struts are either bolted through the joist web or clamped onto the joist.

The strongest and most expensive struts are tubular, fitted with flanges at either end that are bolted through the joist web. Laboratory tests show that a full set of struts, correctly assembled and fitted to arches, can contribute significantly to support-system strength.

A chief drawback for roadway supports, mentioned previously for H-section joists, is the lack of a yield mechanism in the support system as opposed to the yield of the steel itself. One way to overcome this is to use stilts. In theory, when fitted to the base of the arch legs, stilts allow a degree of yielding of the overall system without threatening the integrity of the arch joist or its joints. Many stilt designs have been made;



Courtesy of British Coal Corporation.
Figure 8.7-12 Wrap-around fishplate arch joint

however, all were expensive yet incapable of providing consistent yield under uncontrolled field conditions. Thus, the use of stilts eventually ceased.

Although the use of stilts ceased, the problem that they were designed to solve did not disappear. The yield mechanism in arches today is provided by packing material between arch and strata or by arch legs that penetrate the floor. The latter mechanism is thought to contribute to floor lift in roadways.

Roadway Support System Strength

With so many variables present in the roadway support system—including the most significant external variable, the loading condition to which each arch in a system is subjected—determination of the strength of a single support or a number of adjacent supports is based more on engineering experience than on science. All conclusions about product or system strength tend to be comparative.

Between 1986 and 1991, the European Coal and Steel Community funded a research project titled "Testing Roadway Support Systems." Project objectives were to provide basic data on the fundamental strengths and characteristics of a range of roadway supports and ancillary equipment. Project conclusions are given below (British Coal Corporation 1991):

- The most severe loading condition for a single arch is a single-point load. As the number of loading points increases and side restraint is offered, the capacity of an arch increases accordingly. The most favorable condition is a uniformly distributed load around the support.
- Support capacity can be increased by reducing support base width, reducing support centers, and increasing section size.
- For fishplated joints, increases in joint strength up to ~80% of the section strength increase overall support strength. Further increases in joint strength do not further increase overall support strength.
- Struts add to the ultimate strength of a support system. Stronger struts have the most favorable effects.

These conclusions, although already fairly well known to mining engineers, were quantified for a range of support types and loading conditions.

German Practice

The T-H-section arch joist was first introduced in Germany in the early 1930s. The principle behind arches made from this section is to allow the portions of the arch to slide relative to one another at the joints, making the arch effectively yield under load without damage to the section itself. The joints are formed by overlaps between the arch legs and the crown, and are secured by clamps. Resistance to yield can be varied by adjusting the torque on the clamp bolts. The T-H section also enjoys the advantage that the strength of the section is similar in both major axes.

Visitors to German mining operations would be hard-pressed to see a roadway support other than the T-H section. German mines are laid out as longwall systems. Gate roads frequently serve two longwalls and are thus subject to considerable closure. They are therefore formed initially much larger than are U.K. roadways, with the expectation that some closure will take place. The T-H section allows major closure without the arch losing its support integrity or suffering excessive damage. The roadway is often backfilled after formation to control the loading condition on the arches.

T-H-section arches have some drawbacks. They are set with the base of the V inward toward the roadway. This causes difficulties in the provision of struts and devices to hang cables and hoses. However, their chief drawback is that the yielding process, because it relies on friction, is difficult to control.

The definitive English-language work on roadway support in Germany is a book published in the United Kingdom by Fritz Spruth (1960). Spruth identified the importance of the joint friction clamp that allows yield and stated that clamps should do the following: (1) yield at a steady load; (2) operate automatically (some early designs relied on the miners having to periodically strike the joint to initiate a degree of yield); and (3) not snag on roadway lagging material during arch yielding.

In conclusion, when comparing U.K. and German arch roadway-support practices, the following can be concluded:

- Performance at yield is clearly better for arches made of T-H sections than for those made of H sections.
- Support dimensions are variable on setting for arches made of T-H sections, within limits; they are not for arches made of H sections.
- Costs are higher for arches made of T-H sections than for arches made of equivalent H sections, mainly due to the clamps.
- Lateral strength is greater for T-H sections and virtually nonexistent for H sections.
- Joints are strong points for arches made of T-H sections, due to overlap; they are often the weakest points for arches made of H sections.
- Versatility is greater for H sections in that they can be readily adapted to U.K. needs to use square-section or flat-topped arches in in-seam retreat roadways or R&P districts. The author is not aware of this need in Germany or of any developments along this line using T-H sections.

As a result of this mix of experiences and operational requirements, the two industries have developed their roadway support technology, as they have developed their mining systems, along different lines.

Face Support

Longwall support offers many challenges to the engineer. Among its basic strata-support capabilities, a support unit must do the following:

- Be capable of being set at a variety of working heights
- Impart an immediate positive load to the roof strata to prevent bed separation in the immediate roof strata and control the movement of bridging beds
- Yield in a controlled manner within a chosen load range
- Provide protective cover against falls of smaller pieces of rock into the working area
- Support the coal face laterally
- Prevent ingress of material or dust from the waste into the working area

Additionally, the support unit must perform the following equally important operational tasks that are not necessarily linked to its ability to act as a roof support:

- Be capable of advancing the face conveyor and then moving the support structure forward
- Be capable of retaining its alignment relative to its neighbors
- Allow personnel access along the longwall
- Have a low collapsed height for ease of transportation among longwall applications
- Provide for both in-support and remote control of support operations
- Provide base positions for coal-face lighting installations

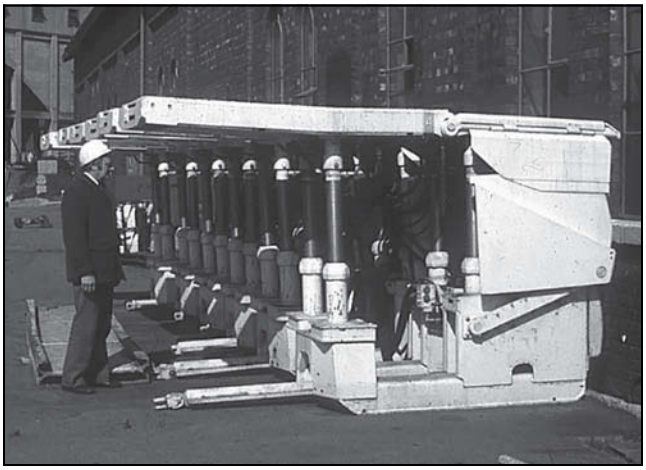
In the United Kingdom, investment in longwall plant and equipment has always been a major feature of the mining industry's capital expenditures, second only to that for new mines or major reconstruction projects at existing mines. By far the largest item in that expenditure is for powered roof supports. Because of their considerable cost, their design features are of significant interest.

Introduction of the lemniscate linkage to powered roof-support designs is arguably one of the most influential technical achievements in longwall mining in modem times. It unlocked the way toward the working of thicker seams in longwall systems. In the United Kingdom, the major powered roof-support manufacturers had developed chock-support technology to its ultimate. This was best demonstrated by the hugely successful six-leg 240-t yield support with full shielding, full-width canopy, and heavy-duty forepole (Figure 8.7-13).

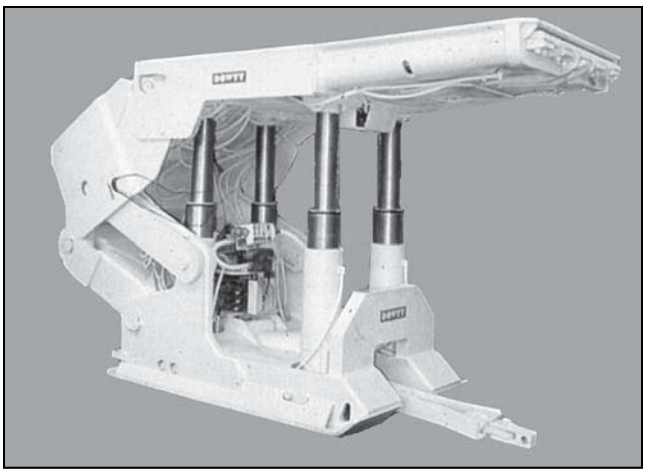
Although designed for extractions up to 2.4 m thick, it was at its best in extractions of 2 m. Attempts had been made to develop chock-support technology for extractions of 3 m, but at this extraction height the stability of a 1.2-m-wide support was questionable, especially with regard to its lack of resistance to lateral loading due to its inability to operate successfully with coal-face support sprags and to be in contact with its neighboring supports.

The lemniscate linkage changed the internal stability of the support structure. It removed all lateral load from the support legs, so that leg restraint need no longer be provided by leg towers. Supports could now be designed with their side shields and canopy side flaps in contact with those of their neighbors, greatly reducing the ingress of material from the waste into the traveling track, which had been a major problem with chock-type supports. Supports could also be designed to advance while retaining contact with the roof, a feature unheard of in conventional chock design.

The lemniscate linkage also addressed another problem: that of spalling coal from the face, a well-known danger to workers that ultimately leads to exposure of unsupported roof at the face and increases the potential for broken ground or major falls. The solution to this problem was the use of face



Courtesy of British Coal Corporation.
Figure 8.7-13 Chock-type powered roof support



Courtesy of British Coal Corporation.
Figure 8.7-14 Four-leg chock shield

sprags. Once again, the lateral resistance offered by the lemniscate protects the legs from any resultant lateral loading. Lemniscate linkage supports became known as shield supports.

The first shield supports used in the United Kingdom were four-leg chock shields (Figure 8.7-14). Operators traveled the longwall within the support structure behind the front legs. However, although they offered positive support to the roof, they did not have a particularly good working range.

Inclined-leg shields soon overtook them in all seam ranges. They enabled the use of shield technology in thinner seams and allowed passage through the face in front of the forward legs, providing better access to coal-face equipment. Additionally, the considerable range of working heights available with these types of support made them easy to transfer among different seams in the same mine, and, indeed, different mines.

Two-leg shields were a natural extension of inclined-leg designs (Figure 8.7-15). The rearward-facing inclined legs were replaced by stabilizing rams between rear shield and support canopy. Interest in their use in the United Kingdom developed as mining engineers became aware of the number of overseas applications. Individual leg ratings of >300 t for

two-leg shields led to a far superior density of support when compared with U.K. four-leg designs. However, the chief argument that slowed their U.K. application was the perception that the forward edge of the base would dig into the soft floors encountered in many U.K. coal mines. This valid concern was overcome by the introduction of the base lift, which lifts the front end of the base during support advance.

How Much Support Is Required?

Support ratings have increased over the last 30 years, in terms of both the *yield load* by which a support is defined and the applied *setting load*. The setting load has increased as a percentage of yield load in the United Kingdom from about 30% to >75%. However, the question has never been satisfactorily answered as to how strong a support should be for a particular circumstance.

The support acquisition process in the United Kingdom was based on the premise that the mine operator would select a support rating, and the manufacturer would supply to that specification and demonstrate by means of rigorous laboratory testing that the support structure and hydraulic legs met that specification. There is a history, especially in Europe, that powered supports have tended to become overdesigned for their duty in terms of nominal load-bearing capacity. However, in view of the substantial investment required to purchase supports, the long, arduous working life expected of them, and the uncertainty always attendant upon soft-rock mining, it has been regarded as prudent to have a significant factor of safety when applying powered supports.

Mechanical testing of support structures, initially identified as a guide to design safety, is now also used to measure design reliability. Such testing involves subjecting structures and hydraulic components to numbers of test cycles approaching those likely to be experienced during production life. These requirements are customer-led rather than imposed by regulatory bodies, as the international market is populated by mine operating companies that are now well aware of what they require from a support and how those features might be demonstrated by the potential supplier.

In 1999, mining regulations were introduced in the United Kingdom to control the application of support in underground mines. These constitute the first revision of the 1966 regulations that were drawn up when powered support designs were comparatively new. Before then, a maximum interval was specified between the coal face and the nearest support member of any given support design. This distance, generally 2 m, was taken as the measure of safe support and was the basic parameter that influenced face design, especially the design of the power loader and the armored face conveyor and its furnishings. Even in an era of relatively low-capacity face equipment, this distance was barely adequate to house the face equipment. As the power and size of all elements in a coal-face package increased, this "prop-free-front" distance plainly restricted potential production that could be won from the face. Some better measure of support had to be found.

Early work by the Mining Research Establishment (Ashwin et al. 1970) demonstrated empirically the minimum mean load density (MLD) required at the coal face to limit convergence during normal operations. At MLD >> 10.76 t/m^2, convergence differs very little from that at MLD = 10.76 t/m^2. At MLD < 10.76 t/m^2, convergence increases. These optimum



Courtesy of British Coal Corporation.

Figure 8.7-15 Two-leg shields, showing good access along the face line

MLD figures were later shown to vary with the seam section. Recommended load densities were then quoted as 11.8 t/m^2 for thin seams, 22.6 t/m^2 for medium seams, and 29.1 t/m^2 for thick seams up to a maximum of 2.4 m.

In 1982, the British Coal Corporation rewrote its internal owner rules governing the use of powered supports, moving away from the single measurements mentioned previously. Their document set down, for the first time, instructions for determining the minimum setting and yield load densities required in the various zones of a coal face. In the face zone, minimum values were set to the following: setting MLD = 7.5 H t/m^2 and yield MLD = 15 H t/m^2 (where H is the seam section, in meters). Thus, a 1-m section requires a setting MLD of 7.5 t/m^2 and a yield MLD of 15 t/m^2; a 2.4-m section requires a setting MLD of 7.5 t/m^2 and a yield MLD of 37.5 t/m^2.

The actual designed setting and yield densities for powered support have increased over the years, reaching values well above these minimum requirements. The focus of attention has since moved from yield density to setting density, with setting densities now approaching within a few percentage points of yield densities.

The question of the validity of MLD as the sole definition of a support's operational characteristic requires closer examination. As it is an average figure, its value must be qualified by reference to the distribution of the load on a support canopy and base. However, load-distribution profiles are somewhat academic in concept because they relate to the ideal situation of a roof beam in full contact along its whole length with a strata profile of matched shape—a long way from normal operational conditions under which powered supports operate. But it does demonstrate that, for a support to operate effectively, consideration must be given to not only MLD but also canopy-load distribution. Five- and six-leg chock supports, because of their geometry, exhibit a reasonably well-distributed load profile over canopy and base. Concern about maintaining an even load distribution in the base was one of the chief reasons that chock shield designs became so popular when the advantages of lemniscate supports were recognized by the industry.

The overall pattern of strata behavior around a longwall face has been widely researched since the early 1950s, for virtually as long as full caving of waste and powered support have been in existence. Difficulties in making measurements

in the strata above a working face have always dictated that information must be gathered from a number of sources—namely, observations of convergence and pressure profiles of the support legs in situ, physical and mathematical modeling, and pure hypothesis. That the debate continued into the 1990s was symptomatic of the uncertainty that surrounded the subject.

When a stratified deposit is extracted by the longwall method, the roof of the longwall is supported temporarily by powered supports until the system has advanced sufficiently that the supports can be removed and the roof allowed to freely collapse (or cave) behind them. This volume of broken rock is known as *waste*. The thickness of roof strata that collapses is governed by the type of strata that forms the roof. Soft friable rocks such as mudstones cave easily when unsupported, breaking into relatively small fragments, the volume of which is greater than the volume it originally occupied as intact rock. This bulking process dictates the height to which caving can occur. Uncaved beds above the caving beds are known as the bridging beds. These remain intact and subside over the caved waste in a manner controlled by the support offered by the unworked mineral ahead of the face, the action of the powered supports, and the compression of the caved material in the waste. This process is known as bulking-factor-controlled caving (Gorrie and Scott 1970; Wilson 1975; B.G.D. Smart, personal communication).

Calculations show that bridging beds commence a distance of twice the height of the mineral extracted at the face above the face roof. This gives an effective value to the weight of rock for which support must be designed. Convergence on a well-supported longwall is caused by the downward movement of the bridging beds being transmitted through the immediate roof beds while they remain intact.

The support offered to the roof by the powered support maintains the integrity of the immediate roof beds but does little to prevent convergence of higher bridging beds. When stronger rocks such as sandstone and limestone form the immediate roof of the mineral being worked on a longwall, they frequently do not cave immediately behind the powered supports but rather cantilever into the waste for some distance. This puts additional load on the supports. Furthermore, when this rock eventually caves, it does not fragment as readily as does weaker material; rather, it forms large lumps in the waste with an effectively much lower bulking factor. The distance up to the bridging beds thus becomes much larger. Beds can also cave to a well-defined parting plane, and the height of the parting plane above the mineral dictates the height of caving. This phenomenon is called parting-plane caving.

As a general rule, the depth of a working from the surface does not directly affect the loading to be expected on the powered supports. Support loads experienced 200 and 1,200 m below the surface can be similar. There is, however, one surprising exception to this rule. Perhaps the most difficult circumstances for longwall support come when working shallow (<150-m depth) seams with competent overlying strata. Caving in these circumstances often extends right through to the surface, such that the powered supports are asked to support a column of strata from working level to surface. In these circumstances, supports designed to yield at ≥1,000 t are applied.

These mechanisms are derived from hypothesis rather than measurements. For this reason, engineers purchasing and installing powered supports for a new application apply a significant factor of safety to ensure that the supports do not, at some time in their working lives, become overloaded.

It has become usual in Europe, with its long history of using powered support, for decisions on new support designs to be made empirically. Empirical selection of supports based on experience in comparable strata conditions rarely fails. The chief criticism that it attracts is that supports can sometimes be overengineered. This is not necessarily a bad practice, as the extra cost of the overdesign (which is difficult to precisely quantify) serves as an insurance premium against the unknown. In all but the thickest seams, yield load densities in the United Kingdom peak at ~100 t/m^2, which normally relates to support yield loads of 650–750 t in the mid-seam range. The cost of achieving this density, for the mine operator, is high and a compromise must eventually be reached between the equally strong desires to increase loading and to restrict the cost of powered supports. Matching support size to an actual application still owes as much to experience as to science and economic reality.

The modern shield support has displayed considerable mechanical reliability and is well equipped to deal with unreliable or unknown geology. However, no amount of technology can maintain high production levels in the most faulted, weak, water-bearing strata or where seam sections reduce. The inflexibility of the longwall method will always make it vulnerable to uncertain geology.

Standardization

In the late 1980s, the European Union issued a directive covering the basic safety of machinery. An amendment to that directive highlighted powered roof supports in mines as a type of machine that requires particularly careful treatment with respect to supply within the European Community member states.

A working group comprised mostly of U.K. and German delegates representing mine operators, the U.K. Mines Inspectorate, and powered-support suppliers put together a series of basic safety standards harmonizing European practice (BS EN 1804 series 2001–2006). However, this standard does not cover the use of support underground. Safety with respect to powered supports is still judged largely by performance under test. The major implications of this exercise are not to restrict the activities of U.K. and German support manufacturers, but rather to control trading within the European Union of potentially inferior products from suppliers outside the European Union.

Design Methodologies in Soft Rock

Siddall and Gale (1992) elegantly summarize a design philosophy appropriate to the identification of support systems to be used in soft-rock environments in the following quotation.

> The science of strata control would be best considered as the application of scientific principles to certain goals:
>
> - Understanding the behavior of rock or strata under known conditions
> - Determining methods to control the strata under known conditions
> - Determining methods to predict the response of strata under conditions other than those for which current experience allows assessment

- Determining methods of controlling the strata within limits dictated by mining operational viability in the conditions predicted outside those for which current experience allows assessment

Scientific principles cannot be applied unless the problems can be defined in a scientific format and the answers obtained can be seen to explain the actual site situation. To achieve this process, scientists and engineers need to:

- Obtain experience of the mine environment and understand the key factors influencing the behavior of the environment
- Determine the factors which can be quantified and those which cannot be directly quantified and may need to be extrapolated or assessed qualitatively
- Undertake measurement of the strata behavior and the influence of mining activity
- Formulate conceptual models of the mechanics involved and the interrelationship of the variables present.

A number of key decisions are required when selecting a support type for a soft-rock excavation. The following key questions must be answered:

- How much closure can be tolerated in the excavation? This question presupposes that some amount of closure may, on occasion, be accommodated within the mining system. This requirement suggests knowing not only the degree of final closure but also the patterns of closure related to both the passage of time and the continuation of working within the excavation. To deal with closure without premature failure, support must incorporate an appropriate yield mechanism—that is, one that provides a level of resistance prior to yield that is largely maintained during the yielding process.
- Is support required for the roof, sides, and floor? In soft-rock excavations, support must be provided not just to the roof of the excavation but also to the sidewalls and on occasion to the floor.
- What are the rock excavation formation methods? In most soft-rock excavations, the time interval between rock excavation and setting of support is an important consideration. Early setting of permanent support or introduction of temporary support measures are essential to providing early resistance to movement of newly exposed rock faces. This early application of support greatly influences the convergence process.
- How significant is cost containment? This is a key element in most soft-rock mining systems, as soft-rock minerals tend to be relatively low in value and therefore production costs must be maintained within tight bounds. The use of supports is a key element in mining costs, and often compromises must be made in choosing between the best support type and an adequate but less expensive alternative.

In designing support systems for soft-rock applications, the following approaches are taken either singly or jointly.

Experience

Support-system design in soft rocks is based largely on previously proven successful practices from the same mine or neighboring mines with perceived similar conditions. It is also often based on the personal experience of individual engineers.

This is a quite satisfactory approach and can provide safe and efficient support systems. But it can also perpetuate overdesign of the support system that can prove ultimately expensive. Although it is quite correct to set additional support within a previously applied system for a new site, only a brave or foolhardy engineer would decide to reduce support density in a mining system based purely on experience without recourse to measurement or rigorous design.

Measurement

A long tradition exists of measuring strata dilation around a mining excavation. In its simplest form, this is done by in-tunnel roof-to-floor and side-to-side convergence tape measurements. In general, the degree of roof, floor, or side displacement can be measured only to the centimeter level, and high levels of accuracy are unnecessary. Patterns of behavior can be identified and variations in the support applied can be evaluated by comparison of closure measurements.

With the introduction of rock-bolted support in coal-mine roadways in the United Kingdom in the late 1980s to early 1990s, a different approach to the design of roadway support was required. Rock bolts were a technology new to both mine management and miners, and confidence had to be gained that these supports were adequate and safe. This process coincided with a general renewal of U.K. health and safety legislation underway at the time. A significant feature of the format of new mining regulations in the United Kingdom was their lack of prescriptive detail. In place of prescriptive detail were general principles and a stated requirement for mine operators to conduct risk assessments of all underground operations before commencement.

The approach adopted to safely introduce the new rock-bolt support technology is to set down a process of geotechnical risk assessment largely based on site investigation, leading to selection of a support-system design. Key elements of this process are knowledge of the geology and stress field together with in-situ experimentation to establish the bond strength between resin and strata at all major roof horizons within the bonded length of the bolt. Knowledge of the typical pattern of strata dilation in relation to its distance from the face once exposed is also needed.

The design of the support system is, in practice, empirically based and site specific, taking into account certain general parameters such as minimum bolt length to be used (1.8 m), minimum bolt density in the roof (1 bolt/m^2), hole diameter ($\leq$7 mm larger than the bolt diameter), and the need to place rock bolts as near to the face of the heading as possible and as soon as practicable after the roof is exposed. Support of the sides is now considered part of the overall design.

Rock bolts, after installation, are subject to detailed and comprehensive monitoring by measurement of strata movement and rock-bolt load. Extensometers are used to measure, with a high degree of accuracy, much smaller displacements experienced between different strata sections in the rock mass around the excavation.

Operating rules also specify the presence of routine monitoring in every roadway in which rock-bolting provided the principal support. Monitoring takes two forms: dual-height telltales and multi-horizon extensometers. Dual-height telltales are installed to at least twice the bolt length (or 4.8 m, whichever is greater) at intervals of $\leq$20 m along the roadway.

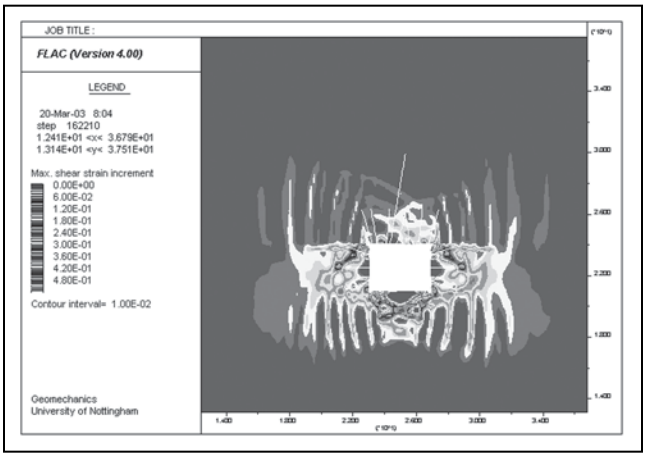


Figure 8.7-16 Typical FLAC output for stress distribution around a mine roadway

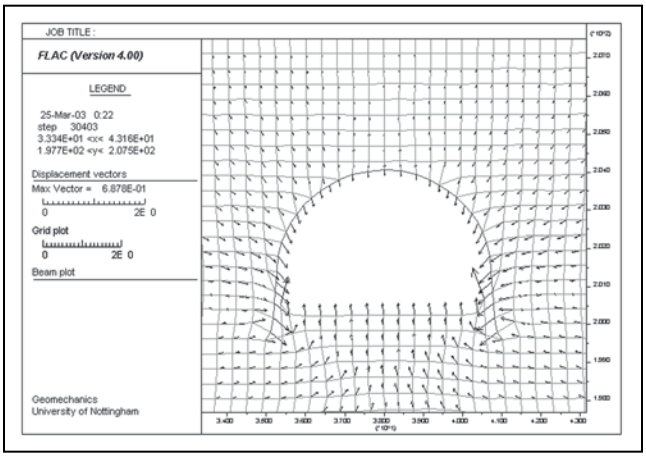


Figure 8.7-17 Typical FLAC output for strata displacement around a mine roadway

Additional telltales are required at every junction and other places where roof instability is known or suspected. Multi-horizon extensometers reading ground movement to ≥7 m above the roof of the roadway are installed at maximum intervals of 200 m. More importantly, the mine manager is required to put into operation a scheme in which these routine monitoring stations are read and reported upon. Supervisors at the mine are given action levels of movement to measure against and actions to take if these levels are exceeded (Great Britain Health and Safety Commission 1996).

Physical Modeling

In the early years of strata-control research, a number of key mining research institutes studied the mechanisms of strata movement and stress redistribution around mining excavations, especially longwall faces, by physical modeling of mining situations using plaster models loaded in large frames. This work was conducted in a number of coal-mining countries, and in its heyday was used to qualitatively explain the mechanisms of displacement that take place away from the mining excavation.

These physical models were expensive in terms of resources, and the results were often limited. The models were heavily dependent on the size of the model domain, the assumptions made, and the scale effects for work of this type. The results gave at best a broad qualitative view of large-scale rock-mass response to mining. They were far less effective in providing practical data on which a support system can be designed.

Numerical Modeling

Before the computer era, analytical solutions were developed that provided a conceptual understanding of rock-mass behavior. However, in most cases they assumed rock to be a linear elastic material, which is a large simplification, especially in soft-rock regimes. For many situations, they could solve only problems that had simple geometry and low dimensionality. Unfortunately, rock masses are generally anisotropic, inhomogeneous, and inelastic due to the presence of geological discontinuities such as joints, fissures, and faults. Most soft-rock mining problems are nonlinear, due to high levels of deformation, and involve complex shapes and boundary conditions that often cannot be described by simple mathematical functions.

With the development of high-speed personal computers in the last two decades, powerful computer-based numerical modeling techniques have become readily available for application to geotechnical materials. Numerical methods have become very popular in academia and, more recently, in industry for solving problems in rock mechanics. The development of these numerical methods has provided the design engineer with a way to predict the stress, displacement, and failure mechanisms of zones adjacent to an excavation formed within rock masses that exhibit complex nonlinear constitutive behavior.

A number of modeling-software products exist for solving geological and geotechnical problems. Both ABAQUS (for finite element analysis) and FLAC (Fast Lagrangian Analysis of Continua) software for finite-difference analysis are popular for determining the stress conditions and behavior of rock masses surrounding an underground excavation. FLAC has been used successfully for many underground problems in rock mechanics and is considered to be an industry standard in both its two-dimensional (Figures 8.7-16 and 8.7-17) and three-dimensional formats.

However, no matter how good a model is, a good prediction cannot be obtained unless the input data is of high quality. Unlike engineering materials, rock mass generally contains a lot of irregular discontinuities in the form of joints, bedding planes, fissures, and faults.

The engineering properties of in-situ rock strata are determined by many factors, such as intact rock properties, nature and orientation of planes of weakness, and presence of groundwater. The stiffness and strength of intact rock can be determined in the laboratory, providing some information about the upper limit of the engineering properties of the rock mass. However, it is practically impossible to determine all of the properties of the many discontinuities by direct testing and measurement. Therefore, the engineering properties of the rock mass may be estimated by means of a rock mass classification system to supplement direct measurement by testing.

Fortunately, several well-recognized engineering rock mass classification systems have been widely used to estimate the strength and stiffness of rock mass, including its discontinuities. As a powerful aid in design, rock mass classification has become an integral part of the empirical design methods popular in the industry and has been invaluable on many

underground construction and mining projects. However, rock mass classification systems alone are not the ultimate solution to design problems and they are now being applied in conjunction with numerical methods.

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CHAPTER 8.8

Ground Control Using Cable and Rock Bolting

William F. Bawden

INTRODUCTION

Ground control is the art and science of maintaining safe and stable conditions at the surface of an excavation in a rock mass at or below the ground surface for the intended service life of the structure. Aggressive mining strategies, design constraints on excavation geometry, or unfavorable arrangements of rock stress, structure, or material properties often lead to the requirement for some form of rock mass improvement to maintain safe and stable conditions. Methods for achieving this improvement are given in numerous standard textbooks and conference proceedings (e.g., Hoek and Bray 1974; Hoek and Brown 1980; Stillborg 1986; Stephansson 1983).

The term *support* is widely used to describe procedures and materials that improve the stability and maintain the load-bearing capability of rock near the boundaries of surface and underground excavations. Rock improvement includes all techniques that seek to increase the strength and decrease the deformability characteristics of the rock mass. This includes the injection of chemical or cementitious grouts, ground freezing, presplitting, and drainage. Within the general category of rock improvement, rock reinforcement is a specific technique with the prime objective of improving the shear and tensile strength of the rock mass adjacent to surface and underground excavations. The term *support* can then be reserved for use in those cases in which the rock mass is truly supported by structural elements that carry, in whole or in part, the weights of individual rock blocks isolated by individual discontinuities or zones of loosened rock (Brady and Brown 2004).

Support and reinforcement are essential components in the design of all surface and underground excavations and are often combined to provide an overall or integrated system. In mining, they enable more economic extraction through steeper slopes in open pits or increased recovery rates from underground operations. In civil engineering, they enable increased excavation rates in tunnels and safer permanent surface and underground facilities (Windsor and Thompson 1993). In this chapter, the dominant reinforcement and support elements will be discussed along with installation procedures and major quality assurance/quality control (QA/QC) issues. Initially, however, some basic terminology will be defined in an attempt to minimize confusion due to the divergent jargon that has evolved in the industry.

ROCK REINFORCEMENT TECHNIQUES

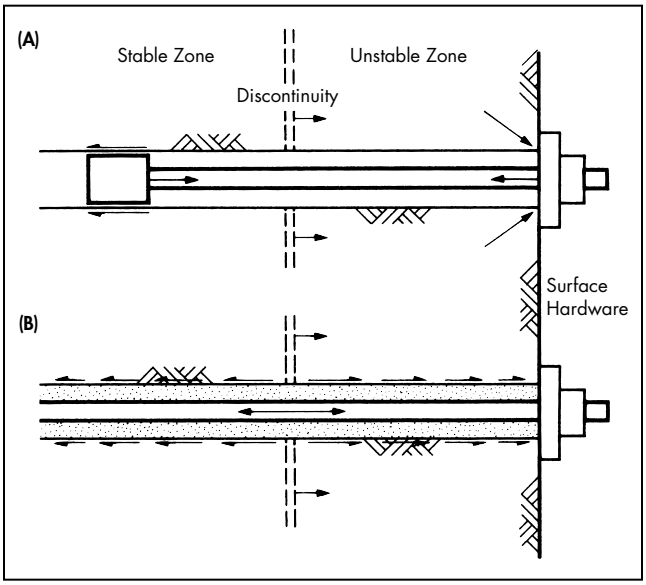
Rock Support Versus Reinforcement

Rock reinforcement is often mistakenly taken to include methods more properly termed support. It is essential to the understanding of the underlying mechanics to differentiate between the concepts of reinforcement and support. The primary objective of rock mass reinforcement is to mobilize and conserve the inherent strength of the rock mass so that it becomes self-supporting. Ground support, on the other hand, should be restricted to structural elements that directly carry the weight or part thereof of discrete rock blocks.

Support and reinforcement should also be classified as active or passive. Active support imposes a predetermined load to the rock surface during installation and can take the form of tensioned rock bolts or cables, hydraulic props, expandable segmented concrete linings, or powered support for longwall faces. Active support is usually required to support the gravity loads imposed by individual rock blocks or loosened zones. A substantial body of work has been undertaken in the Australian coal industry to demonstrate that active support via bolt and cable pre-tension is a critical aspect of strata reinforcement where the objective is to control in-situ ground stresses rather than the weight of potentially unstable strata. Passive support or reinforcement is not installed with an applied loading but develops its loads as the rock mass deforms. In fact, many reinforcement schemes only operate when the rock mass reaches yield. Passive support may be provided by steel arches, timbered sets, or composite packs; or by untensioned, grouted rock bolts, reinforcing bars, or cables, these three often described as dowels.

Strata control describes the support and reinforcement techniques used in coal mining. Although the term is suitable, because it evokes the concept of the control or limitation of strata displacements rather than support, the major function of some strata-control measures is support (Brady and Brown 2004).

William F. Bawden, Pierre Lassonde Chair in Mining Engineering, University of Toronto, Ontario, Canada



Source: Windsor and Thompson 1993.

Figure 8.8-1 Reinforcement load transfer concept: (A) discrete load transfer and (B) continuous load transfer

Load Transfer Concepts for Reinforcement

Fundamental to the understanding of rock reinforcement behavior and the action of different elements and their effect on excavation stability is the load transfer concept. This is shown schematically in Figure 8.8-1 and is composed of three basic mechanisms:

1. Rock movement that requires load transfer from the unstable rock to the reinforcing element
2. Transfer of load via the reinforcing element from the unstable surface region to the stable interior region
3. Transfer of the reinforcing element load to the stable rock mass

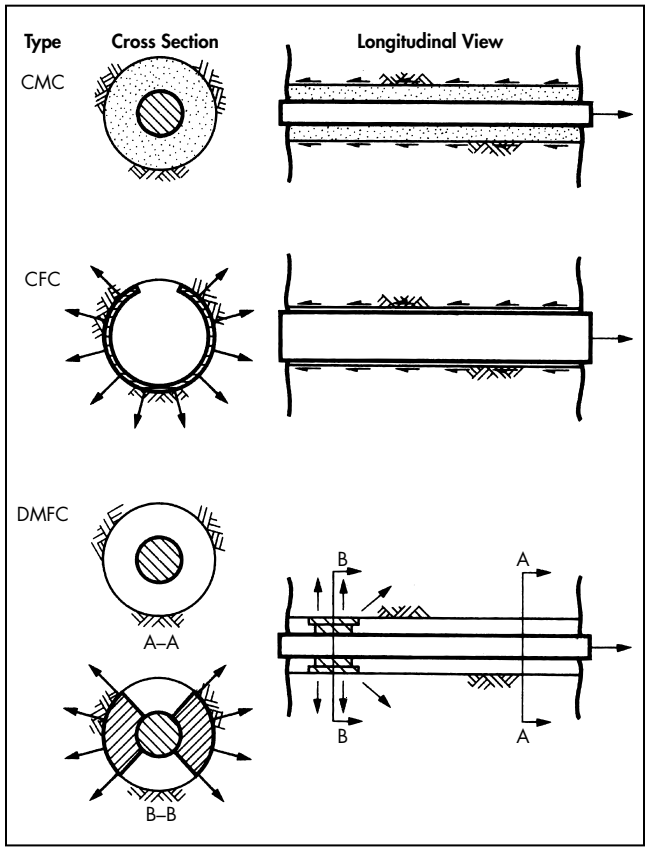
Reinforcing element length can be considered in three general ranges associated with perceived unstable surface regions for different surface and underground excavations: near surface (1.5–3.0 m long), medium depth (3–15 m long), and deep seated (>15 m long) (Windsor and Thompson 1993).

Load transfer can occur by numerous methods, and many different reinforcing devices have been developed. In general, these devices are solid bar, hollow tube, or wire rope elements that are installed within boreholes drilled into the rock. A terminology has been proposed (Windsor and Thompson 1993) that places load-transfer mechanisms for these devices within one of three categories, as defined in Figure 8.8-2. These categories are

1. Continuously mechanically coupled (CMC),
2. Continuous frictionally coupled (CFC), and
3. Discrete mechanically and frictionally coupled (DMFC).

Continuous Mechanically Coupled Elements

These elements rely on a securing agent that fills the annulus between the element and the borehole wall. Generally known as grout, the agent is placed as a fluid and requires a finite setting time before the element is able to provide support. The setting time can vary from a few seconds to several hours



Source: Windsor and Thompson 1993.

Figure 8.8-2 Categories of element load transfer

depending on the type of grout used, which may well affect selection of the device. The major function of the grout is to provide a mechanism for load transfer between the rock and the reinforcing element (CMC in Figure 8.8-2). Reinforcing elements used with grouts are often manufactured with variable cross-sectional shapes. This variation causes a mechanical interlock between the element and the grout and strongly impacts the shear resistance, or bond strength, and the load-transfer stiffness that can develop at the element–grout interface.

Continuous Frictionally Coupled Elements

CFC elements behave similarly to CMC elements, except the CFC element is placed in direct contact with the rock (CFC in Figure 8.8-2). Load transfer results from friction between the element and the borehole and is limited by the radial prestress set up at installation. Any geometrical key that is present occurs by chance due to irregularities in the borehole wall.

Devices in this class consist either of expansion of an undersized section into a larger borehole or contraction of an oversized section into a smaller borehole (see "Frictional Support" section). Borehole size is critical to support performance in this class.

Discrete Mechanically and Frictionally Coupled Elements

DMFC elements tend to be simple bars with either a deformed end region or provision at one end for attaching an expanding anchor and provide either mechanical or frictional load transfer over this relatively short portion of the entire element

length (DMFC in Figure 8.8-2). This interval is commonly known as the anchorage length and is usually limited to less than 500 mm for grouted anchorages and 200 mm for expansion anchorages. The anchorage must have sufficient capacity to mobilize the full material strength of the element. Expansion shell bolts are the most common DMFC element (see the "Ungrouted Mechanically Anchored Rock Bolts" section). With these elements the anchor is expanded outward against the borehole wall because of tension in the element, and anchorage strength may be limited by the strength of the host rock. Grouted anchorages have a lower unit load transfer and offer advantages in weaker and softer rocks. In general, only resinous grout can provide the high-strength requirements for shorter anchorages.

Pre- and Postexcavation Reinforcement and Support Pre-tensioning

In certain applications it may be advantageous to install support elements prior to excavation (i.e., pre-reinforcement), while in other cases it may be better to install the support elements at some discrete time following excavation. In the former case, support elements may better restrain rock mass movements but may be required to sustain higher loads. In the latter case, ultimate support element loads can be controlled by allowing some initial rock movement. This is common in tunneling using concrete linings or shotcrete in weak strata, where one objective is to ensure the long-term integrity of the permanent support (e.g., the New Austrian Tunnelling Method). Rock-support interaction theory provides a framework for understanding the impact of pre- versus postexcavation practice and is covered in numerous standard rock mechanics textbooks (e.g., Brady and Brown 2004; Hoek et al. 1995). The limitations of rock-support interaction theory, however, must be clearly understood, particularly that subsequent extraction processes change the ground stresses, which therefore may require higher levels of support than development conditions.

In certain applications, it is also advantageous to provide the reinforcing element with an initial pre-tension. Posttensioning is the tensioning or retensioning of elements subsequent to installation. It is important to understand that additional tension is likely to develop in the element because of rock mass deformations in response to subsequent excavation, stress change, creep, and so forth, and to allow for this in order to avoid subsequent overstress and rupture of the support elements (Windsor and Thompson 1993).

Encapsulation Techniques

Encapsulation materials can generally be classified as cementitious, usually normal portland cement or resinous, with the latter generally being stronger. Cement grouts are suited to longer reinforcing elements because of availability, ease of mixing, and placement. Resins are restricted to shorter holes primarily because of difficulties in mixing resins at lengths exceeding 3–4 m. A benefit of encapsulation is increased resistance to corrosion, although in highly corrosive environments additional protection beyond simple encapsulation is required.

Resin-grouted reinforcement has found wide application in discrete load-transfer elements. Set times can be reduced to less than 10 seconds, which is particularly important for cyclic and fast advance mining methods where support installation is one of the controls on advance rate. It must be understood, however, that many other factors (e.g., machine availability, operator training, and cutting rates) also strongly influence advance rates. In addition, the anchors can supply design capacity within a few minutes of installation, which improves safety during advance. High early-strength grouts are also available in cartridges (Windsor and Thompson 1993).

Auxiliary Fittings

Various forms of auxiliary fittings have evolved for use with different reinforcing elements that cater to different geological conditions. The fittings are either attached to the reinforcing element at the rock surface or are used to modify the internal response of CMC elements. Special equipment and procedures are required for tensioned applications.

External fittings (e.g., plates, straps, mesh) can be attached to the reinforcing element to provide varying degrees of surface restraint to the rock mass. External fittings are mandatory for discrete coupled devices, while internal fittings are used primarily with CMC devices and may comprise intermittent anchors and/or decoupling sleeves. These combinations of devices can be used to engineer the stiffness of the reinforcing elements to match anticipated rock mass deformations based on predicted failure mechanisms.

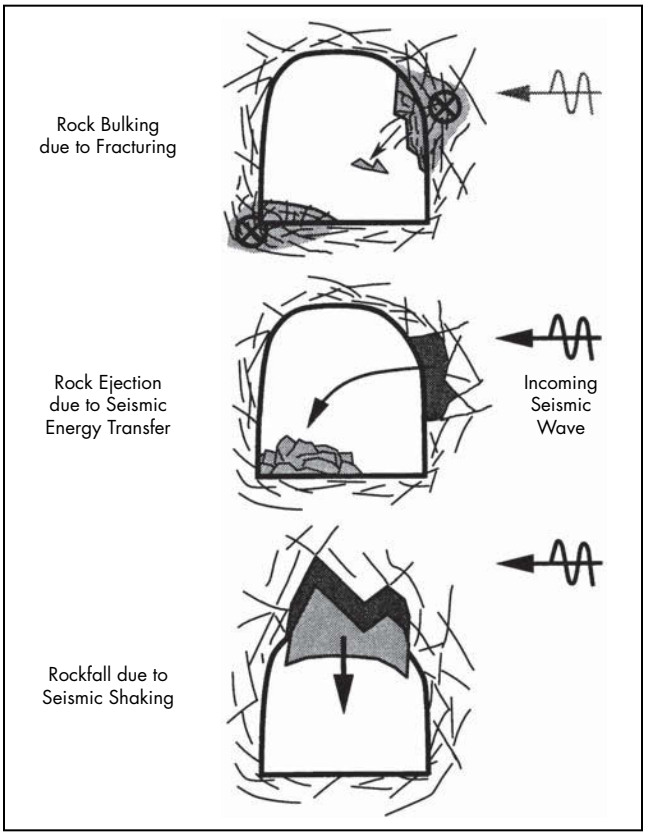
In most applications the number of reinforcing elements in relation to the exposed excavation area is quite low. In closely jointed, fissile, or weathered rock masses where unraveling may be the dominant failure mode, stability can be improved by installing structural elements that span the reinforcing elements. Such support forms a surface retention system that restrains the movement of small blocks and provides a reactive stress to the free span area. This stress is usually low for mesh applications but will be higher for shotcrete. The concept of surface retention support on a larger scale using reinforcement as anchors is important and holds the key to supporting many of the larger, more complex failure modes, particularly for surface excavations (Windsor and Thompson 1993).

Static Versus Dynamic Support

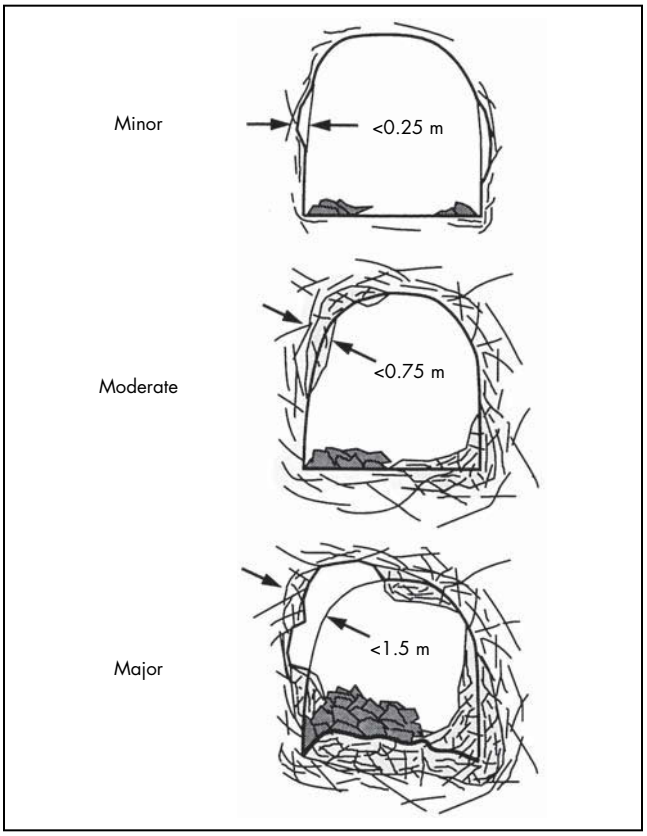
The majority of ground support applications are designed for static rock mass conditions, which may include both simple gravity-driven (e.g., wedge failure and unraveling) and mine-induced stress-driven failure. In the latter case, a stable failure process occurs where most of the stored strain energy in the rock mass is consumed during the fracture and deformation process. Ground support must be designed with sufficient capacity for both anticipated loads and deformations; however, these will accumulate over time, and energy absorption will not be a concern.

Dynamic support elements are generally restricted to ground subject to rock-burst loading. First are some basic definitions (Kaiser et al. 1996):

- A seismic event is a transient earth motion caused by a sudden release of potential or stored strain energy in the rock. As a result, seismic energy is radiated in the form of strain waves. The magnitude of a seismic event is usually determined from the peak amplitude of the strain wave, using a logarithmic scale (e.g., Nuttli or Richter scale).
- A rock burst is a seismic event that causes injury to persons or damage to underground workings. The general and essential feature of rock bursts is their sudden, violent nature.



Source: Kaiser et al. 1996.
Figure 8.8-3 Rock burst damage mechanisms



Source: Kaiser et al. 1996.
Figure 8.8-4 Rock burst damage levels

Rock bursts are generally divided into three classifications:

1. **Strain bursts:** These are caused by high-stress concentrations at the edge of mine openings that exceed the strength of the rock. Events can range from small slivers of rock being ejected from the walls to the collapse of a complete wall as it tries to achieve a more stable shape.
2. **Pillar bursts:** Severe rock bursts, involving thousands of tons, have been caused by the complete collapse of support pillars. These tend to occur in extensively mined-out areas, and the resulting damage can be severe.
3. **Fault slip bursts:** Recognized in the 1980s, this type of rock burst occurs when slippage suddenly occurs along a geological weakness plane. This is the same mechanism as for an earthquake.

Rock-burst magnitude (and hence energy and damage potential) increases from classifications 1 to 3. The damage may occur as rock bulking due to fracturing, rock ejection by seismic energy, or rockfalls due to seismic shaking (Figure 8.8-3). Each mechanism may result in different levels of damage to an excavation and its support system. The damage severity depends on many factors, including

- Failure potential near the opening (i.e., the level of existing wall stress compared to the rock mass strength, which, in turn, depends on rock wall quality);
- Support effectiveness;
- Local rock stiffness;
- Magnitude of seismically induced stress, rock accelerations, or velocities;
- Opening geometry, size, and orientation; and
- Geological structure.

Three levels of rock burst damage severity are defined, as shown in Figure 8.8-4. Damage level determination is usually based on observations of previous damage, where such observations are available, plus analytical methods. The three severity levels are discussed later and are summarized in Table 8.8-1.

Only a limited number of ground support elements are suitable for dynamic loading conditions (see "Spiling" section), and under the most severe conditions even these are restricted by maximum practical support limitations.

ROCK REINFORCEMENT SUPPORT ELEMENTS

Rock reinforcement elements can be obtained in various lengths. The most common are near-surface support elements, generally ≤3 m long, that provide support to the inner shell of the opening and are commonly referred to as primary support. Deeper ground reinforcement can only be achieved either by using cable bolts or by connecting shorter, stiff bolt elements together as discussed later in this section. Such deep reinforcement is commonly referred to as secondary support.

Near-Surface Static Reinforcement (≤3 m depth)

Ungrouted Mechanically Anchored Rock Bolts

Mechanically anchored reinforcement elements, commonly referred to as point anchor bolts or expansion shell bolts, fall into the DMFC category and represent one of the earliest developments in rock mass reinforcement. These elements

Table 8.8-1 Rock-burst damage mechanisms and the nature of anticipated damage

Damage Mechanism	Cause of Rock-Burst Damage	Damage Severity	Thickness, m	Weight, kN/m²*	Closure,† mm	V_e,‡ m/s	Energy, kJ/m²
Bulking without ejection	Highly stressed rock with little excess stored strain energy	Minor	<0.25	<7	15	<1.5	Not critical
		Moderate	<0.75	<20	30	<1.5	Not critical
		Major	<1.5	<50	60	<1.5	Not critical
Bulking causing ejection	Highly stressed rock with significant excess strain energy	Minor	<0.25	<7	50	1.5–3	Not critical
		Moderate	<0.75	<20	150	1.5–3	2–10
		Major	<1.5	<50	300	1.5–3	5–25
Ejection by remote seismic event	Seismic energy transfer to jointed or broken rock	Minor	<0.25	<7	<150	>3	3–10
		Moderate	<0.75	<20	<300	>3	10–20
		Major	<1.5	<50	>300	>3	20–50
Rockfall	Inadequate strength, forces increased by seismic acceleration	Minor	<0.25	$<7g/(a + g)$	NA§	NA	NA
		Moderate	<0.75	$<20(a + g)$	NA	NA	NA
		Major	<1.5	$<50\ (a + g)$	NA	NA	NA

Source: Kaiser et al. 1996.
*$a \times g$ are seismic and gravitational accelerations.
†Closure expected with an effective support system.
‡Ve = velocity of displaced or ejected rock.
§NA = not applicable.

apply a positive force to the rock without requiring movement of the rock mass. They are composed of three elements: (1) an expansion shell anchor, (2) a faceplate assembly, and (3) a connecting steel bar.

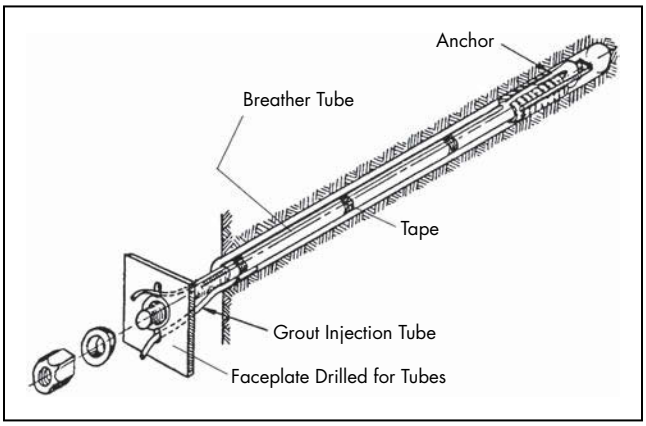
Expansion shell anchors consist of a tapered cone with internal thread plus a pair of wedges held with bail wire. The cone is screwed onto the threaded end of the bolt and the assembly is inserted into the borehole. The hole must be at least 100 mm longer than the bolt or the bail wire will be dislodged and the anchor will not set. Once in place, the anchor is seated with a sharp pull on the bar. Tightening the bolt forces the cone into the wedge, increasing the anchor force. Expansion shell anchors can work well in hard rock but are not good in closely jointed or soft rocks because of the potential for deformation and failure of the rock in contact with the grips.

At the bolt head, either a fixed head (forged head) or threaded end and nut system can be used. A faceplate is always required to distribute load from the bolt to the rock face. A tapered washer or conical seat is required to compensate for the rock surface not being perpendicular to the bolt (i.e., tightening the nut must place the bolt in direct axial tension). A wide variety of faceplates and domed or tapered washers are available. Details of these assemblies are available from the various bolt manufacturers.

The bar element in a mechanically anchored rock bolt is a smooth circular element. Steel grade and bar diameter can be varied to adjust reinforcement element capacity and stiffness.

Grouted Mechanically Anchored Rock Bolts

Mechanically anchored bolts can be changed from DMFC to a CMC category by filling the annulus between the steel element and the borehole wall with grout. Cement grout is used for this purpose and can be emplaced by the use of grout tubes inserted with the bolt (Figure 8.8-5) or through the use of a hollow stem bolt. Grout tubes are not commonly used with mechanically anchored bolts since space constraints restrict the grout tube diameter and therefore the water/cement (w/c) ratio of the grout that can be used (a w/c = 0.4 grout should be used). With a hollow stem bolt, grout is pumped through the hollow core of the bolt and is forced back through the annulus.



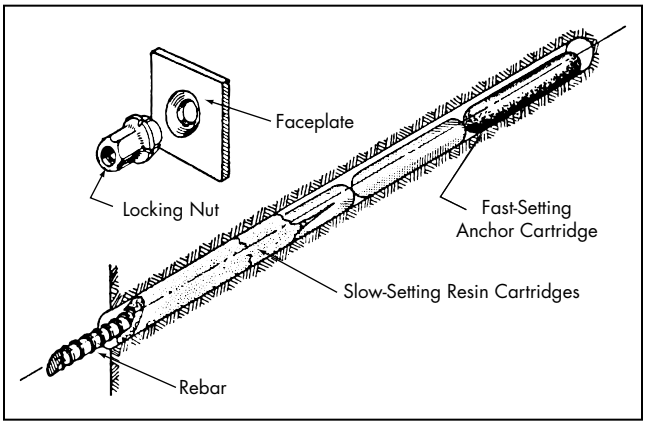
Source: Hoek et al. 1995.
Figure 8.8-5 Grout injection arrangements for mechanically anchored rock bolt

Grouting mechanically anchored bolts provides corrosion protection, stops anchor slip, and ensures that the full bolt capacity can be achieved. Hollow core bolts are available from a number of manufacturers.

Quality Assurance and Control

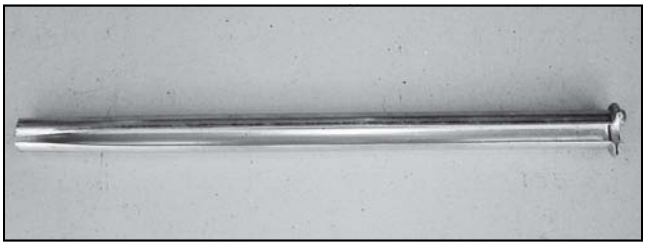
QA/QC is critical for all rock reinforcement elements. Tensioning of rock bolts is critical to ensure all components are in contact and that a positive force is applied to the bolt. Where bolts are required to carry significant load, they are generally tensioned to about 70% of yield. This provides a known load with reserve for additional load induced by rock movements.

Corrosion is one of the primary causes of bolt failure. This can be counteracted by grouting the bolts. The use of hollow stem bolts (e.g., Stelpipe bolt) makes this function simple to accomplish. It is nevertheless a two-stage process and therefore impacts cost. The primary purpose in grouting the bolts is corrosion protection and to lock the anchor in place to stop loosening from blasting. Grout strength is not too



Source: Hoek et al. 1995.

Figure 8.8-6 Typical setup for creating a resin-anchored and grouted pre-tensioned rock bolt



Courtesy of Mansour Mining, Sudbury, Canada.

Figure 8.8-7 Split Set friction stabilizer

important and higher w/c grout can be used (i.e., in the region of 0.4 w/c).

Ungrouted bolts must be retensioned after each blast for several rounds behind the face since blast vibration will cause anchor slip and loss of prestress. A torque wrench can be used to test bolt tension.

Beveled or conical seat washers must be used to ensure that tensioning of the bolt results in increased axial tension in the bar and subsequent seating of the anchor. Otherwise, for applications not perfectly perpendicular to the rock face, high torque on the nut will result from friction between the nut and the plate rather than tension in the bar.

Resin-Anchored Rebar Bolts

Pre-tensioned bolts. Ungrouted mechanically anchored bolts can work loose when subjected to blast vibrations or when anchored in weak rock. Additionally, they are subject to corrosion and provide only relatively soft reinforcement. Resin-anchored rebar overcomes these difficulties. Pre-tensioned rock bolt applications use two-component cartridges containing a resin and catalyst in separate compartments (Figure 8.8-6). The resin cartridges are inserted in the drill hole ahead of the bolt element, which is then spun into them, mixing the catalyst and the resin. Setting of the resin occurs in a few minutes (depending on mix specifications), and a strong anchor is created. The system works for most rock types, including weak rocks. Fully resin bonded bolts should be used for corrosion protection and long-term support.

Pre-tensioning of the reinforcing element is achieved by using a combination of slow- and fast-setting resins. In a pre-tensioning application, a fast-setting resin (~10-second set time) is inserted first, followed by slow-setting resin (e.g., 10 minutes) for the remainder of the hole. The bolt can be tensioned 2 to 3 minutes after installation. The bar element is a deformed reinforcing bar with the toe end cut at about 45 degrees so as to adequately puncture the resin cartridges. The high resin unit cost is offset by the installation speed of the tensioned, fully grouted bolt.

Dowels. Dowels are rock reinforcement elements installed passively in a borehole (i.e., no prestress). Such elements require movement of the rock mass in order to develop a restraining force. Dowels work well when applied close to the advancing face and/or where later mine-induced stress change will cause movement of the rock mass. The most common form is a resin-grouted dowel. In North America, this consists of a resin-grouted rebar dowel where single set time resin is used for full dowel encapsulation. Resin-grouted dowels and rock bolts are in the CMC category of support. Resin-grouted rebar forms a very stiff support, the capacity of which is dictated by the type of steel and the rebar diameter.

Quality assurance and control. Quality control issues with resin rebar support primarily revolve around problems with the resin. Resin QA/QC problems include the following:

- Shelf life is limited (may be as short as 6 months).
- Resins must be stored under conditions conforming to manufacturer's recommendations, including storage temperature and conditions. Freezing or excessive temperature may impact resin characteristics.
- Effective mixing of the resin can be a problem. Cutting the bolt rod at an angle to form a sharp point will help properly puncture the resin cartridges.
- The allowable length of time or number of rotations for spinning the resin is limited. Additional spinning destroys the resin structure and weakens the overall installation (follow the manufacturer instructions in this area).
- Hole diameter is critical, because it exerts a strong control on bond strength and load transfer stiffness.
- In some weak argillaceous rocks, the drill-hole surface can become clay coated and the resin will spin in the hole, causing incomplete mixing.
- The resin initially has a low viscosity, and in highly fractured rock the resin may seep into fractures, leaving voids in resin column.
- The reaction of resins with aggressive groundwater is not certain.

Frictional Support

Split Set bolts. Split Set stabilizers consist of a slotted, hollow high-strength steel tube and faceplate (Figure 8.8-7). The bolts are installed by pushing them into a slightly undersized hole. The radial spring force generated by compression of a C-shaped tube provides frictional anchorage along the complete length of tube. The toe end of the tube is beveled inward to facilitate installation, and the system is quick and easy to install. It can be particularly useful in mild rock-burst conditions where the bolt will slip and not break, and when used with mesh will retain broken rock generated by a mild burst. These bolts can also be used to pin up mesh in highly fractured and yielded ground prior to the application of shotcrete. Provided the load demand does not exceed capacity, this system can work well for short-term support. However, susceptibility to corrosion rules out this system for most long-term support requirements. The bolts are installed as dowels

Table 8.8-2 Typical Split Set bolt properties

	Bolt Type			
Description	**FS33**	**FS35**	**FS39**	**FS46**
Actual bolt diameter, mm	34	35.5	39	46
Weight, kg/m	1.64	1.64	1.88	2.8
Minimum ultimate strength, kN	81.65	81.65	90.7	136.1
Recommended initial load, kN	27–54	27–54	27–54	54–90
Recommended bit diameter, mm	31–32	31–33	35–38	41–45
Bolt length, m	0.45–2.4	0.45–2.4	0.45–3	0.9–3.7

Courtesy of Mansour Mining, Sudbury, Canada.

Table 8.8-3 Swellex premium line

	Bolt Type				
Description	**Pm12**	**Pm16**	**Pm24**	**Pm24C**	**Pm24H**
Type of steel			S275JR/S355MC		
Minimum breaking load, kN	110	160	240	200	200
Minimum yield load, kN	100	140	200	190	190
Minimum elongation, %*	10	10	10	10	10
Inflation pressure, bar	300	300	300	300	300
Profile diameter, mm	27.5	36.0	36.0	36.0	36.0
Original tube diameter, mm	41	54	54	54	54
Material thickness, mm	2.0	2.0	3.0	3.0	3.0
Upper bushing diameter, mm	28	38	38	38	38
Inflation bushing diameter, mm	30.0/36.0	41.0/48.0	41.0/48.0	41.0/48.0	41.0/48.0
Optimal hole diameters, mm†	35–38	45–51	45–51	45–51	45–51
Range hole diameters, mm	32–39	43–52	43–52	43–52	43–52
Connection diameter including welds, mm	NA‡	NA	NA	45	NA
Recommended collar diameter, mm	NA	NA	NA	NA	53–55
Bitumen-coated option	Yes	Yes	Yes	Yes	Yes
Plastic-coated option	Yes	Yes	Yes	Yes	Yes

Source: Adapted from Atlas Copco 2000–2009.
*A5. According to SS-EN 10002-1 (2009), where applicable.
†Typical values.
‡NA = not applicable.

and fall in the CMC category. Table 8.8-2 provides typical Split Set bolt specifications (these may vary for different manufacturers). Split Set bolt pull-out capacity is rated in metric tons per meter (t/m) and is derived from in-situ pull tests. Nominal pull-out strengths vary from 1 to 2 t/m.

Swellex bolts. The original Swellex dowel came in varying lengths and consisted of a 42-mm-diameter tube that is folded into a 25–28-mm diameter during manufacture. When the dowel is inserted into a 32–39-mm drill hole, no push force is required. The dowel is activated by injection of high-pressure water (about 30 MPa), which inflates the folded tube into intimate contact with the walls of the drill hole. In 1993, the original Swellex dowel was replaced with the EXL Swellex manufactured from high-strength but ductile steel. Present products consist of the Swellex premium (Pm) line (Table 8.8-3) and the Swellex manganese (Mn) line (Table 8.8-4) of bolts. The bolts are available in various lengths up to 7 m and in coated versions for corrosion protection. One of the most important features of the Swellex system is the high bolt ductility. The Pm line of bolts can withstand 10% strain while the Mn line can withstand 20% strain. This high ductility without loss of capacity makes the Swellex

Table 8.8-4 Swellex manganese Line

	Bolt Type		
Description	**Mn12**	**Mn16**	**Mnb24**
Type of steel	S355JR	S355JR	S355JR
Minimum breaking load, kN	110	160	200
Minimum yield load, kN	90	120	180
Minimum elongation, %*	20	20	20
Inflation pressure, bar	300	300	300
Profile diameter, mm	27.5	36	36
Original tube diameter, mm	41	54	54
Material thickness, mm	2.0	2.0	3.0
Upper bushing diameter, mm	28	38	38
Inflation bushing diameter, mm	30/36	41/48	1/48
Optimal hole diameters, mm†	36–38	45–51	45–51
Range hole diameters, mm	32–39	43–52	43–52
Bitumen-coated option	Yes	Yes	Yes

Source: Adapted from Atlas Copco 2000–2009.
*A5. According to SS-EN 10002-1 (2009), where applicable.
†Typical values.

bolt useful in rock-burst and high-closure applications. An additional advantage of this system is the speed of installation compared to conventional rock bolts and cement-grouted dowels.

Quality Assurance and Control

Split Set bolts. Corrosion is a potential problem with all Split Set bolts because of the thin steel section of the tube that is in direct contact with the rock. Although galvanizing the tube helps delay corrosion, much of the galvanizing may be scraped off during installation, and the impact of this on bolt life is uncertain. These bolts are generally not suitable for long-term support and should not be used in areas with aggressive groundwater. Split Set bolts have limited capacity and type SS 39 will only mobilize about 5 t on pullout. In many cases, actual field loading conditions will reduce active capacity to no more than 50% of the pullout load. The bolt capacity is sensitive to hole gauge, and the bolts have extremely low shear resistance due to the limited steel cross-sectional area.

Swellex bolts. Corrosion of Swellex dowels is of concern because the outer surface is in direct contact with rock and the steel section of the tube is thin. Coated Swellex bolts are now available that help to reduce corrosion susceptibility. The Swellex coatings should be less impacted by installation procedures than is the case with galvanized Split Set bolts. The long-term support characteristics of Swellex bolts, particularly in aggressive groundwater conditions, are nevertheless unknown. Swellex dowels can be tested for corrosion by repressurizing the bolts; if the bolt holds the pressure, it has not been perforated by corrosion. These bolts are also sensitive to hole gauge and are weak in shear due to the limited steel cross-sectional area.

Spiling

Spiling is a method of advancing a tunnel in collapsing soil or very loose rock (e.g., poor ground, fault gouge). Low-angle holes are drilled ahead of the face using a sacrificial drill bit and hollow rods that may be coupled to achieve the desired length. Grout is then injected through the hollow rods to consolidate the ground between the closely spaced anchors, forming a reinforced umbrella or diaphragm, and the next tunnel round is then advanced below this support. Following the advance, additional ground support is installed and another set of anchors is drilled and grouted, and the tunnel is again advanced. The self-drilling anchors are critical to this process. Numerous bit sizes and styles are available for varying conditions. Grout may be injected through perforations in the bit, or the hollow rods may also be perforated. There are numerous manufacturers of such systems.

Near-Surface Static Design Guidelines

Ground support design is a complex topic and a specialist area, the details of which are well beyond the scope of this chapter. At the simplest level, for static design, there are empirical design techniques that are relatively easy to apply (providing one fully understands the limitations of the supporting database and associated design methodology), and examples of these are provided in this section. The most basic design uses rules of thumb such as those provided by the U.S. Army Corps of Engineers (Table 8.8-5). Empirical ground support design charts have also been developed around rock mass classification charts (e.g., the rock mass rating system and tunnelling rock mass quality index [Q] system—see Chapter 8.4). Figure 8.8-8 shows the chart for estimating support categories based on tunneling quality index Q (Palmstrom and Broch 2006). More sophisticated support designs incorporate various numerical modeling techniques (e.g., Unwedge, Phase2, Flac3D). Discussion of these techniques is beyond the scope of this chapter.

Near-Surface Dynamic Reinforcement Elements

Near-surface reinforcement elements suitable for use under dynamic loading include cone bolts, Swellex bolts, and Split Set bolts:

- Cone bolts were originally developed in South Africa and were later modified by Noranda Technology for use in North America as a resin bolt. The MCB33 (Modified Cone Bolt for 33-mm holes) cone bolt is a smooth bar threaded at one end, with forged cone and mixing blade at the other end and is manufactured and distributed by Mansour Mining in Sudbury, Canada. It is installed in a specially formulated Coneloc or Fasloc resin and is mixed by the resin mixing blade. A plastic sleeve is installed over the shaft of the bolt to debond the resin from the bar, thus allowing the cone to yield or plough through the resin. The cone bolt load-displacement characteristic curve is shown in Figure 8.8-9. Cone bolts show high-ductility and energy absorption characteristics. Cone bolts should provide high-shear resistance.
- Swellex bolts have high-ductility (10%–20% strain, assuming they are anchored beyond the bulking zone) and high energy absorption characteristics. Swellex bolts are weak in shear and, if they become locked in the hole, the theoretical ductility may not be achieved.
- Split Set bolts show high ductility but low energy adsorption and low-load capacity (Figure 8.8-9). The bolts are weak in shear and if they become locked in the hole will rupture at low loads.

Medium-Depth Reinforcement Design (3–15 m)

Static Reinforcement Elements—Cable Bolt Support

A conventional cable bolt is a flexible tendon consisting of a number of steel wires wound into a strand, which is grouted into a borehole. Cable bolts are normally installed in regularly spaced boreholes to provide reinforcement and support for the walls, roof, and floor of underground or surface openings. Cable bolting is a versatile form of support, because strands can bend around fairly tight radii, making installation of long bolts from confined openings possible and, because they can be fabricated with a number of configurations of the steel wires (e.g., plain, bulb, nut case), a variety of performance characteristics can be achieved. In North American hard-rock mining, cable bolts are made from 15.6-mm, seven-wire reinforcing strand (manufactured to ASTM 416 standard), having a yield strength of about 20 t at 0.8% strain and an ultimate strength of 24 t at 3.5%–4% strain.

In theory, cable bolts can be placed to any depth but for pragmatic reasons are seldom installed to depths exceeding about 20 m. Multiple cables can be placed in the same hole to increase tensile capacity, if the borehole diameter is large enough. In addition, face restraint can be attached in the form of plates, straps, and mesh. Cable bolts are commonly used in combination with other support systems such as shotcrete, rock bolts, and mesh. Figure 8.8-10 shows twin-strand bulb cable bolts coiled to about 1 m in diameter. Plain-strand

Table 8.8-5 Typical rock-bolt rules of thumb

Parameter	Empirical Rule
Minimum length	Greatest of
	(a) 2 × bolt spacing
	(b) 3 × thickness of critical and potentially unstable rock blocks*
	(c) For elements above the spring line:
	Spans <6 m: 0.5 x span
	Spans between 18 and 30 m: 0.25 × span
	Spans between 6 and 18 m: interpolate between 3 and 4.5 m
	(d) For elements below the spring line:
	Height <18 m: as (c)
	Height >18 m: 0.2 x height
Maximum spacing	Least of
	(a) 0.5 x bolt length
	(b) 1.5 x width of critical and potentially unstable blocks*
	(c) 2.0 m†
Minimum spacing	0.9–1.2 m
Minimum average confining pressure at yield point of elements‡	Greatest of
	(a) Above spring line:
	Either pressure = vertical rock load of 0.2 x opening width or 40 kN/m^2
	(b) Below spring line:
	Either pressure = vertical rock load of 0.1 x opening height or 40 kN/m^2
	(c) At intersections: 2 x confining pressure determined in (a) and (b)§

Source: Adapted from U.S. Army Corps of Engineers 1980.
*Where joint spacing is close and span relatively large, the super-position of two reinforcement patterns may be appropriate (e.g., long heavy elements on wide centers to support the span, and shorter, lighter bolts on closer centers to stabilize the surface against raveling).
†Greater spacing than 2.0 m makes attachment of surface support elements (e.g., weld mesh or chain link mesh) difficult.
‡Assuming the elements behave in a ductile manner.
§This reinforcement should be installed from the first opening excavated prior to forming the intersection. Stress concentrations are generally higher at intersections, and rock blocks are free to move toward both openings.

and bulb-cable are the configurations most commonly used in North American hard-rock mining. Alternately, in the Australian coal industry, the standard long cable tendon has around 63 t capacity and the heaviest just over 80 t capacity.

Quality control issues with cable bolts deal with grout quality and cable cleanliness. Cables can be grouted either toe to collar using 0.35 w/c grout or collar to toe (i.e., breather tube method) using 0.4 w/c grout. High w/c grout reduces cement strength and stiffness and results in significant reduction of cable bond capacity, commonly resulting in stripping failure. The use of bulb cable helps overcome grout quality control to some degree. Plates are attached using a barrel-and-wedge assembly. The barrel and wedge are susceptible to rust, which causes the wedges to lock up and the assembly will slide off at the seating load (nominally 5 t). Copper coat grease (or equivalent) should be applied to the inner surface of the barrel and the outer surface of the wedges to prevent corrosion. A detailed coverage of all aspects of cable bolt practice is given in Hutchinson and Diederichs (1996).

Connectible Super Swellex. The Pm24C Swellex bolt can be used to achieve deep anchorage by connecting two or more threaded sections. The bolt has a 200-kN breaking load and achieves 10% strain. Swellex is commonly used in combination with other support systems such as shotcrete, rock bolts, mesh, and so forth. QA/QC issues are the same as for ordinary Swellex. Under static support conditions, these support elements may be too soft (i.e., may allow too much ground movement).

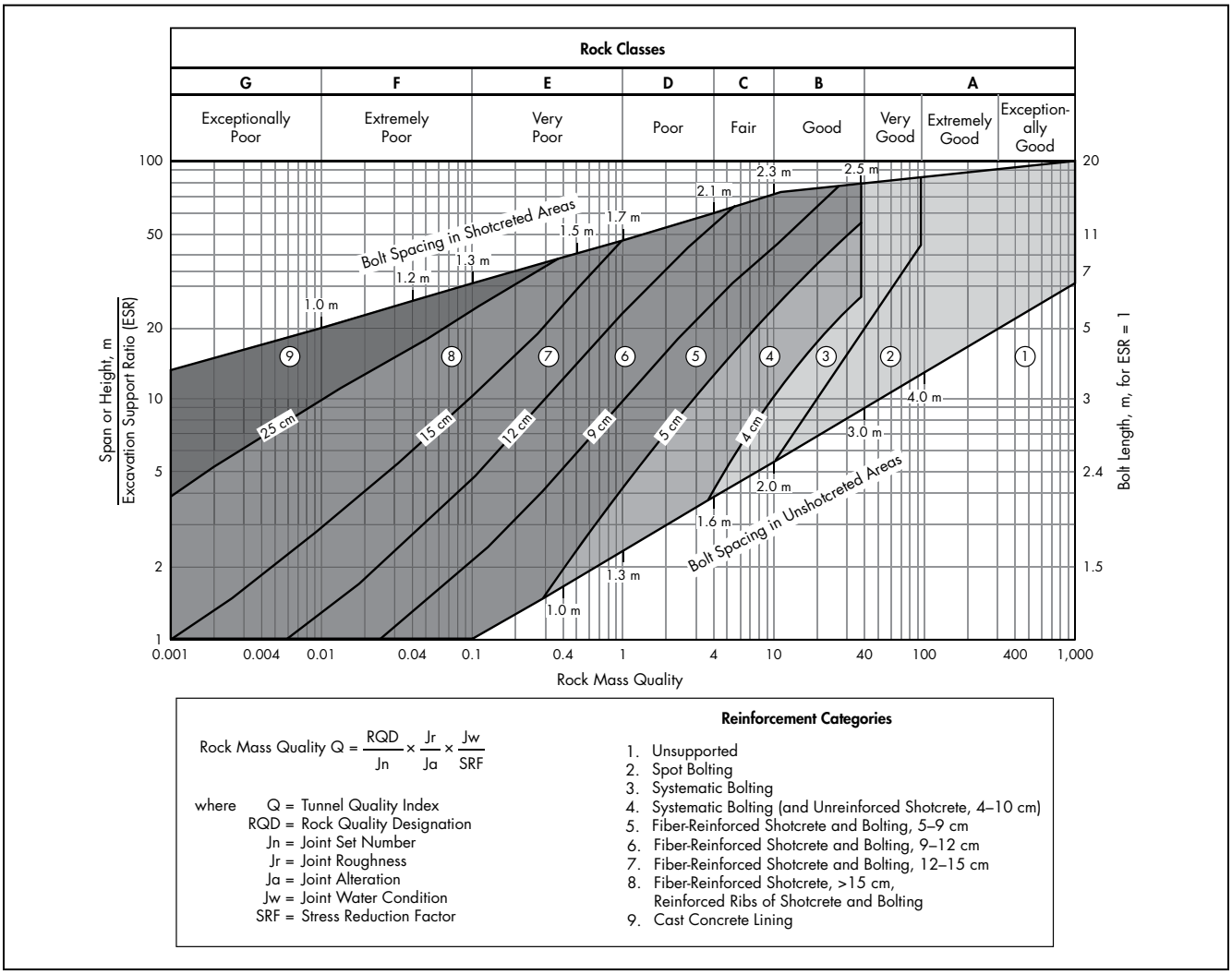
Static support design. For cable bolt support in hard-rock mining, static support design is generally done empirically using the Stability Graph Method (Hoek et al. 1995), which is widely used in North American and Australian hard rock mines. Numerical modeling can be used to help establish required support depth. Connectible Swellex can be used to replace cable bolts, but the Swellex density must be adjusted to achieve equivalent system capacity.

Dynamic Reinforcement Elements

Dynamic support elements suitable for medium depth conditions include debonded cable bolts and connectible super Swellex bolts.

Debonded cable bolts are created by placing a plastic sheath over a plain-strand section of the cable bolt. When the cable bolt is grouted in the borehole, the plastic sheath acts to debond the cable strand from the grout. The debonded section is then allowed to stretch freely, achieving 0.8% strain at yield and 3.5% strain at rupture. Each meter of debonded cable therefore permits ~8 mm of stretch at yield and 35 mm of stretch at rupture. Normally at least 2 m of cable is exposed at the toe, beyond the debond section. The cable may be secured at the collar by attaching a plate and barrel-and-wedge assembly. Alternately, a section of bulbed cable can also be left exposed at the collar. The use of debonded bulbed cable bolts under dynamic loading is discussed by Bawden and Jones (2002). QA/QC issues are the same as for static applications.

Pm24C Swellex bolts can be used to provide deep tendon support under dynamic loading.



Source: Palmstrom and Broch 2006.

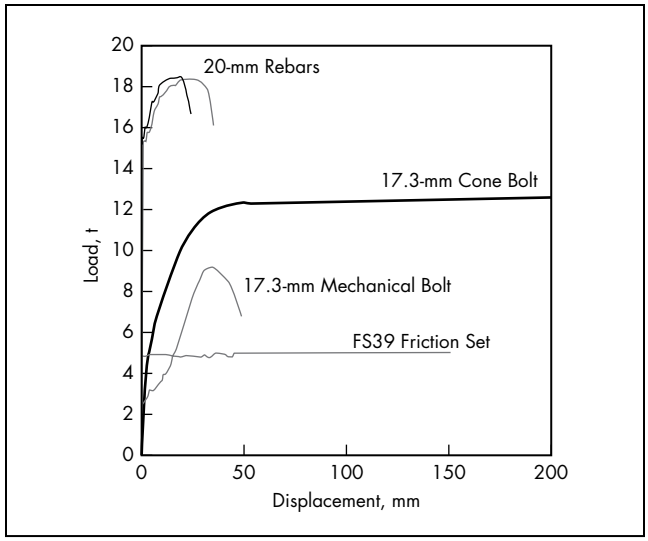
Figure 8.8-8 Estimated support categories based on tunneling quality index Q

Dynamic Support Design

Dynamic support design effectively implies rock-burst-resistant support design and is a function of the size of the seismic event and the location of the event hypocenter relative to the mine infrastructure in question (Figure 8.8-11). Local site factors such as the orientation of the infrastructure can also play a significant role in support and ground performance. Support ductility is required to achieve displacement capacity and energy absorption capacity (Figure 8.8-12). Tables 8.8-6 and 8.8-7 provide displacement and energy absorption criteria for most common support elements. Further coverage on this topic is beyond the scope of this chapter. A thorough coverage is provided in the *Canadian Rockburst Support Handbook* (Kaiser et al. 1996). Mesh is discussed in more detail later in this chapter.

Deep-Seated Ground Anchors (>15 m)

These long anchors tend to be arranged in larger cross-sectional areas to handle the greater volumes of unstable material. Used extensively in civil engineering and less so in mining, they are usually composed of a large number of individual elements that act together as a composite reinforcing element. To enable prestressing, they are usually discretely



Courtesy of Mansour Mining, Sudbury, Canada.

Figure 8.8-9 Load-displacement characteristic for a cone bolt versus selected conventional reinforcing elements

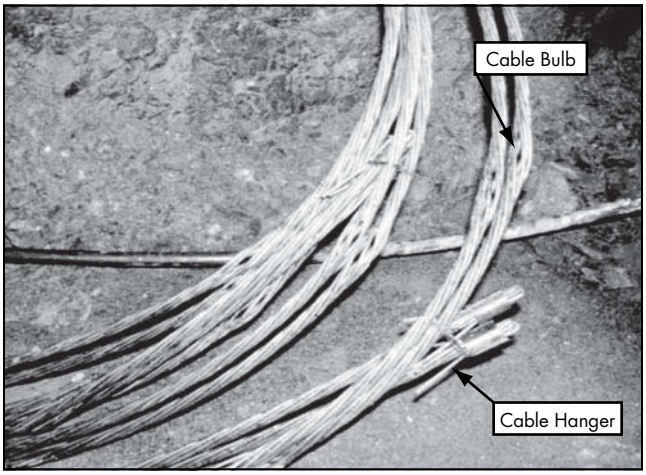
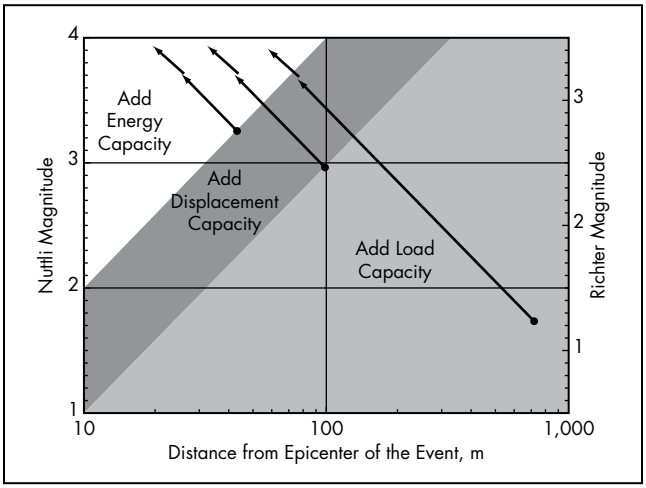
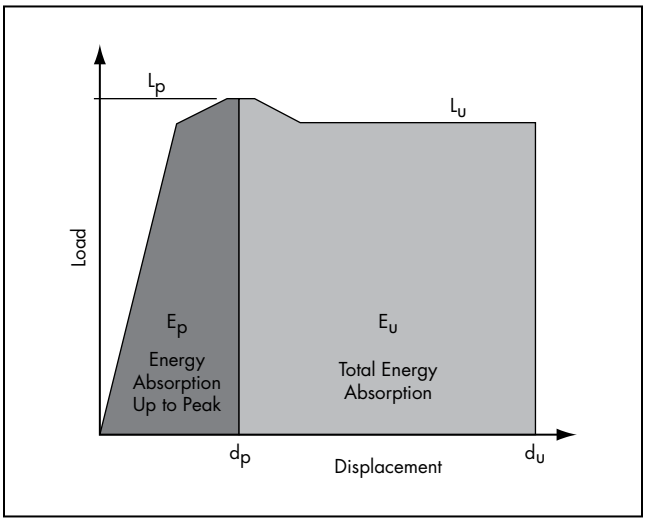


Figure 8.8-10 Coiled twin-strand cable bolts



Source: Kaiser et al. 1996.

Figure 8.8-11 Selecting the most important characteristics of a support system for burst-prone ground



Source: Kaiser et al. 1996.

Figure 8.8-12 Generic load-displacement curve for holding or reinforcing element

coupled over a fairly long anchorage and may be used to impart considerable load to the surface of the rock mass by way of large, rigid, built-in stressing blocks situated at the free surface. These devices often play a critical role in maintaining stability and are subject to stringent installation quality assurance by proof testing programs.

High-shear-capacity elements (shear keys) may be in the form of universal steel sections, large diameter tubes, or commonly railway lines cast in concrete. These elements are most commonly used as pre-reinforcement in surface excavations. The large cross-sectional area of the steel elements provides high shear resistance to transverse displacements (Windsor and Thompson 1993).

GROUND SUPPORT

Screen or Mesh

Screen or mesh is used to form a surface retention support system to cover the rock mass between the reinforcing tendons that are usually spaced at 1 to 2 m centers. The mesh is attached to the reinforcing tendons by the plate assembly or by push plates installed after the reinforcing tendons have been installed. There are several types of mesh construction, including welded wire mesh, chain link mesh, and expanded metal mesh. The wire gauge and construction technique control the load capacity and stiffness of the mesh. The mechanical properties of several common mesh types are provided in Table 8.8-7. Mesh serves the additional purpose of providing reinforcement and ductility for plain shotcrete. Welded wire mesh with a minimum 100-mm hole size is recommended to ensure that the shotcrete can penetrate and bond to the rock.

Straps

Steel straps (6.35 × 100 mm) are sometimes attached between reinforcing tendons to increase the footprint of the plates and to help secure specific wedges. Hole locations are preformed in the straps, and it is often difficult to install bolts at the required location and to make the straps conform to irregular rock surfaces. This style of strap has largely been replaced with zero-gauge welded wire straps (Figure 8.8-13), which are easy to install and pull tight to the rock surface and have proven very effective.

Shotcrete

Shotcrete is the generic name for cement, sand, and fine aggregate concretes, which are applied pneumatically and compacted dynamically under high velocity. Shotcrete can be applied in either a dry or wet form. In the dry application, shotcrete components, which can be slightly predampened to reduce dust, are fed into a hopper with continuous agitation. Compressed air is introduced through a rotating barrel or feed bowl to convey the materials in a continuous stream through the delivery hose. Water is added to the mix at the nozzle. In underground mining, dry shotcrete components are normally purchased premixed in 1,000-kg bags for ease of material handling (Hoek et al. 1995).

In the case of wet mix, the shotcrete components and the water are mixed before delivery into a positive displacement pumping unit. Low-profile agitator trucks have been developed for underground mining applications. Shotcrete is usually mixed in a surface batch plant and delivered underground using either agitator trucks on a ramp system or by slick line down the shaft.

Table 8.8-6 Recommended design parameters for reinforcing and holding elements*

Element	L_p, kN	L_u, kN	d_p, mm	d_u, mm	E_u, kJ
19-mm resin-grouted rebar	120–170	NA†	5–10	10–30	1–4
16- mm 2-m mechanical bolt	70–120	NA	10–40	20–50	2–4
16-mm-grouted smooth bar	70–120	50–100	5–10	50–100	4–10
16-mm cable bolt	160–240	NA	5–10	20–40	2–6
16-mm 4-m debonded cable	160–240	NA	120–160	120–160	10–25
Split Set bolt	50–100	40–80	10–30	80–200	5–15
Yielding Swellex	80–90	80–90	10–20	100–150	8–12
Yielding Super Swellex	180–190	180–190	10–20	100–150	18–25
16-mm cone bolt	90–140	90–140	10–20	100–200	10–25

Source: Adapted from Kaiser et al. 1996.
*d_p = peak displacement; d_u = ultimate displacement; E_u = ultimate Young's modulus; L_p = peak load; L_u = ultimate load.
†NA = not applicable.

Table 8.8-7 Load-displacement parameters for different types of mesh*

Mesh Type	L_p, kN	d_p, mm	d_u, mm	E_p, kJ/m²	E_u, kJ/m²	$E^†$, kJ/m²
#9 gauge welded wire	12–18	100–150	125–175	0.5–1.0	1–4	1
#6 gauge welded wire	24–28	125–175	150–225	1.5–2.5	4–6	3
#4 gauge welded wire	34–42	150–200	175–250	2.5–4	6–9	5
#9 gauge chain link	32–38	400–450	>400–450	3–4	10–12	7
Expanded metal	24–26	500–650	>500–650	6–7	8–10	7

Source: Adapted from Kaiser et al. 1996.
*d_p = peak displacement; d_u = ultimate displacement; E_p = peak energy; E_t = ultimate Young's modulus; L_p = peak load.
†Suggested energy absorption for design purposes.

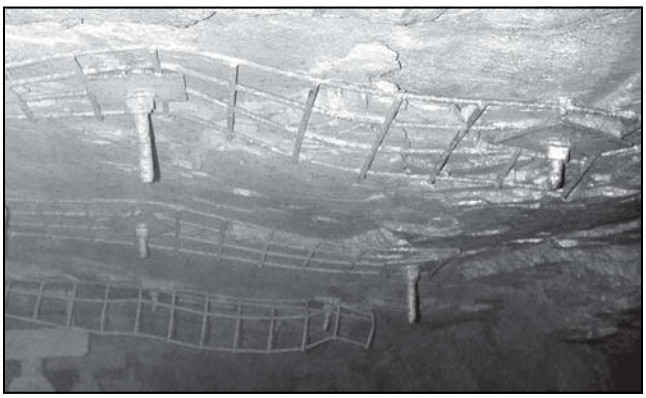


Figure 8.8-13 Zero-gauge straps

Shotcrete is normally applied in layers from about 50 mm to >100 mm thick. Because plain shotcrete is very brittle and will crack and spall off with small movement of the rock mass, it should always be reinforced. Welded wire mesh attached to the reinforcing tendons will provide reinforcement if plain shotcrete is placed over it. Alternately, fibers (steel or plastic) can be added to the mix to create what is normally called Fibercrete. The welded wire mesh and/or fibers provide tensile strength and ductility to the shotcrete to withstand movement of the rock mass. Shotcrete may be placed as an entire ring or only over selected sections of the excavation profile. A typical shotcrete application should achieve a uniaxial compressive strength of about 35 MPa after 28 days.

Wet-mix shotcrete is the most popular form because dry-mix shotcrete tends to have high rebound volumes, increasing unit costs, and usually results in inaccessibility of the area due to dust. However, wet-mix shotcrete has higher capital cost. QA/QC issues with shotcrete revolve primarily around preparation of the surface to be sprayed (the surface must be washed down clean in order to achieve proper bonding of the shotcrete) and control of the sprayed layer's thickness. Areas of active research with shotcrete today include the use of rapid curing shotcrete (via the addition of accelerators, etc.), deciding when it is safe to work under the shotcrete, and balancing the improvement in curing time against the inevitable loss of longer-term properties. Additional discussion of shotcrete design is provided by Hoek et al. 1995.

Concrete Liners

The three main types of concrete liners are poured, sprayed, and segmental:

1. Poured concrete liners are constructed by pumping concrete behind moveable forms (usually some type of slip from assembly). The poured concrete liner trails the advancing face by a discrete time/distance increment and is usually considered to be a secondary support system. Poured concrete liners are common in modern shaft-sinking operations and in some civil tunneling operations.
2. Sprayed concrete liners are created through the application of one or more shotcrete layers. Using a remote application arm, sprayed liners can be applied right up to the advancing face.
3. Segmental concrete liners are normally erected mechanically behind advancing tunnel boring machines and usually form the permanent liner for the excavation. This method is common in civil applications but is seldom used in mining.



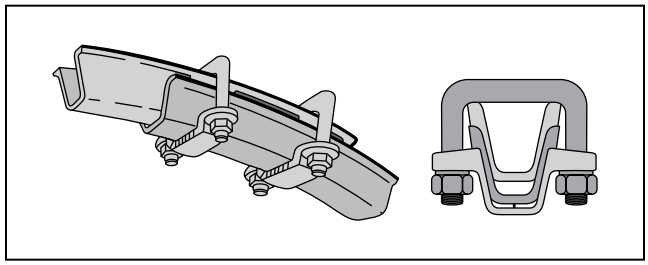
Figure 8.8-14 Steel arches with wooden lagging

Rigid Steel Sets

Steel arches or steel sets are used where high load-carrying elements are required to support tunnels or roadways. A wide range of rolled steel sections are available for this application. Where the rock is well jointed or becomes well fractured after the excavation is made, the spaces between the sets may be filled with steel mesh, steel, or timber lagging (Figure 8.8-14). Steel sets cannot be preloaded against the rock, and their efficacy depends on the quality of the blocking provided to transmit loads to the steel set. Improper blocking may lead to point loading of the set, often resulting in buckling failure of the legs. Today, steel sets are sometimes shotcreted in place, providing a more uniform loading and much stiffer response (Brady and Brown 2004). While steel sets give the impression of high load-carrying capacity due to their bulk, in terms of carrying capacity per weight of steel, they are often not that efficient. In most mining situations, they are an absolute last resort due to the difficulties involved in their erection, particularly under difficult ground conditions.

Yielding Steel Sets

Steel arches were widely used historically to support roadways in coal mines where they are often required to sustain large deformations. These deformations can be accommodated using yielding arches containing elements designed to slip at predetermined loads (Figure 8.8-15; Brady and Brown 2004). The modern U.S., South African, U.K., and Australian coal industries primarily use roof bolts and tendons, and this trend is continuing into other developing countries such as India and China. Steel sets present an impediment to efficient



Source: Hoek et al. 1995.

Figure 8.8-15 Assembly of a sliding joint is a top-hat section steel set

coal mining and are generally not considered as effective for ground control as reinforcing bolts and tendons. Hence they have been phased out over the past 30 years other than in arduous conditions around localized structures.

Shotcrete Pillar Support

Shotcrete pillars can be constructed by first forming a square cage of rebar anchored in boreholes in both the floor and the back of the opening. Each rebar should be 3 m long, with 1 m protruding from the hole. Additional rebar can be tie-wrapped to this rebar to complete the reinforcing cage. Steel half-culverts can be used to form the pillars, with angle iron tack welded so the half culverts can be bolted together. Shotcrete should be tremmied into the form, which should then be vibrated to remove air voids. Finally, the gap at the back is sprayed full of shotcrete. A 0.9-m-diameter shotcrete pillar using a 24-hour compressive strength of 15 MPa would carry a load of 1,200 t.

Timber and Steel Support

Timber support is no longer widely used in North America. The simplest form of timber support is the post and cap, or prop and lid, consisting of a single upright with a plate above. The post may be made of timber or steel, or may be a hydraulic prop. Timber posts, sometimes called sticks, are widely used to assist in supporting the mined-out area behind the stope face in the mining of narrow, reef-like metalliferous ore bodies. These sticks are usually 100 to 200 mm in diameter and are designed to support or help support the dead weight of the first 1 to 2 m of rock in the hanging wall. A typical 200-mm-diameter stick can have a short-term load-carrying capacity of 60 t (Korf 1978–1979), but for long-term use and low rates of load application, lower design capacities should be used.

Supports that are stiffer and have higher load-carrying capacities than timber posts or sticks of the same diameters are provided by pipe sticks in which a stick is press fitted into a steel pipe, typically 150 mm in diameter with a 4-mm-thick wall (Steinhobel and Klokow 1978–1979).

Sticks are passive support and require significant deformation to develop the full load-bearing capacity. They are also difficult to set properly, commonly leading to local falls of ground and increased loosening of the back. Their use is restricted to narrow openings or else they will act as slender columns and fail in buckling (Brady and Brown 2004).

In larger openings, timber cribs can be installed for local back support (Figure 8.8-16). Such timber cribs must be blocked against the back, and their effectiveness is highly dependent on the quality of the blocking. Timber

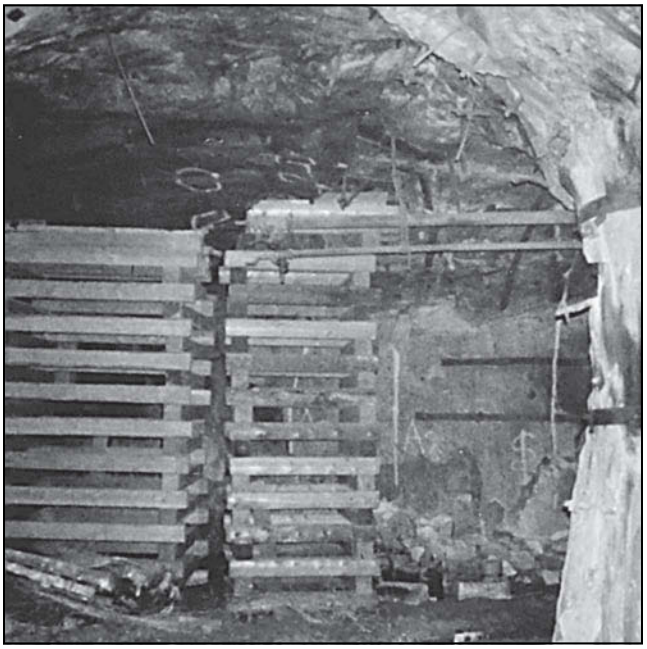


Figure 8.8-16 Timber crib

cribs can also be backfilled with waste rock to increase their load capacity. Timber cribs are a passive support and require significant ground movement in order to mobilize full load-bearing capacity. An issue not often discussed with timber support is shrinkage. If green timber is used, timber cribs can shrink from the roof by 100 mm over a relatively short period.

Hydraulic Props

Hydraulic props are set with a pre-load to provide active support and suffer from none of the disadvantages of sticks, although they are obviously much more expensive. The load bearing capacities of individual props may vary from as little as 5 t for a very light prop to more than 100 t for a 0.3-m-diameter prop. In the deep-level gold mines of South Africa, rapid-yielding hydraulic props are widely used to provide concentrated, active support of the hanging wall close to the face. Their rapid-yielding capability allows the energy released by rock bursts to be absorbed rapidly and safely, thereby minimizing the damage caused (Brady and Brown 1985).

A major risk with hydraulic props are leaks in the system, particularly longwall hydraulic supports, as mentioned earlier.

SAFETY

The majority of ground support is installed to ensure the safety of mine-operating personnel, and, where properly utilized, rock-fall-related injuries and fatalities have been dramatically reduced over the past 50 years. The statistics now indicate that most ground-control-related injuries occur during the installation of ground support (where workers are often in close proximity to unsupported ground) and are not due to subsequent ground instability. Today, it is common practice in many mines to use mechanized equipment to install primary support. In Australia, it is common practice in development headings to install Split Set bolts and screen using the jumbo on advance. In North America, it is more common to use mechanized bolters. In either case, the operator is protected by a steel canopy. In addition, mechanized scaling units can be used to scale the fresh heading prior to support installation. Another approach is to apply a flash coat of shotcrete (about 50 mm thick) and then to bolt through the shotcrete. Fiber-reinforced shotcrete can be used and may, in certain conditions, negate the need for screen.

WORK-FORCE TRAINING

At the end of the day, once all of the geotechnical design work is done and state-of-the-art products procured for use, it is all handed over to a group of miners to install. The quality of the final product is dependent on how well the work force is trained, how well the design is communicated, and on the quality of equipment provided. The importance of proper operator training cannot be overstated and in many areas the need for operator training in ground control principles is enshrined in the legislation.

MINING LEGISLATION AND DUTY OF CARE

In many areas of the world, and particularly in the developed world, the legislation pertaining to ground control has changed dramatically since about 1990. In most areas, a ground control management plan is mandatory and includes all manner of requirements above and beyond the type of ground support. A thorough discussion of this is beyond the scope of this chapter. Readers are encouraged to search out and carefully review the applicable local legislation.

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CHAPTER 8.9

Mine Subsidence

John P. Harrison

INTRODUCTION

The phenomenon of subsidence is an inevitable consequence of the extraction from underground of any resource—be it solid, liquid, or gas. During recent years, with the trend toward larger mines and increasing requirements for environmental protection, it is no longer possible to ignore it. Worldwide, great attention is devoted to the subject and its methodical study, and strict regulations for its control have been introduced by government agencies in many regions to protect the public interest.

The problems associated with subsidence have been recognized since antiquity. Agricola's *De Re Metallica* of 1556 talks about "a mountain or hill…subsiding by its weight" as a result of mining (Agricola 1556). The early- to mid-20th century saw many developments in the understanding and prediction of subsidence, motivated by legal action resulting from severe damage to surface structures, communications, and agricultural resources caused by underground mining. It was the defense against unjustified claims that required improved understanding of subsidence phenomena. Thus, the major objectives of subsidence engineering are

- Prediction of ground movements,
- Determining the effects of such movements on structures and renewable resources, and
- Minimizing damage due to subsidence.

Consequently, subsidence engineering entails much more than the prediction of ground movements: It requires knowledge of geomechanics (both soil and rock mechanics), structural geology, hydrology and hydrogeology, mining methods and techniques, construction procedures, agriculture, socioeconomics, and mining and property law.

Although the effects of fluid extraction on subsidence have been widely investigated, they are beyond the scope of this chapter. Nevertheless, mining may lower the local water table, and this can induce ground movements that cause surface damage. This must not be overlooked.

Finally, the term *subsidence,* as used in this chapter, encompasses the complete range of surface effects associated with the mining of minerals and not just the vertical displacement of the surface as is sometimes implied in the literature.

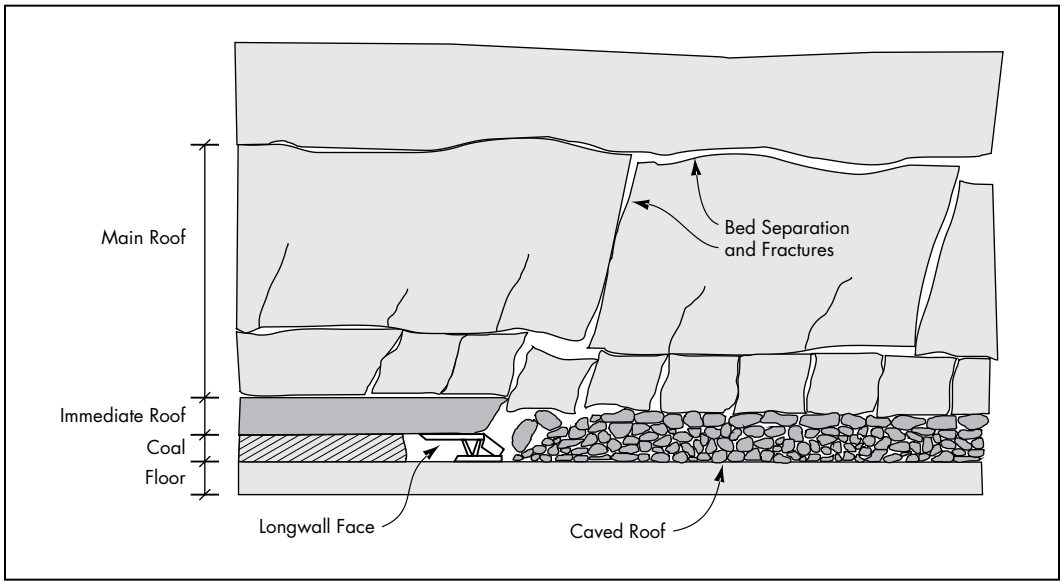
PRINCIPLES OF SUBSIDENCE

Development of Subsidence

The creation of any subsurface opening perturbs the stress state in the surrounding material. This perturbation produces deformations and displacements of the material, the magnitudes of which depend on the degree of the stress change, the spatial extent over which it occurs, and the nature of any rock support or reinforcement. If sufficiently large, these changes can cause the rock around a mine excavation to collapse into the mined void (Figure 8.9-1). The ground movements associated with such collapse tend to propagate to the ground surface, with the deformations and displacements experienced there being termed *subsidence*. Surface subsidence generally entails both vertical and lateral movements, and may be discontinuous (steps, cracks, or cavities form at the surface) or continuous (the surface deforms smoothly).

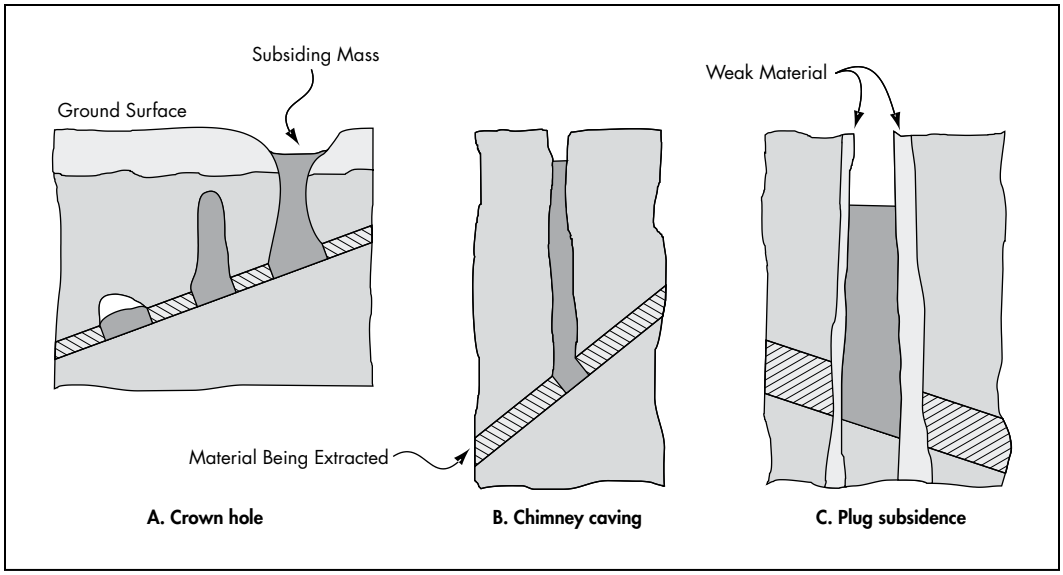
Discontinuous subsidence is generally of limited areal extent and is characterized by large vertical displacements. It occurs when material overlying an extraction zone collapses into the void, and its form depends on the mining method, the geometry of the extraction zone, and the geomechanical properties of the rock above the extraction zone (Figure 8.9-2). Crown holes may form because of roof or pillar collapse of shallow room-and-pillar (R&P) mines. A similar phenomenon is chimney caving (also known as piping or funneling), which occurs when a collapse at the mine level migrates upward (sometimes over distances of many hundreds of meters) through weak overlying materials. Both crown holes and chimney caving can occur above solution cavities, which themselves may be either a deliberate or inadvertent effect of solution mining. The presence of weak structural features (e.g., faults or boundaries between different geological materials) may lead to plug subsidence, in which a large plug of material falls suddenly and instantly downward into the mine void; the speed and suddenness of the process means this is particularly dangerous. Mining methods such as block caving

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Source: Adapted from Peng and Chiang 1984.

Figure 8.9-1 Caving and fracturing behind and above a longwall face



Source: Adapted from Brady and Brown 2004.

Figure 8.9-2 Forms of discontinuous subsidence

and sublevel caving also lead to discontinuous subsidence, but in these operations, use of and access to the surface area affected by the subsidence is generally prohibited.

In the case of continuous subsidence above laterally extensive extraction zones such as longwall coal mining operations, observations of subsidence profiles or troughs above the mined areas have shown they can be characterized on the basis of shape, in particular the absence or presence of an essentially horizontal central region. This leads to the concepts of subcritical, critical, and supercritical conditions (Figure 8.9-3), with the critical width being the minimum necessary to cause maximum surface subsidence. The position where the vertical displacement is effectively zero is known as the *limit of subsidence*, and the angle between a line joining this to the edge of the extraction zone and the vertical is termed the *angle of draw* or *limit angle*. The geometry shown in Figure 8.9-3 indicates that critical width is uniquely defined in terms of the mining depth and the angle of draw, and that the surface area affected by extraction is greater than the area of the workings themselves. The angle of draw has been found to be site-specific, as it depends on the geomechanical properties of the overlying materials; Table 8.9-1 gives various examples. In some areas, particularly those where the overlying rocks are particularly strong, a minimum extraction width has been found, below which no significant subsidence occurs (Sheorey et al. 2000); this is generally termed the *noneffective width*.

The minimum amount of mining required to lead to maximum subsidence of a point *P* is the extraction of a circular area whose diameter is equal to the critical width, and this circular

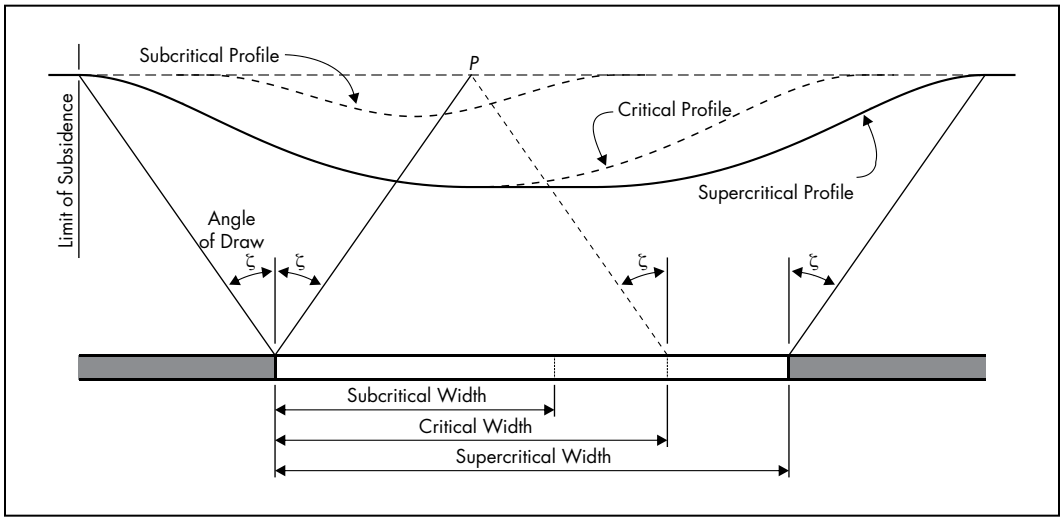


Figure 8.9-3 Continuous subsidence profiles above laterally extensive extraction zones

Table 8.9-1 Examples of angle of draw

Country	Angle of Draw, degrees
Australia (Newcastle coalfield)	35
Austria (Tertiary coal basin)	41–42
France (North)	35
Germany	30–45
India (Pench and Kanhan coalfields)	41
Japan (Kuho II mine)	40–50
Netherlands (Limburg coalfield)	35–45
Russia	30
Slovakia (Ostrava-Karvina coalfield)	25–39
South Africa	11
United Kingdom	25–35
United States (Central)	0–8.5
United States (East)	15–27
United States (West)	12–16

Source: Adapted from Bell and Donnelly 2006.

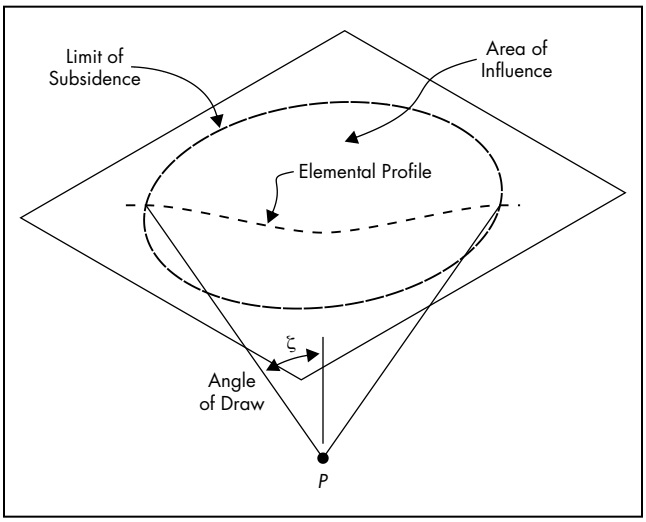


Figure 8.9-4 Concept of the area of influence

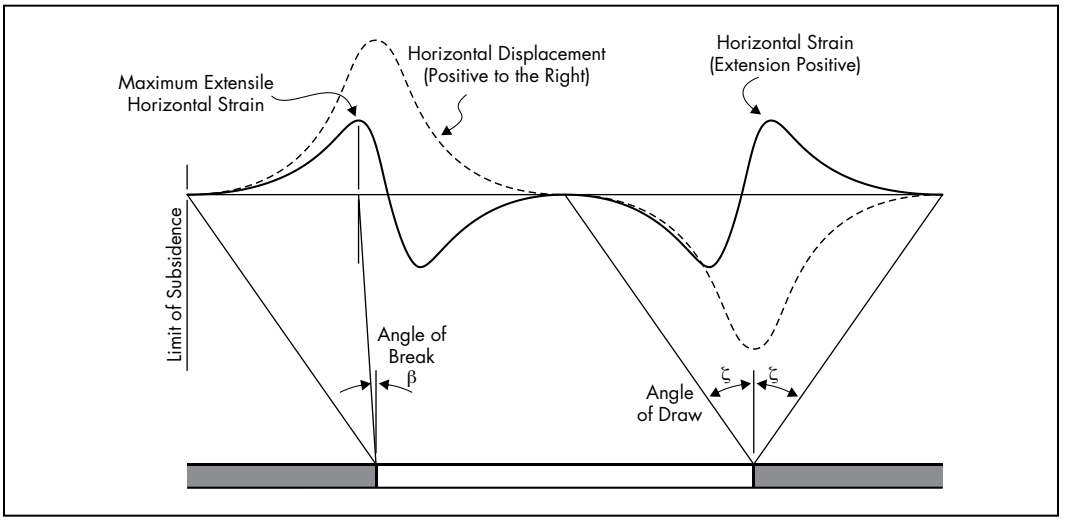
region is known as the area of influence. As every part of this area contributes in some way to the subsidence, extraction of material at a single point produces an elemental subsidence profile (Figure 8.9-4) that is circular in shape. The subsidence developed above any mining operation can be thought of as being formed by the summation of many such elemental profiles.

As subsidence occurs, points on the ground surface move horizontally toward the center of the mined area as well as vertically downward. Vertical displacement is greatest at the center of the mined area and zero at the limit of subsidence, with horizontal movement varying from a maximum almost vertically above the edge of the mined area to a minimum (essentially zero) at both the center of the mined area and the limit of subsidence. As the horizontal displacement is not constant, horizontal strain is induced at the ground surface. These strains are extensile (i.e., dilatant) outside of the boundaries of the mined area and contractile within them (Figure 8.9-5), although across the central region of the subsidence profile above critical- and supercritical-width workings the contractile strains are zero. Note that the terms *tensile* and *compressive* strictly refer to forces and stresses, and the terms *extensile* and *contractile* refer to strains. The inclination to the vertical of the line connecting the edge of the mined area to the surface point exhibiting the maximum extensile strain is called the *angle of break* or *angle of fracture*—not to be confused with the angle of draw defined earlier.

As a mine working is progressively extended, it begins at a subcritical width and, depending on the extent of the extraction, may pass into critical and supercritical widths. As the working face moves, so the horizontal extensile and contractile strain regions also move, meaning that a given point on the ground surface may experience different strain states as mining progresses. This is of critical concern for damage of surface structures. Finally, as Figure 8.9-6 indicates, the zone of subsidence reflects the shape of the extraction zone.

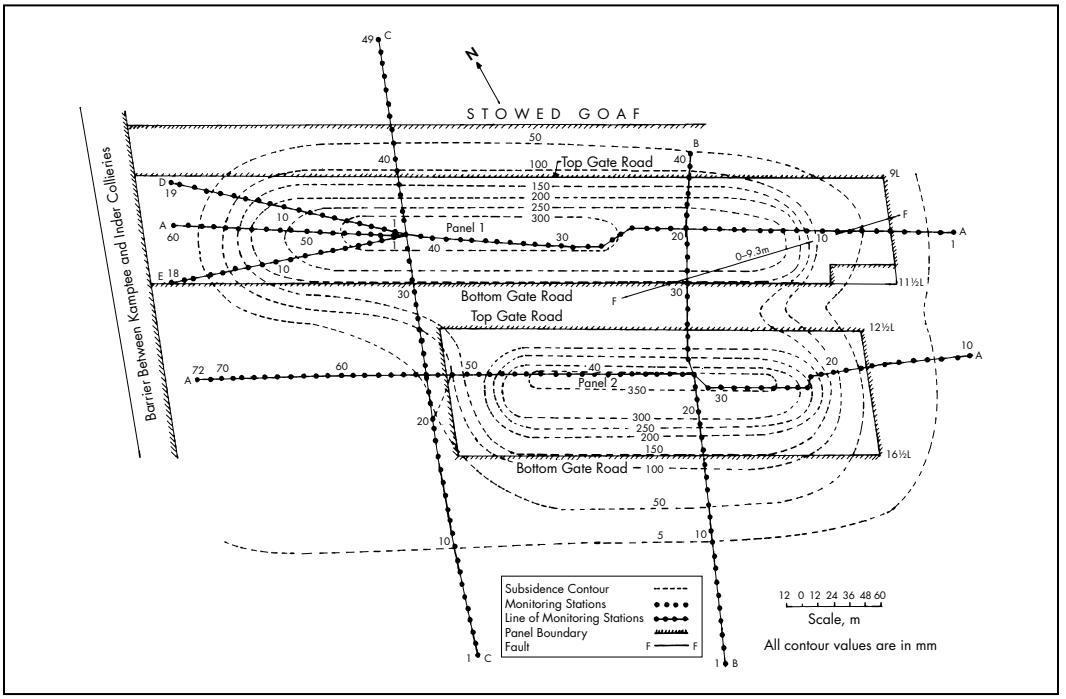
Factors Affecting Mine Subsidence

Experience has revealed that many geological and mining parameters besides the width of the extraction zone can affect the magnitude and extent of subsidence. The number



Source: Adapted from National Coal Board 1975.

Figure 8.9-5 Horizontal displacement and strain profiles over a critical width opening



Source: Singh and Singh 1998; reprinted with permission from Elsevier.

Figure 8.9-6 Subsidence contours above longwall coal panels

and interrelation of these factors means that predicting in an accurate, quantitative manner the magnitude and time to subsidence onset is generally not straightforward.

Extraction Thickness

The thicker the material extracted, the larger the amount of possible surface subsidence. Note that it is the actual extracted thickness, not the in-situ thickness, that must be considered. Where mining takes place in several overlying mining horizons, subsidence is related to both the total extracted thickness and the sequence of horizon extraction.

Mining Depth

For supercritical longwall operations the maximum subsidence amount is unrelated to depth, but for other operations both the magnitude and time to onset of subsidence are dependent on depth.

Inclination of Extraction Horizon

Asymmetric subsidence occurs when the zone being mined is inclined. The subsidence profile is translated in a downdip direction with both the limit angle and the horizontal strains increased downdip and reduced updip.

Degree of Extraction
Reducing the amount of material extracted will reduce the amount of subsidence. Thus, lower extraction ratios tend to both reduce and delay the onset of subsidence.

Mined Area
The critical width of a mined void must be exceeded in all directions if maximum subsidence is to develop. This is especially important if overlying competent materials are present, as these may tend to form bridges in the direction of subcritical width and thus decrease subsidence.

Method of Working
The amount of subsidence is largely controlled by the degree of caving induced by the mining method (e.g., complete subsidence for block caving and longwall mining, and zero for R&P) together with the amount of support offered by any backfilling. Nearly immediate, but predictable, subsidence occurs with longwall mining, whereas with R&P operations both the magnitude and onset of subsidence are largely unpredictable.

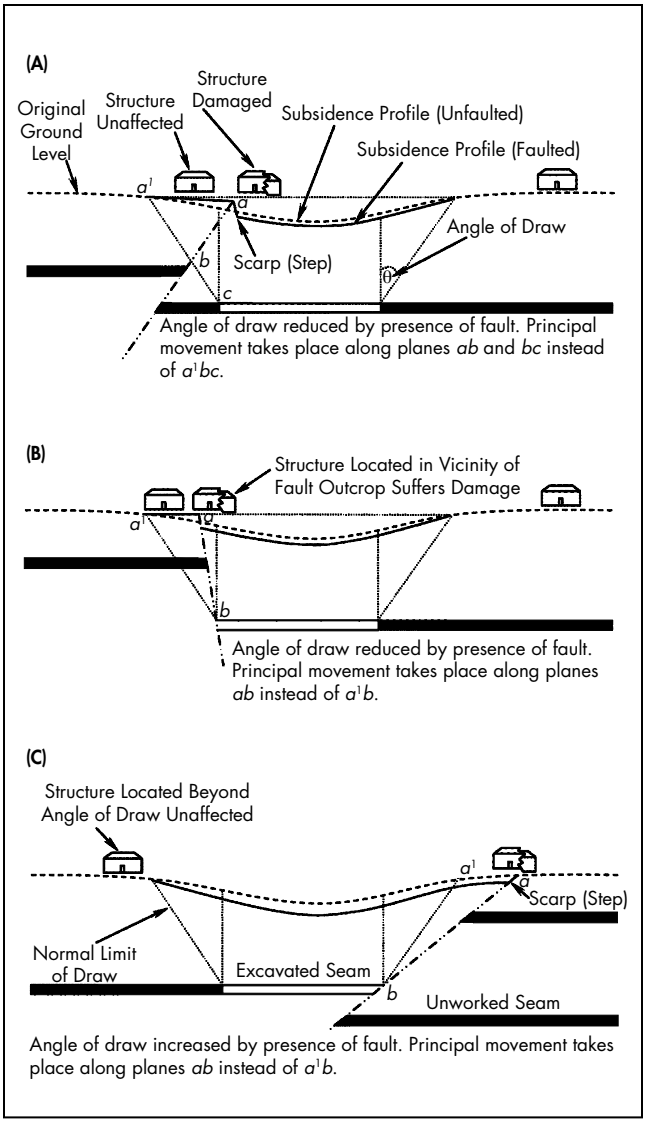
Extraction Rate
Surface subsidence follows the face as it progresses, and so to minimize the effect of strain and tilt on surface structures, a fairly rapid, constant face advance rate should be adopted (Legget 1972).

Competence of Surrounding Materials
Since subsidence propagates from the mine level, the mechanical behavior of the rock adjacent to the mined void directly affects the initiation of subsidence. Weak roofs and floors accentuate subsidence, whereas strong materials can delay or even prevent collapse and hence subsidence. Strong, massive materials above the mine level are able to withstand the effects of extraction for a prolonged period and hence defer the occurrence of subsidence.

In-Situ Stress State
High horizontal stresses may foster formation of an arch in the material overlying a mined void, thereby attenuating subsidence. However, arch formation is a complex phenomenon, depending on many geomechanical parameters: It cannot be guaranteed, and arches may fail suddenly and catastrophically.

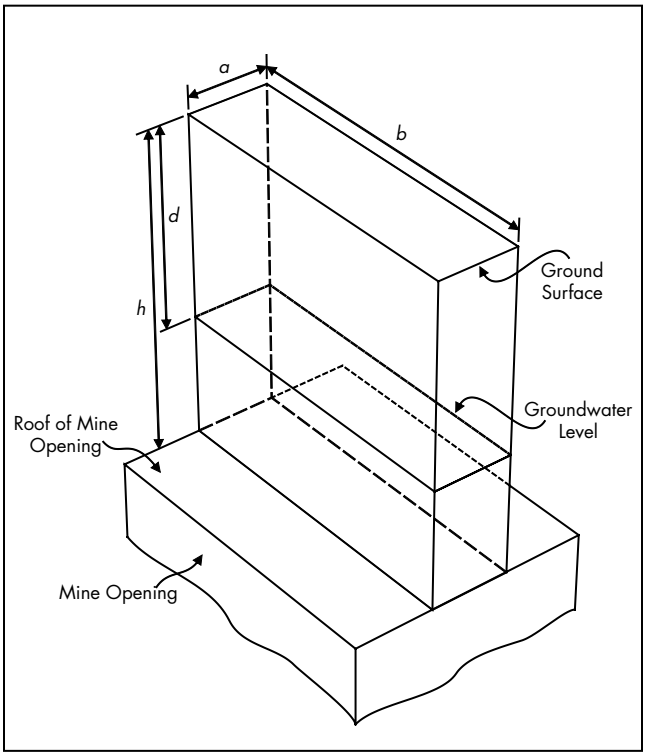
Geological Discontinuities
The existence of faults, folds, and the like may increase and localize subsidence potential so strongly that in areas of adverse geological conditions the effects of the other parameters can be discounted. As mining perturbs the stress state in the ground, fault planes may become critically stressed and hence undergo shearing. The associated release of strain energy can be rapid and dramatic, producing seismic events, or can be more benign, resulting simply in the ground surface being displaced into a series of steps. Displacements will be concentrated toward faults, although horizontal strains may become negligible in their immediate vicinity. Structures that straddle faults may be severely damaged while nearby buildings remain relatively intact (Figure 8.9-7). Joints and fissures in the strata affect subsidence behavior in a manner similar to faults, but on a smaller scale.

Near-Surface Geology and Surface Topography
The nature of any near-surface soils and unconsolidated rocks affects subsidence development, with both the thickness and mechanical characteristics of these materials being important. For example, cracks and fissures may form in stiff clays, whereas soft clays may deform plastically and cohesionless sands may flow down into fractures in the underlying rocks. Specialist geotechnical advice may be needed in order to determine the behavior of structures in or on these materials. Sloping ground tends to emphasize downward movements because of gravity, and extensile strains may increase on hilltops and decrease in valleys, although there is evidence that the effect of topography on strain is unpredictable (Ewy and Hood 1984). Deeply incised valleys, particularly those in regions of high horizontal in-situ stress, may display valley bottom vertical subsidence that is markedly less than predicted for a horizontal ground surface, and horizontal closure greater than predictions of horizontal strain would suggest (Waddington and Kay 2003). These phenomena seem to be analogous to natural periglacial cambering and valley bulging in weak rocks.



Source: Bell and Donnelly 2009, reproduced by permission of the author and the Geological Society of London.

Figure 8.9-7 Localization of structural damage due to faulting



Source: Adapted from Brady and Brown 2004.
Figure 8.9-8 Equilibrium analysis of chimney caving

Hydrogeology
Deformation of the strata around mined areas may alter hydraulic gradients, resulting in either the flooding or draining of surface areas and the formation or draining (in aquifers) of underground reservoirs (Booth 2002). Rocks may be weakened by changes in saturation, and, in carbonate (e.g., limestone) areas, caverns or karst may develop over a period of time. If surface runoff from precipitation or water from leaking pipes is allowed to percolate into ground fractured by mining, the increased groundwater pressure can reduce the effective stress, thereby inducing shear on faults.

Elapsed Time
Subsidence does not occur instantaneously but over a period of time. In R&P operations, subsidence may only develop a long time (possibly centuries) after the mining is complete, when pillar degradation leads to roof collapse. In caving operations, surface displacements may occur almost immediately after the face passes below an area. However, as noted, the presence of strong, competent layers overlying an opening can delay this.

Discontinuous Subsidence
The wide range of circumstances leading to discontinuous subsidence means that no generic method is available for predicting or analyzing the phenomenon. Instead, the principles of geomechanics must be applied on a case-by-case basis. As all discontinuous subsidence involves the movement of clearly defined bodies of rock or soil, it is often straightforward to develop a free body diagram of the system and examine its equilibrium.

Figure 8.9-8 represents an example of such an analysis for the case of a rigid prismatic block of rock whose vertical faces follow the Coulomb shear strength criterion. For a groundwater level of $0 \le d \le h$, the factor of safety, F, can be calculated as

$$F = \frac{2}{\gamma}\left[c' + kh\frac{\gamma}{2}\tan\phi' - \frac{\gamma_w \tan\phi'}{3h}(h-d)^2\right]\frac{a+b}{ab} \qquad (8.9\text{-}1)$$

where
a, b, d, and h = block geometry
c' and ϕ' = shear strength of the vertical faces
γ and γ_w = unit weight of the rock and water, respectively
k = ratio of horizontal to vertical in-situ stress in the block

By setting the factor of safety F to unity, these equations can be used to determine the critical value of the span b leading to caving.

Other analyses that are applicable in the context of discontinuous subsidence are pillar collapse and roof stability.

Continuous Subsidence

Subsidence Measurement and Monitoring
The differential settlement and horizontal strain developed during subsidence tend to be critical in terms of structural damage. As these are related to vertical displacement, measuring and monitoring subsidence displacements is of prime importance. To be of maximum benefit, monitoring should commence before any mining activity begins and be continued for as long as either ground movement is likely to occur or surface structures are liable to be affected by movement.

Traditionally, subsidence measurement and monitoring has been undertaken using customary precise leveling techniques. Measurements must be made on structures or markers that respond directly to the subsidence of the rock surface, and it is essential that all survey points are unaffected by surface movements (e.g., fluctuations in elevation due to freeze–thaw action or changes in saturation). Although many forms of survey marker have been devised, their essential characteristics are a long steel tube or bar topped with a disk or hemispherical dome of rust-resistant metal, grouted into the rock at a depth appropriate to the local conditions, and surrounded by concrete for protection.

Although the markers need to be sufficiently close to allow changes in the local subsidence gradient to be detected, economy of cost generally will require the spacing to be as large as possible. The U.K. National Coal Board recommended a spacing of $0.05D$, where D is the depth to the mine workings (National Coal Board 1975). In the United States, a spacing of $0.05D$ to $0.1D$ has been used. The accuracy of the measurements should be such as to detect horizontal strains of 1×10^{-4}, which is about one-tenth of the strain that causes structural damage. The measurement procedures may be obtained from any text on surveying, but attention needs to be paid to details such as the area to be covered, surface topography, labor requirements, and survey frequency.

Vertical displacements may be measured directly by trigonometric leveling (precision optical or laser), differential leveling, or tilt measurement. When using a theodolite, vertical angles must be measured correctly to one-half second of arc. With precise leveling, a micrometer direct reading to about 1.5 mm should be employed. An inclinometer with a sensitivity of 10 seconds of arc is generally adequate for subsidence measurements.

The increasing availability of high-resolution satellite imagery means that indirect measurement of vertical displacement using satellite interferometry is becoming more widespread as a subsidence monitoring tool (Wright and Stow 1999; Carnec and Delacourt 2000). Although displacements obtained in this manner currently have a resolution limited to the order of a few millimeters (Ge et al. 2007), the technique has many benefits, including removal of the need for surface surveys, regular automatic acquisition of data, and measurements not limited to specific profiles. For new mines it is also possible to use imagery obtained before mining in order to obtain baseline conditions. With the continual deployment of ever more powerful remote sensing satellites, it is reasonable to assume that interferometric techniques will become the future standard method of subsidence monitoring.

Subsidence Prediction

Subsidence prediction techniques may be characterized as either phenomenological or empirical, with phenomenological techniques using computer-based numerical modeling to reproduce subsidence by simulating the behavior (both continuous and discontinuous) of the rock around a mine, and empirical methods using various models and formulas based on observations and experience of actual mining subsidence. Empirical methods include influence functions, profile functions, and graphical techniques. Although empirical techniques have formed the basis of most subsidence predictions to date, continuing developments in numerical modeling and subsurface characterization using geophysics suggest that phenomenological techniques will become preeminent.

Phenomenological methods. Phenomenological methods are computational methods based on the principles of mechanics and assume some particular material model to be applicable to the rock. Initial work concentrated on elastic, viscoelastic, plastic, and elastoplastic continuum models (e.g., Marshall and Berry 1967; Crouch 1973), but recent work has moved toward discontinuum and hybrid continuum-discontinuum approaches (e.g., Wu et al. 2004; Zangerl et al. 2008). These are likely to develop further and become more prevalent because they more closely capture the fragmentation that occurs during subsidence than do continuum approaches.

Application of phenomenological methods requires sophisticated software (the complexity of the physical processes involved in subsidence means that simple numerical techniques are seldom appropriate) and suitably skilled analysts; although powerful and comprehensive, they cannot be used in an impromptu manner.

Influence functions. Influence functions apply the concept of the elementary cone (Figure 8.9-4), with the extraction zone being discretized as a large number of small elements of simple geometry and the subsidence at any point on the surface being obtained from the superposition of the influence of each extracted element. The functions themselves are empirical, and the method has achieved widespread application (Ren et al. 1987). It has been found to be particularly applicable to mine workings with irregular or complex geometries, but, as implementation of the method is reasonably straightforward, it is a pragmatic approach to predicting subsidence above any mining operation.

At their simplest, influence functions take the form

$$p(r) = k_1 S_{\max} f(B, r, k_2) \tag{8.9-2}$$

where

$p(r)$ = contribution made to the subsidence at some location on the surface by extraction of a small area located a horizontal radial distance r away
k_1 and k_2 = constants (Brady and Brown 2004)
S_{max} = maximum subsidence associated with critical width extraction (usually determined empirically)
B = critical width

In addition, as the subsidence over an element of extraction is assumed to follow an inverse square law (Kratzsch 1983), influence functions contain a term that is some function of r^2/h^2. The inclusion of the critical width in Equation 8.9-2 means that more complex functions are required when either the ground surface or the extraction zone is not horizontal. For the latter case, the use of an angle of draw that varies with orientation relative to the dip direction of the extraction zone has been proposed (Lin et al. 1992).

The complete subsidence s at a point on the surface is given by integrating Equation 8.9-2 over the entire extraction area A:

$$s = \int_A p(r)\,dA \tag{8.9-3}$$

Evaluating this integral is challenging for all but the simplest of influence functions and extraction geometries, and so a computer-based approach is usually used. One technique, illustrated in Figure 8.9-9, divides the area of influence of the function into a number of annular sectors and integrates the function over each sector to give an influence coefficient (Ren et al. 1987). Only those sectors that lie within the extraction area for a given point of interest are used in the calculation, and so determining subsidence becomes a case of simply summing the influence coefficients for each such sector. The calculation is repeated for a number of points of interest, and a contour map of subsidence determined. A major advantage of this approach is that the calculation is easily programmed and can readily be performed on a computer spreadsheet.

Calculation of the influence coefficients requires integration of the influence function over the area of each annulus. The influence of the ring between r_i and r_{i-1} is then

$$I(i) = 2\pi \int_{r_{i-1}}^{r_i} p(r) r\,dr \tag{8.9-4}$$

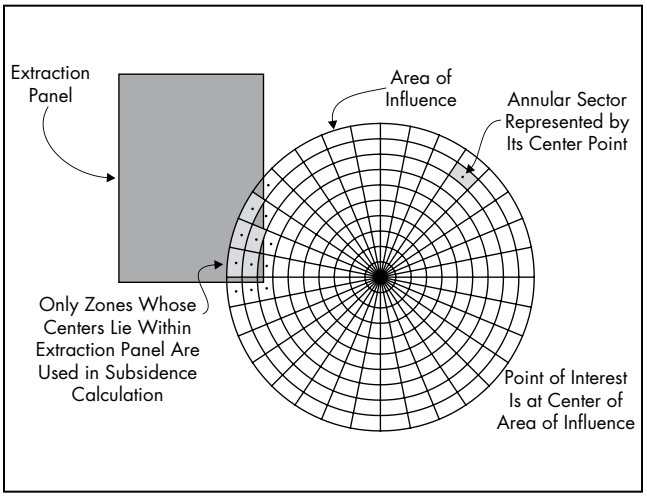
The sum of all *I(i)* is required to equal unity, and the results of Equation 8.9-4 must be scaled to ensure this is the case. As an example, take the case of the stochastic, or exponential, function (Ren et al. 1987):

$$p(r) = \frac{1}{R^2} \exp\left(-\pi \frac{r^2}{R^2}\right) \tag{8.9-5}$$

where R is the radius of the area of influence. Substitution of Equation 8.9-5 into Equation 8.9-4 yields

$$I(i) = \exp\left(-\pi r_{i-1}^2 / R^2\right) - \exp\left(-\pi r_i^2 / R^2\right) \tag{8.9-6}$$

and using this influence coefficients for a scheme using 10 annuli and 64 sectors are given in Table 8.9-2. With this scheme each annular sector contributes 1/640 to the overall subsidence



Source: Adapted from Whittaker and Reddish 1989.
Figure 8.9-9 Calculation using influence functions

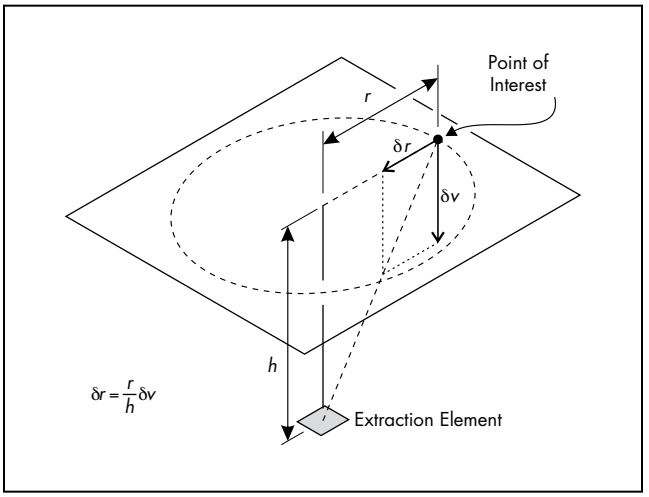


Figure 8.9-10 Focal point concept

Table 8.9-2 Influence coefficients for exponential function

Annulus	1	2	3	4	5	6	7	8	9	10	Sum
Outer radius	0.1*R*	0.2*R*	0.3*R*	0.4*R*	0.5*R*	0.6*R*	0.7*R*	0.8*R*	0.9*R*	1.0*R*	
Integral	0.0309	0.0872	0.1282	0.1488	0.1490	0.1332	0.1082	0.0806	0.0554	0.0353	0.9568
Annulus influence	0.0323	0.0911	0.1340	0.1555	0.1557	0.1392	0.1131	0.0842	0.0579	0.0369	1.0000
Element influence × 10^{-3} (64 sectors)	0.51	1.42	2.09	2.43	2.43	2.18	1.77	1.32	0.90	0.58	

at a point, and so any errors associated with simplifying each sector to its center point is very small.

A related influence function that has found widespread application is

$$s(r) = \frac{n}{R^2}\exp\left[-n\pi\frac{r^2}{R^2}\right] \tag{8.9-7}$$

where n is a site-specific constant (Brady and Brown 2004). For any particular site, comparison of calculated and observed values of subsidence will allow determination of the appropriate value of the constant n.

A function proposed recently is

$$p(r) = \frac{1}{(1+4n)k^2h^2}\left[\begin{aligned}&\exp\left(-\frac{\pi}{k^2}\frac{r^2}{h^2}\right)\\&+n\exp\left(-\frac{\pi}{4k^2}\frac{r^2}{h^2}\right)\end{aligned}\right] \tag{8.9-8}$$

where

n and k = constants related to the maximum subsidence magnitude and the shape of the subsidence profile, respectively
h = mining depth (Álvarez-Fernández et al. 2005)

This latter function has been shown to be effective for very complex mining geometries (González-Nicieza et al. 2005; Díaz-Fernández et al. 2009).

Finally, as illustrated in Figure 8.9-10, the so-called focal point concept can be used to calculate the horizontal radial displacement (Ren et al. 1987) associated with each elemental value of vertical displacement. The resultant values of horizontal displacement at neighboring points of interest can then be used to calculate horizontal strain.

Profile functions. Profile functions are mathematical functions representing a complete subsidence profile at the surface along a traverse either parallel or perpendicular to the face of a longwall panel. Although they are, in effect, integration of an influence function along a particular traverse, profile functions are nonetheless selected empirically and the controlling constants determined from subsidence observations. Provided that sufficient suitable observation data exist to allow calculation of the constants, this method can be applied to a wide range of geological conditions, and profile functions have a history of application in several countries (Hood et al. 1983). A major limitation of profile functions is that they are applicable only to simple mine geometries, such as longwall panels.

Profile functions take the form

$$s(x) = S_{\max} f(B,x,c) \tag{8.9-9}$$

where

$s(x)$ = subsidence at position x
$S_{\max}$ = maximum subsidence associated with critical width extraction (usually determined empirically)
B = critical width (Brady and Brown 2004)
x = distance from the coordinate origin
c = site-specific empirical constant or function (Hood et al. 1983)

Differentiating such a function once with respect to x gives the surface slope and twice gives the curvature. It has been suggested that relating calculated curvature to structural damage is more valid than attempting to determine horizontal strain from a profile function (Whittaker and Reddish 1989).

Various exponential, trigonometric, hyperbolic, and error functions have been used as profile functions, but one that has been found to apply well to supercritical conditions in both the United Kingdom and United States is

$$s(x) = \frac{1}{2} S_{\max}\left[1 - \tanh\left(\frac{b}{h}x\right)\right] \quad (8.9\text{-}10)$$

where x is measured from the inflection point of the function (i.e., where $s = \frac{1}{2}S_{max}$), b is a constant, and h is mining depth.

Application of such a function requires values for S_{max}, b, and the location of the inflection point relative to the mined panel, and these can only be obtained from subsidence monitoring at either the mine to which they are to be applied or one in similar circumstances. For example, following extensive monitoring of a longwall coal panel in Illinois (United States), values of b of about 11 for transverse profiles and between 8 and 9 for longitudinal profiles have been reported (Hood et al. 1983). These values should be considered in the light of work that suggests $b = 5$ is appropriate for the United Kingdom (King et al. 1975); the disparity between the two sets of results confirms the need to obtain locally derived data.

Graphical methods. Graphical methods involve using previously obtained subsidence data, normalized with respect to one or two mining parameters, to develop charts from which subsidence associated with other mining operations can be predicted. These methods effectively entail graphical representation of a profile function but require considerable subsidence data and are generally restricted to similar areas and mining conditions from which the data were collected.

Probably the most comprehensive example of a graphical approach is that set out in the *Subsidence Engineers' Handbook* (National Coal Board 1975), which was developed for use in the coalfields of the United Kingdom. The basis of the approach is that strain and tilt were found to be related to maximum subsidence and mining depth in the following ways:

$$\begin{aligned} +E_{\max} &= 1{,}000 \times k_1 \times S_{\max}/h \\ -E_{\max} &= 1{,}000 \times k_2 \times S_{\max}/h \\ \omega_{\max} &= 1{,}000 \times k_2 \times S_{\max}/h \end{aligned} \quad (8.9\text{-}11)$$

where

$+E$ and $-E$ = extensile and contractile strains, respectively
k_1, k_2, and k_3 = constants of proportionality (Brady and Brown 2004)
h = mining depth
ω = tilt

The constants vary with the ratio of extraction width to mining depth and location relative to the center of the working face, and so the various charts presented in the handbook are effectively graphs of the constants in terms of these factors. The maximum subsidence is generally dependent on the mining depth and both the extraction thickness and width; Figure 8.9-11 shows the relation derived by simplifying to straight lines the comprehensive curves relating subsidence to extraction thickness given in the *Subsidence Engineers' Handbook*.

Because the charts used in this method are only applicable strictly to the area and conditions for which the source observations were obtained, they should not be applied to other circumstances. Instead, charts should be developed on an area-by-area basis (see, for example, the charts developed for use in New South Wales, Australia [Holla and Barclay 2000]).

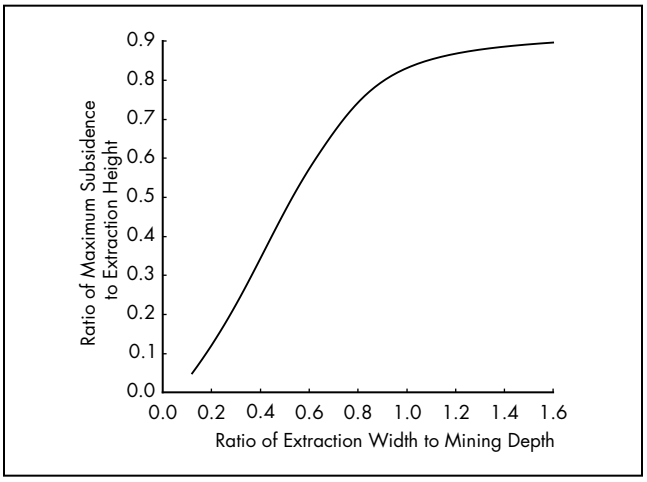


Figure 8.9-11 Simplified relation between maximum subsidence and extraction height, extraction width, and mining depth

Long-Term Effects

The duration of subsidence resulting from mining is composed of two distinct phases: active and residual. Active subsidence consists of those movements occurring contemporaneously with mining operations, while residual subsidence consists of those that occur later than this (either following the cessation of mining or the passing of a zone of influence). The magnitude and duration of residual subsidence has particular relevance to structural damage and legal obligations, but both of these vary markedly with the mining method and the geomechanical characteristics of the ground.

Numerous observations worldwide suggest that the magnitude of residual subsidence associated with longwall panels rarely exceeds about 10% of the total subsidence, with the duration of these movements typically varying from a few weeks to about 10 years (Table 8.9-3). Prediction of duration generally uses site-specific empirical relations; one such example from the United Kingdom is

$$t = \frac{2d\tan(\zeta)}{a} \quad (8.9\text{-}12)$$

where

t = time (years) for subsidence to complete
d = mining depth (m)
ζ = angle of draw
a = face advance (m/yr) (National Coal Board 1975)

However, further work is needed to fully understand time-dependent, or residual, subsidence over longwall panels (Goulty and Al-Rawahy 1996).

With R&P operations, most subsidence results from deterioration and collapse of either the support pillars or the roof between them, or punching of the pillars into the roof or floor. This may occur many years (decades or even centuries) after cessation of mining (Thorburn and Reid 1977; Ivey 1978; Dunrud and Osterwald 1980), and predicting the timing is difficult, as it depends on a large number of factors, including the strengths of pillars, roof, and floor; extent of fracturing;

Table 8.9-3 Duration of residual subsidence over longwall mines

Country/ Coalfield	Residual Subsidence Duration	Reference
United Kingdom	Several months to 3–6 years (strong overburden)	Orchard and Allen 1975
United Kingdom	2 to 4.5 years	Collins 1977
Germany	1 year (Cretaceous overburden), 2 years (sandstone overburden)	Brauner 1973a, 1973b
USSR	2 years (shallow mines), 4 to 5 years (deep mines, >400 m)	Brauner 1973a, 1973b
United States/ Illinois	12 months	Hood et al. 1983
India	16 months	Singh and Singh 1998

presence of water; depth of workings; pillar size; and extraction ratio.

The caving process associated with roof collapse will either migrate to the surface or become stable at depth, depending on the volume of the original void and the strength and bulking factor of the caving material. The surface expression of this subsidence is generally localized depressions or pits. Pillar failures result from either weathering or increased loading due to surface development (Piggott and Eynon 1978) or mining activity (e.g., pillar robbing, other nearby operations, or load transfer due to collapse of adjacent small pillars) and leads to trough-like subsidence. Punching failures (squeezes or crushes) occur when either the immediate roof or floor is weakened or altered by the action of water or other weathering processes and pillars are forced into them. Generally, the subsidence takes the form of a trough or basin. The failure mechanism in this case is similar to that of a structure exceeding the bearing capacity of its foundation. Factors that encourage punching failure include argillaceous materials in the roof or floor, high pillar stresses, and flooded conditions.

Of the wide range of factors that affect the duration of residual subsidence associated with R&P operations, the principal ones include depth of working; mine geometry; mine support and reinforcement; geomechanical properties of the overburden, roof, pillars, and floor (e.g., strength, stiffness, degree of fracturing); and groundwater and mine humidity. This wide variety of interrelated contributory factors means there is no comprehensive model for predicting the duration of residual subsidence over R&P workings. Many case histories are reported in the literature, a thorough review and understanding of which is likely to be more fruitful than attempts to develop predictive models for a particular operation.

Subsidence-Induced Environmental Effects

The surface displacements and deformations characteristic of subsidence will affect any use made of the ground surface. This includes the ground as a foundation for buildings and structures, as well as a resource in its own right; subsidence affects both the built and the natural environment.

As previously outlined, subsidence comprises five major components:

1. Vertical displacement
2. Horizontal displacement
3. Slope (or tilt), the derivative of the vertical displacement with respect to the horizontal displacement
4. Horizontal strain, the derivative of the horizontal displacement with respect to location
5. Vertical curvature (or flexure), given approximately by either derivative of the slope

Vertical displacements alone cause little structural damage, although lowering of the land may adversely affect both urban and rural drainage systems, flow-through transport pipes, groundwater regimes, and vertical alignment of roads and railways. Similarly, uniform horizontal movements of the ground surface also cause little damage to structures. However, spatial variation in vertical displacement causes formation of slopes that can interfere with drainage systems and the vertical alignment of roads and railways, and can cause unacceptable tilt in tall structures such as chimneys. Surface horizontal strains cause most of the damage to structures located above mined areas. Extensile strains can lead to breaks in pipes, electricity or communications lines, and roads, and cracks in masonry or similar brittle structures. Contractile strains lead to distortion and buckling of steel structures, pipes, and railway tracks; damage to walls; and failure of the weaker parts of masonry structures (e.g., around door and window openings).

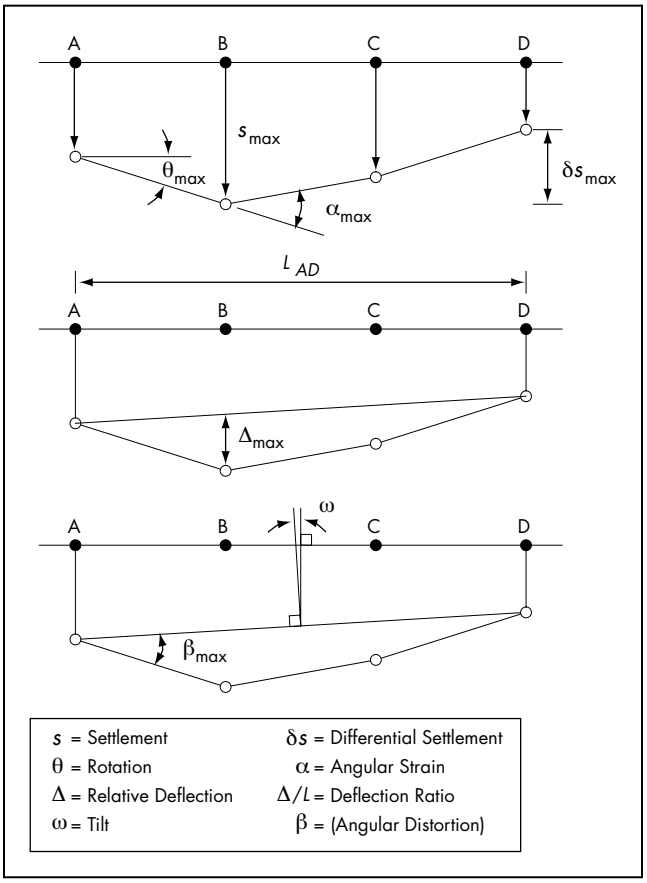
In most cases it is a serviceability, rather than an ultimate limit state, that governs whether or not subsidence damage is tolerable (Wahls 1994). Quoting other work, Wahls suggests that "movement is *not* tolerable if damage requires costly maintenance and/or repairs, *and* a more expensive construction to avoid this would have been preferable." The concept of *tolerable damage* is important. In general, it is related to the consequences of any damage and the ease and economics of repair, but in the context of mining it is also related to the cost and economics of changed or abandoned mining plans, if needed to reduce the effects of subsidence.

Work continues worldwide to codify the effects of deformation on structures. For example, European Standard EN 1997-1:2004 (i.e., Part 1, Geotechnical Design, of Eurocode 7, often referred to simply as EC7; CEN 2004) presents guidelines for limiting values of structural deformation and foundation movement. This code requires designers to consider overall settlement, relative (or differential) settlement, rotation, tilt, relative deflection, relative rotation, and horizontal displacement of structures (see Figure 8.9-12 for definitions), and gives guidelines for critical values for many of these parameters. Thus, the maximum acceptable relative rotations for open-framed structures, infilled frames, and load-bearing or continuous brick walls are likely to lie in the range from about 1/2,000 to about 1/300, with a maximum relative rotation of 1/500 being acceptable for many structures. As these ratios apply to sagging mode deformations, the code suggests that the values be halved for hogging modes (i.e., vertical displacement of the edges more than that of the parts between). Similarly, it is suggested that for normal structures with isolated foundations, total settlements up to 50 mm may be tolerable, with larger settlements being acceptable provided they do not cause excessive relative rotations, tilting, or other problems such as with the services entering the structure.

Finally, the code stresses that these guidelines "apply to normal, routine structures." For other structures, specialist advice should be sought.

Buildings and Structures

Although EC7 requires the effects of differential settlement, horizontal strain, angular distortion, and tilt to be recognized, it is the complex interaction between the magnitudes of these deformations and the characteristics of the



Source: Adapted from CEN 2004.

Figure 8.9-12 Definitions of structural deformation

structures themselves that determines the nature and amount of damage.

Extensile horizontal ground strain tends to produce vertical and step-like cracks in brick walls, generally of uniform aperture. Extension cracks in floor slabs usually occur at right angles to the direction of the extensile strain, with the sides of the cracks showing little or no shear offset. Contractile strains lead to bulging of walls and buckling and heaving of floor slabs and rigid coverings such as floor tiles. Angular distortion (or relative rotation) is a measure of shearing and leads to diagonal cracks in masonry and brick walls and their plaster coverings, and binding of doors and windows.

The concept of tolerable damage has led to the development of classification schemes such as that shown in Table 8.9-4. The information in this table has a history stretching back to the middle 20th century and is widely accepted. However, the tolerable damage concept means that this table and others like it should be considered as guidelines; for example, distorted window frames may be tolerable for some buildings, whereas even slight cracking may be intolerable for others.

These damage criteria are often embodied in various charts. As examples, Figure 8.9-13 shows how horizontal strain and angular distortion combine, and Figure 8.9-14 shows how horizontal strain and length of structure affect the damage classification.

Damage may be caused to bridges by horizontal ground strain resulting in the movement of the supports of piers either toward or away from one another. Differential vertical settlement or distortions in the horizontal plane may bring about complex and often serious effects on the decking and arches (National Coal Board 1975). It is generally difficult to determine the location and nature of the ground movements based on visual observation of damage. Often compressive damage leads to crushing and spalling of concrete decks, with combined compression and extension due to bending causing opening and closing of construction joints in abutments. More substantial damage is characterized by distress in the superstructure, inward horizontal movement of abutments, jamming of beams and girders against the back wall of the abutments, and serious damage to the bearings (Moulton et al. 1985). These findings are summarized in Table 8.9-5.

Across a wide range of other structures, consensus seems to be converging on a value of 1.0×10^{-3} for tolerable horizontal strain (see, for example: Nishida and Goto 1970; Lackington and Robinson 1973; Lee 1977; Institution of Civil Engineers 1977; Kratzsch 1983). However, specialist advice should be sought for nonroutine structures.

Public Utilities and Communications

Roads and airport runways. Subsidence principally affects roads and roadways through formation of cracks and undulations in the surface; distortions to both horizontal and vertical alignment; damage to ancillary structures such as drains, fences, and curbs; deterioration of base courses and subgrade; and ponding of surface water. Structures used for high-speed operations (e.g., runways and motorways) are particularly vulnerable to such damage. The formation of tensile cracks on the pavement surface (usually coinciding with the position of the rib-sides in the mine workings) and compression ridges near panel centers (Donnelly and Melton 1995) are the most common forms of damage. As highway bridges are generally more susceptible to subsidence than the highways themselves, bridge damage criteria may be used to give conservative limits for highways.

Railways. Railway authorities are generally protective of railway infrastructure and will set quite prescriptive limits on subsidence effects. One of the first effects of subsidence on railways is rider discomfort, which may require the reduction of maximum permissible speeds. Changes in ground slope may adversely affect track performance by formation of localized depressions or creation of gradients greater than permissible for a given type of traffic. The reversals of surface horizontal strain (i.e., from zero to extensile strain initially, followed by contraction) observed over longwall panels as the working face progresses are particularly damaging (Institution of Civil Engineers 1977). At higher levels of contractile strain, rail tracks have a tendency to snake or bend, and in more extreme cases, the rails themselves may be forced from the track.

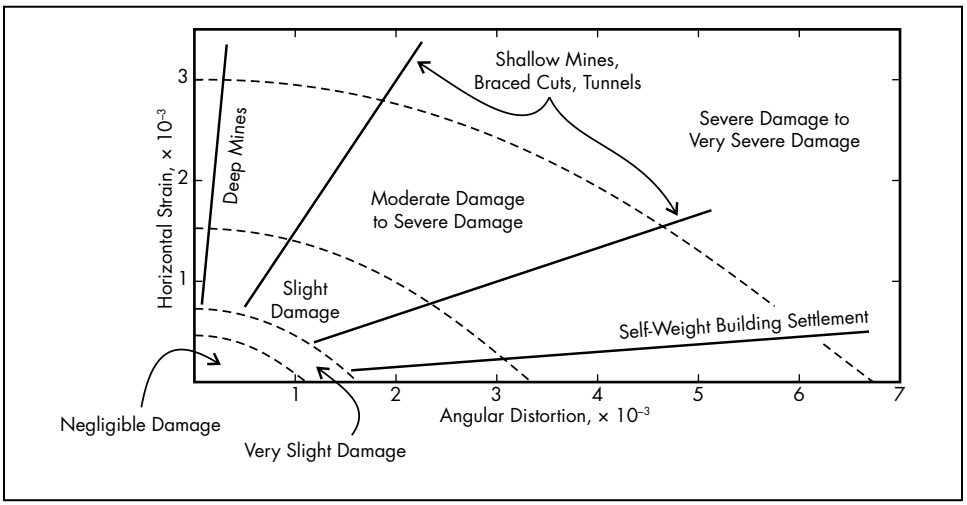
The extent to which a railway line is affected by ground movements is related to types of traffic involved, speed limits, types and construction of track, preventive and remedial works, and nature and magnitude of ground movements. Serviceability limits of 2.0×10^{-3} and 10.0×10^{-3} for horizontal strain and slope, respectively, have been suggested (Kratzsch 1983).

Pipelines. Generally, pipelines are laid below the ground surface and so deform in response to ground movements. If the magnitudes of these movements are such that either the pipeline or its joints or couplings are unable to accommodate the strains and rotations developed in them, the pipeline will fail. In the case of thin-wall welded steel pipe, this failure may

Table 8.9-4 Classification of building damage

Class of Damage	Description of Damage*	Repair	Approximate Width of Cracks, mm†
Negligible	Hairline cracks	None	<0.1
Very slight	Cracks in exterior brickwork visible upon close inspection; possible isolated slight fracture in building	Fine cracks easily treated during normal redecoration	<1
Slight	Several slight fractures inside building; visible external cracks; doors and windows may stick slightly	Cracks easily filled; redecoration probably required; some repointing may be required for weather tightness	<5
Moderate	Doors and windows sticking; weather tightness often impaired; utility service may be interrupted	Cracks may require cutting out and patching; recurrent cracks can be masked by suitable linings; repointing and possibly replacement of a small amount of exterior brickwork may be required.	5–15, or several cracks >3 mm
Severe	Windows and door frames distorted; floor slopes noticeably; walls lean or bulge noticeably; some loss of bearing in beams; utility service disrupted	Extensive work involving removal and replacement of sections of walls, especially over doors and windows	15–25, also depends on number of cracks
Very severe	Beams lose bearing, walls lean badly and require shoring; windows broken by distortion; danger of instability	Major work involving partial or complete reconstruction	Usually >25, depends on number of cracks

Source: Adapted from Wahls 1994.
*Location of damage in the building or structure affects assessment of degree of damage.
†Crack width is only one aspect of damage and should not be used solely as a measure of it.



Source: Adapted from Boscardin and Cording 1989.
Figure 8.9-13 Building damage in terms of angular distortion and horizontal strain

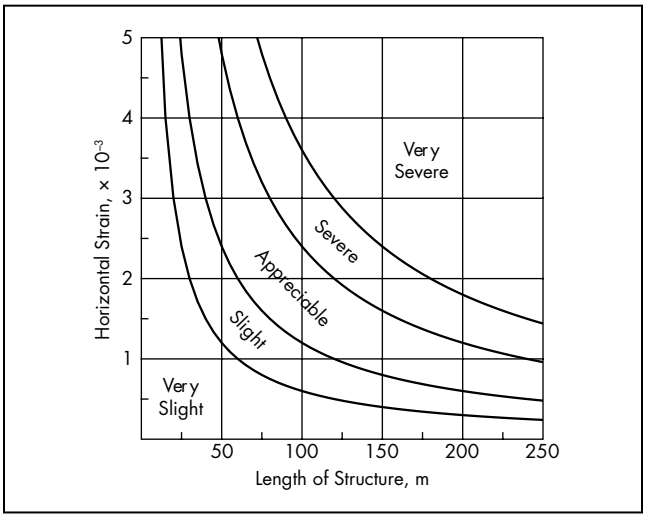
be one of serviceability, in that the pipe undergoes local buckling; for other cases, it is likely to lead to either interruption or loss of service. The consequences of these depend on the contents of the pipeline and the nature of the service it provides.

The resistance a pipeline offers to ground movements is dependent upon such factors as the mechanical properties of the pipe material, the rotation and pull-out capacity of the couplings, connections to other structural elements, corrosion resistance of pipe and joints, state of repair, and installation technique (Trautmann and O'Rourke 1985; Abdoun et al. 2009). For pipelines constructed with flexible joints, the joints constitute the weakest component in a pipeline. Thus, the location of joints with respect to the subsidence profile together with the rigidity of the pipeline will significantly influence its susceptibility to damage, which is focused on whether subsidence-induced displacements can cause leakage at the joints (Luo et al. 1998). For pipelines constructed with welded, heat-fusion, or rigid bolted joints, the joints are as strong as the pipeline itself, and induced stress becomes the critical factor. Curvature and horizontal strain at the ground surface resulting from subsidence induce stress in a pipeline, and this must be added to the normal pipeline operating stresses.

To minimize subsidence effects, buried pipelines can be uncovered and actively releveled while subsidence is taking place. In all cases where substantial ground deformation is anticipated, suitable pipe layouts should be used to accommodate these; pipeline manufacturers supply standard details for such layouts.

Subsurface Structures

The deformations associated with mining occur at all depths within the rock above an extraction zone, as indicated in Figure 8.9-15, and so subsidence damage is liable to affect subsurface structures in addition to those at the ground surface. As Figure 8.9-15 suggests, the deformations and hence damage experienced by subsurface structures depend on their



Source: Adapted from National Coal Board 1975.

Figure 8.9-14 Structural damage in terms of length of structure and horizontal strain

locations relative to the extraction zone. Of course, mining operations may mean that such a zone will extend laterally, and in this case a subsurface structure may experience a range of strain regimes.

Shafts are the structures most critically affected by subsurface deformations. Although it is possible to locate shafts outside the area of influence of any workings—that is, within a protective shaft pillar—this may lead to uneconomic sterilization of resources for deep mines in laterally extensive deposits such as coal seams. In those cases where shafts cannot be protected within a pillar, it is important that they are designed and constructed to be serviceable under all of the vertical and horizontal deformations that the mining will induce. The tolerable deformations will depend on factors such as type of shaft lining, shaft use (i.e., ventilation or hoisting), and the presence of water-bearing zones.

The effect of subsidence on smaller mine structures, such as access drives and chambers for machinery and services, depends primarily on their position relative to the extraction zones. Below and at the level of extraction, the increased stress due to excavation rather than deformation is the most problematic. With appropriate design, all such structures should remain serviceable throughout mining. Structures within the caving material immediately above an extraction zone are unlikely to survive, regardless of their form of construction. At greater distances above the extraction zone, where the fracturing and deformation of the rock are more limited, it may be possible to design and construct structures that remain serviceable. The same holds true for structures in overlying soils.

Agricultural Resources

Essentially, subsidence damage caused to agricultural resources is characterized by loss of use or reduced productivity. Subsidence generally affects agriculture through interference with the established hydrology, for example, formation of surface fissures in areas of extensile strain, leading to erosion and reduced water retention; changes to ground slope, leading to either increased runoff and erosion or reduced runoff and waterlogging; disruption of surface drainage patterns and systems; and contamination of groundwater with deleterious minerals such as sulfides and chlorides (Singh and Bhattacharya 1987). Subsidence features such as sinkholes can also affect the hydrology and in extreme cases can lead to sterilization of the resource or loss of livestock and equipment. Although forests and grazing lands are generally less susceptible to subsidence damage than are cultivated lands, the types of damage to which they are vulnerable are broadly similar.

Table 8.9-5 Tolerable deformations for bridges

Movement Type	Tolerable Magnitude	Source
Angular distortion	$4.0–5.0 \times 10^{-3}$	Moulton et al. 1985
Differential settlement	25 mm	DiMillio 1982
	25–50 mm	Moulton et al. 1985
Horizontal	35 mm	Moulton et al. 1985

Hydrological Resources

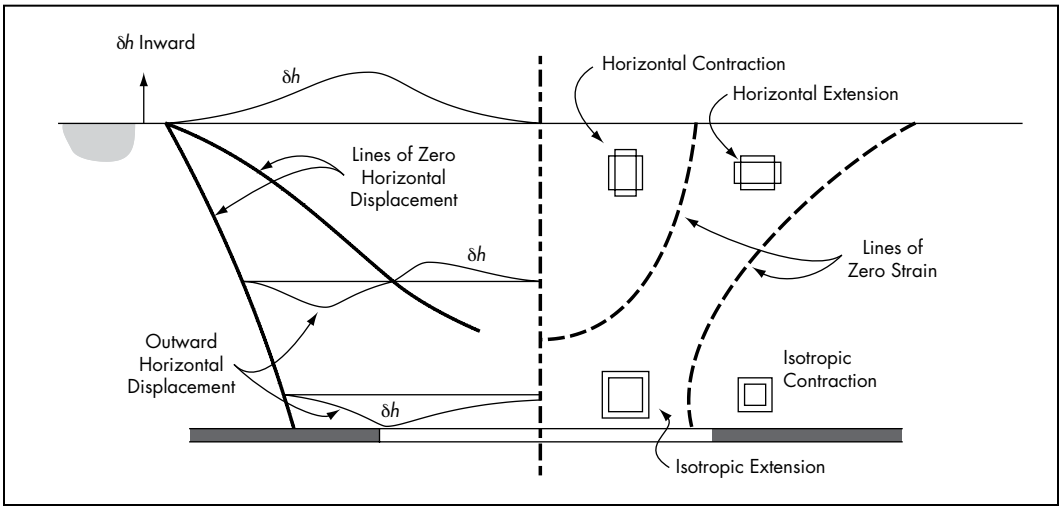
If zones of extensile strain form beneath a body of surface water, there is the risk that fissures may develop in the underlying ground, leading to partial or total loss of water. If the body of water is extensive (e.g., a lake or the sea), then catastrophic inundation of the mine workings can occur. To prevent this latter occurrence, strict conditions are placed on working beneath water, and these can be used to offer conservative guidelines for prevention of subsidence damage. For example, the U.S. Bureau of Mines recommends a safety zone around and beneath any body of surface water, as shown in Figure 8.9-16, and gives guidelines for mining beneath such a safety zone (Babcock and Hooker 1977).

For the case of aquifers, subsidence-induced fissuring may significantly affect the hydrogeological and geochemical regime (Singh and Bhattacharya 1987; Booth 2002). Water table depression is common because all underground openings act as sinks, but the presence of impermeable materials in the overburden may inhibit this. Structural features such as faults and folds and the nature of the rocks themselves may have a significant influence on subsidence effects. Flow rates in surface streams may change, and the general loss of water can lead to reduced plant growth. The increased fracturing and changes in water flow paths can lead to oxidation of a number of minerals, leading to reductions in water quality and hence usability. This can be particularly important for potable water supplies. The limits set for undersea mining will probably lead to conservative conditions for mining beneath aquifers.

Pseudo-Mining Damage

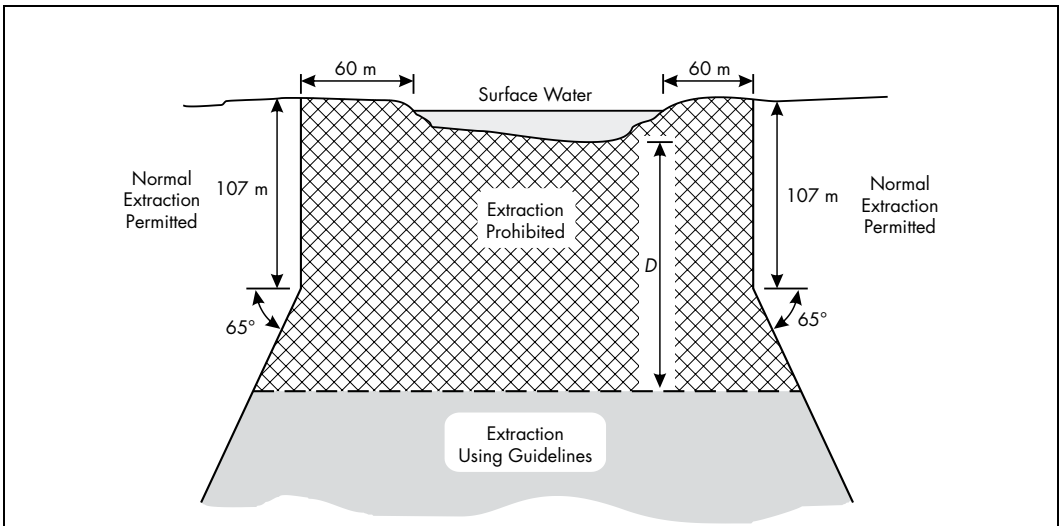
Inhabitants of mining areas may blame mining activity for any damage observed in local structures or lands, especially if compensation is available for mining-related damage. Because damage may be due to effects other than mining, it is important that any other causes are recognized and the true source of the damage properly determined.

Differential settlement can occur in various ways. All foundations settle when first loaded, and so differential settlement can happen when old and new foundations are combined; this commonly occurs when old structures are extended laterally. Foundation settlement is particularly pronounced on clayey soils, and laterally varying soil properties can lead to differential settlement. Likewise, constructing a foundation on fill material, especially when the fill is improperly compacted or the fill only partially underlies the structure, can give the same effect.



Source: Adapted from Kratzsch 1983.

Figure 8.9-15 Subsurface horizontal deformations determined using continuum finite element analysis



Depth of Safety Zone
Total Extraction: $D > 60t$
Room and Pillar: $D >$ max ($5s$, $10t$)

Principal Extraction Guidelines
Total Extraction: Total extraction thickness < $60D$
Horizontal extensile strain at bed of water body < 8.75×10^{-3}
Room and Pillar: Pillar dimensions must satisfy

$$D > \left[\frac{552}{\sqrt{t}} + 20\left(\frac{w}{t}\right)^2\right] \Big/ \left[4.921\left(1 + \frac{s}{w}\right)^2\right]$$

where
t = Extraction Thickness, m
s = Room Span, m
w = Pillar Width, m

Source: Adapted from Babcock and Hooker 1977.

Figure 8.9-16 Safety zone beneath a body of surface water

Some clayey soils change volume dramatically as their water content changes, shrinking as they dry and swelling as they become wet. Such changes can occur seasonally, can be the result of leaking water pipes and sewers, or can be due to planting or removal of nearby vegetation. Similar effects occur in fine-grained soils from freezing and thawing effects. As soil volume change by itself leads only to vertical strain, the absence of horizontal strain in a structure can be used to exclude mining as the cause of damage.

Changes to the hydrological regime can similarly lead to damage. Building activity, irrigation, or drainage associated with major excavations can all remove water from the ground. Inadequate or faulty drainage can lead to rising groundwater levels and saturation of the ground. Seepage into subsurface structures can result, as can deterioration of materials such as wood and plaster. However, hydrological changes resulting from mining activity are subsidence damage and must be recognized as such.

Poor-quality construction and building materials can increase the likelihood of damage to structures, as can natural wear and tear. Construction and maintenance records, together with an assessment of anticipated mining damage, can help identify these effects. Poor detailing whereby materials of widely different coefficients of thermal expansion are affixed to one another can lead to either cracking or buckling as the ambient temperature changes. Buckling in hot weather is particularly troublesome for poorly detailed, linearly extensive steel structures such as pipelines and railways.

Many components of buildings and structures are susceptible to chemical attack. In the case of ferrous components embedded in masonry or concrete, the volumetric expansion associated with the formation of rust can lead to tensile splitting and spalling of the host material. With steel window and door frames, cracked glazing may occur. Concrete is susceptible to a number of adverse chemical reactions. The presence of siliceous or carbonate aggregates in concrete can cause alkali–silica or alkali–carbonate reactions, whereby the reactive compounds swell due to absorption of water, and the swelling leads to formation of random patterns of cracking. Sulfate attack from either minerals in the ground or the aggregate, or ions dissolved in groundwater, can lead to similar swelling and cracking.

Dynamic loads from seismic events, machinery in industrial plants, or road and railway traffic can lead to cracking in brittle construction materials and differential settlement due to consolidation of foundation materials. Substantial surface movements on slopes can also be generated by various natural processes, such as freeze–thaw and solifluction (flow of saturated soil down a slope).

Extraction of fluids, particularly from aquifers and petroleum reservoirs, can also lead to substantial subsidence and damage. Such activities are usually licensed and highly regulated, and as such it will be straightforward to determine the degree of damage due to such operations.

Control and Prevention of Damage

The measures that can be implemented to control and minimize subsidence damage fall into the categories of

- Adoption of particular mining techniques,
- Postmining stabilization,
- Architectural and structural design, and
- Comprehensive planning (Singh 1984).

Each of these comprises a number of devices. In all of these, comprehensive calculations are required to identify the economically optimal approach or combination of approaches.

Adoption of Particular Mining Techniques

The principal measures to consider are partial mining, changes to the mine layout, harmonic mining, backfilling, and changing the extraction rate.

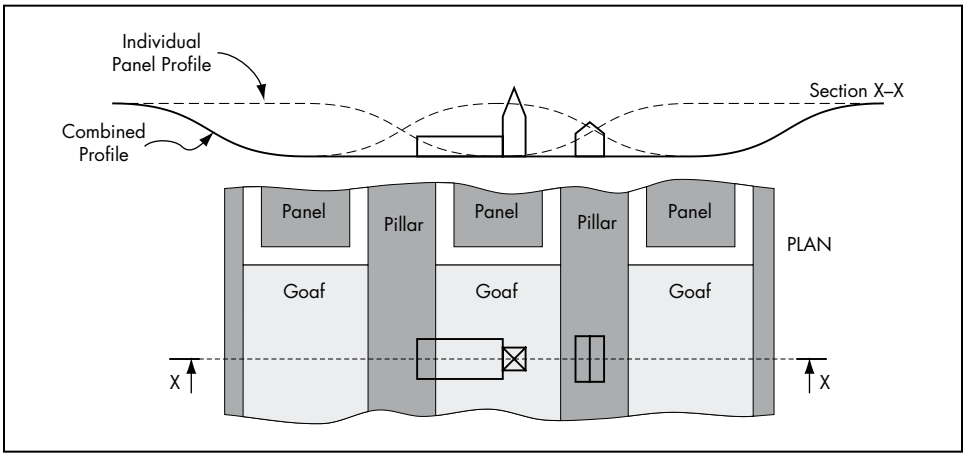
Partial mining generally involves such measures as leaving protective pillars beneath critical surface infrastructure. As noted previously, with deep mines this is liable to lead to sterilization of substantial resources and hence be rendered uneconomic. Any structure supported by a protective zone will remain at its original elevation and thus end up at a higher level than the surrounding area once subsidence has taken place. This may lead to problems with transport links and utilities. In coal mining, partial mining can also involve extracting less than the full thickness of a seam. Apart from subsidence control, this can also be beneficial if either the immediate roof or floor is weak or susceptible to weathering.

Other partial mining techniques, such as electing to use an R&P operation or mining subcritical widths in order to reduce the maximum subsidence, are in some respects changes to the mining layout. Such changes involve locating and orienting extraction zones so as to limit deformations in surface structures to tolerable levels. For example, aligning the center line of a supercritical extraction zone with a surface structure will reduce the effects of changes in horizontal strain.

It is possible to adopt a mining layout whereby sized pillars separate adjacent and parallel extraction zones, so as to generate essentially uniform subsidence across the complete mining area (Figure 8.9-17). Clearly, for optimal benefit, this requires extraction of adjacent zones to proceed simultaneously. This is an example of harmonic mining, whereby extraction of adjacent zones is scheduled so as minimize changes or magnitudes of horizontal strain. As Figure 8.9-5 shows, regions of extensile and contractile strain exist above the rib-side of an extraction zone, and so by simultaneously mining two appropriately separated zones it is possible to substantially minimize the horizontal strain. However, harmonic mining severely constrains the mine operation and, in a mechanized environment, requires substantial investment in mining equipment. As a result, it is generally only applied where protection of surface facilities is paramount and mining costs can be disregarded.

Backfilling a mined void is well known to reduce subsidence. However, in mining operations that generate little waste material, the material for backfilling will need to be imported and the economics of this will need careful assessment. Where suitable material exists, separate infrastructure from that required for extraction may be needed for its emplacement. The reduction in subsidence damage brought about by backfilling benefits all surface facilities and may be particularly appropriate in areas where maintenance of surface drainage regimes is required.

Finally, subsidence development depends to some extent on extraction rate (Goulty and Al-Rawahy 1996). Thus, it is worthwhile to select an extraction rate that minimizes subsidence damage. Generally, higher rates are appropriate for unfractured rocks and lower rates for fractured rocks. However, after the equipment for a mechanized mine has been selected and commissioned, it is difficult to increase the extraction rate should that be found necessary.



Source: Adapted from Singh 1984.

Figure 8.9-17 Pillar widths between adjacent extraction zones sized to give uniform vertical movement

Postmining Stabilization

Stabilization of complete mine sites extending over many hectares may be achieved by backfilling (as previously outlined), grouting, or—in the case of shallow voids beneath derelict or unused land—complete excavation and backfilling. Grouting, which has found widespread application, involves pumping or injecting a cementitious material into the mine void using an array of boreholes drilled from the surface. High-pressure injection may be necessary if the grout is required to penetrate joints or fissures in order to stabilize a fragmented rock mass. Stabilization of individual structures generally involves either localized grouting of the mined void or, in the case of shallow workings, underpinning with new foundation structures, such as deep piles.

Architectural and Structural Considerations

When structures are to be built in areas of known or future mining activity, designs should be adopted that will tolerate the anticipated ground movements. This may include locating and orientating structures so they expose their shortest dimension to a subsidence trough (Figure 8.9-14 shows how damage decreases with decreasing length of structure) and avoiding the surface outcrop of any faults (Figure 8.9-7).

Many design techniques are available to produce structures tolerant of subsidence. For structures that are small in plan, it may be appropriate to adopt a rigid foundation and superstructure that is capable of withstanding differential settlement and horizontal strain. Such structures will tilt as a rigid body, which needs to be fully taken into consideration. In other cases, flexible structures are designed to ride over the subsidence wave associated with a mining face and do so by permitting free ground movement below the foundation, ensuring sufficient superstructure support and withstanding ground deformations without loss of structural stability. Concrete floor slabs no longer than about 18 m, which contain both top and bottom reinforcement and are founded on granular material, are characteristic of these designs. Trenches around the structures, backfilled with granular material, can be used to absorb horizontal strain. With utility buildings such as low-rise industrial units and warehouses, some minor damage may be tolerable. In these cases, it may be most economic to adopt a design that is a hybrid of the rigid and flexible approaches, and undertake minor repairs as required. For structures where avoidance of tilt is crucial, jacks or similar leveling devices may be necessary between the foundation and the superstructure.

Specific design details that are appropriate for domestic buildings include provision of movement joints and use of structural connections capable of withstanding both extensile and contractile strains; window and door frames positioned so as not to weaken the overall structure; internal walls finished with plasterboard rather than plaster; floors and roof secured to the walls; omission of bay windows, porches, and unreinforced masonry; outbuildings detached from main buildings; provision of increased falls to all gravity drainage systems, including gutters; no paving immediately adjacent to buildings; flexible paving materials; and light boundary fences rather than walls.

In the case of existing structures, overall repair expenses may sometimes be reduced if the structures are suitably modified prior to the commencement of subsidence (Kratzsch 1983). Measures that can be taken include provision of temporary supports and/or strengthening to parts susceptible to damage; supporting screens, partitions, and the like independent of the walls and floor; installing wall ties or temporary corbels to roof trusses likely to be pulled from their seats; taping windows, especially those with metal frames, to prevent flying glass; removing stained-glass windows to safe storage; temporarily installing pretensioned steel mesh around exterior walls; excavating trenches around buildings to below foundation level (without disturbing the foundations), backfilling with compressible material and, if required, covering with flexible paving; cutting slots in rigid pavements, floors, and superstructures to leave units about 18 m long; removing parts of structures (e.g., connecting corridors); and removing complete units from within rows of buildings.

In all cases where structures are damaged by subsidence, no repair work should take place until subsidence is complete. As Figure 8.9-5 shows, an approaching working face is preceded by a zone of extensile strain that quickly becomes one of contraction. As a result, cracks should not be filled until all movements have stopped and any debris that has fallen into cracks should be removed before the contractile strain develops.

Comprehensive Planning

Successful implementation of any surface land use or mine plans requires extensive knowledge of the requirements of each and complete collaboration between the various interested parties. It is essential that everyone (i.e., not only the general public but also mine personnel, developers, government and utility officials, and public interest groups) likely to be affected by subsidence is completely aware of all the issues.

Principles to be applied in the planning of development include the following:

- Avoid construction across faults.
- Construct only specially designed structures over shallow workings.
- Locate structures in areas where little strain is anticipated.
- Locate significant individual structures over areas that have either completely subsided or will not be mined.
- Locate linear structures such as canals, motorways, and railways so they can be uniformly lowered along substantial lengths.
- Avoid locating important structures near mine boundaries because coordination with several mine operators and surface landowners is onerous.

It may be necessary to bar construction from particularly risky areas, which should then be used for open spaces.

Development of areas likely to undergo subsidence must be socially and culturally acceptable as well as economically justifiable. This implies that the associated development plans must not only be agreed to by the mine owners and landowners but also be open to public comment prior to adoption. Any changes to such plans require the same public input.

Through a wide range of regulations—including environmental impact assessments, environmental protection, building regulations, safety requirements, and mining regulations—national, regional, and local government authorities exert considerable control over both subsidence and the development of land that is vulnerable to subsidence damage. In various places worldwide, mine owners have statutory obligations to prepare—and gain approval for—subsidence management plans that are circulated to all interested parties for comment and suggestion before being approved. It is likely that in the future this requirement will become universal.

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CHAPTER 8.10

Tailings Impoundments and Dams

Thomas Kerr and Bryan Ulrich

INTRODUCTION

This chapter is an introduction to the techniques for and challenges associated with safe storage of waste residues from the mining industry, known as tailings. In most instances, tailings comprise crushed and ground rock and soil particles but in some cases may contain precipitates and/or residual metals or minerals from the ore or the process. Because they often form one of the largest volumes of waste from a mining and milling operation, together with their physical and chemical characteristics, tailings disposal is one of the largest elements of risk management and cost for a mine. Such risk is captured by the following quote: "The management of the disposal of mill tailings is of critical importance to the success of any mining project. Failures of tailings facilities have resulted in loss of life, devastating environmental damage, the closure of mining operations, dramatic declines in share value and, in some countries, the personal liability of the mine management" (Brown 2002).

In recognition of these consequences, the mining industry has, since 1980 or so, focused much attention on tailings management and improving the design, operation, surveillance, and closure of tailings facilities. Methods are now available to provide secure tailings storage over the full life cycle of a facility, but the main challenge is consistency in application. A second challenge is to maintain this momentum and continuously strive for improvement. Society will continue to demand more from the mining industry, and safe tailings management will be at the center of this demand.

The tailings characteristics are unique from mine to mine and are linked to the ore and the processes used to extract the mineral of value from the ore. Tailings produced at a phosphate mine, for example, will behave considerably differently than tailings produced at a gold mine. Similarly, uranium tailings will behave differently than those from an oil sands extraction operation. Each tailings facility is also unique because each site has different geological, hydrogeological, topographical, climatological, and seismological conditions that must be taken into account. The upshot therefore is that each tailings storage facility must be designed, operated, maintained, and ultimately closed in a project-specific manner. Although this requires dedicated resources and continuous planning and attention to detail, the business case for practicing sound tailings management is that a consistently well performing tailings facility adds considerable value to a mine in terms of reduced risk.

This chapter provides a summary of the key considerations related to present-day sound tailings management and is aimed chiefly at engineered, aboveground tailings storage facilities. Information on in-lake disposal, marine disposal, in-pit disposal, and tailings placement as underground backfill is largely beyond the scope of this chapter. The chapter discusses some of the risks associated with tailings storage facilities and their potential mitigation measures, and it presents some of the tools available to the mining industry to manage tailings responsibly through the full life cycle of a mine.

The design, construction, operation, and ultimate closure of a tailings facility can be complex, involving a broad spectrum of scientific and engineering specializations. This chapter does not attempt to cover these in detail. Rather, it should be considered as a starting point in gaining an understanding and appreciation for the types of issues that may need to be addressed in developing and managing a tailings facility. Specialist technical input is readily available from qualified people in the mining industry and companies that have assisted the mining industry with tailings management, and it is strongly recommended that these resources be utilized in any tailings facility project. Notwithstanding, it is important for nonspecialist individuals working with tailings facility design teams to have at least a basic understanding of the numerous considerations that must be adequately developed for proper design, and this chapter is intended to provide that knowledge.

The components of tailings facility design generally include

- Siting studies to select the most appropriate location and configuration;

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- Site geotechnical and geological investigations to characterize subsurface conditions and identify potential construction material borrow sources;
- Geochemical studies and modeling to estimate the tailings solids and water quality to assist in developing the environmental management plan;
- Hydrology and hydrogeologic studies to understand surface and groundwater flows, directions, quantities, and qualities to assist in developing the environmental management plan;
- Water balance calculations to define the logic for managing water within, to, and from the tailings facility and to estimate the associated flows and volumes;
- Environmental and social studies to define the prefacility or baseline conditions and estimate the changes or impacts (positive and negative) that may be expected when operations begin;
- Field and laboratory testing to develop engineering parameters to be used in analyses; and
- Engineering analyses to select the various components of the tailings facility, as well as their configurations and specifications, often including the following:
  - Embankments, including zonation;
  - Filters and drains;
  - Basin area lining system and underdrainage;
  - Tailings slurry rheology and dewatering to select an appropriate consistency for deposition;
  - Tailings delivery and deposition systems;
  - Fresh or nonimpacted water management systems, including perimeter diversions; and
  - Process water management systems, including tailings water recovery and reuse.

Often the design of each of these components is iterative with data collection and progresses in steps: from conceptual development to prefeasibility studies to feasibility design and final design. In addition, most tailings facilities are developed in stages and expanded throughout the life of the mine, and thus the design does not occur as a single step but is an ongoing process. These concepts are discussed in greater detail in the following sections.

TAILINGS MANAGEMENT PRINCIPLES

The consistent application of good practice and continuous improvement in tailings management over the life of a tailings facility (in some cases in perpetuity) requires the use of a comprehensive information and resources management system. Such a system, often referred to as a framework for tailings management, addresses the generation, organization, dissemination, documentation, implementation, and continuous upgrading of information, procedures, and policy matters related to a tailings facility over its life cycle. The life cycle of a tailings facility typically includes

- Initial site selection, planning, and design;
- Construction of stage 1 or the starter facility;
- Operation and periodic construction of subsequent stages; and
- Decommissioning, closure, and postclosure.

A framework for tailings management not only defines the engineering, operation, maintenance, and surveillance activities for the facility, it defines the roles and responsibilities for the personnel involved. This includes management in terms of policy and procedural reviews and the provision of appropriate resources for completing the necessary tasks. The framework also addresses procedures for managing change in personnel, design, and operating criteria and emphasizes proper and consistent documentation through all stages of the life cycle. In addition, the framework discusses engagement and open communication with all identified stakeholders external to the mine.

A good overview of the need for a tailings management framework is provided with examples in the document by the International Council on Metals and the Environment (ICME 1998). The introductory chapter, titled "A Management Perspective," begins with the following statement:

> The challenge to the mining industry is to assure itself, governments, local communities, and other stakeholders that it is capable of meeting its responsibilities to manage tailings deposition in a manner which achieves the highest standards. It is not enough to rely on the fact that the technology is available to ensure all environmental objectives are met. The mining industry must go beyond technical excellence and demonstrate that it has the commitment, and the management systems and skills, to be able to manage tailings in keeping with the expectations of its stakeholders.

An excellent guideline for developing a tailings management system is provided by the Mining Association of Canada (MAC 1998), which describes the five principal elements of the MAC management system that are continuously evaluated throughout the life cycle of a tailings facility. The five elements are

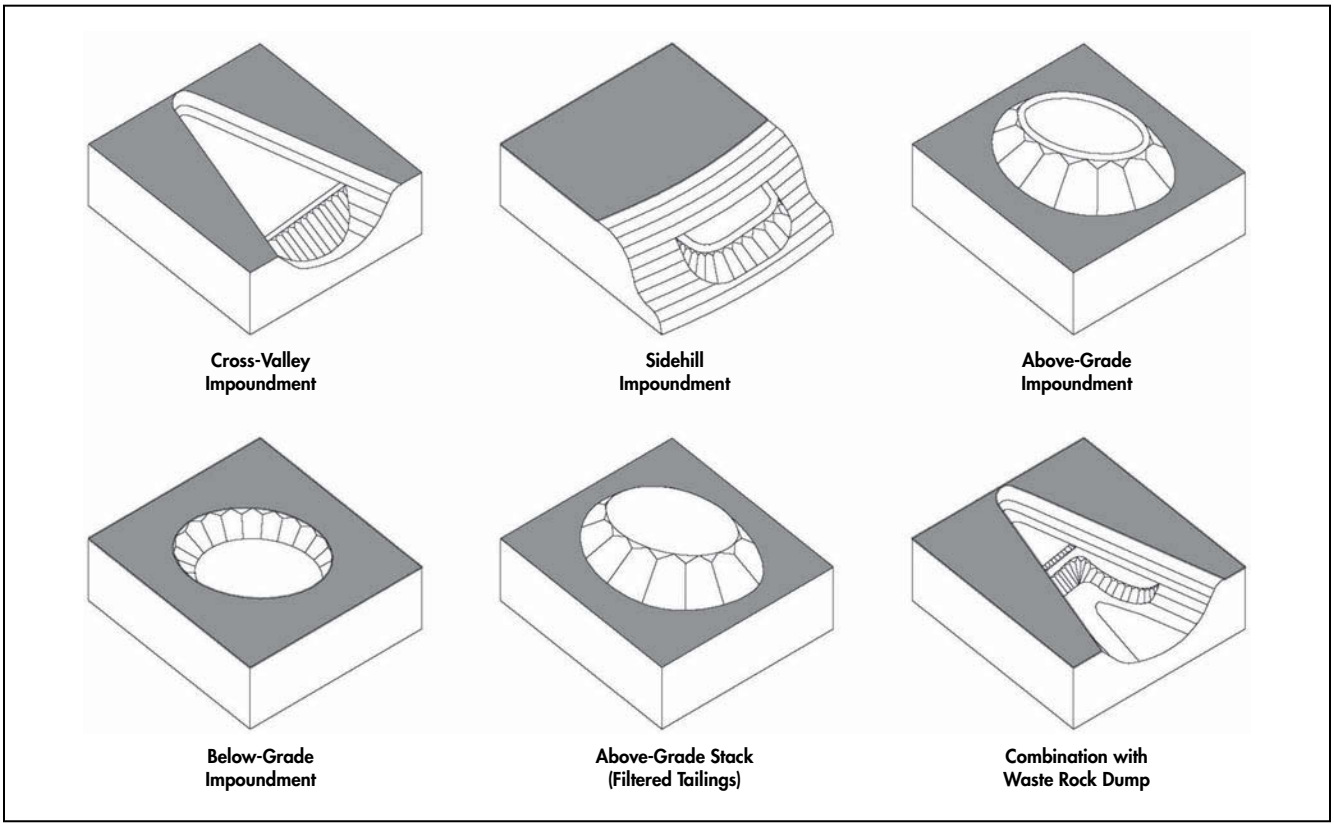
1. Policy and commitment of the company,
2. Planning,
3. Implementing the plan,
4. Checking and corrective action, and
5. Management review for continuous improvement.

In keeping with the ICME statement, it is pertinent to note that two of the five components of the MAC guide are policy and commitment of the company, and management reviews. The MAC guide contains checklists of activities for each component under each life cycle stage to serve as a starting point in developing a framework.

Another guide for developing a tailings management system is provided by the Australian government's Department of Industry, Tourism and Resources (DITR 2007). Section 4.0 of the guide describes the components of a tailings management system, and Section 5.0 describes the guide's use in leading practice.

TYPES OF TAILINGS FACILITIES AND CONSIDERATIONS IN THEIR SELECTION

Because tailings characteristics vary from mine to mine and each site is different, tailings facilities can take a variety of configurations, with different operating principles, methods of delivery and deposition, solids management and facility development, solution management systems, and closure plans. The large majority consists of on-land impoundments developed behind berms, embankments, or dams. Figure 8.10-1 gives some examples.



Source: Adapted from Zahl et al. 1992.
Figure 8.10-1 Types of tailings facilities

In many facilities, the tailings are deposited in slurry form onto sloping beaches developed from previous deposition. Surface water drainage, consisting of tailings process water and precipitation falling on the tailings facility, runs down the slope to a low point on the beach where a surface water pond is maintained, typically at a safe, prescribed distance from the embankment. Water is often reclaimed from this pond for reuse in the mill, and, if the climate is wet, excess water may need to be released after any necessary treatment has been completed.

Control of the beach, including the slope angle, length, and direction, and thus control of the pond location, is accomplished in part by the slurry deposition system. Such systems may consist of a perimeter distribution pipe with numerous valve-controlled offtakes for a high level of control or a few single-point outlets if less control is required. Some facilities using thickened tailings to produce a slightly steeper beach slope may employ a central discharge point or series of points whereby the beach develops radially outward and storage is provided in a conical deposit contained behind a low-height perimeter berm. In some cases where the tailings contain sulfide minerals that may turn acidic if oxidized (common, for example, in porphyry-type ore deposits), the tailings facility must be designed to reduce the risk of acid generation. In such a case, there are two common approaches:

1. Develop the facility as a water impoundment and place the tailings under water (which has a low oxygen diffusion coefficient and thus reduces the risk of oxidizing the tailings). This can only be done in areas where rainfall runoff or other water supply is sufficient to maintain a water cover.
2. Utilize a multiple offtake or spigot-type of tailings deposition system that enables the points of slurry discharge to uniformly blanket each area of the beach and frequently rotate the points around the facility such that each area becomes covered with fresh tailings before the underlying tailings can turn acidic. Usually the tailings at the time of discharge from the mill contain excess alkalinity from the milling process, and the key is to cover the previous tailings layer with the next layer before this alkalinity is consumed.

Other more unique disposal methods may also be adopted depending on site-specific circumstances; some of these are described later in the "Unique Tailings Management Schemes" section.

Materials for Construction

Many types of structures are built to contain tailings (such as embankments) and to protect the surrounding environment (such as liners and drains). Often the embankments are classified by the materials used to construct them and by the shape of the cross section. Embankments may be earth fill or rock fill or both, and in some cases may be concrete- or membrane-faced. Roller-compacted concrete dams have also been used.

The use of waste products from the mining process, including waste rock from the mine and the tailings from the milling process, can be economically and environmentally beneficial in certain cases. For example, waste rock can

often be directly hauled from the mine to the embankment at a relatively low overhaul cost. Tailings can be processed using hydrocyclones or controlled deposition to separate the tailings into coarse and fine fractions, and the coarse fraction can be used for embankment construction. The use of mine waste products can reduce the need for excavating natural materials and thereby reduce the overall area of land disturbance. It is, however, necessary to fully characterize the waste materials before using them in embankment construction. Mineralized waste rock may be subject to oxidation and release of metals that could cause environmental impacts. Oxidation can also lead to mechanical breakdown of rock particles, which can result in a loss of strength. The coarse fraction of tailings can also be subject to oxidation, which would result in acid drainage and/or can be subject to water and wind erosion.

The alternative to using waste materials is the use of borrowed natural materials. The availability of different types of materials—for example, low-permeability soils, sand and gravel, and durable rock—may drive the design of the containment structures. However, excavation of these materials from within the facility's storage basin can have the added advantage of increasing the facility's storage capacity.

Staged Development

Because tailings are generated and deposited progressively over the life of a mine, the development of the storage facility can be sequenced and built in stages. These stages may include dam construction, basin raises, or lateral expansions. The advantages of staged development include

- Deferral of some of the capital costs over the life of the mine;
- Allowance for design adjustments in the latter stages as experience and knowledge are gained from the early stages;
- Allowance for adjustments to account for changing conditions, including hazard consequence, if the land use around the facility changes;
- Increased use of waste rock or drained or cycloned tailings produced over the life of the mine; and
- Facilitation of a slow loading rate to maintain stability in staged development at sites where soft soils form the embankment foundation.

Staged development also has some disadvantages:

- Over time, unless properly managed, some of the original design objectives and supporting information may become lost or unclear as personnel change.
- The quality of the construction may vary from stage to stage if different contractors, materials, and/or quality assurance and control procedures are used.
- These disadvantages can be countered with good documentation, adequate resources at all times through operations, and a commitment by management and the operating staff to continuous improvement. These are all aspects of a sound tailings management system.

The staging sequence and schedule for the facility's operation should be defined at the time of the initial design of the facility. They are usually connected to the mine plan because that determines the schedule of ore production and thus, by association, the schedule of tailings production. The mine plan also determines the waste rock production over the life of mine in case the waste rock is used for facility construction. However, most mine plans change frequently, and, thus, any change that could affect the tailings facility should trigger a review of the staged development and tailings deposition plans. However, care must be taken when making significant changes, because changing the construction materials schedule may affect the following:

- Elevations and alignments of tailings delivery and water reclaim systems
- Dam foundation stability (e.g., if the dam is on a soft clay foundation and its rate of rise increases, an undrained shear failure in the foundation may be triggered)
- Construction material properties for embankments, drainage layers, and so forth (by virtue of the reduced availability of the most suitable materials, which may result in performance changes that could detrimentally affect impoundment behavior)
- Tailings deposit stability (e.g., if the tailings are being relied upon for structural support of a perimeter slope and the rate of rise increases without adequate drainage, an undrained shear failure in the slope may occur)
- Tailings deposit density and compressibility (e.g., if the rate of rise of the tailings increases without adequate drainage, the degree of consolidation of the deposit could decrease as more pore water is trapped in the voids, resulting in a lower density and a longer period for the new rapidly placed tailings to consolidate)

Basin Liner Systems

The requirements for lining a tailings facility basin in part depend on the geochemical characterization of the tailings to be stored, the groundwater characteristics below the basin, and regulatory requirements. Regulatory requirements can be either performance driven or prescriptive in nature. Performance-driven regulations usually set limits on seepage rates or levels of impact on nearby groundwater, whereas prescriptive regulations mandate a specific liner design on the basis of the waste to be contained. In either case, the chemistry and rates of potential seepage from the tailings facility are the motivating criteria in liner design. Sites that may not require a liner, if the chemistry of the fluid is adequate and the regulations permit, will generally meet one or more of the following criteria:

- Low-permeability ground conditions cause the seepage rate into the ground to be low.
- Hydraulic containment is provided by the groundwater regime.
- Tailings are drained with minor or no pore pressures (i.e., filter stack or subaerially deposited tailings in a dry climate) such that the potential is low for seepage outflows.
- Groundwater is deep in the bedrock such that the seepage path length between the tailings deposit and groundwater is long enough to allow for attenuation of flow and elemental constituents.
- Groundwater is of naturally poor quality such that the seepage inflows will not significantly degrade existing conditions.

The difficulty in adopting either the first or second of these criteria is in conclusively demonstrating during the site investigation phase of the design that no discrete leakage pathways will exist (Brown 2002).

For protection of groundwater where tailings impoundments cannot meet one or more of these criteria, it is necessary to design and construct a liner over the impoundment base. Liners may be as simple as the selective placement of low-permeability soil to cover outcrops of pervious bedrock or granular soils. Alternatively, liners may consist of a multilayered system of low-permeability soil and/or geomembrane layers, in some cases with a drain between certain layers for seepage removal. In many cases, it is conventional practice to incorporate a drainage layer above the liner, which functions as an underdrain beneath the tailings to reduce the pressure head on the liner, reducing seepage through the liner. Another benefit of such underdrainage is that it provides bottom drainage for the tailings, promoting accelerated consolidation and ultimately a higher strength in a shorter period of time. However, in some cases the tailings immediately above the underdrain may consolidate to a low permeability, thereby restricting further drainage from the tailings mass. In this case, the benefit of the underdrain is to remove head on the liner only. The drainage layer typically consists of at least a 300-mm (12-in.) thickness of granular material, with perforated pipes at selected intervals within the drainage layer. The pipes are installed to increase the hydraulic flow-carrying capacity of the system. A filter layer, usually consisting of nonwoven filter fabric, is also used to prevent the ingress of tailings solids into the drain.

Low-permeability soil liners have some distinct advantages over geomembrane liners:

- Because of their thickness, the initial seepage front through the liner has significant time to travel.
- Concentration of solutes in the seepage water may be reduced as a result of dispersion, diffusion, and adsorption by the soil.
- The liner will consolidate under loading by the tailings deposit, resulting in further reductions in its permeability.

A disadvantage of soil liners is that seepage takes place through the entire area of the liner. For large facilities, even if the permeability of the soil liner is low, the overall seepage rate can still be significant. Other disadvantages are that soil liners are erodible and subject to desiccation cracking, frost heave, and osmotic consolidation (Brown 2002).

Geomembrane liners are formed of thin plastic membranes. Typical plastics used for these liners include high-density polyethylene (HDPE), linear low-density polyethylene, and polyvinyl chloride. The liners are placed in sheets, and the seams are welded or fused in the field to create a continuous membrane layer. Although the plastic materials are effectively impermeable, the liners invariably have a finite permeability due to leakage through pin holes, seaming defects, and water vapor permeation. Typical leakage rates for well-installed geomembrane liners are approximately 0.3 to 3 m^3/d/ha (30 to 300 gal/d/acre) of liner (Brown 2002).

A composite liner comprises a geomembrane liner placed in direct contact with an underlying soil liner. This combines the best of both liner types (a small unit leakage rate over a small area) and results in leakage rates substantially lower than either liner used individually. A main factor controlling the leakage rate through a liner is the hydraulic head on the liner. For a single liner without an underdrain between the tailings and liner, the head can be significant. The head on the overall liner system can be controlled by the installation of an inner liner separated from the outer or lower liner by a drainage layer. The drainage layer is designed to have sufficient capacity to remove leakage from the inner liner without significant buildup of hydraulic head on the outer liner. The drainage layer between liners may be used as a leak detection system to measure leakage through the upper liner (Brown 2002). In this application, the drainage layer is often termed a *leakage collection and recovery system*.

Embankment Types

Four basic configurations of embankments are used in tailings facilities: downstream, upstream, centerline, and modified centerline.

Downstream Embankment

The downstream embankment is constructed in stages so the centerline of the embankment crest moves downstream with each stage. Because this embankment type uses the largest amount of construction material, it is often the most expensive option. Its performance is completely independent of the physical properties of the tailings deposit, and for this reason in some jurisdictions (such as Chile) this is the only option available for a tailings embankment.

Upstream Embankment

The upstream embankment is constructed in stages with each stage constructed as a berm on the tailings beach immediately upstream of the previous stage. The centerline of the embankment crest therefore moves upstream with each stage. Because this embankment type uses the least amount of construction material, it is often the lowest-cost option. Upstream dams rely entirely on the strength of the tailings deposited upstream of the berm for support. A large proportion of tailings facilities constructed using the upstream method have utilized a spigoting method of deposition to produce a coarse fraction on the upper beach for strength. The spigoting method consists of numerous, closely spaced, and independently controlled offtakes that allow the tailings slurry to be deposited in a low-velocity manner over the beach for maximum segregation so that a free-draining, coarse sand deposit forms in the structural zone. Major failures of embankments constructed by the upstream method for the most part have been due to a lack of drainage in the structural zones and have led to the ban of upstream construction methods in some jurisdictions. Interestingly, no upstream dam has failed that has been rigorously designed using modern engineering principles to ensure that the embankment is adequately drained and the phreatic surface is controlled.

Centerline Embankment

The centerline embankment is constructed in stages so the location of the centerline of the embankment crest does not change with each stage. The upstream toe of each embankment stage is constructed slightly over the tailings beach, but the majority of each new stage is founded on the previous embankment stage. This method relies on some strength and structural support of the tailings for the upstream slope but does not rely on the tailings strength for overall stability. Liquefaction of the tailings as a result of earthquake loading could result in some localized instability of the upstream slope of the most recent stage, but this would not result in significant damage to the structure. The centerline method is a compromise between the higher-cost downstream embankment and the higher-risk upstream embankment.

Modified Centerline Embankment

A variation of the centerline embankment is the modified centerline embankment. In this method, the embankment crest centerline moves slightly upstream, thus reducing the quantity of construction materials required in the downstream shell zone of the embankment. Modern analytical techniques are used to determine how far the embankment crest centerline can be moved upstream with staged construction and still keep the embankment independent of the tailings' strength for overall stability. These techniques have resulted in significant cost savings without compromising embankment stability. Many modified centerline embankments have been designed, permitted, and constructed in Canada, South America, and the United States.

Liquid–Solid Separation in the Tailings Facility

A key factor in the selection and design of many tailings facilities over the last 20 to 30 years is liquid–solid separation, which has the benefits of

- Increasing the density of the deposit, thus increasing the storage efficiency and reducing the potential for liquefaction under static conditions and dynamic loading;
- Increasing the stability of the deposit because the shear strength of deposit tailings is derived from particle-to-particle contact of the solids (the liquid has no shear strength);
- Reducing the amount of long-term drainage from the deposit, which requires collection, monitoring, and release, possibly continuing well after the facility's closure (possibly requiring water treatment);
- Reducing the amount of long-term consolidation of the deposit, thus reducing the amount of settlement that the final surface may undergo, which can undermine the integrity of the cover; and
- Recovering as much of the liquid as possible from the tailings so that it can be recycled to the mill for reuse in the process, thus reducing the amount of water required to be added from an external supply. This is often the main factor in dry climates where outside water sources may be scarce.

In some cases, enhanced liquid–solid separation can also result in a slightly steeper slope angle on the deposit, which may improve storage efficiency, but caution should be exercised because the actual slope angle is affected by a number of factors and may be less steep than expected; and a steeper slope may be more susceptible to liquefaction.

Enhanced liquid–solid separation can be achieved in a few ways. One is a deposition method that discharges the slurry at a low velocity over the deposit, allowing the solid particles to drop out of suspension into the next layer while the liquid flows down the slope. In this approach, the tailings slurry is introduced at the top of the deposit through a number of adjacent but concurrently operated offtake points (often termed *spigots*) such that each point discharges a relatively small flow rate onto the deposit. By strategically relocating the points of active deposition, each newly deposited layer can be allowed to consolidate, and, in dry climates, to air dry, further enhancing liquid–solid separation while deposition is occurring elsewhere. This approach is sometimes referred to as the subaerial method. If a significant amount of clay is in the tailings, the addition of a flocculant to the slurry just before subaerial deposition may further enhance liquid–solid separation. Mechanical dewatering by thickeners (often in concert with flocculant addition) is another method. In recent years, significant advancements have been made in thickening technology and product development such that mechanically thickened tailings are becoming more common.

Maximizing liquid–solid separation may not always be the objective of a tailings facility design. For example, in wet environments where precipitation or run-on into the facility together with the separated liquid from the tailings produces an amount in excess of that needed for recycle, less emphasis may be placed on liquid–solid separation and more on securely storing the higher-moistures content tailings. In this case, the objective is to reduce the amount of excess water generated in the facility that requires releasing.

Tailings Continuum

The consistency of a tailings slurry has often been defined in terms of the weight of solids in the slurry as a percentage of the total weight of the slurry, which is termed the *slurry density* and expressed as percent solids. This convention is being amended, however, because the weight of solids is a function of the specific gravity of the parent material, and two slurries containing the same amount of water will have different solids contents if the specific gravities of the solids are different. The convention now being adopted is to use yield stress, a strength parameter, of the slurry as the indicator of its consistency. In rheological terms, *yield stress* can be defined as the shear stress that must be applied to the slurry in order to initiate its movement, which is applicable to the design of a transport system. In geotechnical terms, yield stress can be defined as the undrained shear strength of the slurry at rest, which is relevant to the slope of a beach on which a fresh layer of tailings will cease flowing.

In any tailings slurry, as water is removed progressively and the percent solids increases, the yield stress will also increase in a nonlinear relationship. Tailings in the higher-water-content region consist of conventional (low density), thickened (moderate density), and heavily thickened (high density) slurries, and have yielded stresses that are generally between 0 and 200 Pa. As the degree of thickening increases progressively across this region, the propensity for sedimentation and segregation of the solid particles from the slurry reduces. These tailings can typically be transported by centrifugal-type slurry pumps, although at the top end of the range, positive displacement pumps may be necessary.

Tailings with higher solids concentrations have a paste consistency, which is sometimes defined as nonsegregating. Typical yield stresses for paste are often up to several hundred Pascals, and these tailings typically require positive displacement–type pumps for transport. The liquid limit of the tailings is often in the paste region.

Tailings with consistencies greater than heavily thickened consist of filter cake material and are typically placed and compacted into the tailings facility by mechanical instead of hydraulic methods. Dewatering to a cake consistency requires filters, typically vacuum or press-type filters.

The broad spectrum of tailings consistencies, from low to high density, is termed the *continuum* because the boundaries are not well defined. Historically, tailings have been produced and disposed at the low-density end of the continuum. However, over the last 10 to 20 years there has been an increasing move toward thickened and paste disposal in concert with a significant advancement in the technology to dewater tailings. The key advantage of thickening

is increased water recovery for reuse, although reduced long-term consolidation of the deposit and/or higher storage densities may also occur. But on a case-by-case basis, these factors need to be weighed against the cost of thickening, transporting a higher-yield stress material, and managing additional recovered water. Thus the selection of any tailings methodology on the tailings continuum is the subject of trade-offs.

One challenge in the design of a tailings facility and delivery system is that the rheological properties of the tailings can change between the plant and disposal facility. Thus, it is important to begin the design at the point of deposition into the tailings facility and design the tailings facility using criteria consistent with the desired point on the continuum and then work upstream to the plant. The effects of shear thinning of the slurry or paste in the pipe and pumps must then be accounted for when designing the thickener.

SITE CHARACTERIZATION

Climate and Hydrology

The climate and hydrology of a region are influenced by patterns of temperature, wind, and atmospheric moisture, and patterns of surface runoff, respectively. The total amount and timing of runoff dictate many aspects of a tailings facility, ranging from environmental concerns (such as what level of impact the facility might have on water quality) to engineering concerns (such as how much storage is required for temporary storage of seasonal or storm runoff, or whether stormwater diversions or a spillway are needed, and, if so, how large they should be).

Typically, little or no site-specific climatic and hydrologic data are available for most development sites at the time of initial design. In most instances, a data collection program is initiated during project feasibility and environmental impact studies, but, to a large degree, these data are for short-term periods and of limited use at the time of project design. As a result, the climatic characterization of a project area is generally conducted on the basis of regional data, which may be extrapolated to the project site according to known or suspected weather patterns, similarities of watershed characteristics, and an understanding of the fundamentals of hydrometeorologic systems, including lapse rates, orographic effects, and rainfall/runoff mechanisms. A correlation of the regional data to the short-term site-specific data is often attempted to support the extrapolation.

The hydrometeorologic characterization of a site typically involves estimates of temperature, evaporation, precipitation (rain and snow), and runoff on both average monthly and annual bases, including some measure of variation. Also required are values of extreme precipitation and flow, usually on a daily basis and usually presented in terms of likelihood of occurrence. In addition, in colder climates, patterns of snow accumulation and melt are needed, often with depths of frost penetration and ice accumulation.

Geology

The regional geology typically defines bedrock types (lithology and alteration) and structure (faults and locations, unconformities, and fracture types and their frequencies, orientations, and strengths) in the site vicinity. Specific site investigations should then be carried out to investigate and accurately map these features on a more concentrated scale. Their location and characteristics can be important to the site characterization and type and configuration of tailings facility to be built. Regional geology maps can usually be obtained from various government agencies. Generally, smaller-scale maps in the order of 1:1,000,000 are easily obtained. The availability of larger-scale maps (i.e., more highly detailed maps), such as 1:250,000 or 1:100,000, is more variable because of the economical, political, and geological characteristics of the area. Many industrialized countries have government geology departments at the federal and provincial or state level, although many developing countries only have limited geological data at the federal level. In addition, the size and population of the country, state, or province may dictate the map scale. Another source useful for gathering regional information is the Internet. Many countries have maps in digital form for viewing and printing.

After regional maps have been obtained, it is recommended that a qualified person carry out a field reconnaissance to verify the mapping. Larger-scale mapping should be carried out if these maps do not exist or the geological or structural setting at the proposed site is complicated. It is important at this stage to identify weak or otherwise unfavorable bedrock units or active faults. These identifications will aid designers and form the framework for detailed site investigations.

Terrain Analysis

Terrain analysis assists in characterizing the origins of and landforms associated with the surficial or near-surficial materials (soils and rocks) in the vicinity of the tailings facility. This analysis is designed to identify and classify material types (potentially for construction) and to identify potential natural hazards that could impact the tailings facility. The terrain analysis will not necessarily determine accurate characteristics of the materials but is a useful tool for defining where detailed site investigations are required.

There are two levels of terrain analysis. The first is at a desktop level and comprises aerial photography. The second involves ground reconnaissance and mapping to verify or modify the observations made from the air photos. The objectives of these two levels are to define as many of the following attributes as possible: bedrock geology, quaternary geology, landforms and geomorphology, soil types, weathering, erosion and deposition modes, slope instability, climate effects, vegetation, hydrogeology, geotechnics, volcanic activity, seismicity, natural dams, human activity, and land use. This list is quite exhaustive, and many of these attributes will not be present at a given tailings facility site. However, it is prudent to initially check for all of them.

In areas of steep, wet terrain, it is recommended that 1:15,000-scale air photographs be used for the first level of reconnaissance. Steep terrain is generally characterized as having significant portions of the area with slopes greater than 50% or 27°. If drier, flatter terrain is present in the site vicinity, 1:40,000-scale air photographs are adequate. If reconnaissance flights and air photo production can be acquired, this exercise can be completed in conjunction with topographic surveys.

The second level of terrain analysis comprises ground verification of the air photo interpretation. The requirements for ground reconnaissance are based on the terrain attributes identified in the first stage. Generally, areas where the slopes are greater than 50% require foot traverses, while areas of flatter terrain can be completed by ground checks supported by helicopter or vehicle. Unstable or potentially unstable areas

identified during the air photo interpretation must be checked in detail by foot traverses and 1:5,000-scale mapping.

The results of the terrain assessment should be made into a terrain hazard map, outlining areas of potential natural hazards or geological risks, construction materials, and targeted dam sites.

Geotechnical Site Investigation

Geotechnical site investigations for a tailings facility need to be carefully planned with an initial design concept in mind and should be conducted to an increasing level of detail as the facility moves from scoping level to preliminary and on to detailed design. Further, site investigations should investigate the geologic interpretation of the site developed during the previous site-characterization exercises. The key objectives of site investigations for tailings facilities are to

- Confirm potential natural hazards identified during the terrain analysis phase;
- Characterize foundation materials through sampling and laboratory index tests, such as particle size distribution and plasticity;
- Characterize existing groundwater conditions through drilling investigations, including in-situ hydrogeologic properties of the foundation soils such as permeability;
- Determine geotechnical properties of the foundation soils, such as shear strength, compressibility, and permeability; and
- Confirm availability and characteristics of the earth-fill or rock-fill materials required to construct the facility according to the proposed design concept, including mine waste materials from open-pit development.

Typically, test pits, drill holes, and seismic refraction (in some cases, electrical conductivity or gravity) surveys are sufficient tools for site investigations for detailed design of a tailings facility. Test pits are used to investigate potential construction material borrow sources and the shallow foundation conditions for the tailings dam or basin. Test pits alone may be sufficient for scoping level designs. To excavate test pits, hydraulic excavators with minimum 5-m (16.5-ft) depth of excavation are typically used. Drilling investigations of the foundation are required for the feasibility and detailed design of a tailings dam.

A properly planned and executed site investigation includes careful selection of the drilling method(s) to ensure the necessary data are collected. It is strongly recommended that a qualified geotechnical engineer supervises all drilling activities to ensure sampling and testing are completed at appropriate locations and following appropriate methods, and that appropriate information is being collected.

Electronic cone penetrometer testing (CPT) is now commonly used to gain continuous profiles of in-situ soil properties in sands, silts, clays, and tailings deposits. Often these tests are carried out using a tip fitted with a water pressure transducer to record pore pressures (CPTu) and in some cases the CPTu can be equipped to carry out in-situ seismic tests in which the cone is equipped with a geophone to record the arrival of shear waves from a surface source. CPTu is an excellent method to extend the data and fill in gaps from more traditional drilling, sampling, and standard penetration testing (SPT). For gravelly materials, the Becker drill is a common method that provides samples and penetration data that can be correlated to the SPT.

Samples collected from the investigations are tested in a soils laboratory for index, strength, compressibility, and permeability characteristics. For the latter two types of tests, undisturbed samples of foundation materials are recommended. For materials from potential borrow areas, index tests will likely include moisture density or compaction testing to confirm their suitability for placement in the tailings dam, basin liners, or drainage systems. Strength and compressibility tests are also required for fill materials and should be performed on remolded samples compacted to the density criteria that will be required by the construction specifications. Permeability tests are required for materials to be used in high- or low-flow applications such as filter drains or core and liner zones, respectively.

Seismicity

In regions of high or even moderate seismicity, often the seismic loading controls the stability and thus design of a tailings dam. Consequently, a seismicity review of the region where the project is located should be carried out during the initial stages of project planning. Initial studies at the conceptual or prefeasibility level may be limited to a review of existing information or maps regarding the regional seismicity, and preliminary seismic design parameters may be obtained from seismic hazard maps for the region, if available. In some areas of the world these are available through the U.S. Geological Survey and/or the Global Seismic Hazard Assessment Program, both of which have Web sites. However, for feasibility and final design stages of a tailings facility, more sophisticated methods of analysis are typically required, including both deterministic and probabilistic methods of seismic risk analysis.

A probabilistic analysis defines a unique probability of occurrence for each possible level of ground acceleration experienced at a site, using methodology based on Cornell's probabilistic analysis (Cornell 1968). The likelihood of earthquake occurrence within defined seismic source zones is determined by examining seismicity data. Using historical earthquake records for the region, magnitude–frequency recurrence relationships are established for each potential earthquake source or fault zone.

Unlike the probabilistic analysis, the deterministic method does not account for the likelihood of a predicted ground acceleration occurrence. Seismic source zones or fault systems in the region are defined, and maximum earthquake magnitudes are assigned to each source based on the characteristics of the fault, including type and length. Expected ground motions, including acceleration, are then transmitted to the site using attenuation relationships. The maximum acceleration produced by this procedure is referred to as the maximum credible acceleration and the corresponding earthquake as the maximum credible earthquake (MCE). The MCE is defined as "the largest reasonably conceivable earthquake that appears possible along a recognized fault or within a geographically defined tectonic province under the presently known or presumed tectonic framework" (ICOLD 1989).

The seismic design parameters selected for the design of a tailings facility are dependent on the seismicity level in the region and the geologic and tectonic conditions at and in the vicinity of the site. They are also dependent on the hazard consequence classification of the dam or facility. For high or very high consequence facilities, the MCE may be adopted as the maximum design earthquake (MDE), which is often defined as the earthquake that a facility must be able to withstand

without catastrophic failure. For lower-consequence facilities, a lesser earthquake from the probabilistic analysis may be selected for the MDE. Another design earthquake is the operational basis earthquake, which the facility must be able to withstand without impeding its operation and is usually selected from the probabilistic analysis.

ALTERNATIVES ANALYSIS AND SELECTION

The selection of a site, configuration, and in some cases the design basis for a tailings facility requires a systematic and defensible method to properly account for the pertinent factors that apply. These often span a wide range of subjects and in some cases may conflict with each other. Usually a few steps are involved. The first is the identification of potential facility sites and configurations on topographical maps. Preconcept level sizing is based on the anticipated tonnage of tailings to be stored and an estimate of the in-storage density. The second step is often a fatal-flaw assessment of the identified options to eliminate those that for one reason or another could not be developed. Fatal flaws may include

- A site on land that cannot be acquired or developed (possibly because of land status or ownership, restricted areas, an ecosystem or water resource that cannot be affected, a community on or adjacent to the site that cannot be relocated, or utilities that cannot be moved); and
- A site at risk of being affected by a natural hazard, such as a major active fault, unstable ground, or a major avalanche or flow debris chute.

The third step in an alternatives analysis is to compare and rank the options remaining after the fatal flaw assessment by scoring them against appropriate comparative criteria in a structured matrix. Depending on the level of detail required, the matrix may be relatively simple or complex. In the early stage of planning, a simple approach may be warranted if the list of comparative criteria is short (due to limited information on the project) or if the initial number of options is large. A more detailed comparison of a few selected options can then follow.

A more detailed comparison uses a tiered multiple-accounts matrix method, often utilizing a three-tiered structure for grouping similar or related criteria together (Robertson 2004; Kerr et al. 2003). The first tier consists of the major account headings, such as technical, economic, environmental, or social. The second tier consists of subaccount headings, which are grouped under each of the account headings; for example, "capital cost of the initial stage of the tailings facility," which would be positioned under the economics account. The third tier drills down further under the subaccounts level and contains the specific comparative criteria that will be used to score and rank the options; for example, "capital cost of the initial stage dam" under the "capital cost of the initial stage of the tailings facility" subaccount. Numerical weights are applied at the three levels of the matrix to reflect the relative importance of one criterion, subaccount, or account to another criterion, subaccount, or account at the same level but not across levels. Individual scores are then assigned to each option at the criteria level, and the results are calculated as weighted output scores for each option at the subaccounts, accounts, and overall levels.

At the overall (or upper tier) level, a comparison of the weighted output scores gives the overall ranking of the options. At the accounts and subaccounts levels, the comparison gives the ranking of facilities for each account or subaccount. Understanding which options are considered better or worse at these levels may be just as important as at the overall level. The multiple accounts matrix approach also allows for the calculation of *discriminatory values* at the subaccounts and accounts levels, which define the accounts or subaccounts that had a greater or lesser influence on the weighted scores at these levels.

One of the greatest advantages of the multiple-accounts approach is that by applying the weights in the manner described, experts in different fields can provide input to only their appropriate accounts or subaccounts, with others providing input elsewhere while not biasing the overall results.

In many regulatory environments, it is now often mandatory to demonstrate that a program of options identification, analysis, and appropriate ranking was adopted to support the selection of the site and design basis for a tailings facility. The multiple-accounts matrix approach, if used properly, can serve as a thorough, transparent, and defensible tool for this.

Table 8.10-1 Terms and definitions associated with the solid, liquid, and air phases

Solid	Liquid	Air
V_s = volume of solid particles	V_w = volume of water	V_a = volume of air
W_s = weight (mass) of solid particles	W_w = weight (mass) of water	W_a = weight (mass) of air (negligible, taken as zero)

CHARACTERISTICS OF TAILINGS

Tailings are often composed of solid, liquid (usually water), and gas (usually air) phases in various relative volumes or masses. The liquid and air phases occupy *voids* between the solid particles. If air is absent, the voids consist only of liquid, and the tailings are said to be *saturated*. Some useful terms and definitions related to these phases are shown in Table 8.10-1.

The total weight or mass of tailings is composed of the W_s and W_w components, while the total volume of tailings is made up of V_s, V_w, and V_a components.

The water or moisture content of tailings is defined as the ratio of the weight or mass of water to the weight or mass of solids and is usually expressed as a percentage. It can be defined as

$$\text{moisture content (mc)\%} = (W_w/W_s) \times 100\%$$

The bulk density of tailings at a point in a deposit is defined as the total weight or mass of the tailings in a unit volume and can be expressed as

$$\text{bulk density } (\rho_b) = (W_s + W_w)/(V_s + V_w + V_a)$$

The dry density of tailings at a point in a deposit is defined as the weight or mass of solids in the tailings in a unit volume and can be expressed as

$$\text{dry density } (\rho_d) = W_s\,(V_s + V_w + V_a)$$

Dry density is a commonly used parameter in the design and monitoring of a tailings deposit because the quantity of tailings produced from the mill is usually defined in dry tons (or metric tons) and is derived from the dry weight of the ore feed after the mineral of interest is removed.

Bulk and dry density can be related to each other by the moisture content using the relationship

$$\rho_b = \rho_d * (1 + (mc/100))$$

The void ratio of tailings is defined as the ratio of the voids volume in the tailings to the solids volume and can be expressed as

$$\text{void ratio (e)} = (V_w + V_a)/V_s$$

The degree of tailings saturation is defined as the ratio of the water volume in the tailings to the voids volume and is usually expressed as a percentage. It can be expressed as

$$\text{degree of saturation } (S) = [V_w/(V_w + V_a)] \times 100\%$$

wherein saturated tailings ($S = 100\%$) have no air in the voids and thus $V_a = 0$.

The specific gravity of tailings solid particles is defined as the ratio of the weight of solid particles in the tailings to the volume of the solid particles and can be expressed as

$$G_s = W_s/V_s$$

where G_s is specific gravity.

The terms previously described can be interrelated using the following equations:

$$(e) = (G_s * \rho_w/\rho_d) - 1$$
$$\rho_d = (G_s * \rho_w)/(1 + e)$$
$$\rho_b = (G_s * \rho_w) * (1 + mc)/(1 + e)$$
$$S = mc * G_s/e$$

where ρ_w is the density of water (1 t/m^3).

Plasticity and Related Issues

The tailings continuum described previously was based on rheology. From a geotechnical point of view, tailings are generally considered to exist in four states of consistency (from high to low moisture content): viscous fluid, plastic, semisolid, and solid. If the tailings contain clay minerals, the plastic range is significant. A property associated with the clay minerals, *plasticity* can be defined as the ability of the clay to deform without cracking. Plasticity results from the electrochemical behavior of the clay particles and the attractive bonds that remain between them even after relatively large deformations. It is often quantified by the plasticity index (PI), which is the difference between the boundaries of the following consistency states:

- Liquid limit (LL) is the moisture content of the clay at the high end of the plastic range, at the boundary between the plastic and viscous liquid states. Remolded clayey materials at this boundary have a low, undrained shear strength, often in the 1-to-2-kPa range.
- Plastic limit (PL) is the moisture content of the clay at the low end of the plastic range, below which the clay changes from a plastic consistency to a moist semisolid material or cake. Remolded clayey materials at this boundary often have an undrained shear strength in the order of 100 kPa (or possibly higher).

The PL and LL values are termed the *Atterberg limits* and are determined from tests in which samples of the material are remolded (deformed plastically) under defined procedures. Given the correlations with remolded strength, the Atterberg limits define the range of moisture contents over which the strength of the material increases by approximately 100 times. Most tailings of hard-rock origin have a low clay content and thus low plasticity. In terms of moisture content change, the difference between the higher and lower end of the plasticity range is low. In some cases the difference between the PL and LL is zero, and these tailings are termed *nonplastic*.

From a tailings engineering perspective, the important considerations related to PL, LL, and PI are

- Tailings with a low PI (e.g., of hard rock or low clay content) can gain significant strength with only a minor reduction in moisture content; and
- Tailings with a high PI require a significant amount of water to be removed, usually under load or evaporation (the process of consolidation), to gain significant strength.

The boundary between the semisolid and solid states is defined as the shrinkage limit, which is important because it defines the moisture content below which no further volume reduction or shrinkage will occur with additional drying or moisture loss. At this point the particles have reached such a dense packing that any further moisture loss results without a further reduction in void ratio. The result is a reduction in saturation as air enters the voids to replace the ongoing water loss.

Permeability and Drainage

Basic Principles for Flow

The amount of flow that can pass through saturated porous media such as tailings is dependent on

- Permeability coefficient of the tailings, which is defined as the amount of flow that can pass through a unit cross-sectional area of the tailings under a unit hydraulic gradient; and
- Hydraulic gradient, which is the amount of head loss that takes place along the flow path divided by the length of the flow path.

Darcy's equation defines this relationship:

$$Q = k * i * A$$

where

Q = volumetric amount of flow per unit of time (m^3/s)
k = coefficient of permeability of the tailings (m/s)
i = hydraulic gradient, which is equal to the change in head or the head loss divided by the flow path length, or $\Delta h/\Delta l$
A = cross-sectional area of flow (m^2)

The permeability coefficient of tailings is influenced by the particle sizes and their distribution, density (or void ratio), and degree of saturation of the material.

Shear Strength

Loading Cases

The proper analysis of stability of a tailings facility structure requires use of the most suitable loading case(s), shear strength parameters, and drainage conditions. The loading cases may include

- Long-term, steady-state conditions;
- End of construction conditions;

- Rapid loading (or unloading);
- Staged construction conditions;
- Earthquake loading;
- Postearthquake conditions;
- Rapid drawdown; or
- Hybrid cases.

Not all of these cases are addressed in this text, and the reader is directed elsewhere for further reading.

Total and Effective Stress Shear Strength Parameters

Total stress is defined as the total amount of force applied to an area of material divided by the area, whereas effective stress is the portion of the total stress that is taken up in particle-to-particle contacts. The difference between the two is the pore water pressure. Thus, total stress is defined without regard to the portion of the stress that is carried by the soil particles and the portion carried by the water. It is the same for the undrained and drained conditions.

The shear strength of soils or tailings is controlled by effective stress, whether failure occurs under drained or undrained conditions. The relationship between shear strength, or fully mobilized stress (S), and effective stress (σ') can be represented by a Mohr–Coulomb strength envelope. The relationship can be expressed as

$$S = c' + \sigma' \tan \theta'$$

where

c' = effective cohesion
σ' = effective stress
θ' = effective stress angle of internal friction, typically referred to as the effective stress friction angle

Strength envelopes for soils or tailings are established by performing shearing tests on samples consolidated to a range of effective confining stresses, and if the tests are triaxial, the results can be plotted on a Mohr stress diagram. Effective stress and total stress envelopes can be established.

Seismic and Liquefaction Considerations

Tailings liquefaction is a major challenge in the design of tailings facilities and can be triggered by both cyclic and static shear stresses. Liquefaction occurs when a soil loses a significant portion of its strength or stiffness for a relatively short time but long enough for slope-stability or bearing capacity failures to occur. Because it is unpredictable and occurs under various conditions, liquefaction has been viewed as a difficult problem. There are many recorded cases of liquefaction failure of mine waste and quite likely many more undocumented cases (Been and Li 2009).

Liquefaction, including occurrences due to earthquake loading or from static causes, requires close assessment, typically by specialist practitioners. Various methodologies assess the liquefaction potential of mine tailings. In-situ methods are often employed for such an assessment, using applications such as the SPT, CPTu, seismic CPTu, field vane shear test, Becker hammer test, or self-boring pressure meter. Various laboratory approaches also assess the liquefaction potential of soils, such as cyclic simple shear and cyclic triaxial testing.

Typically, a tailings facility designer will assess the likelihood of liquefaction due to earthquake shaking and static loading. For earthquake-induced liquefaction, laboratory or in-situ testing results are typically required, with in-situ testing often the preferred method. An example approach for assessing earthquake-induced liquefaction is the use of the CPT piezocone, a tapered electronic probe that is advanced into a soil deposit typically using a hydraulic ram, such as a geotechnical drill rig. The piezocone is equipped with load cells and transducers to measure the device's penetration resistance at its tip, friction along its side shaft, and pore pressures that develop during penetration. This pore pressure has been termed *dynamic pore pressure* by some practitioners, but the authors suggest it should be termed *quasi-static pore pressure*, because the piezocone is a quasi-static test. By pausing the penetration, the quasi-static pore pressure is allowed to dissipate to reestablish the equilibrium pore pressure, which represents the in-situ pore pressure at that point in the deposit.

By conducting several of these dissipation tests, it is possible to develop a pore pressure profile throughout the deposit's depth. This profile is useful in understanding the effective stresses in the vicinity of the probe and helps assist to establish whether the material is contractive or dilative. Data from the piezocone tests are combined with parameters pertaining to the design earthquake event (i.e., earthquake magnitude and ground acceleration) to complete the analysis. Procedures for completing an earthquake-induced liquefaction analysis use piezocone, seismic CPTu, SPT, and Becker hammer (Harder and Boulanger 1997). The art and science of liquefaction assessment is continually developing, and the National Center for Earthquake Engineering Research (NCEER) report has been updated several times (Robertson and Wride 1998; Robertson 2004; Arango 1996).

Once the likelihood of the occurrence of earthquake-induced liquefaction is established, it is necessary to develop strength properties for the soil zones that are predicted to liquefy (Yoshimine et al. 1999; Olsen and Stark 2002). The Yoshimine method produces a more conservative analysis (Robertson 2004).

Geochemistry

Characterization of both the solid and solution portion of the mill tailings is an essential step in the development of the tailings management plan. The acid-generating potential and metal-leaching characteristics of the solid tailings mass and the chemistry of the liquid effluent will affect the design of the tailings containment facility and be important considerations in solution management and reclamation planning. Acid rock drainage (ARD), metal leaching, and contaminant/metal release from mine tailings facilities are recognized environmental concerns. To ensure that natural aquatic systems are not significantly degraded or that human or other receptors are not impacted, it is important to fully understand the tailings material by conducting a thorough waste characterization program.

A phased approach to waste characterization is often a prudent way to proceed. The relatively inexpensive static tests can be performed on a large number of tailings solids samples, using the results of the initial testing to plan for additional, more detailed testing requirements. This stage of the characterization program will include acid–base accounting (ABA) or equivalent to determine the relative balance of potentially acid-generating and potentially acid-consuming minerals; leach extraction testing to measure the soluble components of the samples; and other testing to determine trace element content of the waste. Whole rock analysis and mineralogical descriptions may also be conducted. Trace element

testing will usually consist of a full suite of metal analyses (inductively coupled plasma–mass spectrometry [ICP-MS]/inductively coupled plasma emission spectrometer). Trace element concentrations will indicate which constituents are naturally high in the waste and may be a concern for future leaching. A comprehensive list of testing methods can be found in the Canadian Mine Environment Neutral Drainage documentation (MEND 1991).

ABA, or the determination of the relative amounts of acid-generating and acid-neutralizing minerals in a sample, can be accomplished through a number of test procedures, with the Sobek acid–base account test and the modified acid–base account test being the most common. These tests measure the acid potential (AP), also called the maximum potential acidity, of the sample based on its sulfur or sulfide content. The sample's neutralizing potential (NP) is determined by titrating a pulverized sample of the material with an acid, with the resulting NP representing the acid-neutralizing capacity of the sample. The sample's net neutralizing potential (NNP = NP – AP), ratio of NP to AP, and paste pH are also determined. The results of the ABA testing are compared to general guidelines to assess whether the samples are likely to generate acid. Guidelines may vary for different areas but are generally based on the NNP and the ratio between NP and AP (NP/AP).

Static testing provides valuable insight into the acid-generating and metal-leaching potential of the waste. It does not, however, provide information about the rate of acid generation or neutralization and should not be used to predict drainage water quality in the field. In addition to static testing, simple leaching tests such as synthetic precipitation leaching procedure, meteoric water mobility procedure, or net acid-generating leachate analyses will provide preliminary data on potential contaminants of concern.

The next step in the characterization process for the tailings solids is kinetic testing. If static testing results show high variability between samples, a relatively high number of samples should be considered for kinetic testing. If the static testing results are largely consistent, fewer samples are needed for the kinetic testing phase, because similar results indicate less variability in the samples. Kinetic testing is used to confirm the samples' acid-generating or acid-neutralizing characteristics while determining the reaction rates for acid generation and neutralization. Kinetic tests are most often conducted in a laboratory, where chemical weathering is simulated over time in cells or columns. The composition of the leachate collected from kinetic testing can be used to predict drainage water quality in the field if concentrations are corrected for the effects of climate and particle size. The leachate should be analyzed for total and dissolved metals (ICP-MS), conductivity, total dissolved solids (TDS), a full set of anions and cations, and pH. Kinetic testing procedures include humidity cell, humidity column, column leach, Soxhlet extraction, and field plot tests.

It is equally important to characterize the tailings solution or the aqueous portion of the tailings slurry. A sample of the tailings solution from the pilot test work should be analyzed for a range of parameters, including total and dissolved metals (ICP-MS), nutrients, and reagents and reagent by-products used in the process, TDS, and turbidity. For example, if cyanide is used in the process, the effluent should be tested for the full range of cyanide species (total cyanide, free cyanide, weak acid dissociable cyanide, strong acid dissociable cyanide, cyanate, and thiocyanate). If warranted, speciation of metals of concern should be determined because the species of metal present significantly affects the availability and toxicity of the metal. Effluent concentrations are compared to in-country effluent guidelines or the World Bank effluent guidelines. A toxicity test of the effluent is also often required to ensure that the effluent is not acutely toxic to aquatic life. If discharge to nearby waterways is considered, effluent concentrations, coupled with the site hydrology, can also be used to perform water quality modeling exercises. Predicted water quality concentrations are then compared to in-country/international water quality guidelines or criteria for aquatic life protection (Brown 2002).

WATER MANAGEMENT

A key part of the tailings facility operations is the management of water in the facility. Water enters the tailings facility as process water in the tailings slurry, direct precipitation, and runoff from surrounding undiverted catchments. It is lost in the tailings deposit as pore water in the voids and to evaporation and seepage. In almost all cases, water is recycled to the mill for reuse in the process. Generally, if the total quantity of the water lost to permanent storage in the voids, evaporation, and seepage is greater than the quantity of water entering the facility from precipitation and runoff, the facility is considered to be in deficit, and makeup water is required to sustain the milling operation. If the opposite applies, the facility is in surplus, and excess water may need to be treated and released or adequate storage provided to contain it.

Precipitation and evaporation usually vary seasonally, which results in water surpluses some months and required makeup water in other months. Water can be stored in the surface water pond in the tailings facility during the wetter months and used as makeup water in the drier months. A detailed water balance that accounts for the water inflows and losses on a monthly basis is required for a tailings facility. It should consider climatic variations to ensure that makeup capacity for dry periods and treatment or storage capacity for wet periods are sufficient. The water balance is used to predict the variation in size of the surface water pond in the tailings facility pond on a seasonal basis as the facility is filled.

A further aspect of water management is to ensure that the tailings facility has adequate capacity at all times to store, route, or otherwise handle runoff from extreme precipitation events. Overtopping as a result of storm runoff is one of the most common causes of tailings facility failure. It is imperative that adequate storage is provided to safely contain the surface water pond within the tailings facility at all times. Particular focus should be on the wet season, with the addition of the design storm plus required freeboard for wave run-up. If sufficient temporary storage cannot be provided, it is often prudent to include an emergency spillway, although in a stage-developed facility, constructing a spillway for each stage may be complex.

Water Balance

The general hydrologic equation describing the water balance of a typical tailings management facility (concentrator, tailings, supernatant pond, other storage ponds, piping, and pump back) is given by

$$P + Q - E \pm \Delta S - D = 0$$

where

P = precipitation on the tailings facility and contributing catchment area

Q = runoff inflows to the tailings facility, including runoff and water liberated from the tailings slurry minus outflows from the tailings surface water pond, consisting of reclaim water to the plant and excess water discharges to treatment (if necessary)
E = evapotranspiration (evaporation plus transpiration)
ΔS = change in storage in the tailings facility and associated ponds
D = deep percolation (unrecoverable by drains and/or plants at closure)

Water balances are used to evaluate the size and configuration of the components for either water storage or flow discharge as well as to quantify the design water supply and evaluate environmental issues (Fenn et al. 1975; van Zyl et al. 1988; Tindall and Kunkel 1999; Kunkel and Lishnevsky 2009). Water balance calculations can be made using simple spreadsheets or can be quite complex in nature and must consider plant operations, potential power outages, and extreme storm events to ensure the development of a complete assessment.

Water Storage

The average and maximum monthly design sizes of the tailings surface water pond throughout the mine life can be determined by various calculations. The authors prefer the following two methods: (1) average monthly pond volume plus a design storm event, and (2) monthly percentage chance of exceedance analysis. For providing adequate pond storage capacity, the larger result of the two should be considered. However, the largest monthly pond size from either method may vary from year to year throughout the operating life of the mine, and this should be accounted for in the staged expansion plans for the tailings storage facility.

In the first method, the averages of the monthly tailings pond volumes are calculated for each of the 12 months of the year and combined with the volume contributed by the selected design storm event. The design storm volume may vary throughout the months of the year, so the maximum of these two components is sought.

In the second method, a frequency analysis is performed on the various January-to-December pond volumes, often to define the 0.001% (representing a 100,000-year storm event), 0.01% (a 10,000-year storm event), 10% (a 10-year storm event), and 50% (a 2-year storm event) chance of exceeding pond volumes.

Water Removal Systems

The control of the supernatant pond is probably one of the most important procedures in managing a tailings system. Inadequate pond control can result in overtopping, increase in pore pressures, reduction of freeboard, high seepage rates, and embankment settlement (Engels and Dixon-Hardy 2009). These conditions can lead to instability and a high risk of problematic situations. For a conventional impoundment (particularly upstream and centerline embankments), it is essential that the ponded water be kept to a minimum volume, well back from the embankment crest, and the freeboard be sufficiently high all along the tailings embankment. Suitable monitoring and management of the supernatant pond are required to safely manage the tailings.

Decant systems for tailings management should be designed for daily management of the supernatant pond as well as storm event surges. The design of the decant system should allow for a high surge capacity of stormwater to compensate for near future storm events. If the pond cannot drain fast enough (typically within weeks), then the freeboard of the embankment may be lost if a near future storm occurs.

The three most common methods of water control within a tailings management facility are (1) drains associated with subaerial deposition, (2) free-standing or sidehill riser decant towers, and (3) decant barges. Each of these water removal systems is briefly discussed in the following subsections.

Drains (Subaerial Deposition)

In subaerial tailings deposition, the slurry typically discharges out of drop bar just ahead of a rising tailings beach. This leads to some dissipation of energy as the tailings slurry exits the drop bar onto the beach. The tailings then flow at a relatively low velocity over the beach, which allows for liquid–solid separation. The points of active discharge are frequently alternated around the sides of the tailings management facility to form a thin-layered, drained, and stable tailings deposit against the embankment.

For this type of facility, the beach is sloped into the central area of the facility so that surface water draining from the tailings and runoff from precipitation are displaced away from the embankment. A small surface water pond is maintained in the central area directly against a blanket underdrain that covers the lined base of the facility, which slopes down to a low point in the facility. The underdrain comprises a layer of free-draining gravel within which a network of perforated, corrugated polyethylene tubing pipes is installed, and this layer is sometimes covered with a geotextile to filter the tailings solids. The geotextile is covered with another layer of gravel and cobbles for erosion and protection of the geotextile from ultraviolet radiation exposure.

A reclaim pond (or return water pond) stores the water being decanted from the facility. The pond is situated outside the confining walls of the tailings storage area a short distance from the embankment toe. The water in the reclaim pond is either sent to treatment/polishing ponds for discharge to the environment or pumped back to the plant for use in the processing operation. If the climate is suitable, some reclaim water can be sent to evaporation ponds or sprays.

Decant Tower Systems

A decant tower is an intake structure consisting of a hollow tower (riser) that is connected to a horizontal conduit or pipe that normally travels beneath the impoundment and through/under the tailings facility embankment. The riser is extended as the level of tailings in the impoundment rises. The decant tower skims off the clear water from the surface of the supernatant pond and carries it away by gravity through the underlying conduit (European Commission 2004).

Decant towers can be effective at removing ponded water from a tailings facility but can be difficult to operate. As more tailings are disposed of in an impoundment, decant tower systems come under increasing stress. The ever increasing weight of the tailings can crack and damage a decant conduit that flows underneath and through an impoundment facility (Engels and Dixon-Hardy 2009). Sidehill decant systems utilizing stop logs are less likely to have operational problems because the decant tower is always accessible from shore, and the placement of stop logs is relatively easy.

One major disadvantage is that the water has to be continuously positioned around the decant tower; unlike a decant

barge, a tower cannot be relocated. If a decant tower becomes inoperative, emergency pumping or spillways need to be implemented. Any tailings facility operating plan should have precise contingency plans documented in case a decant tower becomes inoperative either by isolation, blockage, or failure.

Decant Barges

A decant barge consists of a floating platform that houses pumps used to reclaim water from the supernatant pond to the processing plant or reclaim pond. Unlike decant towers that are gravity fed, a decant barge requires electrical power to operate the pumps that decant the water from the supernatant pond. This increases operating costs because a constant and reliable power source is required to ensure the pumps operate, as a power failure results in no water being decanted. It is good practice to have standby pumps and diesel generators to use in an emergency or when a decant barge cannot cope with rapid inflows during storm conditions or when process water is required in large quantities. Before power or equipment failures occur, there should be emergency procedures and response plans that can be implemented in both normal and storm conditions in order to rapidly mitigate any decant problems (Engels and Dixon-Hardy 2009).

The capacity of the decant barge should be adequate to remove day-to-day decant demands as well as stormwater accumulation. The barge should also be situated in an easily accessible location for maintenance and inspection purposes, ideally against the side of a valley wall (for a valley impoundment) or against the side of a jetty wall in an impoundment area where ponded water is the deepest. The water depth below the barge can influence the clarity of the decant water being extracted by the barge.

As the supernatant pond location changes and/or the tailings volume increases, a decant barge or submersible pump can be moved. For valley impoundments or in-pit disposal, the decant barge or pump is generally retracted to keep the equipment close to the valley or pit walls. This makes it easier to access and prevents the use of heavy anchoring to control varied movement, which can be expected the farther away the equipment is from surface anchoring points. Each time a barge is moved to other cells, the plant water demand should be recalculated to ensure the barge is capable of meeting the water demand in the new cell (Engels and Dixon-Hardy 2009).

TAILINGS DELIVERY AND DEPOSITION

Tailings are transported from the mill to the tailings management facility using any of the following methods:

- Pump and pipeline systems
- Gravity flow in open channels and pipelines (widely used in Chile and Peru for large-tonnage systems)
- Belt conveyor and trucks (limited to low-tonnage filtered tailings operations)

The following points should be considered for the design of a pump and pipeline tailings delivery system:

- Ideally the tailings facility should be located in relatively close proximity to the mill to minimize capital and operating costs.
- The elevation difference between the mill and tailings facility has an important bearing on the operating complexity and cost of the delivery system. If the tailings are transported uphill, the additional elevation change increases the pumping requirements and may lead to greater capital and operating costs. For tailings facilities located downslope of the mill, the pumping requirements can be considerably reduced. However, if the elevation difference exceeds the pipeline friction losses, choke stations or drop boxes are required to dissipate the excess energy, which can complicate the system operation.
- For thickened tailings systems where the tailings facility is some distance from the mill and/or at a higher elevation, locating the thickeners adjacent to the tailings facility rather than at the mill may have cost advantages.
- Centrifugal pumps are suitable for most tailings applications. High concentration non-Newtonian tailings applications may require the use of positive displacement pumps. Typically the capital costs of the systems will be greater than comparable centrifugal pump systems, although the operating costs may be reduced because of the increased pump efficiency and lower maintenance costs.
- Tailings delivery pipelines are generally constructed aboveground using HDPE (low-pressure applications) or unlined carbon steel piping. For especially corrosive or abrasive tailings, lining steel pipe with HDPE, rubber, or polyurethane has cost benefits. In certain applications, cementitious type pipes are sometimes used.
- Provision must be made along the tailings pipeline corridor for the containment of possible spillages associated with blockage clearance or pipeline failure; this is especially important near water features such as lakes and rivers. Some applications may require the use of double contained piping. To minimize the likelihood of pipeline failures, the pipeline condition must be regularly monitored.
- In cold climates provision must be made for draining the pipeline to prevent freezing during pipeline or pumping system shutdowns.

CONSTRUCTION CONSIDERATIONS

Detailed designs for construction of tailings management facilities comprise technical specifications for the work and detailed drawings issued for construction. These documents can be included in a contract for facility construction by a third-party contractor or can be used by the mine to carry out the construction using its own equipment and staff. The design of a tailings facility, like that of most large civil structures, is based on the available topographic and geologic mapping and information generated by the site investigation into soil and rock, groundwater, and surface water conditions. It is inevitable that conditions encountered in the field during construction will differ to some degree from those assumed or estimated in the design, and thus it is good practice that a representative of the design engineer be on-site to provide technical direction for construction. This will ensure that variations in site conditions are recognized and that the design is modified as required to meet the design intent. This service can be and often is integrated with the quality assurance/quality control (QA/QC) role. QA/QC is required during the construction of a tailings facility so that conformance of the work with the technical specifications and construction drawings is measured and recorded. This continuity from the design through construction is also important from a liability perspective. A designer that has no involvement with the construction can take the position that any shortcoming is a result

of the construction or changed conditions that were evident only during construction, while the constructor conversely can take the position that the design had shortcomings.

Continuity of qualified outside consultants over the life of a facility can also be important as, by its nature, the mine may have a high staff turnover. Often the design engineer provides the only continuous presence on a tailings facility over the mine life.

An important but sometimes overlooked aspect, tailings facility construction is the preparation of detailed as-built drawings and a record of construction for the completed facility. The preparation of these important items takes place at the end of the project when budgets may be nearly exhausted. They are invaluable however, as the basis for ongoing staged construction and if any problems arise with the facility's performance.

MONITORING

Operation, Maintenance, and Surveillance

Operation, maintenance, and surveillance (OMS) are critical aspects of any successful tailings management facility. The procedures should be documented in an OMS manual, and regular checks or audits made to confirm they are being followed. The procedures should be modified and the OMS manual revised as necessary.

An OMS manual should be in place upon commissioning and maintained through closure. It should provide a clear, documented framework for actions to be taken for a wide variety of occurrences. The OMS manual should also provide a reliable basis for measuring the facility's performance and for demonstrating due diligence of the operating team. The detail level of an OMS manual should reflect site-specific requirements. The need for revision may be triggered by changes in such matters as

- Hazard classification of the dam,
- Operational performance of the facility,
- Project personnel or the company organizational structure,
- Regulatory or social considerations,
- Life cycle and/or design philosophy (MAC 2002),
- Geology, and
- Climate.

An excellent guideline on tailings facilities OMS is available through MAC (MAC 2002). An OMS manual should provide the user with information in a clear, logical format that is easy to read and implement, and supporting documents should be clearly identified and referenced. The user should be able to easily identify what is required and how to access the needed information. The manual should allow the facility's performance to be compared to expectations, design criteria, and operating intent, particularly in the event of significant incidents.

Confirmation That Design Objectives Are Being Met

Initial designs for tailings facilities are established on the basis of engineering estimates about the behavior of the embankments, the deposited tailings, seepage and groundwater conditions, and overall water management. It is essential that sufficient instrumentation is installed and that a comprehensive monitoring and surveillance program is implemented to measure actual performance against the design objectives. The monitoring program should include a formal inspection carried out at least annually by a senior engineer with full knowledge of the design and operational criteria for the facility. The results from the monitoring program should be reviewed on a regular basis, preferably by the design engineer, to assess whether the design requirements are being met. A regular review of monitoring data can provide early warning of developing problems and allow remedial action to be taken before problems develop into a major concern, or worse, a failure. Many regulatory jurisdictions require that monitoring reports be prepared on a regular basis, which fully describe the tailings facility operation over the reporting period and record the volumes and types of tailings that have been deposited, the formal inspections findings and resulting recommendations, and the data from instrumentation and their interpretations (Brown 2002).

Monitoring may consist of visual observation of the embankment and an assessment of data from piezometers and other instrumentation. Preventive maintenance, based on the timely observation of potential problems, can help maintain the stability of the structure, control seepage, and contain costs. Distress signals, such as cracking, sloughing, saturated areas on the downstream face, unusual plant growth, and settlement, indicate potential deficiencies in the structure, but without proper instrumentation it may be difficult to accurately interpret the extent of the problem. Piezometers, pressure gauges, survey monuments, and borehole slope inclinometers can be used to indicate developing trends in the facility's behavior. Observations made from these instruments, combined with daily operators' logs, which show dates and locations of deposition, meteorological conditions, and so forth, can help in the analysis of the facility's condition (Vick 1990).

Instrumentation

Visual observations alone are an insufficient means of monitoring the safety of a tailings facility. Instrumentation should be installed in the embankment and/or its foundation to monitor changes that may be critical to stability and to help predict unstable conditions. Instruments can be installed to measure pore water pressures, seepage flows, embankment movements, and earth pressures (EPA 1994).

Pore water pressures are measured with piezometers, which encompass several types used to monitor pore pressures, such as standpipes (isolated tip and fully slotted) or pneumatic, vibrating wire, or silicon strain gauge ceramic tip. They may be installed during the construction of a tailings facility or later in a borehole. Each type of piezometer has various advantages and disadvantages, a full discussion of which is beyond the scope of this chapter. Piezometers should be installed in the structural zone of embankments, in drains, and in the tailings deposit. A well-conceived installation of piezometers can play a crucial role in understanding the pore pressure distributions in a tailings facility, and care should be taken to install this equipment to provide information on critical sections of the facility for use in slope-stability analyses.

Measuring slope movements can also play a critical role in facility monitoring. Simple methods for measuring embankment deformations can be easily employed. Markers can be installed on the surface aligned in a straight line-of-sight to permit rapid detection of movement during periodic observations. Survey methods may be used to enhance monitoring of tailings facilities especially if visual observations indicate possible slope instability. The simplest example of a survey method consists of a line of survey points installed along the embankment, slope, or toe of the slope. These markers

are surveyed shortly following installation and periodically thereafter.

A more advanced device for assessing deformations within an embankment is a borehole inclinometer, which consists of a cased borehole in which the lining is equipped with two sets of orthogonal longitudinal guiding tracks. A slope indicator device is lowered into the borehole and records the verticality of the lining in two directions at right angles to each other and at constant depth increments. These directions are usually parallel to and at right angles to the slope. By integrating the recorded slopes of the borehole lining, the deformation throughout the casing can be assessed.

Slope cross sections and available freeboard for pond water levels may also be important aspects of the monitoring program, especially for upstream or cycloned facilities and for traditional paddock dams. Flows from drains should also be measured and samples tested periodically for water quality.

Safety Audits

The operator of a tailings facility should implement an annual audit and review of the facility to help assess whether the operations remain consistent with the design criteria. The annual audit should be undertaken by a suitably qualified and experienced person, preferably the original designer or an independent third-party individual.

Periodic operational audits provide a valuable status report of actual facility performance compared to the design parameters, expectations, and assumptions. The records provide an ongoing history of the facility and can be vital for sites with frequent personnel changes. An audit can assist with tailings management planning, dam construction scheduling, and improvements to the overall operation.

A comprehensive periodic audit and review report should include the following:

- Updated site plan
- Updated survey plan of the facility including cross sections and contours of the embankment and tailings beach
- Certification from a qualified and experienced person that any construction (since the previous report) meets appropriate engineering and safety standards
- Data about the engineering properties of construction materials and comparisons of those properties with the design criteria
- Updated data on the properties of the tailings stored in the facility
- Reconciliation of the stored volume and densities of the tailings with the design values
- Calculations of the deposition rate against capacity and of the remaining facility capacity in terms of time and volume
- Water balance data
- Climatic conditions over the period between reports
- Data from daily inspections and periodic monitoring during facility operation
- Review of the safety and stability monitoring results
- Review of the environmental monitoring results
- Information on the operation of diversion channels, drains, and freeboard
- Information on the location and depth of boreholes and their proposed monitoring program
- Inspection and maintenance schedules for tailings pipelines and other pertinent equipment
- Information on the operations in terms of risk assessment
- Planned operations for the next reconciliation period

UNIQUE TAILINGS MANAGEMENT SCHEMES

Owing to the wide variety of tailings types, sites, settings and environments, and operational requirements, each tailings facility is unique. This has prompted the development of some unique schemes to enhance the performance and/or reduce a specific facility's risk and cost. A few examples are given.

- **Disposal of tailings within other mining facilities.** At the La Quinua mine, which forms part of the Yanacocha complex in Peru, a new, thickened tailings management facility has been commissioned within an active heap leach pad where the heap leach ore pile is used to provide full containment of the tailings (Kerr et al. 2009).
- **Mud farming.** Fine tailings placed in large paddocks in a dry environment often form significant surface crusts from evaporative forces, which reduce subsequent evaporation and thus densification and stiffening of the underlying tailings. The surface may be *farmed* using a modified bulldozer equipped with long floats and Archimedes screws. This practice is used at the Wagerup alumina refinery in Western Australia.
- **Pit tailings disposal.** Tailings are placed in mined-out open pits for cost-effective storage and to limit the overall footprint of the mine. In some cases, to limit oxidation, operations place potentially acid-generating tailings in a reducing environment below the groundwater level. Care must be taken to determine that the tailings will not contaminate the groundwater, which is often accomplished by establishing and maintaining a flow gradient into the pit. This is practiced at various operations worldwide, including some of the uranium mines in northern Saskatchewan.
- **Algae introduction into the tailings stream.** This method promotes algae growth on the surface of the tailings deposit to suppress dust generation. It is being investigated at the Sierrita mine in Arizona (United States).

Research is also being carried out on a number of potential tailings improvement initiatives that have not yet made it to full-scale implementation. These include

- Use of foam to transport filtered tailings in a pipeline instead of conveyors or truck haulage, in an effort to reduce cost;
- Addition of fibers into a tailings stream that will become integrated into the deposited mass and provide added strength in the structural zone; and
- Mixed disposal of tailings and mine waste rock in a single mass such that the rock component provides strength, and the tailings, if maintained saturated, reduce the oxygen diffusion into the mass, thus reducing the potential for acidic conditions to develop if the rock or tailings have significant sulfides.

LESSONS LEARNED

Although mining and tailings disposal have been part of human history for thousands of years, the disposal of tailings into dedicated facilities has been practiced only since the early 1900s. Initially the construction of these facilities was by hand, due to a lack of earthmoving equipment, which prompted the

use of upstream techniques for most dams. By the 1940s and 1950s, the availability of large earthmoving equipment and engineering principles for water storage dams, in terms of placing selected materials in certain zones in compacted horizontal lifts, began to be applied to mine tailings facilities. In the 1960s, largely as a result of a few earthquake-induced failures of tailings dams, the phenomenon of liquefaction began to be understood and accounted for in design, although it is only now that a related phenomenon, static liquefaction, is becoming well understood. Planning for the safe management and containment of water in tailings facilities is also relatively new, having become a specific design and monitoring subject in the 1960s and 1970s. More recent is the recognition that chemical stability is a crucial aspect of tailings management, and over the last 20 years much research has been carried out to, among other things, control acid drainage from sulfide tailings deposits.

Notwithstanding the relatively rapid advancement in tailings management engineering, the mining industry has been saddled with a somewhat checkered history, in part due to tailings facility failures and incidents. Like any other field of human endeavor, learning from its mistakes is vital. It has been reported that "there have been no unexplained failure events" and concluded that "In all cases over the past thirty years, the necessary knowledge existed to prevent the failure at both the design and operating stage, but that knowledge was not used" (Martin et al. 2002). This position was updated in 1998: "The mining industry has the technology to safely design, build, operate and decommission tailings facilities. This technology must be consistently applied for the safe and environmentally responsible management of tailings" (MAC 1998).

With those statements as a backdrop, it is worth investigating the lessons learned from past incidents. The following trends are reported for tailings facility incidents (Martin et al. 2002):

- Active impoundments are more susceptible to tailings dam failure than inactive facilities.
- The correlation between upstream constructed dams and the likelihood of incidents is strong, although the large number of upstream dams in the database must be recognized.
- Seepage-related phenomena are the main failure mode for non-upstream tailings dams.
- Earthquakes are of little consequence for most non-upstream dams.
- For inactive impoundments, overtopping is the primary failure mode.

The International Commission on Large Dams Bulletin 121 (ICOLD 2001) provides a comprehensive report of these lessons, drawing from a range of tailings management facility failures and incidents. The main causes of failures and incidents identified were

- Lack of water control in the facility leading to overtopping and/or seepage erosion (piping),
- Lack of construction control, and
- General lack of understanding of the features that control safe operations.

Tailings facility failures were due to (in order of prevalence)

- Slope instability,
- Earthquake loading on upstream dams,
- Overtopping,
- Inadequate foundations, and
- Seepage.

Successful planning and management of tailings facilities could benefit greatly from involvement of stakeholders; thorough investigations and risk assessments; comprehensive documentation; and tailings management integrated into mine planning, operations, and closure (ICOLD 2001). It is also suggested that benefits can be realized by adoption of a comprehensive tailings management system and maintaining continuity of staff, including consultants, to the greatest extent possible.

Specific examples of tailings facility failures where lessons can be learned include the Stava tailings facility in Italy (Ulrich 1996) and the Merriespruit tailings facility in South Africa (Fourie et al. 2001).

Stava Dam Incident

On July 19, 1985, two tailings dams near Stava, Italy, catastrophically failed, resulting in the destruction of two villages and causing extensive property damage. Two hundred and sixty-nine lvives were lost (Genevois and Tecca 1993). A total of 190,000 m^3 (154 acre-ft) of liquefied tailings flowed down the Stava valley. The flow reached the village of Tesero, a distance of 4 km (2.5 mi) from the mine, in only minutes. The failures occurred without any warning (Berti et al. 1988) and are not attributed to earthquake shaking or stormwater flooding.

These dams were never engineered, at least not in the conventional use of the term; however, the initial construction of the upper dam was based on the successful operation of the lower dam. The dams were constructed with minimal design effort. Site investigations were never carried out in any form until after the failure, lab testing was never conducted, and design drawings were never made. Surveying was never attempted at the site, and no instrumentation of any kind was ever installed.

The failure of the upper dam was caused by the presence of a high phreatic surface within the sand shell leading to the initial failure, which strained sufficiently to produce liquefaction of the sandy materials. "The critical factor was direct ingress of water into the sand shell of the embankment and subsequent saturation of a portion of it" (D'Appolonia and Morgenstern 1988).

When the saturated zone had incorporated a region of the sand shell approximately 7.5 m (24.6 ft) wide, failure was incited under drained conditions. Because the sand shell was in a loose state, the straining that took place generated positive pore pressures, sufficient to induce liquefaction under static loading conditions (i.e., static liquefaction), and the sand shell began to flow. A condition of undrained loading of the slimes resulted from this loss of confinement, and the slimes deformed in an undrained manner (D'Appolonia and Morgenstern 1988).

The relationship between the stability of a dam and the location of the phreatic surface within a dam has been well recognized by the geotechnical community for a considerable time. Records of pond water level indicate that the pond was at its historic maximum elevation at the time of failure. In addition, there is ample evidence that appreciable seepage existed in the sand shell just prior to failure. Given such information, it would appear that the dominating factor causing the failure was related to the location of the phreatic surface (D'Appolonia and Morgenstern 1988).

The failure of the Stava dam stands as a grim reminder to the mining industry of the need for proper design, operations, and monitoring of tailings facilities, and as an important reminder to the geotechnical community of the need to understand the possibility of failure mechanisms such as static liquefaction and how subtle changes in operation practices may lead to significant changes in important characteristics such as seepage patterns within a tailings facility.

Merriespruit Tailings Failure

Failure of the 31-m- (100-ft-) high gold tailings facility just outside of the village of Merriespruit, South Africa, in February 1994 resulted in the death of 17 people and widespread damage to the village and the surrounding environment. The failure occurred shortly after a 50-mm (2-in.) rainfall event. Approximately 600,000 m^3 (21.2 million ft^3) of liquid tailings flowed from the facility, through the village of Merriespruit, stopping about 3 km (1.9 miles) downgradient.

It is accepted that the primary cause of the failure was overtopping, which resulted in large-scale removal of tailings from the slope face (Wagener et al. 1998). The post-failure investigations found that the dam overtopped at the breach location and water flowed over the crest of the dam for 1 to 2 hours. Removal of tailings from the outer slope would have exposed tailings inside the facility that had previously been confined. Conventionally, however, gold tailings produced from Witwatersrand quartzites are considered to be strongly dilatant (Blight 1998), and thus even if confinement were removed, the tailings should not have moved for any significant distance, because for a strain-hardening soil the resistance increases continuously during undrained loading (Zhang and Garga 1997). In addition, the outer zone of the tailings dam was considered to be well consolidated due to the thin layer deposition procedure wherein layers of tailings are placed sequentially around the perimeter and allowed to drain, desiccate, and consolidate (Blight and Steffen 1979). Undoubtedly, it came as a surprise to the geotechnical community in South Africa that 600,000 m^3 (21.3 million ft^3) of gold tailings would liquefy and flow as far as 3 km (1.9 mi) (Wagener et al. 1998).

The fact that no gold tailings dam in South Africa had previously failed with such devastating consequences probably contributed to the complacent response to the excessive seepage and toe sloughing that occurred prior to the dam's catastrophic failure. Conventional stability analyses carried out a few months prior to the failure indicated a satisfactory factor of safety against slope instability. Based on a traditional geotechnical engineering approach to slope stability, perhaps the perception that the facility was inherently safe can to some degree be excused. Many practicing geotechnical engineers today, if presented with the information available at the beginning of February 1994, might also come to the conclusion that the tailings impoundment, although not in a completely satisfactory state, was not intrinsically unsafe. It is only when the concepts of static liquefaction are invoked that incipient instability might have been predicted. Such a prediction would have been difficult, if not impossible, without either piezocone profiles or information regarding the in-situ void ratios and knowledge of steady-state soil mechanics.

The failure of the Merriespruit tailings dam caused a major reconsideration of operating procedures of tailings dams in South Africa and contributed in some measure to the development and publication of a Code of Practice for Mine Residue Deposits (South African Bureau of Standards 1998). Although such improvements are commendable, it is important not to lose sight of the fact that the Merriespruit failure was a single, isolated incident and that many dozens of gold tailings dams in South Africa have been successfully and safely operated for nearly 100 years. The stark lessons learned from Merriespruit should not be forgotten: Poor pool control, subaqueous tailings deposition, and inadequate freeboard are all well-known sources of operational difficulties on tailings dams. The Merriespruit failure simply emphasized this point, with tragic consequences.

CURRENT AND FUTURE TRENDS

The mining industry has made, and continues to make, many significant improvements to tailings management. Such changes come under the following categories (Martin et al. 2002):

- Management practices
  - Creation of guidance documents
  - Initiatives by the mining companies
  - Use of qualified engineering consultants
  - Response to regulatory trends
- Tailings management technologies
  - Improved geotechnical designs
  - Designing for geochemical issues
- Metallurgical improvements
  - Control of ARD (when applicable)
  - Management of cyanide (when applicable) and other chemical reagents

Increasingly, guidance documents are being prepared by the mining industry and other stakeholders including regulatory authorities, as described previously in the "Tailings Management Principles" section. Such documents provide guidelines for a wide variety of topics ranging from nontechnical issues, such as the development of management frameworks and identification of management responsibilities, to more technical matters, such as OMS of tailings facilities.

International standards are frequently mentioned yet they are not well defined. In general, they consist of a compendium of standards and guidelines by lending agencies such as the World Bank and its International Finance Corporation (IFC), mining industry initiatives, and regulatory agencies. The IFC has recently produced an environment, health, and safety guidance document for mining (IFC 2007) that includes tailings and water management. The Mining, Minerals and Sustainable Development (MMSD) project of the International Institute for Environment and Development (IIED) produced its report "Breaking New Ground" (IIED 2002), which includes a section on large-volume wastes, including tailings. The United Nations under its United Nations Environment Programme (UNEP), the International Council on Metals and the Environment (ICME), and more recently the International Council on Mining and Metals (ICMM) have also produced case study and risk management guideline documents for tailings. The broad objectives of these documents are generally to

- Monitor, interpret, communicate, and implement improvement trends in mining and the environment;

- Provide direction and influence to international and national government polices and company practices related to improving environmental management, regulations, and practice; and
- Seek cost-effective solutions to environmental challenges in mining.

Mining companies themselves are also producing several initiatives to improve mining operations globally. In addition to the MAC initiatives described earlier, one other such initiative is the International Cyanide Management Code that is administered by the International Cyanide Management Institute (ICMI) for the manufacture, transport, and use of cyanide in the production of gold (ICMI 2009). This code is typically referred to simply as the cyanide code. The cyanide code is a voluntary program for the gold mining industry, illustrating the commitment of mining companies to, among other things, improve tailings facility design and operation, and promote

- Responsible management of cyanide used in gold mining,
- Enhance the protection of human health, and
- Reduce the potential for environmental impacts.

Companies that become signatories to the cyanide code must have their operations audited by an independent third party to demonstrate their compliance with the code, the results of which are published and divulged to the public. In addition, many international mining companies have written their own in-house guidelines, much of which can be found on their company Web sites. Several companies have promulgated such undertakings as internal audits of their procedures and management systems, forming review boards with industry experts from internal and external sources, especially for significant, high-profile undertakings. Probably the biggest current initiatives are improved designs of tailings facilities, including the use of thickened, paste, and filtered tailings, and incorporating closure and reclamation objectives into the design.

The practice of tailings management has come a long way in a relatively short period thanks to (1) its establishment as a dedicated topic of engineering and environmental science, and (2) the commitment made by many mining companies, industry associations, and regulatory bodies for improvement. However, continuous improvement must be sought. The following topics warrant attention in the near future.

Improved Consistency of Good Tailings Management Practices Across the Industry

It is now accepted that the technology exists to address all currently contemplated engineering and environmental science issues related to safe and secure tailings disposal (although it is also accepted that technology improvements will continue to be made). However, it is also recognized that the technology is not always used or used adequately. The industry is working to put tools in place that will help rectify these gaps but is not yet at the finish line. Widespread adoption of full life-cycle tailings management systems, as outlined in the "Tailings Management Principles" section, will help, as will acceptance by all mine owners that a well-developed and safely operating tailings facility has value and is not simply a cost. Good documentation is fundamental to this endeavor because, over the life of many tailings facilities, key people come and go and the current people must be aware of the thinking to date.

Improved Recognition of the Uniqueness of Each Tailings Facility and the Need for Appropriate Expertise

Because each tailings facility is unique, an off-the-shelf or cookie-cutter approach to design is not acceptable. This is why mining industry guidelines are just that—guidelines. They do not and must not replace sound project-specific engineering and environmental science provided by appropriately qualified firms or individuals. Knowledge and experience are fundamental to take advantage of the lessons learned and to fully understand and appreciate the nuances associated with the site conditions, tailings properties, and the facility's operating principles.

Improved Recognition That Tailings Facilities Are Permanent Features with an Indefinite Life

Tailings dams differ in many ways from water dams in that they are generally permanent structures unless the tailings deposit is subsequently removed (which can happen if the tailings are reprocessed or the area is evacuated for mine expansion). Thus, indefinite time frames may need to be contemplated when designing and operating a facility. Planning from the beginning for closure and a postclosure period is gaining acceptance, but planning for indefinite closure may be more appropriate. Although peak deterministic (equivalent to a very long return period) earthquake and storm events are now being used, the long-term durability and potential degradation of materials for seepage barriers, drains, and structural zones are not always contemplated. In addition, the implications of future land-use changes that could increase the hazard classification of the facility should be appreciated.

Increased Application of Dewatered Tailings in Appropriate Circumstances

The technology associated with dewatering tailings for bulk surface disposal is improving rapidly in response to demand by the industry. The most pressing need is in dry climates where the value of water is high and thus the benefit of recovering and recycling water is also high. This is particularly true at the large copper mines in Chile where significant volumes of water are lost to evaporation from the tailings facility. The biggest challenge is associated with the large tonnages of tailings produced and thus the large dewatering capacity required.

A second challenge is the cost. In recent years substantial advancements have been made to reduce capital (equipment) and operating (power) costs, and, as further experience is gained, additional capacity expansions and unit cost reductions can be expected. Understanding the behavior of thickened, paste, or filtered tailings when deposited is also a key challenge, particularly in terms of predicting shear strength and liquefaction potential, and beach slope angles for high-density tailings. Often the decision to use thickened or paste tailings, in addition to increased water recovery, is because development of more steeply sloped deposits reduces the height required for containment structures. Care must be taken, however, to avoid overpredicting the slope if the beach length is large, because this may result in the need for subsequently constructing an unexpected rise to the structure.

Dust Control

Dust generation has become a problem for some tailings facilities with large exposed beaches in windy environments. Often the dust is generated from sandy tailings in the upper portion

of a segregated beach because the sandy tailings are unable to retain the capillary moisture that finer tailings can, which helps to hold the particles on the beach. However, in areas where freeze–thaw cycles are significant, both sandy and finer tailings can be dewatered and loosened from the beach, creating a dust concern. Current methods for controlling dust include the installation of snow fences in strategic high-impact locations, application of water to moisten the dry and dust-prone areas, or the application of tackifiers to adhere the loose particles to the beach. A more innovative approach involves the injection of algae into the tailings stream to produce a biological cover on the exposed beach surface. Thickened tailings can help in areas where freeze–thaw is not an issue because the degree of particle segregation is reduced.

Placing Greater Emphasis on Integrating a Tailings Facility with Other Mining and Mine Waste Facilities

Placement of tailings underground in mined-out workings as backfill has been successfully done in numerous mines worldwide and is partly responsible for the origins of the tailings dewatering industry because dewatering was required to give the tailings substantial strength and stiffness. However, the disposal of tailings in combined surface facilities with other mine wastes, such as waste rock from the pit or heap leach pads, is less common. The La Quinua thickened tailings management facility previously described is unique because it is the only one in which the tailings are fully contained within an active heap leach ore pile. Combining these facilities requires careful consideration of the design and operating principles of each and achieving compatibility. This may not always be possible. However, if compatibility can be reached, the benefits are large and include cost, land-use, and closure efficiencies. Extrapolation of this example to tailings within waste rock storage facilities may hold similar promise of successful applications.

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CHAPTER 8.11

Waste Piles and Dumps

Marc Orman, Rich Peevers, and Kristin Sample

The terms *mine waste piles* and *dumps* refer to piles of waste rock or leached ore that carry little or no economic value at the time they are placed. As commodity values rise and process methods gain efficiency, waste piles and dumps may be reclassified as ore and gain value. Also, the waste material may be valuable at some future time as an aggregate source, for use in riprap, drain material, or other process method that recovers the commodity at lower grades or has lower acceptable rates of return. Heaps are ore piles that are amenable to a leaching process, both with and without the use of liners, and share physical characteristics with piles and dumps.

HISTORIC PERSPECTIVE

Ever since humans began to extract materials of value from rocks, waste or other lesser-valued material has been left behind after the extraction process. For early miners, waste dumping was simply a matter of pushing the waste out of the way, either down a slope or to any other available area. Frequently, these waste materials ended up in drainage basins, rivers, and lakes where they caused environmental harm. In 1884 in California (United States), hydraulic mining was essentially outlawed by the Sawyer decision because the mine waste in the rivers had led to flooding after streams and rivers became choked with solids. This law, handed down by the Ninth Circuit Court in the case of *Woodruff v. North Bloomfield Gravel Mining Company*, became one of the first environmental decisions in the United States (U.S. Circuit Court 1884).

Over time, other regulations and laws have emerged, and standard practices have evolved to minimize the environmental damage and potential hazards associated with the disposal of mining waste. Enlightened mining companies now deal with their waste products in a responsible manner, especially when negative impacts on the public may result because of improper disposal. Nevertheless, improper disposal and handling of mine waste continue to pose environmental hazards across the world. While the hazards associated with mine waste disposal have decreased in most developed countries, it continues to be problematic in regions where regulations and environmental laws are not strict and enforcement is lax.

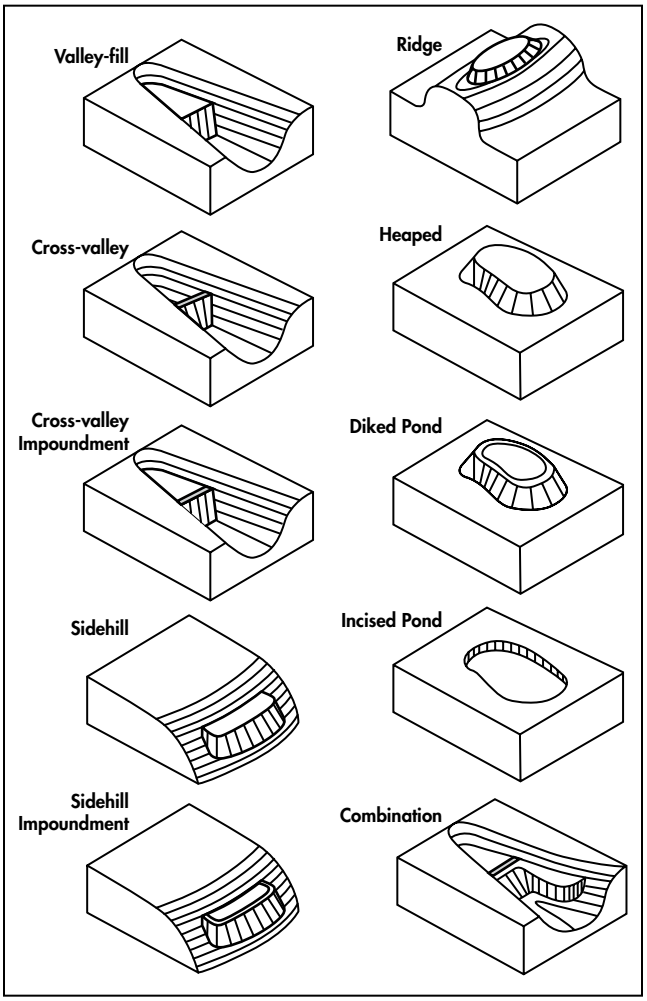
A significant contributor to mine waste dumps is heap leaching, which is a relatively new form of mining where low-grade ore is piled over large surface areas and irrigated with solutions. The resulting pregnant solution is then processed to recover the desired commodity. After the leaching is completed, the leached ore becomes a waste product. On a permanent pad, the ore material is stacked and leached in lifts until the pile reaches the final design height. Modern leach pad facilities can be hundreds of meters high and cover thousands of square meters in area. Alternatively, on a dynamic pad, a single thin lift (5 to 10 m [16.4 to 32.8 ft]) of ore is stacked and leached on the pad at any one time, after which the leached ore is removed and stored in a waste dump. In the past, environmental issues were not a main consideration for heap leaching. However, in response to environmental regulation, facilities have evolved, and many now utilize geosynthetic liners incorporating a leak-detection provision.

The earliest full-size leaching projects in the United States were for copper in the form of dump leaching with natural containment. Subsequently, with the introduction of cyanide for leaching gold and silver, soil liners came in vogue in the late 1970s to the mid-1980s (Breitenbach and Smith 2007b). Since then, use of geosynthetic clay liner, high-density polyethylene (HDPE), low-density polyethylene (LDPE), linear low-density polyethylene (LLDPE), and polyvinyl chloride (PVC), as well as asphalt impregnated geotextiles and a few others, have become the standard liner materials for heap leaching.

TYPES OF WASTE PILES, DUMPS, AND HEAPS

This section provides a description of waste piles, waste dumps, and heap leach pads (both lined and unlined). Although these types of facilities are similar, the liner aspect introduces an additional potential for failure along the liner as part of design. On the other hand, for unlined facilities it is important to consider the materials' geochemistry (both the ore as well

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Source: Zahl et al. 1992.

Figure 8.11-1 Mine dump configurations

as the resulting pregnant solutions) and the site's hydrology to ensure that natural water resources are adequately protected.

Configurations

Collectively, waste pile, dump, stockpile, or a leach heap can be referred to as waste structures. As such, their layout generally falls into the following categories, depending on the type of waste, the purpose of the waste structure, and the physical constraints at the site. Each of the configurations is shown in Figure 8.11-1 and discussed in further detail in the following paragraphs.

A *valley-fill* waste structure, as the name indicates, fills a valley. Many of the lined valley-fill leach pads require some type of stability berm at their toes. Construction of a lined valley-fill leach pad would begin at the toe berm and progress up the valley. Construction of a waste dump (not a leach pad) usually begins at the upstream end of the valley, and dumping proceeds along the downstream face (as shown in Figure 8.11-1). For a heap leach facility, stacking should begin at the toe and proceed up the valley to avoid slope-stability problems.

The top surface is usually sloped to prevent water ponding. Stormwater run-on can be controlled by constructing diversion channels up-gradient of the facility. In steep terrain, where the facility is going to take a long time to fill, it may be more economic to construct a rock drain below the facility to pass stormwater. Subdrains may also be needed below the structure to control seepage from natural springs and material drainage.

A *cross-valley* structure crosses the valley, but the valley is not completely filled up-gradient. The structure is usually designed with a rock drain at the bottom of the valley to control the storage and/or discharge of stormwater flows, or a water diversion system must be installed up-gradient to provide drainage around it. This type of structure could also be used as a retention dam for fine coal or waste slurries, in which case the design must conform to applicable regulations for dams and impoundments.

A *sidehill structure* lies along the side of a slope but does not cross the valley bottom. This structure may be constructed to impound either water or mine waste slurries (and therefore would need to conform to applicable dam regulations). As with a cross-valley structure, a sidehill embankment should also be designed and constructed with either stormwater diversion channels or rock drains to control the storage and/or discharge of flood flows. In some cases, the hillside may require benching and/or a keyway at the toe to increase the stability of the facility.

A *ridge embankment* straddles the crest of a ridge, and waste material is placed along both sides. Unlike the cross-valley or sidehill configurations, this type of structure is typically not used to impound fine-grained material or water. In some cases, one or both sides of the ridge may require benching and/or a keyway at the toe to increase stability.

A *diked embankment* is constructed on nearly level terrain and can either impound fine-grained or coarse-grained mine waste. By definition, this type of embankment is composed of two parts: a down-gradient containment dike and the embankment or dump itself. These two parts may or may not be isolated from one another by liners. If fine wastes are impounded by coarser waste, the structure is considered a *dike*. If the embankment is homogeneous and coarse, the embankment is termed a *heap*, such as a heap leach pad.

Leach Dumps or Heaps

Leach heaps consist of low-grade ores spread or stacked on large platforms where the pile is irrigated with leaching solution to leach out the recoverable product of value. Although heap leaching has been used mostly for precious metal and copper ores in the past, it is now also being used for other products, such as uranium and nickel. In recent times, even municipal wastes have been leached using similar methods to accelerate the decomposition of waste and add capacity to the facility. Heaps are normally placed on impermeable liners of natural and synthetic materials (discussed in more detail later in this chapter).

Dumps usually refer to material piles created by end dumping. Run-of-mine ore is sometimes simply dumped instead of being stacked on a leach pad and leached for economic recovery of the contained commodity (a process known as *dump leaching*). The same procedure is often used for secondary recovery from leached ores. Dumps are generally placed on natural soil or rock subgrade surfaces that have been demonstrated to have some degree of natural solution containment and are normally located on sloping ground or in a valley to promote drainage to the toe.

Stockpiles

The term *stockpile* refers to any pile of material that is placed for future use. This can include material with either proven or potential value, material for structural fill, or other materials obtained from borrow pits or removed from stripping projects. Waste rock or processed material to be used as backfill can also be categorized as a stockpile. These materials, which are stored for processing or future use, appear much the same as waste rock except they are normally isolated from waste materials so they may be recovered at some later time as economically as possible and without being contaminated with waste. Stockpiled material, such as the ore itself, may be chemically unstable, and the stockpile may require liners, caps, and/or stormwater diversion structures to prevent water infiltrating the pile and causing water contamination.

Placer Waste and Tailings Deposits

During placer mining for gold or aggregates, the practice of washing sand and gravel to recover minerals can produce tailings with particle sizes ranging from coarse to fine (<75 μm) and wash water, which should be treated. The coarse waste fraction can be disposed of using one of the methods previously described; however, the fines portion is similar to the tailings from a milling operation. Considerations for these types of wastes include the placement and storage of the tailings and treatment of the wash water to meet discharge requirements.

With physical constraints of space limitation and the rising cost of conventional impoundment methods for tailings storage, the use of process items such as thickeners and filter presses to put tailings in piles or mounds has become more common. With the removal of additional moisture, alternative disposal methods such as thickened tailings, paste backfill, treated paste backfill, and dry stacking become viable options, which can add capacity to the facility.

IMPACTS OF WASTE DUMPS

Waste dumps and heaps have several (actual and potential) impacts on the environment, which must be considered as part of their permitting and design. These impacts include disturbance of the land, water quality issues, slope stability, and visual effects. In the past, waste dump disasters have led to the contamination of surface and groundwater, as well as massive slides, which have buried communities.

Planning waste disposal facilities requires evaluating the regulatory constraints, identifying an appropriate site, designing the structural and environmental integrity of the facility, developing an operating and maintenance plan, and developing a reclamation plan for future land use (Center and Zlaten 1982, and Ritcey 1989, as cited in Zahl et al. 1992). From the design point of view, the specific issues to be considered are the contamination potential of the waste, slope stability, the condition of the waste structure's formation under normal and seismic loading, and ways to control water (both internal and external) to the dump.

Water Quality

Water quality impact issues associated with waste, unlined dumps, or poorly constructed heap leach facilities can be a major environmental concern. Waste rock should be thoroughly tested at the design stage for acid-generating and metals-leaching potential to ensure that water resources are adequately protected.

To provide background information on flows and water quality, groundwater and surface water samples should be collected before construction begins. These measurements and samples should be collected throughout the year so that seasonal fluctuations can also be monitored and effectively evaluated. All drainages and aquifers in the vicinity of the project should be tested to ensure that water quality for the entire project area is well understood before the project begins.

Initial testing of the water samples should include major cations and anions, metals, nitrates, dissolved and suspended solids, salts, and organic compounds, as well as other constituents that may emerge as relevant during the process and involve potential changes to water chemistry. Samples should be collected from dedicated monitoring wells and surface sampling locations, both up- and down-gradient of the project site.

A water quality monitoring plan should be prepared to document sample locations, sampling frequencies, and protocol for collecting the samples. At a minimum, the plan should contain the following items:

- Identification of the surface and groundwater sources
- Monitoring objectives
- Description of water quality parameters
- Sampling point descriptions and a map of their locations
- Analytical procedures
- Data quality control objectives
- Data management and quality control details
- Sampling equipment to be used
- Sample preparation and handling procedures
- Chain of custody and data sheets to be used
- Reporting requirements

Land Disturbance

Wherever mine waste is placed, the natural environment is changed, and the process is therefore classified as land disturbance. The initial disturbance creates the potential for sedimentation of natural waterways caused by erosion and water quality degradation, which are among the major potential impacts of waste dump construction. Although waste dumps can be designed to minimize the impacts of land disturbance and blend in with natural surroundings as part of reclamation in some locations, these disturbances have been perceived by some as highly destructive to the environment. Specifically, in California, all metallic mines are now required to use waste rock to backfill all open pits as part of the state's mine reclamation requirements. The U.S. Office of Surface Mining requires restoration to approximate original contours for surface coal mining. These requirements can add considerable cost to final closure.

Since most waste structures are not compacted, the volume of a pile or a dump can be much greater than the volume of the pit, which adds further to the issues of how to hide, or at least reduce, their impact.

Visual Impacts

Visual impacts from mine waste dumps and leach pads can be a major concern for mines located in the vicinity of populated areas or where the facilities will be visible from roads and highways. Visual impact and viewshed studies are now performed routinely in many areas of the world as part of initial mine permitting.

In areas where the color of the rock blends with the natural color of the terrain, visual impacts will be less than in areas

with sharp color contrasts. In flat areas, hills develop, and, in mountainous terrain, ridge tops appear and grow, and drainages are filled. By maintaining slope angles that are similar to natural slopes, visual impacts may be reduced, and many companies are now designing dump surfaces to simulate the original topography. However, contrasts in colors from the natural vegetation to rock and topsoil can take several years to blend together as the revegetated slopes take hold following reclamation.

A visual impact study may include the following components, as described by the Federal Highway Administration (FHA 1981):

- Description of the project setting and the major viewsheds
- Photographic study of the project from the major views
- Description and analyses of the existing visual resources and responses from people in the area
- Renderings of the project alternatives' views
- Assessment of the visual impacts of the project alternatives
- Possible methods to mitigate the adverse visual impacts

As part of the visual impact study, maps are usually produced and show the areas from which the project would be visible according to different design options. The design options typically include several different ultimate elevations and possible configurations of the waste dump or heap.

DESIGN OF WASTE DUMPS

This section provides an overview of waste dump design. Further details regarding the design of waste dumps may be obtained from the following recommended SME-AIME publications and from several other references cited throughout this chapter: McCarter 1985a, 1990; and Hustrulid et al. 2000.

Proper planning and design require a thorough understanding of the material properties of the waste rock or ore, liner interface strengths in the case of a lined facility, and foundation conditions. In the case of a dump or heap leach, groundwater and seepage properties of the ore must also be understood in order to properly design these types of facilities. Studies would include a field investigation consisting of mapping of soils and rock; drilling boreholes; monitoring well installation; excavating a test pit; sampling waste rock, ore, and foundation materials; laboratory testing; and analyses.

Slope Stability

Slope instability and failure are major issues for all types of mine waste dumps and heap leach operations. The risks and environmental impacts of waste dump instability are a major concern for both mine operators and regulators. A slope failure in a waste structure could cause injuries and disruption of operations because of equipment burial or closure of an access or haul road. Slope failure in a heap leach pile can lead to a liner failure and the potential release of pregnant solution, which may result in contamination of groundwater resources, as well as a loss of revenue. In either case, there are clean-up and remediation costs. Proper preplanning and design are imperative to avoid these types of costs.

Numerous factors affect waste dump or pile stability, including site topography, dump geometry, rate of stacking and lift thickness, geotechnical properties, method of construction, equipment loads, phreatic surface, and seismic forces—all of which must be considered in the evaluation of the waste structure's stability over its design life. Generally, limit equilibrium analysis using one of the several prevalent approaches is considered adequate to evaluate slope stability of waste dumps.

Failure Modes

The basic failure modes of waste dumps must be considered during the stability evaluation and design. Detailed descriptions of identifiable waste dump failure modes and appropriate analyses are described by many in the literature (e.g., BCMDC 1991; Caldwell and Moss 1985). Each of the main failure modes are shown in Figure 8.11-2.

Surface or edge slumping. The most common failure mode is edge slumping (crest slumping), where a thin wedge of material translates down the slope, parallel to the dump face. This shallow failure typically originates near the crest of the dump because of oversteepening. Cohesive or low-permeability waste materials allow the development of oversteepened slopes. End dumping the waste in thick lifts or pushing material over the dump crest also leads to a higher risk of over-steepening and edge slumping. Edge-slumping failures often occur after heavy precipitation, which leads to increased pore pressures in the low-permeability waste. In coarse rock-fill dumps, oversteepening of the crest may develop due to initial interlocking of the blocks (BCMDC 1991).

Plane failure similar to edge slumping may occur deeper within the waste dump materials. In this case, sliding occurs along a single plane of weakness within the dump, which may have been created because of a zone of poor quality waste or from dumping waste on top of snow or ice. The plane of weakness parallels the dump slope or daylights at the dump face.

Shallow flow slides. Flow slides are shallow slumping failures of saturated or partially saturated waste. Typically triggered by rain or snowmelt, they result in material flowing down the slopes due to shear failure or collapse of the soil structure.

Rotational circular failures. Rotational circular failure (mass failure along a curved failure surface) may occur within the waste as a result of excessive dump height, additional loading induced during an earthquake, weak or fine-grained waste materials, reduction in toe support, and/or high pore-water pressures. Rotational failure surfaces may also extend into the foundation if the soil is weak or high pore pressures develop, such as within a deep fine-grained soil deposit. Creep failure is also a type of rotational failure, with widespread rotational shearing characterized by bulging at the dump toe (BCMDC 1991).

Base failure (spreading). Base failure may occur if a thin, weak base layer is placed over the foundation, especially if the foundation is inclined. If a slope wedge of the waste dump translates laterally along a shear surface, the foundation soils may spread and be squeezed ahead of the advancing dump toe. This phenomenon, known as foundation spreading, may result in progressive failure of the overall dump (Vandre 1980; BCMDC 1991).

Block translation. Block translation (planar sliding) may result from any of the inducing factors mentioned for rotational failure and is favored by steep foundation slopes and a thin, weak soil cover or lined surface. The bulk of the dump slides as a rigid block along a plane of weakness. This weak plane may be within the foundation soil, along the interface between the dump and the foundation, or along a liner interface.

Liquefaction. If the soil foundation or the waste dump itself is composed of liquefiable materials, and high pore-water

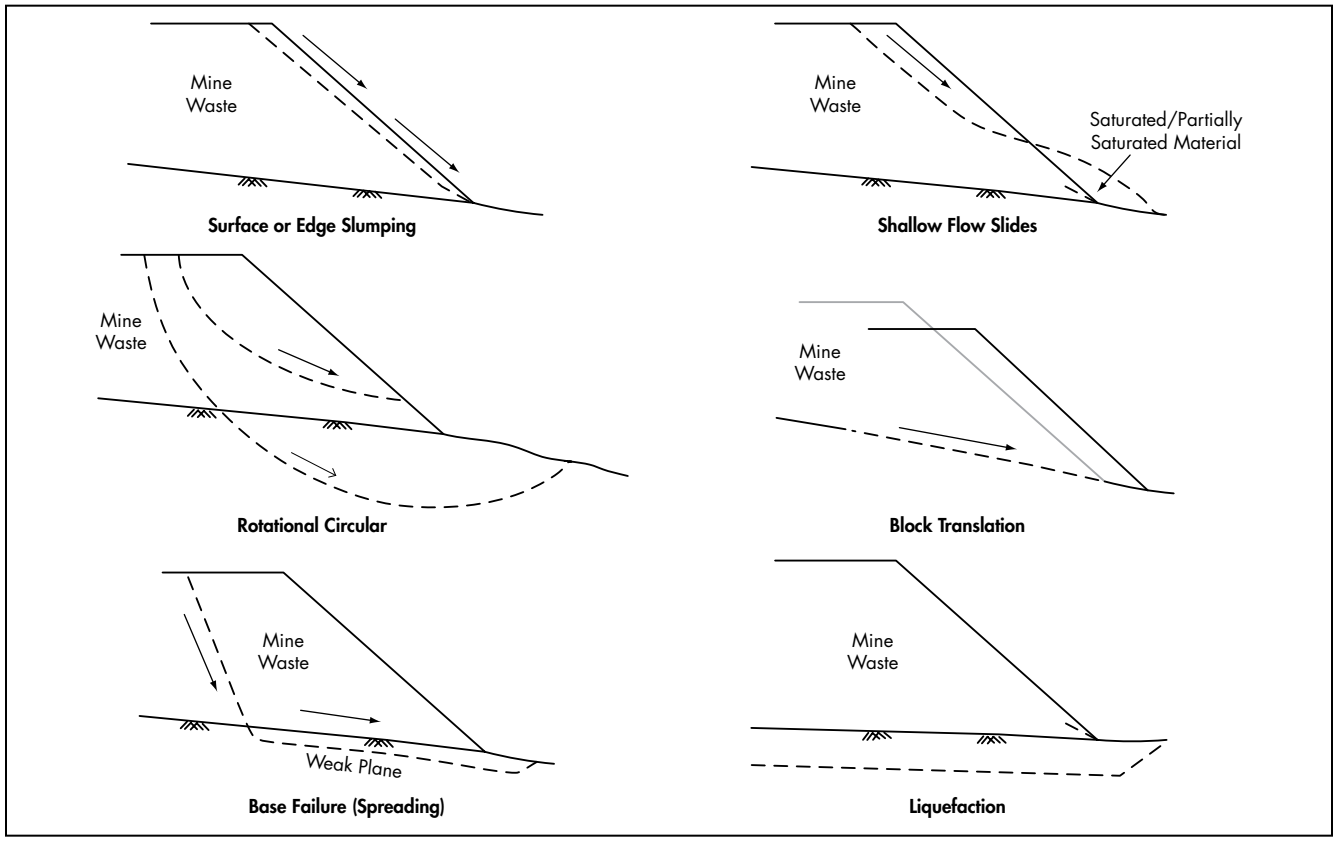


Figure 8.11-2 Failure modes

pressures exits, then liquefaction may pose a significant stability risk. If liquefaction occurs in the foundation, the entire dump may be translated or there may be progressive failure (BCMDC 1991).

Factors Affecting Slope Stability

To properly design a mine waste dump for stability, the following details should be considered:

- Site topography and location
- Dump geometry, rate of stacking, and lift thickness
- Geotechnical properties of the waste, liner system (if applicable), and foundation
- Methods of construction and equipment loading
- Seepage, phreatic surface level within the dump, and the solution collection system
- Seismic forces and liquefaction potential

The size and complexity of the project, as well as the consequences of dump failure, will typically control the extent of the investigation performed to obtain this information. The investigation should be thorough enough to identify all adverse conditions and to provide reasonable certainty that the parameters used in the design are appropriate (Vandre 1980).

Site topography. Based on economics, dump-site locations are typically selected to minimize the distance between the waste source and the disposal area. The waste may be disposed of in an area completely outside of the pit, or in-pit dumping may be preferred.

During the investigation stage of design, the topographic information gathered should include the entire drainage area that may affect the dump, as well as identifying those areas that would be affected should a dump failure actually occur. Should a failure occur, the inclination of the dump foundation will be an important factor in the dump stability as well as run-out distance. Experience shows that foundation slopes steeper than 25° typically result in lower factors of safety for slope stability. On the other hand, topographical features providing lateral support or toe buttressing will improve the stability of the waste dump.

Dump geometry and stacking method. The geometry of the waste dump depends largely on the dumping method, as well as the topography of the site. The two common construction methods for waste dumps include end dumping and stacking material in lifts or layers. If the material is end-dumped from the crest of the waste dump, the material will flow down the slope and rest at or near the angle of repose, with the larger particles rolling down to the toe of the dump (Couzens 1985). The angle of repose for mine waste rock typically falls within the 35°-to-40° range, leading to steep side slopes. The factor of safety for the slope of an end-dumped waste pile is close to 1.0. The slopes are generally not flattened or compacted until closure of the waste dump.

In comparison, layered or stacked dumps allow for a higher factor of safety to be maintained, because they are constructed in a more controlled manner from the bottom up. The layers can be placed and compacted to increase the density and strength of the material. However, except for the heap

leach piles, layered waste dumps are not always feasible, as they require relatively flat topography (Vandre 1980).

Waste dumps constructed from end dumping are more likely to have a loose, collapsible particle structure within the dump than those constructed from the layered method. Collapse will result in localized *arching*, which leads to reduced normal pressures and shear strengths (Vandre 1980).

The exterior slopes of heap leach pads and waste dumps are typically constructed as steep as practical during mining operations to maximize the tonnage contained in the dump. Slope-stability analyses are used to determine the maximum allowable overall slope angle, including benches, for maintaining stable slope conditions to the planned ultimate dump height (Breitenbach 2004).

Smith and Giroud (2000) examined the effect of ore placement direction on the stability of a geomembrane-lined heap leach pad and concluded that stacking ore in the down-gradient direction results in a less stable structure than stacking in the up-gradient direction typically would.

Geotechnical properties—mine waste. The geotechnical properties of mine waste materials vary significantly between projects and even between different phases of the same project. The density, saturation, and shear-strength parameters of the materials forming the dump slope affect the failure mode and the calculated factor of safety (FS) against sliding. Other useful information for design includes the particle size distribution, specific gravity, permeability, compression index, soils classification, and degradation behavior of the waste materials. These parameters are generally based on laboratory tests. However, field practices and construction procedures are often not completely simulated in the laboratory for various reasons (e.g., equipment limits, time and budget restraints), and therefore engineering judgment is required in selecting properties for stability analyses. Verification testing is often required during construction to ensure that the parameters used during the design were reasonable, accurate, and appropriate.

Waste rock is coarse material typically classified as cobbles, rocks, or boulders with some fines. As previously stated, the angle of repose for mine waste rock typically ranges from 35° to 40° and is based on factors such as particle size and shape, fall height, specific gravity, and amount of water present. The density of waste rock materials typically ranges between 1.6 and 2.2 t/m^3 (100–137 lb/ft^3), depending on whether the material is loose or compacted (Williams 2000). In heap leach pads, for example, the ore is purposely stacked in a loose state to maintain a high permeability, as required by the leaching process. As subsequent lifts are placed, the density of the lower lifts increases as they are compacted by material placed on top, and therefore the shear strength of the lower lifts typically increase (Smith and Giroud 2000). Stacking or dumping mine waste in thick lifts results in significant variability of the in-place density within each of these lifts.

Understanding the shear-strength behavior of the waste material is important for evaluating the slope stability of the waste dump. Waste density and gradation variability, along with differences in normal and confining stresses (e.g., inside the pile versus at the toe or on the slope face), result in heterogeneous shear strength throughout the pile. Generally, a linear-strength envelope with a single friction-angle value over the entire range of stresses may be assumed for the stability analysis. However, dump heights achieved these days result in a much wider range of normal stresses in the pile, over which the strength envelope does not necessarily remain linear, and this nonlinearity of the strength envelope must be considered in the stability analysis.

The dominance of cobble- and boulder-sized rock fragments in typical waste rock imparts a dilatant behavior under low effective normal stresses and significant crushing of contact points at high stresses, as demonstrated in the case of rock fill (Barton and Kjaernsli 1981). The friction angle of the rock fill is strongly stress dependent and will be significantly lower for material at the base of the dump (due to higher normal loads) than for material near the toe of the dump (under low loads). Barton and Kjaernsli (1981) estimated that the effective friction angle of rock fill increases by between 4° and 8° for every 10-fold decrease in effective normal stress. The shear strength of rock fill is also influenced by the rock-fill dry density, void ratio, unconfined compressive strength, uniformity coefficient, maximum grain size, fines content, and particle shape.

Laboratory testing of the mine waste is often too limited to accurately represent the potential material variability of a large volume of waste under various loading conditions. Therefore, the shear strength of the mine waste for design and analysis purposes must often be estimated based on various inputs, including current laboratory test results, previous experience, the behavior of similar materials, and published literature (Vandre 1980; K.P. Sinha, personal communication).

Another aspect to consider during design is the effect of weathering on geotechnical properties. Waste materials that were assumed to be durable may weather or be altered in some other way, which decreases slope stability. For example, weathering of feldspar-rich rock may result in formation of clay, decreasing the effective friction angle and inhibiting rapid drainage.

Geotechnical properties—foundation. The foundation is a critical factor in the overall stability of the waste dump. The dump-site investigation should identify the general geology of the site and any adverse geologic and soil conditions. The soil cover and rock weathering depths should be determined and the materials should be classified for design. Particular attention should be paid to the presence of shallow groundwater, discharge areas, landslides, creeping slopes, organic soils, clays, and dip slope bedrock structures (Vandre 1980). The subsurface exploration may include sampling, in-situ testing, and borehole geophysics, and should cater to obtaining the critical parameters for design.

After soil and rock samples have been obtained during the investigation, laboratory testing should be performed to identify the pertinent geotechnical properties of the materials. The classification, strength, permeability, and consolidation properties of the foundation materials, and how these properties are affected by time or saturation, should be determined. The shear strength and thickness of the foundation soil is an important parameter for slope stability and the dump failure mode. Permeability of the foundation material will affect the generation of pore water pressures in the foundation, affecting the dump stability and limiting the permissible dumping rate. Foundations consisting of low-plasticity silts and clay soils have been blamed for forming shear failure surfaces of several large (>10 Mt [11 million st]) dump failures (Zavodni et al. 1981). Consolidation parameters are used for calculating expected settlement of the foundation; excessive settlement could have serious implications in terms of the liner and collection system in case of heap leach piles and dump failure in general.

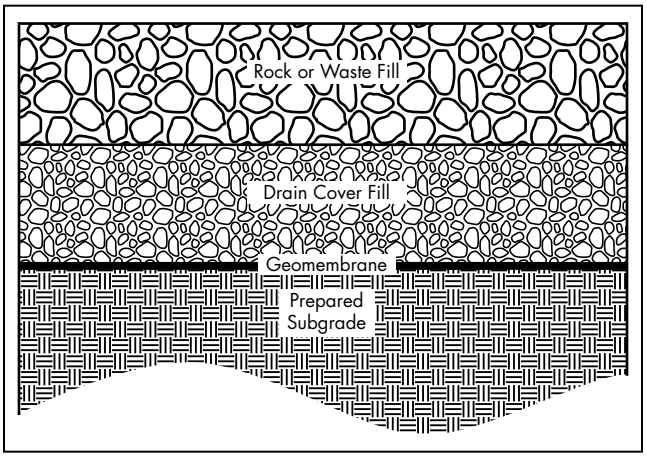


Figure 8.11-3 Example of heap leach pad liner system

Geotechnical properties—geosynthetics. Within the last 20 years, gold, silver, and, more recently, copper leach pads have been constructed with geomembrane-lined foundations (Breitenbach 2004). Typically, LLDPE or HDPE is used as the base liner. The decision is based on the elongation, strength, and other requirements of the application, as well as economic reasons. PVC liners have been provided in specific cases, mainly for economic considerations. The liner interfaces with the overliner (the drainage material), the subgrade, or the ore material itself (in case of interlift liners) create planes of weakness in the leach pile. An example of a geomembrane-liner system for a heap leach pad is shown in Figure 8.11-3. Slides in lined facilities usually occur by wedge failure along the geomembrane interface with geotextile or low-permeability subgrade (Breitenbach 2004), this being the weakest link in the chain. Thus, the soil–liner interface strength parameters may become the most critical data for evaluating heap leach stability. The soil–liner interface strength depends on several factors, including normal load, rate of applied shear, soil type, density, water content, and drainage conditions, as well as liner thickness, flexibility, and texture (Sample et al. 2009).

Just as with the waste and ore material, soil–liner interface strengths may also exhibit a nonlinear strength envelope, with the friction angle generally decreasing as the normal stress increases. Thus, as heap leach piles are extended to greater heights, decreases in the interface friction angle used for the stability analysis should be considered for the liner interface.

To select an appropriate minimum FS against slope failure, the designer must consider whether peak or post-peak (residual) strengths were used for the liner interface in the stability analysis. One method to ensure conservative design for wedge failure of a heap leach pad is to assume post-peak (residual) strengths for the liner system. Numerous studies of shear stresses for geomembrane–soil interfaces based on direct shear testing have been published, and the conclusions regarding peak versus post-peak strengths have been mixed. Post-peak strengths as low as 50% of peak strength have been observed for geomembrane–clay interfaces (Byrne 1994; Stark and Poeppel 1994), while other studies indicated that no strain-softening (i.e., reduction in strength with straining) behavior occurred (Koerner et al. 1986; Masada et al. 1994). Valera and Ulrich (2000) recommend the use of post-peak shear strength for soil–liner interfaces in stability analyses of heap leach pads, because the interface may reach residual strengths because of minor strains caused by installation and initial loading. Residual strength conditions may also be reached because of cyclic loading during an earthquake (K.P. Sinha, personal communication). Sharma et al. (1997) observed that the reduction in HDPE–soil interface strength after peak stress was greater when the plasticity index of the soil was more than 30.

Groundwater and phreatic surface. The effects of water on the stability of mine waste dumps can be difficult to evaluate, and measures should be taken to prevent excess water from entering the dump. In order to accurately assess the stability of the waste dump, a seepage analysis should be performed to establish flows through the dump and the height of the phreatic surface. Water pressure buildup within the dump will lower the FS for slope stability, and the potential for increases in the phreatic surface should be considered.

Within heap leach pads, the phreatic surface is often assumed to be some height above the base liner (e.g., 1 to 3 m [3.3 to 9.8 ft]), based on the design of the collection system. Because of the leaching process, leach pads present a combination of extreme base pressures and high moisture conditions not present in other lined facilities, such as landfills (Thiel and Smith 2004). In addition, leach pads are sometimes located in highly seismic areas, raising concerns about liquefaction due to sudden pore-pressure buildup.

An increase in the foundation water table may significantly decrease the FS for a deep failure through the foundation material, while perched water within the dump may lead to surface failures. Flow parallel to the surface of the slope may also decrease the FS significantly.

Seismic forces. In seismically active regions, the slope stability of the waste structure is also evaluated for seismic loading conditions. The seismic loading, although dynamic and cyclic in nature, is generally treated as a superimposed equivalent set of static loads, and the stability analysis for this case is referred to as the pseudostatic analysis. For these analyses, the two-dimensional mass in the limit equilibrium slope-stability model is subjected to a horizontal acceleration, which represents inertia forces due to earthquake shaking and is equal to an earthquake coefficient multiplied by the acceleration of gravity. The earthquake coefficient, or pseudostatic coefficient, is selected based on a specified design earthquake. Often a percentage of the maximum design acceleration in bedrock may be used for the pseudostatic analysis. However, selection of an appropriate pseudostatic coefficient may rely heavily on engineering judgment and is often debatable. Also, materials within the waste dump may undergo a significant loss of strength during earthquake shaking, which may not be entirely understood or defined from the laboratory testing. Therefore, while pseudostatic analyses are a simple and convenient tool, they should serve primarily as a screening method as to whether significant displacement may occur during the design earthquake. If a low FS is calculated in the pseudostatic analysis (e.g., <1.0), then significant displacements may occur, and displacement (deformation) analyses should be performed.

Dynamic analyses with numerical tools provide a more sophisticated alternative to pseudostatic analyses. Analyses may be performed with tools such as the finite difference program FLAC, and available finite element method and boundary element method programs. Use of these tools during design may depend on project budget, design requirements, and available resources.

For waste dumps, the greatest stability risk posed by earthquakes is typically liquefaction of foundation materials, although liquefaction may occur in susceptible waste materials as well. If liquefaction occurs in the foundation, the entire dump may be translated or there may be progressive failure (BCMDC 1991). Liquefaction due to seismic events is typically limited to 20 m (66 ft) in depth or shallower, due to the beneficial effects of confining pressure against liquefaction susceptibility (Thiel and Smith 2004). Simplified procedures to evaluate liquefaction resistance in soils have been widely discussed in the literature (e.g., Seed and Idriss 1971; Seed 1979; Ambraseys 1988; Suzuki et al. 1995; Arango 1996; Andrus and Stokoe 1997; Olsen 1997; Youd and Noble 1997; Robertson and Wride 1998; Youd and Idriss 2001). The paper by Youd and Idriss (2001) is a summary of commonly used procedures and provides recommendations for design.

General Design Considerations

All waste dumps have some risk of instability, whether due to an inadequate design process or unforeseen variability of assumed parameters. The issue of addressing uncertainty in geotechnical design has been discussed in depth by numerous authors (Duncan 2000; Christian 2004; Whitman 1984; Christian et al. 1993). The trade-off between the costs of a thorough geotechnical investigation versus the risks of design uncertainty has long been a challenging management decision in geotechnical projects. For mine sites, significant investment is typically made in exploration and estimating mineral resources, and the geology of a mine site is often more thoroughly documented than other types of geotechnical projects. Nevertheless, the engineering properties of the soil and rocks relevant to slope stability receive less emphasis. Baecher and Christian (2003) observed that the areas of geotechnical concern, such as slopes and waste disposal facilities, are usually associated with mine costs rather than revenue, and, therefore, significantly less money is devoted to their site characterization and laboratory testing.

One may ignore the uncertainties involved in a design, take a conservative approach, rely on observational methods (Peck 1969), or attempt to quantify the uncertainty. Geotechnical projects, in general, may include a combination of these methods.

Factor of safety. The most common way to take the conservative design approach is to require a minimum calculated FS for slope failure. The methods used to calculate the FS are described in detail in Chapter 8.3. The minimum FS selected for design allows for some margin of error between the assumed conditions and those that actually exist in the field, and should consider the following, as outlined by Vandre (1980):

- Consequences of instability
- Thoroughness of the geotechnical investigation
- Reliability of the design assumptions
- Ability to predict adverse conditions
- Possible construction deviations from design
- Engineering judgment based on past experience

The FS is calculated for normal loading conditions, as well as for seismic loading when the project is located in a seismically active area. In general, a minimum FS of 1.3 (for shallow failures) to 1.5 (for more significant failures) is considered acceptable for long-term (static) conditions (NAVFAC 1982; Vandre 1980). The FS required for extreme adverse conditions, such as the design seismic event or temporary slopes, is typically lower than that required for long-term stability of final waste slopes, and a range of 1.1 to 1.3 is generally accepted.

Reliability. For significant structures, such as waste dumps and heap leach pads, it is critical that sources of uncertainty in the stability analysis be acknowledged early on and considered in the overall design approach. As with any project, economics and other physical constraints such as space limitation do not always allow for an overly robust design. In an effort to quantify uncertainty and provide a level of confidence in the safety and reliability of a design, probabilistic methods have been developed and implemented in many slope-stability software packages. Reliability methods are often used in the design of open-pit mine slopes but not as commonly in designing heap leach pads and waste dumps. When selecting appropriate values for the input parameters of the stability analysis, the level of uncertainty in the data and the assumptions that are made must be clearly identified and considered in the design.

Simplified deformation analyses. Analyses may also be performed to evaluate seismically induced deformations. The pseudostatic analysis method can be used to calculate the yield acceleration of the sliding mass. This yield acceleration may then be used in simplified procedures for estimating earthquake-induced deformations, such as those provided by Makdisi and Seed (1978) and Bray et al. (1998). Determination of acceptable deformation limits may depend on several factors, such as regulations, engineering judgment and previous experience, and acceptable risk.

In summary, slope failure may occur in waste dumps by a variety of failure modes, which include surface slumping, shallow flow slides, rotational circular failures, base spreading, block translation, and liquefaction. In geomembrane-lined heap leach pads, slides typically occur by wedge failure along the critical interface of the liner system. Engineering judgment and experience must be used when selecting the appropriate analysis method for these potential failure modes, as well as when selecting input parameters for the dump materials and foundation. The reliability of the stability analysis results depends on whether the design assumptions are representative of the actual waste dump conditions.

Settlement

Waste rock settlement occurs because of particle reorientation, weathering of high clay-content materials, weakening of inter-particle bonding due to water, and transport of fine particles through the dump (Williams 2000). The rate of settlement is affected by dump height, loading rate, location within the dump, and material type (Zavodni et al. 1981). Settlement is more predictable and usually less in layered dumps than in end-dumped embankments.

During placement of the waste material, initially self-weight settlement may occur or crest settlement may happen because of compaction or surface sloughing from oversteepening (Zavodni et al. 1981). After waste placement, primary settlement and creep settlement occur at a decreasing rate with time and have been shown to continue for more than 10 years after dump construction (Williams 2000). The majority of the settlement, however, occurs within the first months after construction (Zavodni et al. 1981).

As the dump materials become saturated, there is a reduction in strength, and collapse settlement may occur (Williams 2000), especially in loose, end-dumped waste piles. The potential for collapse can be minimized with adequate compaction (Vandre 1980).

Under dry conditions, settlements of 0.3% to 7% of the waste dump height have typically been reported (Naderian and Williams 1996). However, settlements of more than 20% of the total dump height have also been documented (Zavodni et al. 1981).

Various techniques can be used to monitor deformations of waste dumps with time. These methods include onsite inspections, surveying, photogrammetry, extensometers, inclinometers, settlement cells, and laser beacons (McCarter 1985b). The appropriate monitoring methods are selected based on the waste dump height, material, and method of construction. Robertson (1982) describes the development and operation of effective waste dump monitoring systems.

Seepage and Drainage

The same fundamental seepage principles used in the design of earth dams and levees should be considered in the design of waste piles and tailings storage facilities (Cedergren 1989). Understanding fluid flow through waste dumps is important for evaluating both stability and environmental risks. Most mine waste dumps and leach piles are usually unsaturated, and accurate seepage and contaminant transport modeling requires determining unsaturated soil properties (Fredlund et al. 2003). However, unsaturated soil behavior is less understood than saturated behavior, and unsaturated properties and flow modeling are not always included as part of the waste dump and heap leach design. In fact, most geotechnical seepage calculations are based on saturated soils. The fundamentals of seepage through porous media are explained in detail in Chapter 8.2. The soil properties used in unsaturated flow modeling are briefly introduced here.

The soil parameters used in unsaturated flow modeling are derived from nonlinear equations using laboratory test data and are generally referred to as the hydraulic conductivity function and the water storage function. To model seepage through an unsaturated pile, these functions are required for each material in the flow path (Fredlund et al. 2003). Various methods of determining unsaturated soil parameters for input in waste dump models are described in detail in Fredlund et al. (2003). Some of these methods are also summarized here.

The hydraulic conductivity function (HCF) represents the conductivity of the unsaturated material at various water contents. The HCF can be measured in the laboratory or estimated using the methods of Brooks and Corey (1964), van Genuchten (1980), Campbell (1973), and Fredlund and Xing (1994). Many software packages allow users to select one of these methods when entering input parameters into the seepage model. Soil–water characteristic curves (SWCCs) represent the relationship between the water content of the soil and the soil suction, and can be measured in the laboratory using a variety of devices. The SWCC is also used to determine the water storage function, which relates the change in water content to the change in soil suction. This relationship becomes highly nonlinear as the soil desaturates (Fredlund et al. 2001).

The saturated hydraulic conductivity represents the limiting condition for unsaturated flow and is generally measured as such in the laboratory. However, if laboratory data are not available, there are multiple methods for estimating the saturated hydraulic conductivity of a material indirectly. The formulas typically relate the hydraulic conductivity to the grain size distribution of the material. Some of the available methods include those by Hazen (1892), Kozeny (1927), Carman (1938, 1956), Rawls and Brakensiek (1989), Alyamani and Sen (1993), and Sperry and Pierce (1995).

Design and construction elements can significantly affect seepage and drainage through waste dumps. The top surface of the waste dump should be graded to prevent surface water from flowing onto the slopes. Since the 1990s, geosynthetic *raincoats* have been used on heap leach pads in high-rainfall areas to minimize storm runoff flows into the collection ponds (Breitenbach 2004; Smith 2008). These raincoats also serve as protection against erosion and damage to the agglomerates (Breitenbach and Smith 2007a).

When waste rock is dumped, the coarsest fraction often ends up at the bottom of the dump, creating a *rock drain* at the base. Depending on topographic details, such rock-fill drain sections can be significantly large and a useful tool for controlling flow, especially in places such as valley bottoms where a watercourse already passes. If the flow capacity of the rock drain is exceeded, the phreatic surface may rise, lowering the stability of the waste dump. Therefore, understanding the hydraulic behavior of rock drains is important for waste dump design. Hansen et al. (2005) have provided some insight into this issue. Additionally, the Rock Drain Research Program was completed in Canada to study the characteristics of rock drains and their environmental effects (Fitch et al. 1998).

In heap leach pads, a properly designed and operating solution collection and liner system is critical for retrieving pregnant leach solution, as well as for controlling phreatic surface levels within the heap. The most versatile and preferred liner system currently used for heap leach pads consists of a low-permeability soil layer overlain by a geomembrane with a drainage layer of crushed rock (overliner) on top of it (Breitenbach 2000). However, in the drier and remote areas of South America, the geomembrane with the overliner is generally considered adequate. The geomembrane liners are specified by their material type, thickness, and surface roughness, and are designed on the basis of initial and final loading conditions and the expected strains produced in the liner. A properly selected overliner or drainage material and a stringent construction quality-assurance program during installation are crucial to performance of a liner system. The overliner material is specified in terms of gradation, or maximum and minimum particle size, in order to avoid puncturing the geomembrane, provide adequate support to the leachate collection pipes, and facilitate adequate drainage. Key concerns for liner system selections are summarized in Table 8.11-1.

Erosion

Erosion is a natural process that cannot be stopped, only controlled. Erosion on material stacked at the angle of repose can be hazardous, because of the risk of material failure and catastrophic movement downslope, as well as sedimentation and contamination of downstream waters. Reclamation and closure of waste dumps or piles usually requires regrading for reduction in slope and seeding of vegetation. Both of these efforts will dramatically reduce erosion. Large dumps, mounds, or piles are designed to control and collect runoff and prevent material failure. The final reclaimed landform is also an important element in long-term erosion control.

It is much easier and less costly to avoid contamination before it occurs than to clean up after the fact. Because of this, regulators and industry are designing facilities that, from their inception, reduce the potential for harmful effects to the environment.

Table 8.11-1 Key concerns for liner system selection

Engineering and Design Concerns	Construction Concerns
• Liquid containment: liner integrity • Operational and closure stability: interface friction strength, flexibility, nonplanar anchorage • Chemical and temperature compatibility • Subgrade and overliner: gradation, permeability, lift placement, compaction, surface preparation • Long-term exposure: ultraviolet (UV), oxidation/aging, animals, and other biological attack • Puncture resistance: subgrade and overliner fill type • Flexibility: differential foundation settlement, installation, puncturing • Tensile, tear, and seam strength: liner uniformity, thickness • Contact between composite geomembrane liner and low-permeability clayey soil subgrade	• Shipping to site: container rolls versus boxes • Installation, deployment, and seaming • Ease of repair: local liner expertise, equipment • Site access: storage area, perimeter access, slopes • Dynamic and static loading conditions: cover fill, roads, traffic, ultimate load • Weather and climate: UV, wind, rain, ice, temperature changes, stress cracking, expansion/contraction • Cold weather installation and cover: frozen subgrade, safety • Grade change adaptability: steep grade gravitational forces, corners, benches, pipe boots • Tie-ins for expansion facilities • Overall cost to construct: materials, labor, schedule

Source: Mark E. Smith and RRD International (Adapted from Smith 2008).

The erosion potential of the waste needs to be characterized, and, where possible, higher erosion-potential material should be capped with a material that has lower erosion potential. An example is the sodic waste rocks in Australia.

Rock Dumps

Rock dumps are generally designed with a slight grade on the top deck to allow rainfall to flow to a collection system and be conveyed to a collection pond. A similar system is placed at the toe of the dump as well. The water in the ponds is tested regularly and treated if required. The collection systems are normally part of the larger mine-wide stormwater control plan. A good stormwater management plan, which will prevent ponding of water against the safety berm at the crest of the dump, helps avoid washouts of the slide slopes on active rock dumps. Generally, the operational toe of a dump will be offset to accommodate the ultimate toe of the reclaimed dump, at a 2:1 (horizontal to vertical) or 3:1 slope, allowing room for minor ravel and washouts on the side slopes.

Another important erosion consideration for slopes is their shape. Concave slopes can reduce erosion (McPhail and van Koersveld 2006). Naturally eroded features are concave in shape, and by emulating this with wider catch benches on the lower elevations of the dump or pile, eroded material from the upper levels is slowed and deposited on the lower levels.

The configuration of the dump design is an important consideration in stormwater management planning. The run-on controls for a valley fill are much more complicated than for a ridge crest or heaped dump, as the entire design storm flow of the drainage needs to be conveyed around the dump or pile. In all cases it is important to keep rainfall from native ground separate from what falls on the dump, as the latter may be contaminated, whereas the former should not be.

Erosion has typically been modeled in civil applications using the U.S. Department of Agriculture Universal Soil Loss Equation, of which there are several variations, including the original Universal Soil Loss Equation (USLE), the Modified USLE, and the Revised USLE. However, this model is for agricultural situations where the slopes are much flatter than those used for waste rocks dumps. Several computer codes such as SIBERIA and CEASAR are being used that utilize digital terrain models and mathematical algorithms to predict both erosion and deposition. The various versions of the USLE calculate erosion loss only. These codes have their own disadvantages as well, such as the need for rigorous calibration (Hancock 2009).

Leach Pads

Leach pads are designed to allow leaching solution to pass through the stacked material, which is then collected on a liner system with collection pipes to be conveyed to the process facility. These systems catch and collect all meteoric water as well, and the ponds must be sized to capture a design storm. A detailed *water balance* is usually calculated to size the ponds and to understand the water needs of the pad and the process facility. In a properly designed leach pile, the material stacked on the pad should have a high enough infiltration rate to prevent excessive solution flowing on the surface and on the side slopes. Any such flow is captured in lined trenches around the pad.

ACID ROCK DRAINAGE

Acid rock drainage (ARD) occurs whenever unoxidized sulfide material is exposed to the atmosphere and water. Dumps, piles, or stacks of material are particularly susceptible to ARD due to the permeability of the material, availability of atmospheric oxygen, and amount of material that can come in contact with meteoric and surface water flow. This topic is covered in detail in Chapter 16.5 and is briefly touched upon here.

Design criteria for ARD prevention for large dumps or mounds include chemical characterization of the material, acid–base accounting (ABA), compartmentalizing the dump/pile into discrete cells for material buffering control, and run-on/runoff control.

Chemical Characterization

Some large mines in Nevada (United States) use ABA and build the waste rock facility to confine potentially acid-generating material in cells composed of acid-consuming material. This creates a net acid-neutralizing environment. In order to do this, good characterization of the material needs to be completed. With modern production analytical capability and mine dispatch systems, material that is not ore can be characterized and routed to a specific location on the dump. If the material balance is not net acid neutralizing based on the ABA, the dumps may need to be placed on a low-permeability layer and capped upon closure.

Waste characterization can also include tests for total and soluble metals, such as the U.S. Environmental Protection Agency's (EPA's) toxicity characteristic leaching procedure and the State of California's waste extraction test. Testing for pH in water flowing from dumps is important, because lower-pH water is more likely to contain metals that have been leached out of the waste rock.

Run-On and Runoff

The run-on component of meteoric water is controlled based on a stormwater management plan. Stormwater collection systems need to be well thought out and based on the mine plan, topography, and required maintenance. A mine-wide

stormwater management plan is required for mines in the United States, and these plans are site specific.

The runoff component of meteoric water is controlled by the waste facility's design and may be included in the overall stormwater plan. Important considerations for the runoff plan include material classification, treatment requirements, and appropriate sizing of ponds and catchments. In the case of a waste rock dump, water that infiltrates the dump will be contained at the toe of the dump. Only the surface flow will be contained on the top.

CLOSURE AND RECLAMATION

Closure and reclamation of heaps and piles are necessary for environmental, ecological, and health and safety reasons, and in some instances for economic reasons such as to recover bonds posted during the permitting or construction phase.

The closure process typically requires detoxification of heap leach facilities, and reclamation usually means decreasing the slopes of heaps and dumps, covering the area with growth media, and reseeding vegetation where appropriate. Rock dumps that have acid drainage issues require ongoing treatment of the water flowing from these facilities. Detoxification of leach pads is required in the United States and typically involves lengthy periods of rinsing to reduce the cyanide or other toxic content of solutions circulating in the heap.

Sloping the waste facility to moderate slopes of 3:1 (horizontal/vertical) or flatter is typically required. Capping of dumps and leach facilities with semipermeable capping material allows for the establishment of a growth medium for planting vegetation, which is the best way to prevent erosion. As discussed previously, the final landform should be concave if possible, with shallower slopes at the base of the facility. In addition, many agencies are requiring certain randomness to the final landform, avoiding stretches of linear slopes and ridges. Many operations do concurrent reclamation, where they slope and plant segments of their facilities to reduce the overall operating footprint and possibly to recover a portion of their bond.

An important factor to consider is the longevity of the closure system. Many waste facilities are looking at closure periods in hundreds of years, and waste facilities containing radon or other radioactive material are looking at thousands of years of containment. Natural material will last longer than synthetic materials, and this must be considered in the design of a facility handling radioactive material.

RADIOACTIVE WASTE ROCK

Some waste rock can be radioactive and may require special design considerations. Uranium mill tailings have received a lot of attention because of their radioactive properties and as a result are designed for long-term disposal. Phosphate mining and processing produce phosphogypsum tailings, which may also contain trace levels of radioactive material (FIPR 2010).

Phosphogypsum tailings have been used as fertilizers and for other uses. However the EPA has banned the use of phosphogypsum with an average radium-226 concentration of >10 pCi/g (picocuries/gram) for agricultural application (FIPR 2010). As a result of phosphate mining, currently 0.909 billion t (1 billion st) of phosphogypsum waste materials are stacked in the state of Florida, and about 27.3 million new metric tons (30 million short tons) are generated each year.

Uranium tailings contain low levels of radioactive radium-226. Ra-226 has a half life of 1,620 years and decays into the odorless and colorless gas radon-222, which has a half life of 3.8 days. Inhalation of Ra-226 is known to lead to lung cancer. Because of the radioactive properties of uranium tailings, the standard practice is to design the impoundments for long-term disposal, typically 1,000 years. To avoid erosion over this type of time frame, slopes of the piles need to be minimized, and natural forms of containment should be utilized.

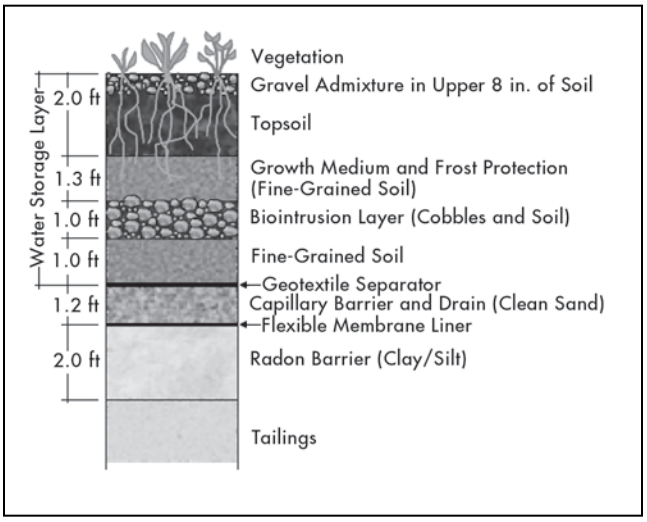


Figure 8.11-4 Typical cover for uranium mill tailings

In the United States, the design of uranium tailings impoundments and covers falls under regulations in the Uranium Mill Tailings Radiation Control Act of 1978. These regulations require that a cover be designed to produce reasonable assurance that the radon-222 release rate does not exceed 20 $pCi/m^2/s$ for a period of 1,000 years to the extent reasonably achievable, and in any case for at least 200 years when averaged over the disposal area for at least a 1-year period. In some cases at inactive sites, the regulations allow for a radon concentration of <0.5 pCi/L above the background concentration. The regulations also state that the tailings should be disposed of in a manner that no active maintenance is required to preserve the conditions of the site.

The typical cover includes, from bottom to top, the following layers (thicknesses are variable):

- 0.61-m (2-ft) radon/infiltration barrier consisting of clay
- 0.15-m- (0.5-ft-) thick capillary break layer consisting of coarse sand/fine gravel
- 1.07-m- (3.5-ft-) thick water storage soil layer consisting of fine-grained soil
- 0.15-m- (0.5-ft-) thick surface erosion protection layer (soil/rock mixture) consisting of 80% soil, 20% riprap boulders
- Vegetated surface for water balance control

The actual thickness of the radon/infiltration barrier in a specific case would be based on calculations of radon flux at the surface of the compacted soil layer. An example design is shown in Figure 8.11-4. The soil type would be selected from available borrow sources that can satisfy performance requirements for permeability and radon attenuation. The compaction requirements would be determined with tests and calculations of saturated hydraulic conductivity and radon attenuation.

A uranium mill tailings cover calculator is available on-line at www.wise-uranium.org/ctch.html. This calculator

determines the radon fluxes and concentrations in multilayer uranium mill tailings and cover systems, and optimizes the cover thickness to satisfy a given flux constraint. The calculator is a clone of the RAECOM (Radiation Attenuation Effectiveness and Cover Optimization with Moisture Effects) code (Rogers and Nielson 1984). The input data include

- Radium-226 activity concentration (if the value is unknown, it can be estimated from the grade of ore processed in the uranium mill);
- Radon-222 emanation fraction (the fraction of the total amount of Rn-222 produced by radium decay that escapes from the soil particles and gets into the pores of the soil);
- Radon-222 effective diffusion coefficient;
- Porosity;
- Moisture content; and
- Minus #200 sieve fraction.

Typically, the effective diffusion coefficient of radon in unconsolidated soil material with low moisture content is in the order of 1.0^{-6} m^2/s (1.08^{-5} ft^2/s). The upper limit is represented by the radon diffusion coefficient in open air, which is approximately 1.1×10^{-5} m^2/s (1.18^{-4} ft^2/s). At the lower extreme, in a fully saturated soil material, the radon diffusion coefficient may be as low as 1.0^{-10} m^2/s (1.08^{-9} ft^2/s).

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PART 9

Infrastructure and Services

CHAPTER 9.1

Electric Power Distribution and Utilization

Reinhold A. Errath

POWER DISTRIBUTION IN MINING

This chapter introduces the engineer to electrical power use in mining and concentrator plants, drawing upon mining equipment designs of the last decade.

Today's mine power systems are complex and subject to numerous technological and environmental constraints. It is no longer possible to treat the subject with the indifference shown in the past. Power distribution equipment is partly stationary and partly mobile, and subject to extreme levels of dust, moisture, and vibration. Mining and concentrator machinery create electrical loading that is often cyclic and variable. Designing and maintaining such an electrical distribution system is challenging and demanding, and requires specialist knowledge of both mining and electrical engineering. Effective mine management requires that those who are responsible for production and safety be familiar with the mine's electrical distribution system.

Typical mining and concentrator-plant power requirements today are ~120 MW for a 100,000-t/d (metric tons per day) plant. In the utility substation, power is transformed from transmission distribution voltage to commonly 22 to 33 kV, and then carried to major load areas to supply large industrial consumers directly or to power distribution substations that step the voltage down.

It is the responsibility of the mining company to select the voltage best suited to its needs. The choice depends primarily on the amount of power purchased. It is not safe to assume that a power company can serve a large mine complex from existing primary distribution lines or even from the transmission system. The problem stems from the fluctuating nature of mine loads. For example, large excavators in surface mines can require high peak power for a short time, followed by regenerative peak power, all cycling within a span of 45 seconds. Fluctuating loads can create voltage and frequency variations beyond the limit set for other utility customers. Similarly, most large draglines and shovels require 4.16 to 30 kV to start up.

History of Electrical Use in Mines

In the late 1800s, electricity was introduced into mines in the form of direct current (DC) for rail haulage. Direct current was used because most systems at that time were powered by DC generators. Direct current has a number of advantages for haulage, the most outstanding of which is that series-wound DC motors have excellent traction capabilities.

In the early 1920s, the first electrically driven mining machine for underground use, the coal cutter, was introduced. This was followed almost immediately by the loader, also driven by DC motors. In mines with rail haulage, trailing cables supplied power to the machines from trolley wires and rails.

In the late 1930s, the next significant increase in power consumption came with the introduction of the shuttle car. The car was battery powered at first. Later, an automatic reeling device to handle a trailing cable was added to overcome the deficiencies of batteries; the cable was connected to the haulage power system. This equipment, when combined with the cutters and loaders, placed additional demands on the DC distribution system.

In the late 1940s, DC was again expected to provide the power as continuous mining machines began to be used extensively in coal mines. However, continuous mining equipment typically consumed more energy than did the various conventional mining machines that they replaced. Consequently DC now proved unsatisfactory in most cases, and the high current demand created enormous voltage drops in the distribution system. The DC supply system was therefore separated from the haulage system, but even this proved inadequate. Voltages at the machines were so far below rated values during peak operating periods that even moderate efficiency was unachievable. The increasingly large cable sizes required to supply the needed power were difficult to handle. The use of three-phase alternating current (AC) motors and distribution was the obvious answer.

Mines quickly converted from DC to AC for both distribution and high-horsepower loads in mines when higher voltages were permitted by the authorities. As a result of these conversions, mine power systems generally now have two voltage levels: distribution and utilization. The substation transforms the utility voltage down to distribution levels. Power is distributed through conductors from the substation to the power center (sometimes called the load center). The

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power center is either a portable switch room or fixed installed "e-rooms" where voltage is transformed to low-voltage utilization levels: today typically 660 or even 480 V. However, despite this reference to voltage levels, the terms *distribution* and *utilization* describe functions of the power system, not specific voltage ranges.

Originally, primary AC distribution was made at 2,300 or 4,160 V. In most mines, these levels were later increased to 7,200 or 13,200 V. The principal reason for the increase was that, at higher voltages, the same load draws correspondingly less current. Cables can thus be smaller, which reduces costs and also distribution losses. From the beginning, 480 V AC was most popular for utilization, despite the fact that drive powers progressively increased as continuous mining proved successful. The downside was that additional drive power required larger trailing cables. Most operations compensated by raising the rated motor voltages to 550 V. Recently, manufacturers have produced machines with 950-, 2,400-, and 4,160-V motors to further overcome problems with trailing cables.

Whereas series-wound DC motors were universally used for underground rail haulage, the first large motors used for surface mining were shunt-wound DC because of their constant-speed characteristics. Initial distribution for electric shovels was also DC.

Technological advances soon made AC power systems superior and AC motors were tried with some success. However, by 1927, AC/DC motor–generator (MG) sets and the invention of the Ward–Leonard (W-L) control concept caused these efforts to be abandoned. The new control system enabled motor speed to be varied by altering the armature voltage while maintaining a constant voltage across the shunt field. The MG sets functioned as onboard power-conversion units, thereby establishing the use of AC distribution in surface mines. Sets driven by synchronous or induction AC motors, W-L control, and DC motors established the standard, and even now the combination is still used on older mining excavators.

The W-L concept is no longer used for new installations. Rather, the standard today is the frequency converter and the AC motor. The multiple-drive (sometimes called multidrive) configuration, which is standard now for shovels with more than one motor, is a fully 4Q operating system, where 4Q signifies four quadrants—that is, motor and generator modes in both directions of rotation.

Power Terminology

Several terms are used to describe the operation of a power system. These terms are applicable not only in system design, operation, and maintenance but also in utility company billing.

The sum of the electrical ratings for all equipment in an electrical operation gives the total connected load, expressed in kilowatts (kW), kiloamperes (kA), kilovolt-amperes (kVA), or amperes (A). Many loads operate intermittently, especially for mining equipment, with varying load conditions. Accordingly, power demand is frequently less than the connected load. This fact is important in the design of a mine power system. The system should be designed for what the connected load actually uses, rather than the total connected load. Obviously, these considerations have great impact on power system investment.

IEEE has standard definitions for load combinations and their ratios because of the importance of assessing equipment power demands. The most important definitions are the following (IEEE 141-1986):

- **Demand:** Electrical load for an entire complex or a single piece of equipment averaged over a specified time interval. The time or demand interval is generally 15 minutes, 30 minutes, or 1.0 hour. Demand is generally expressed in kilowatts, kilovolt-amperes, or amperes.
- **Peak load:** Maximum load consumed or produced by one piece or a group of equipment in a stated time period. It can be the maximum instantaneous load, the maximum average load, or (loosely) the maximum connected load over the time period.
- **Maximum demand:** Largest demand that occurs during a specified time period.
- **Demand factor:** Ratio of the maximum demand to the total connected load.
- **Diversity factor:** Ratio of the sum of the individual maximum demands for each system part or subdivision to the complete system maximum demand.
- **Load factor:** Ratio of the average load to the peak load, both occurring in the same designated time period. This can be implied to be equal to the ratio of the actual power consumed to the total connected load in the same time period.

For example, consider a power cable supplying several mining sections in a mine or concentrator plant. The sum of the connected loads on the cable, multiplied by the demand factor for these loads, yields the maximum demand that the cable must carry. When applied to current, this is the maximum amperage. Good demand factors for mine power systems are in the range 0.7–0.8, depending on the number of operating sections; the lower value is used when there are fewer producing units.

The demand factor can be extended to include estimates of average load. For instance, the sum of the average loads on a cable multiplied by the demand factor provides the average load on the cable. A prime application here is for approximating the current that a conductor is expected to carry. The demand factor and the diversity factor can be applied to many other electrical applications, such as estimating transformer capacities, protective circuitry continuous ratings, and the load that a utility company must supply.

The load factor can be used to estimate the actual loads required by equipment. Here the total connected load multiplied by the load factor yields an approximation of the actual power consumed. The average load factor in underground mining tends to be low, due mainly to the cyclic nature of machine operation but also to the use of high-horsepower motors for performing specific functions, sometimes for only a small fraction of the possible running time.

The peak load is normally one basis that utility companies use to determine power bills. Most often, one month is the specified time period. Demand meters are often installed at the utility company's metering point.

Design Criteria

The goal of the engineer is to provide an efficient, reliable electrical system at maximum safety and for the lowest possible cost. The types of information made available to the engineer include the following:

- Expected size of the mine
- Anticipated potential expansion
- Types of equipment to be used
- Haulage methods to be used

- Available power from the utility company
- Amount of capital assigned for the electrical system

Designing a safe, reliable electrical power distribution network for an industrial plant requires detailed planning to avoid installation, commissioning, and operational problems and limitations of future plant expansions. Basic questions regarding mine operation scenarios need to be answered, and careful attention must be paid to the following:

- Location of equipment, altitude, and earthquake zone (for installations at >1,000 m above sea level, design must be suitable for operation at higher altitudes; in earthquake-prone areas, design must be as specified in the *International Building Code* [ICC 2009], which replaced the older Uniform Building Code)
- Safety of electrical technicians and other personnel, reliability of operation, simplicity, maintainability, and selective system operation
- Voltage regulation (to handle fluctuations in voltage resulting from changes in electrical load), high/medium/low voltage levels (designated HV/MV/LV) and breaker technology, transformer sizes and impedances, protection monitoring, and control philosophy
- Load flow and network harmonic studies, harmonic distortion levels at the point of common coupling (PCC), electromagnetic compatibility (EMC) issues, and power factors
- System efficiency, grounding and lightening, and potential for expansion

Of these, safety, reliability, and simplicity are closely related, depend on good preventive maintenance, and are of vital concern in the cramped, inhospitable environment of a mine operation, especially in underground mines. Routine and dedicated maintenance should be performed only by electrically trained personal, and training for these tasks must be provided.

Importance must be paid to the mean time between failures (MTBF). The failure rate increases as the number of installed components in a system increases. Similarly, the mean time to repair (MTTR) must also be carefully considered. Safety is enhanced with reliability measures such as adequate interrupting capacity, current-limiting capability, and selective system operation. Adequate interrupting capacity and current-limiting capability ensure protection during a disturbance. Current limiting, when applied to grounding, is perhaps the most significant personnel safety feature of mine electrical systems. Selective system operation is a design concept that minimizes the effect of system disturbances.

Although initial cost is important, it should never be the determining factor in the design of a mine's power system. High-cost equipment can easily offset first costs by reducing operating costs when designs maximize safety and reliability.

Using the data available, the task of the mine electrical engineer is to

- Select one combination of power equipment;
- Provide power or circuit diagrams;
- Estimate equipment, operating, and maintenance costs;
- Set system specifications; and
- Receive and assess proposals from suppliers.

The engineer must possess a firm knowledge of mine power systems and operating plans. No two mines are exactly alike, so there can be no standard mine electrical system. The engineer must resort to fundamental concepts, an awareness of what has worked in the past, and a clear understanding of technological, operational, environmental, and safety constraints.

Power Supply and Distribution Equipment Solutions

The increase in power plant size in the last decade has been accompanied by an evolution in mine power systems toward higher complexity. Key aspects of concern must now include the following:

- Overhead power transmission
- Voltage level
- HV substation
- HV, MV, and LV technologies

Overhead Power Transmission

Power requirements for a 100,000-t/d plant can typically reach 120 MW as a consequence of almost yearly increases in grinding-equipment size involving, for example, huge mine hoists, transport belts, and pumps. In addition, concentrator plants are usually built in remote areas where power supply and network conditions are not always adequate. The trend in load-behavior characteristics over the last few years, from network linear loads to nonlinear loads, requires sufficient network short-circuit power capability to operate the electrical equipment correctly. Adjustable-speed drives constitute 80% of the total installed power in today's modern mine operations, resulting in 70–90-MW nonlinear loads from 120 MW of installed power.

The short-circuit power at the PCC defines whether a plant is normally operable or special measures must be taken. The best engineering practices specify the following:

- If the short-circuit power at the PCC, in megavolt-amperes, is a factor of >6 times the total nonlinear installed power in megawatts, no special measures need be taken. If the factor is <6, consideration must be given to implementing special filter and compensation units, voltage compensators, or, in the worst case, rotating synchronous condensers.
- The minimum short-circuit power at the PCC must be ≥400 MVA for a 100,000-t/d plant. The maximum short-circuit level must be kept within the breaker-protection capability.

There are in principle four remedial approaches.

Approach 1: Replace single lines with two parallel lines. For plants in remote areas, overhead power-line lengths from the next power station can easily be up to 400 km. The transportability of electrical energy is reduced significantly in such cases because of the inductive resistance of the line. A solution is to run not just single lines but two parallel lines (Figure 9.1-1A). For example, a single overhead line whose short-circuit power is 700 MVA at the beginning of the line furnishes, at 300 km, only 40% of its original value (280 MVA); two parallel lines furnish 60% of their original value (420 MVA).

Approach 2: Increase voltage in the lines. An alternative method of boosting short-circuit power in overhead lines is to increase the line transmission voltage (Figure 9.1-1B). For example, a 230-kV overhead line whose short-circuit power is 700 MVA at the beginning of the line furnishes, at 300 km, only 40% of its original value (280 MVA); a 400-kV line furnishes 70% of its original value (490 MVA).

Approach 3: Use direct current. Another solution, for distances of >600 km, is to use DC transmission (Figure 9.1-2).

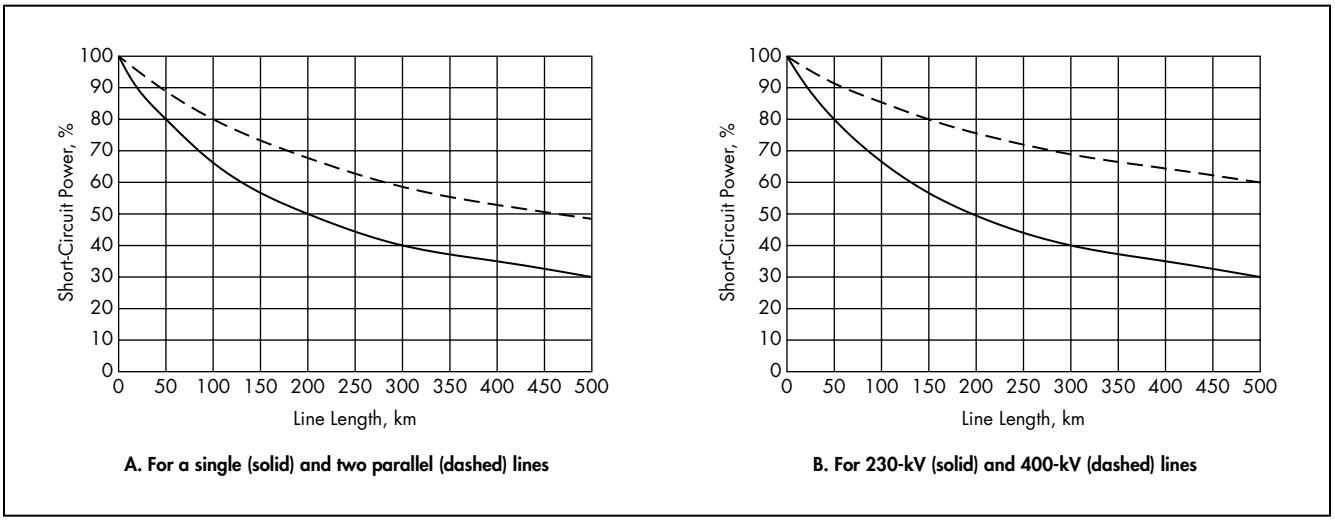


Figure 9.1-1 Short-circuit power as a function of overhead line length

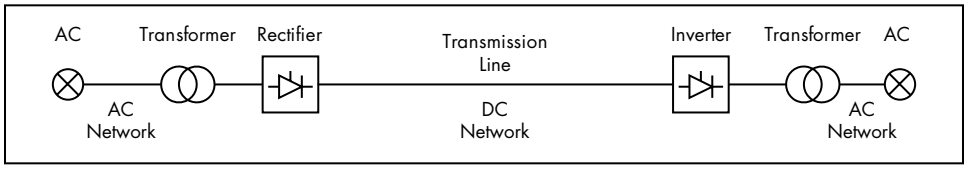


Figure 9.1-2 DC transmission

DC transmission by overhead line or by cable at high voltage (750–1,000 kV) is an alternative at these distances. Relevant factors are the voltage drop from the beginning to the end of the line and the ohm resistance (there is no inductive line resistance). A significant advantage over AC transmission is the nonattenuation of short-circuit power along the line.

Approach 4: Produce supporting power directly at the plant. Power production directly at the plant is technically advantageous. First, there is no attenuation; full power is available at the plant. Second, network short-circuit power on the PCC is improved. This approach makes sense for plants located in remote areas. For example, an additional on-site power generator of 50 MVA increases network short-circuit power to ~350 MVA and solves many problems that are characteristic of weak networks. However, the advantages of this solution are largely outweighed by the disadvantages, including the need to transport fuel to remote areas and the logistical and technical problems at high altitudes.

Voltage Level

The network connection voltage is generally determined by the power utility and depends on the required power of the new facility, capacity of the local network, location of utility substations, and utility network voltages. Typical loads that utility authorities allow for connection to their system are listed in Table 9.1-1.

Utility planning authorities dictate their requirements and rules in most cases when permitting connection to their electrical grid. Typical considerations include the following:

- Ownership of the substation
- Total harmonic distortion limits (set by the authorities to typically <5%)

Table 9.1-1 Typical loads allowed by utility authorities

IEC* Connection Voltage, kV	NEMA† Connection Voltage, kV	Load, MW
11	13.8	20
22	24	20
33	38	40
66	72	60
132	145	150
220	260	200
500	550	500

*IEC = International Electrotechnical Commission.
†NEMA = National Electrical Manufacturers Association.

- Power factor requirements (set by the authorities to typically >0.9)
- Maximum power drawn
- Short-circuit contribution from plant to grid
- Equipment and installation standards that must be fulfilled
- Protection and metering requirements
- Load-shedding systems

HV Substation

After the coupling voltage has been determined with the utility, the HV substation can be considered. Factors to consider include the following:

- Current requirements
- Lighting and voltage insulation levels
- Short-circuit levels
- Environment

Table 9.1-2 Typical loads allowed by utility authorities for HV switchgear and associated equipment

Connection Voltage, kV	Load, A	Short-Circuit Level, kA
66	3,150	40
132	3,150	40
220	3,150	40
500	3,150	40

- Site conditions (potential pollution, earthquake, and atmospheric conditions)

Common engineering practice in the last two decades is to indicate the most-used configuration in terms of load and short-circuit levels for HV switchgear and associated equipment, as shown in Table 9.1-2.

The substation should be located as close as practical to the load without compromising safety and access to the site and plant during construction and future expansions. The location should be chosen so as to minimize contamination from dust sources such as stockpiles and from airborne chemicals. To minimize power outages, redundant incoming power supplies should be considered.

Use of underground cables and surge arrestors rather than overhead line construction should be considered when designing the substation. This solution minimizes exposure to lighting strikes and provides a good safety margin for inadvertent accidental contact with live HV connections by heavy machinery or personnel.

Depending on the configuration from the overhead line (i.e., whether there is one line or two lines from different sources), often multiple HV/MV transformers, 50 to 80 MVA each, are installed. On the MV side, the transformers also feed three or four distribution systems that are connected by selectable tie breakers. This configuration covers all plant operational conditions including failure of one of the transformers. Because of the operational flexibility with this configuration, it is essential that a switching concept for the tie breakers be implemented. The concept must respect not only the plant load but also power factors and network harmonics.

Usually automatically operated tap changers are installed on the primary side of the transformers. Tap changers provide the correct voltage to the plant and mine consumers, and compensate for voltage variations in the HV feeder line or voltage changes due to varying load requirements in the plant. Tap changers are usually switchable in ±2.5% and 5% steps. It is critical that the function of the tap changers be coordinated as soon as the distribution bars on the secondary side are connected (i.e., the tie breaker is closed). One tap changer must be designated master and the others slaves. Later, if more plant loads are added, it can be relatively simple to boost the transformers for more power with forced cooling.

A delta primary connection and wye (Y-shaped) secondary connections are preferred for standard two-winding power transformers, and are commonly used in mine power centers. The wye secondary connection provides an easy means for resistance grounding. The delta primary connection isolates the distribution circuit from the utilization circuit with respect to ground currents. The delta–wye connection stabilizes the secondary neutral point and minimizes production of harmonic voltages. In certain situations, a delta secondary connection may be specified and the primary connection may be a delta or a wye. If neutral grounding is desired or required, a grounding transformer is needed to derive a neutral grounding point.

The point of common coupling (PCC) is the point where the utility ends and the plant starts. This point defines the following:

- Level of voltage distortion from the utility to the plant and contribution from the plant to the utility, needed to design the harmonic filters. The most frequently used standard is IEEE 519-1992.
- Power factor from the plant to the utility, needed to design power factor correction equipment. A value of >0.95 is generally required.
- Allowable voltage fluctuations, needed to design equipment for the rated load and to protect it from over- and undervoltages.
- Network short-circuit power, in megavolt-amperes, needed to design harmonic filters and converter transformer impedances.

A typical power distribution configuration for a mine and plant operating at >100,000 t/d is shown in Figure 9.1-3.

HV and MV Technologies

HV and MV technologies relate primarily to the types of switchgear system used. Switchgear systems are of two types: air-insulated switchgear (AIS) and gas-insulated switchgear (GIS) (Figures 9.1-4 and 9.1-5).

Conventional metal-clad AIS systems fully fitted with draw-out circuit breakers (CBs) are in common use. For primary distribution circuits, either vacuum or sulfur hexafluoride (SF6) circuit breakers can be used. Both provide similar operating and technical characteristics. Vacuum circuit breakers are lower in cost but can expose electrical distribution systems to overvoltages from motors during the switching-off cycle. This problem can be eliminated by using fused vacuum contactors (FVCs) rather than circuit breakers. FVCs should be considered for high short-circuit levels or high MV-motor switching cycles. Their use provides MV motor protection, reduces required cable sizes and thus costs, and prolongs device life by reducing the amount of let-through energy incurred.

These relative performances of these three AIS technologies are compared in Table 9.1-3. The cable sizes required for use with various FVC and CB configurations are listed in Table 9.1-4.

GIS systems offer several advantages over AIS systems. They are much more compact (20 × 15 m, as compared with 75 × 75 m). They are available up to 500 kV and are minimally affected by power derating factors. They can easily accommodate a duplicate bus-bar system. They are suitable for use in site conditions that are arduous with high contamination, high altitude, or extreme climatic conditions. They can become cost-effective when consideration is given to substation real estate, civil works excavations, and geographical location. Systems are available that provide visible isolation and equipment grounding facilities for operator verification, thus ensuring safe and practical isolation of the electrical plant.

Primary distribution network voltage should be determined according to a site's distribution distances and power-flow requirements, which depend in turn on site size, location, and conditions. Current requirements, lighting, voltage insulation levels, and short-circuit levels are all important.

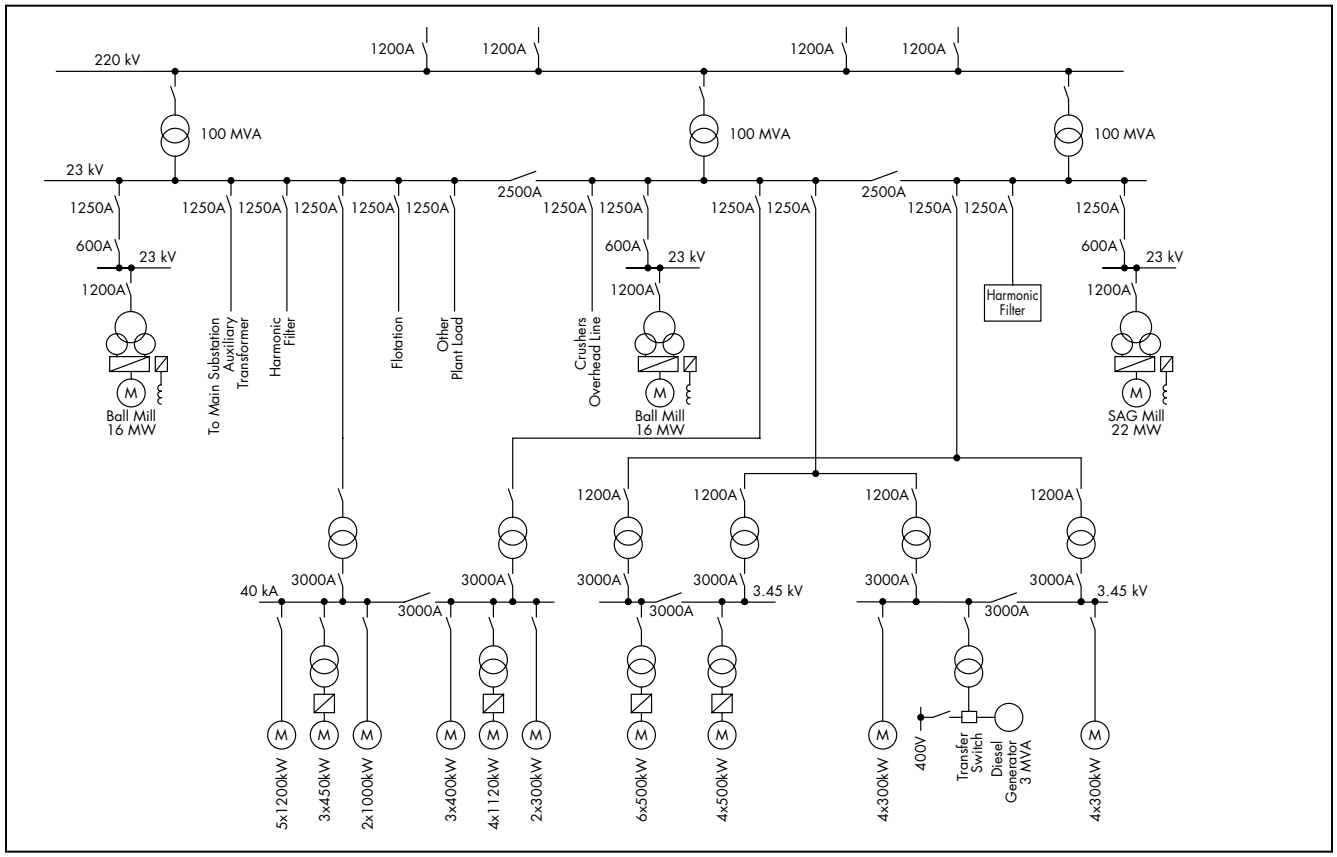
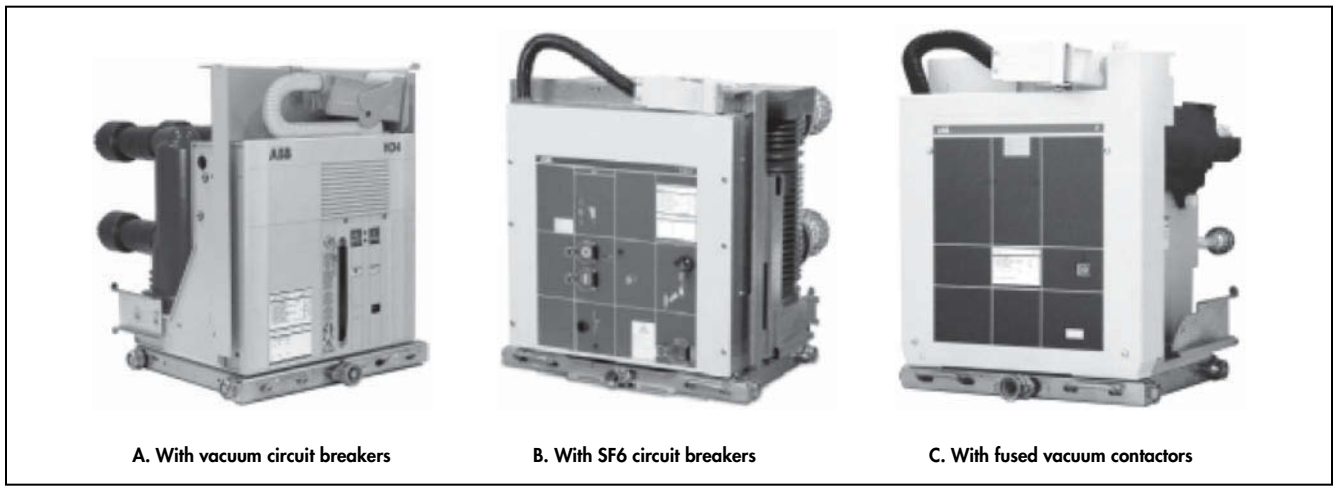


Figure 9.1-3 Typical power distribution configuration for a mine and plant



A. With vacuum circuit breakers **B. With SF6 circuit breakers** **C. With fused vacuum contactors**

Figure 9.1-4 Air-insulated switchgear systems

The cost of cabling between substations should be a determining factor in distribution voltage selection. Substation and associated equipment for MV installations should be designed for full arc fault containment to provide maximum operator safety during installation or in the event of an internal switchgear fault. Consideration should be given as to how to increase distribution voltage when current and short-circuit values rise above the typical values shown in Table 9.1-5.

When supply outages cannot be tolerated, duplicate bus-bar systems should be considered for switchgear. Such systems give additional flexibility to the power distribution system. However, they are relatively costly. Consideration should also be given to emergency power feeds for critical areas of the process.

LV Technologies

Civil installation and transformer costs can be minimized by careful grouping and positioning of motor control centers (MCCs). Redundant or standby drives and feeders should be positioned within the MCC so as to avoid common modes of failure such as cubicle faults and loss of control supplies.

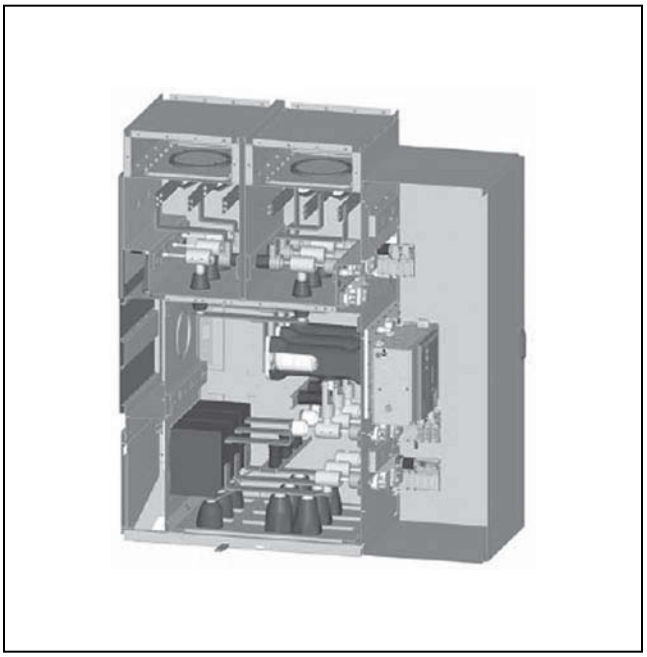


Figure 9.1-5 Gas-insulated switchgear system

Cabling and hardware costs for connecting the electrical and control systems are simplified with the advent of intelligent switchgear systems with electronic protection and monitoring and with control devices with communication facilities.

Larger MCC installations can be accommodated with appropriately large switching-hardware current ratings and short-circuit clearing capacities. Large industrial projects increasingly use 690-V rather than 400-V systems; the relative capabilities of the two systems are shown in Table 9.1-6. Single loads, for instance for a motor, are limited to the values shown in the table due to the need to limit the thermal loading of the connected MCC module.

MCC packing density should also be confirmed to avoid excessive temperatures in the MCC and associated components. As with MV gear, the cost of cabling between areas should be a determining factor in the selection of operating voltage. Substation and associated equipment for LV installations should be designed for full arc fault containment.

For weak networks, any voltage drop upon motor start-up must be considered, as well as the cable length and size between the MCC and the motor. If the voltage drop is outside limits, the motor may not start at all, because the start-up torque capability of a squirrel-cage induction motor is proportional to the square of the voltage drop. In such cases a better solution is to use frequency converters for assisted start-up.

Table 9.1-3 Relative performances of three AIS technologies

	Switchgear Technology		
Performance Criteria	**Vacuum CB**	**SF6 CB**	**FVC**
Mechanical endurance	10,000 operations	10,000 operations	500,000 operations
Electrical endurance	30,000 operations	30,000 operations	500,000 operations
Short-circuit let-through energy	Full short-circuit current	Full short-circuit current	Limited by fuse

Table 9.1-4 Cable sizes required for use with various FVC and CB configurations

	Cable Cross-Sectional Area Required, mm^2			
	Short-Circuit Installation Level = 31.5 kA		Short-Circuit Installation Level = 50 kA	
Motor Full-Load Current, A	**FVC**	**CB**	**FVC**	**CB**
45	35	185	35	300
60	35	185	50	300
75	35	185	50	300
90	50	185	50	300
120	50	185	70	300
150	50	185	70	300
165	50	185	70	300
185	50	185	70	300
225	70	185	95	300

Table 9.1-5 Typical loads and short-circuit levels for MV switchgear and associated equipment

Connection Voltage, kV	Typical Load, A	Typical Short-Circuit Level, kA
6.6	4,000	50
11	4,000	50
22	2,500	25
33	3,150	40

Soft starters are applicable only in certain cases, mainly for limiting network start-up current.

Power Quality Support Equipment

Power quality support equipment includes filter and compensation units, static-voltage compensators, active filters, and synchronous condensers.

Filter and Compensation Units

Filter and compensation (F&C) units must be applied in most cases to comply with IEEE 519-1992 for network quality. F&C units are ideally positioned at the MV level and cover all harmonic-producing loads.

All electrical equipment that consumes nonsinusoidal current from the supplying line must be considered to be a source of harmonic current, and all other equipment such as transformers and AC motors must be considered to contribute to harmonic pollution of the network. Equipment impact can be more or less severe depending on the amplitude and frequency of the harmonic currents. Network impedance also has a significant influence on the magnitude of harmonics.

Frequencies in the spectra of harmonic-generating loads can arise from both constant-frequency and variable-frequency sources.

Cycloconverters and frequency converters clearly produce harmonic-producing loads. Beyond these, the inrush current effects of switching on converter transformers must be taken into account. The magnitude of the inrush current is based on the saturation inductances of the transformer and can reach 6 to 7 times the nominal current. The inrush current contains all integer low-order (including zero-order) harmonics for DC components. Harmonic components decay with a time constant ranging from several seconds for low-power transformers to a few minutes for high-power transformers.

Second-order harmonics are not normally expected in plants, but their presence causes heavy distortion of the network, so their potential to cause negative effects is rated as high. It is therefore necessary to damp second-order harmonics with a filter, even though it is expensive to do so. This is especially important for weak networks with consequently low network natural parallel resonance, because the potential danger of hitting the network with low parallel resonance during transformer switch-on is high and therefore equipment damage is likely.

Cycloconverter input current distorts the supply voltage to an extent that depends on supply system impedance. For strong supply systems with high fault levels, the effect on the supply voltage is minimal. However, for weak supply systems or low fault levels, the supply voltage becomes highly distorted. A highly distorted voltage can disturb the operation of sensitive devices connected to the system or simply increase their losses.

A typical filter configuration for a plant contains groups of switchable passive filters starting with F2 for the second harmonic and continuing with F3, F4, F5, F7, and F11 (Figure 9.1-6). The total filter configuration for a 100,000-t/d plant is ~70 MVAr (megavolt-amperes reactive, the unit of reactive power in an AC power system).

All of the following additional technologies are implemented today in different configurations that support but do not replace F&C systems (Errath et al. 2001a, 2001b). A breakthrough with active filters will be realized only if the switching frequency of the elements increases by a factor of ~10 and prices decrease by a factor of ~5.

Table 9.1-6 Loads and short-circuit levels for LV 400- and 690-V motor control centers

Voltage, V	Load, A	Short-Circuit Level, kA	Single Loads, kW
400	5,000	≤80	≤315
690	5,000	≤65	≤500

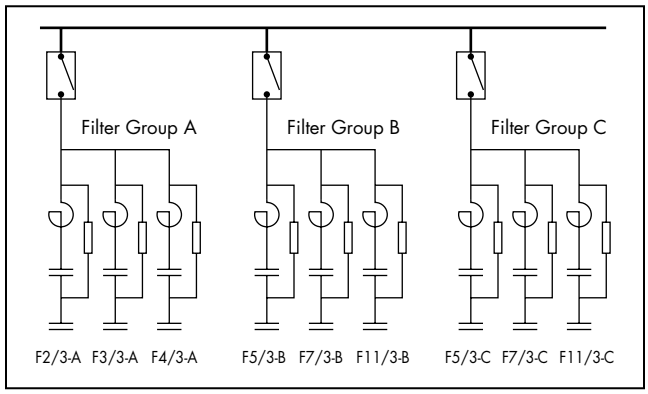


Figure 9.1-6 Typical filter configuration for a plant

Static VAR Compensators

Static VAR compensators (SVCs) control the reactive power (VAR) and are used mainly in weak utility networks (Figure 9.1-7). An SVC holds line voltage at a constant level. Most power systems can be represented by a simple model showing source voltage, bus short-circuit capacity, impedance from the source, feed line to the plant, and plant load characteristics. However, loads are never constant. In the milling part of a mine where the largest single loads are incurred, starting or stopping one drive can affect total plant load by 10% to 20% and change plant bus voltages by 5% or even 10%. To compensate for changes in line voltage, SVCs can be used as switched capacitors to produce stepped increases or decreases in voltage.

SVCs control VAR steplessly (without voltage jumps) within small tolerances (~1% reaction in three sinusoidal cycle times). They raise voltage by adding VARs (adding capacitive VARs) and lower voltage by subtracting VARs (adding inductive VARs), thus enabling the voltage on a bus to be kept constant.

The information needed for regulation is the bus line voltage and the line current. The regulator defines the firing-pulse angle from the SVC. Some SVC layouts are of mixed configuration, in which some capacitors are switched in and out and the rest are regulated by the SVC. This configuration minimizes SVC size, although total capacitive power remains the same. When using an SVC in a utility supply system as a voltage stabilizer, it is advisable to position it as close as possible to the plant, because SVC function must be electrically coordinated with the F&C unit in the plant.

An important consideration when using an SVC is that its large reactive power (~50–100 MVAr for a 100,000-t/d plant) causes network parallel resonance to decrease. What constitutes a reasonable amount of harmonic contribution in the network must also be considered.

Active Filters

In the mineral industry, the use of active filters is still restricted to smaller units of ~4 MVA or multiples of that value connected

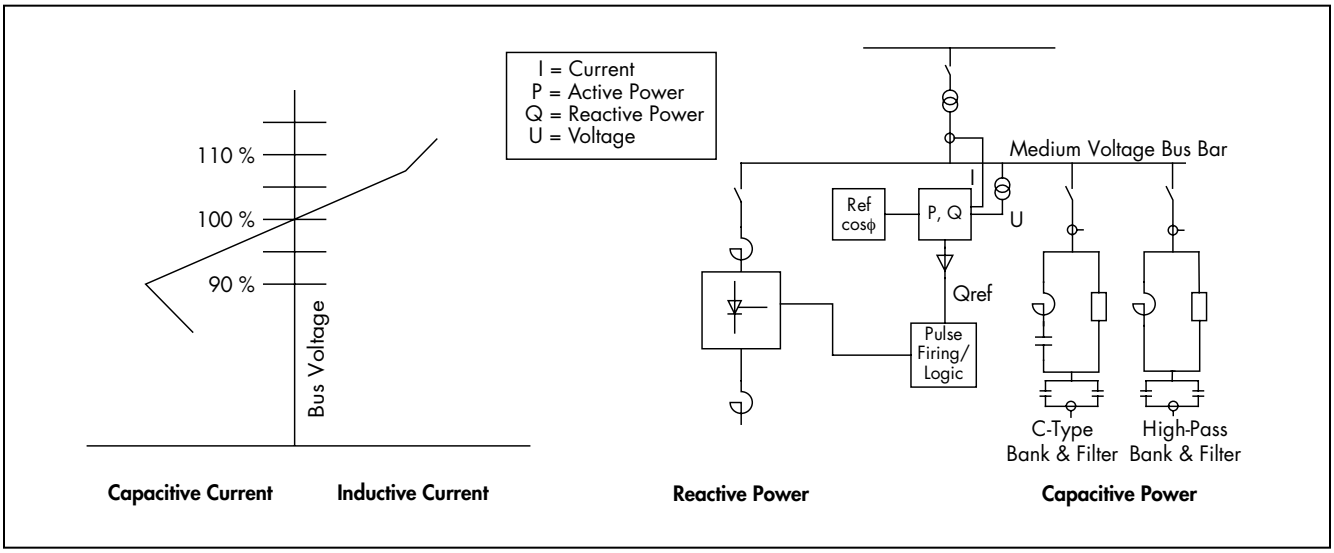


Figure 9.1-7 Static voltage compensator

in parallel. This restriction is due mainly to the actual high investment cost of the active filters and the inability of this technology to cope with higher harmonics.

Synchronous Condensers

Synchronous condensers have played a major role in voltage and reactive power control for many years (Figure 9.1-8). A synchronous condenser rotates or spins freely, unconnected to the network. Its relatively small frequency-controlled motor rotates increasingly faster until it reaches the synchronous speed.

Synchronous condensers have been connected at both subtransmission and transmission voltage levels to improve stability and maintain the voltage within desired limits under varying load conditions. However, they are used mainly to supply a portion of the required converter reactive power and to provide system reinforcement as needed. They are a proven solution for increasing short-circuit power in plants and also increasing the network's parallel resonance.

Synchronous condensers have an inherent advantage over capacitors in that they are functionally synchronized with the power system. Their field is controlled so as to induce them to either generate or absorb reactive power. A further strong advantage is their ability to ride through small network disturbances because of their large accelerated masses.

The short-circuit improvement at the point where a synchronous condenser connects with a 15-MVA condenser is ~100 MVA. This performance efficiency is considered good in that total full-load losses are <1% of the condenser rating.

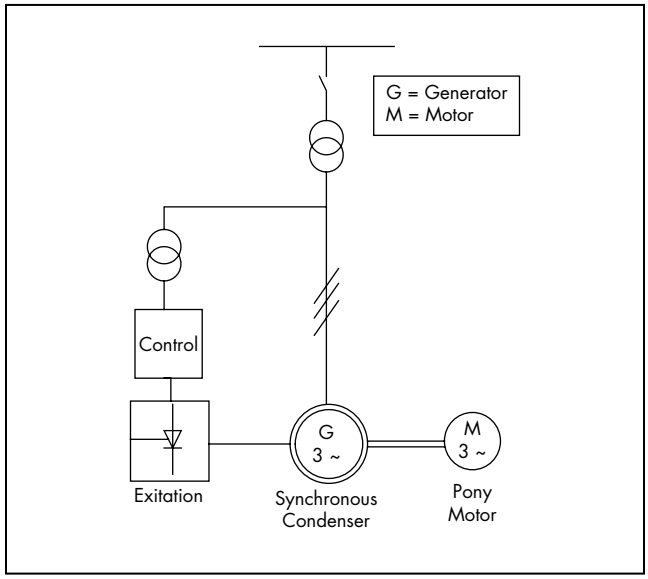


Figure 9.1-8 Synchronous condenser

Drives and Motors

Almost all relevant process drives today are adjustable-speed drives. Traditional power distribution for fixed-speed drives applies now only to smaller motors. Adjustable-speed drives usually have their own distribution philosophy.

SAG Mill Drives

Large autogenous grinding (AG) and semiautogenous grinding (SAG) mills are usually powered by adjustable-speed drives—either gearless mill drives (GMDs) or geared single- or twin-pinion drives. SAG mill drives are larger than AG mill drives in both size and power. In new installations, power ratings for SAG mill drives can range up to 28 MW.

Adjustable-speed drives are always coupled by converter transformers that are connected to the MV distribution system. The transformer is a *converter transformer*, not a distribution transformer. The converter transformer must handle current harmonics and nonstandard secondary voltages produced by the drive.

A general rule for sizing converter transformers is that the sum of their power ratings in megavolt-amperes should be ~1.9 times the power of the drive in megawatts. For example, the 21-MW drive shown in Figure 9.1-9 requires converter transformers with a total power rating of ~40 MVA (= 3 × ~14 MVA). Normally, three converter transformers are connected by one MV breaker to the MV bus distribution system. The secondary voltage of the transformers is relatively low—depending on the cycloconverter configuration, in the

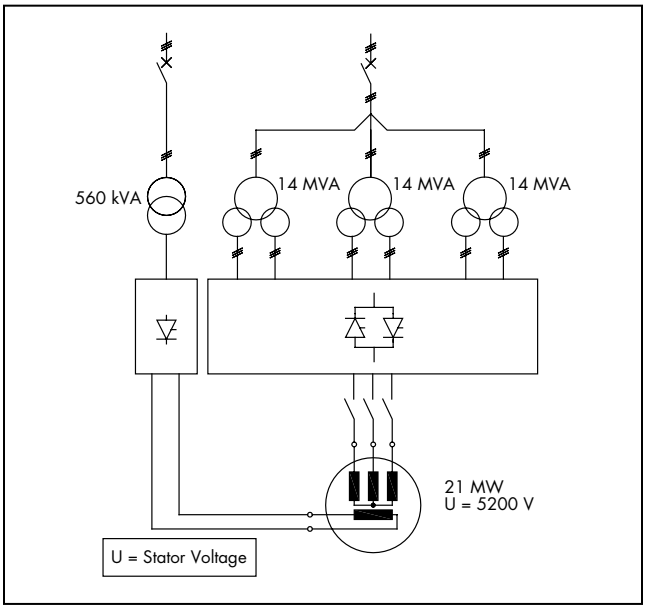


Figure 9.1-9 Converter transformer in a 12-pulse configuration for a 21-MW gearless mill drive powering a SAG or ball mill

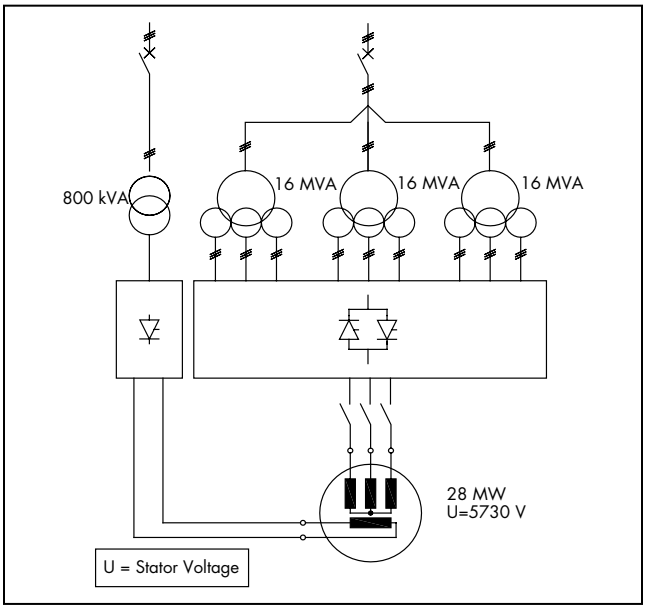


Figure 9.1-10 Converter transformer in an 18-pulse configuration for a 28-MW gearless mill drive powering a SAG mill

range 1,000–1,800 V. An isolation switch is mounted for safety reasons between the cycloconverter and the motor. A controlled rectifier unit is used for excitation and connected by a converter transformer to the MV bus bar with a voltage of <22 kV.

For average-size-drive motors, converter transformers are typically triple-wound. If the primary winding is in a delta configuration and one of the two secondary windings is in a star configuration with a phase shift of 30°, a 12-pulse configuration is possible on the transformer (Figure 9.1-9). Typical start-up torque is up to 130% of the nominal network current. Start-up current is very soft and is not a problem in terms of power distribution. Maximum power is achieved only with the rated torque at the rated speed. Thus adjustable-speed drives are generally considered electrical-network-friendly in comparison with fixed-speed drives, which can draw start-up currents of 600% of the nominal network current.

For >25-MW-drive motors, converter transformers should be quadruple-wound in an 18-pulse configuration because of the physical limits of the semiconductor (Figure 9.1-10) and to minimize creation of harmonics.

Cycloconverters are fuseless. In the event of a short-circuit in the motor or the cycloconverter, current is limited only by the impedance of the converter transformers. Cycloconverters are designed to survive short-circuit currents for 200 ms, which is longer than an MV breaker normally takes to trip (40–80 ms). Normally, three transformers are connected by one MV breaker to the MV bus distribution system. As an example, the motor in cycloconverter operation is designed for 5,730 V.

Ball Mill Drives

For ≤7-MW ball mills, drives are usually fixed-speed and geared. Wound-rotor induction motors with secondary starter or synchronous motors are usually used with adequate start-up equipment to limit the start-up current to the network. In terms of power distribution, drives with fixed-speed motors are more challenging to use than are those with adjustable-speed motors because of their high start-up currents. The first harmonic created is the 5th in the six-pulse configuration, except for interharmonics, followed by the 7th, 11th, 13th, and so on.

For >7-MW ball mills, drives are usually adjustable-speed GMDs. Adjustable-speed geared single- or twin-pinion drives are also used. For adjustable-speed drives in the range 7–22 MW, the primary converter transformer is connected in the same manner as for a SAG mill directly to 13.8, 22, 33 kV on the MV bus. The converter transformer winding vectors are similar to those for the SAG configuration.

For small GMDs, converter transformers are often in a six-pulse configuration (Figure 9.1-11). The primary transformer can be connected to the MV distribution bus to 4.16, 13.8, or 22 kV (maximum).

High-Pressure Grinding Rolls

High-pressure grinding rolls (HPGRs) are another option for grinding or milling (Figure 9.1-12). HPGRs were first used in the minerals industry in the 1990s and are known for their high material-breaking efficiency. Their main application is for hard-rock gold and copper ore. They have twin drives in a typical power range of 2 × 2 MW. A grinding circuit usually consists of two to four HPGR sets followed by two or three ball mills.

HPGRs have standard 2,300-, 3,300-, or 4,160-V motors with reinforced bearings. The drives are mainly frequency converters. One drive receives power from a single triple-wound 5–6-MVA transformer. The other drive handles high-load peaks up to 200% of the nominal torque and must be able to share the load between the two rolls in a fast, controlled manner. This load behavior must be taken into consideration in the power distribution.

Conveyor Belts

Conveyor belts today are equipped, depending on the application, with direct on-line motors and hydraulic or gearbox slip-ring motors with resistor starters or variable-frequency

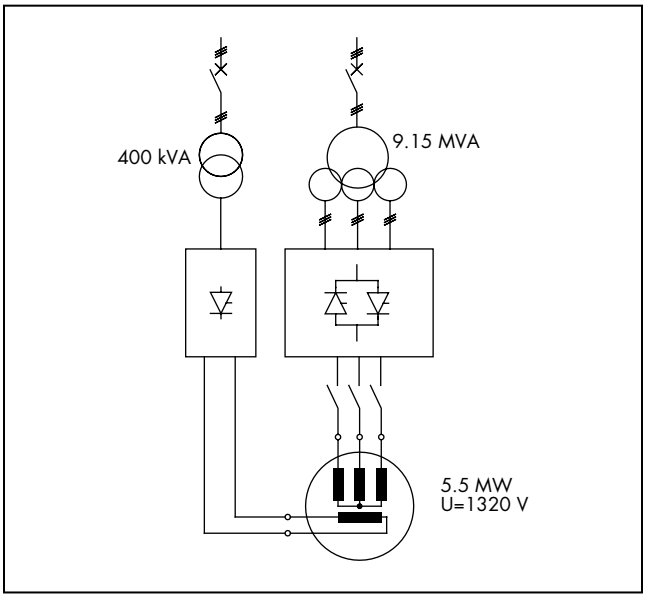


Figure 9.1-11 Converter transformer in a six-pulse configuration for a 5.5-MW gearless mill drive powering a ball mill

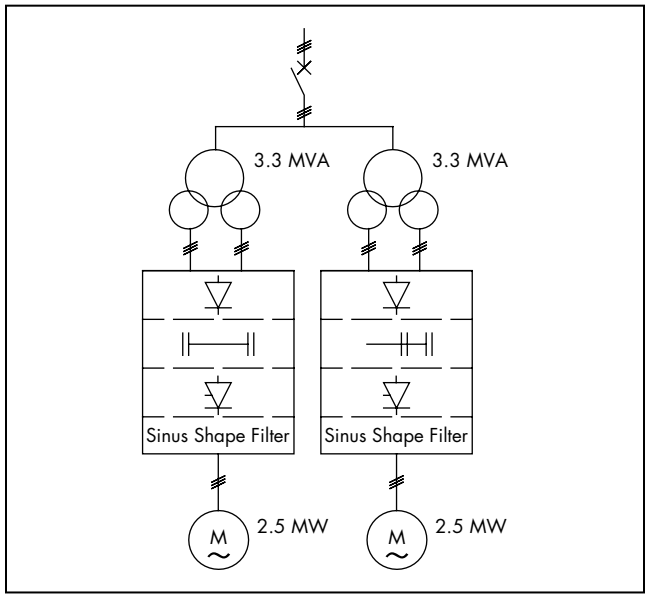


Figure 9.1-12 Single line of an HPGR set

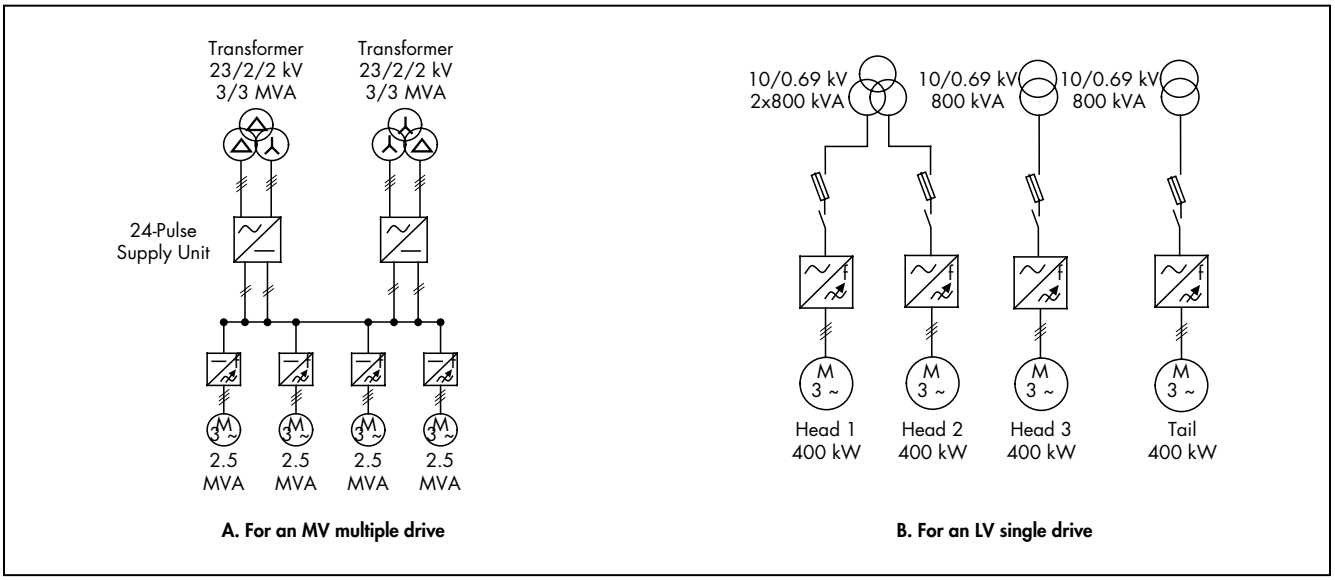


Figure 9.1-13 Conveyor-belt drive systems

drives (Figure 9.1-13). Larger conveyor systems are driven by several motors in the power range from hundreds of kilowatts up to 2 MW at the head and tail end, depending on belt width and length and on conveyor profile.

Overland conveyor belts can curve with uphill and downhill sections of many kilometers, and are controlled mainly by variable-frequency drives (VFDs). Downhill applications use mainly active front-end (regenerative) drives where the drive electrically breaks the belt and braking energy is supplied back to the grid (Lüchinger et al. 2006).

VFD supply voltages are as follows: for LV drives, 500 or 690 V; for MV drives, 3.3, 4.16, or 6 kV. The load behavior is a constant torque load with a relatively low dynamic for start-up and braking. Typical start-up or braking torque is between 1.15 and 1.5 times nominal torque. The supply power range for one conveyor can vary from several hundred kilovolt-amperes to ~15 MVA.

Cyclone Feed Pumps

The drives of cyclone feed pumps have recently undergone significant performance changes. A large plant typically has two to four ~1,500-kW pumps. Pumps are usually controlled by frequency converters and operated at low speed. The pump motor is a scalar-controlled induction motor and often does not have a gearbox but is coupled directly to the pump. To achieve low-speed (600–200-rpm) operation, the motors are designed with 16 to 30 poles. Operation with a frequency converter guarantees a soft start with low network load. The supply voltage of the frequency-converter transformer is from 4,160 V to 22 kV. Advantages of this configuration are that

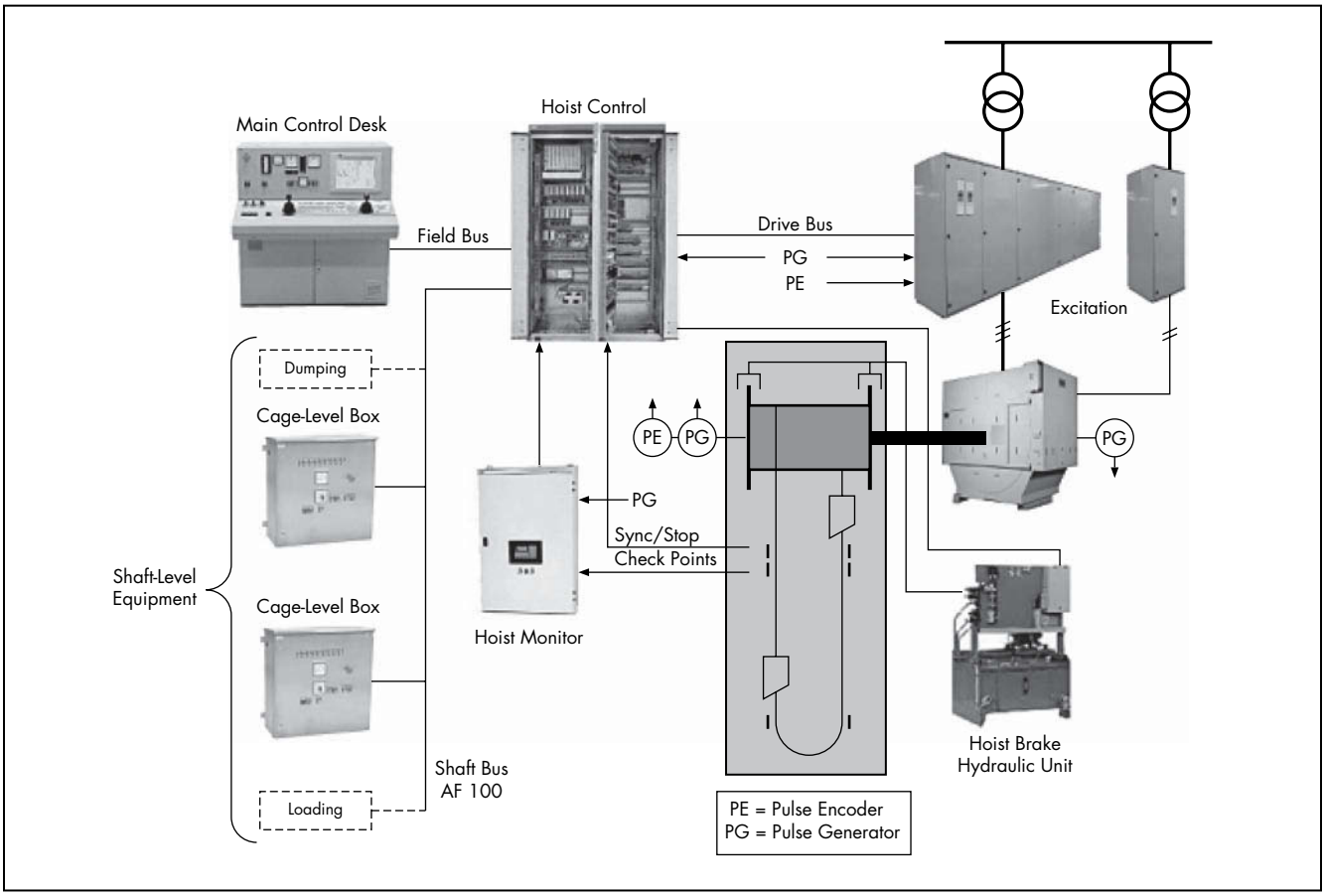


Figure 9.1-14 Friction-type mine hoist

it is adaptable to process requirements, energy savings, lack of energy-wasting throttle valves and maintenance-intensive gearbox, and the high efficiency of the drive system. A disadvantage is the higher investment cost for the motor.

Mine Hoists

Underground mining relies on hoists to transport people, equipment, and ore between the mining zone and the surface. In many mines, vertical shafts are combined with a ramp. However, ramps are not technically or economically viable in mines that are deep or have poor rock stability. In such mines it is common to sink both a production and a service shaft. The shafts can also be used for ventilation purposes. The main considerations for a mine hoist are absolute operational safety, reliability, energy efficiency, and productivity. Grid stability must be considered because of changing load cycles during hoist operation.

Mine hoists are of two types: friction and drum (Figure 9.1-14). Modern hoists are powered by AC motors. Smaller hoists with power requirements of ~1,500 kW normally use a gearbox with high-speed induction motors. Larger hoists use a direct drive with an overhung synchronous motor. Speed control is achieved by a voltage-source-inverter (VSI) frequency converter.

Modern hoists are direct-drive systems; that is, the motor is coupled directly, without the use of a gearbox, to the single- or double-drum hoist. The power of a friction-type hoist system is in the range of 5 to 10 MW. The main components of a hoist system include a synchronous motor, 4Q frequency-converter drive (motoric and generatoric), hydraulic brake system, control system for friction, rope oscillation, and so on. Figure 9.1-15 shows the electrical design for the friction-type mine hoist shown in Figure 9.1-14.

Dragline Excavators

Dragline excavators are one of the largest mining machines for overburden removal in strip mines. They can have any of three major drive systems:

1. MG sets with digital field excitation
2. Conventional AC drives, LV or MV variable-speed with squirrel-cage AC motors and gears
3. Gearless AC drives with gearless ring motors for hoist and drag drums

Most draglines built between 1960 and 1990 use Ward–Leonard MG sets and DC motors. Draglines built today use conventional or gearless AC motors. The new systems are almost maintenance free, more efficient overall, and contain better drive controls.

Supply power usually varies from 4 MVA for small draglines up to 25 MVA for large draglines. Higher voltage and short-circuit capacity are desired for better performance. Standard dragline supply is typically ≥22 kV. Drive-system voltage is typically 6 kV for the synchronous motors of an MG set and 1,400 or 690 V for AC drives.

Draglines with MG-set drive systems usually have two to four sets; some larger draglines even have six sets (two extra

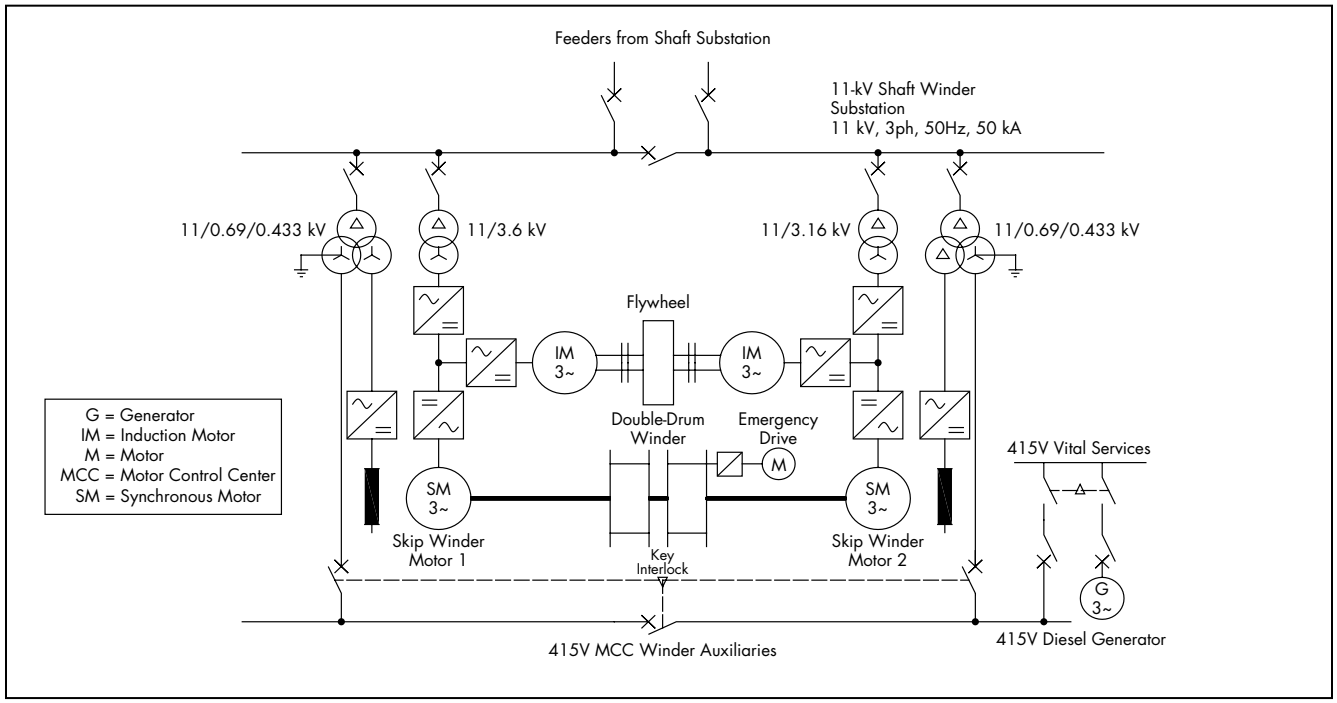


Figure 9.1-15 Friction-type mine hoist system, generic single-line configuration

sets for the walking mechanism). Older systems are commonly upgraded with programmable logic controllers (PLCs) and digitally controlled field-excitation modules. Conventional AC drives have been successfully installed on smaller and even a few larger draglines.

A typical application is a multiple-drive dragline (Figure 9.1-16), where one supply unit feeds several motor inverters over a common DC bus bar. The system shown in the figure has two hoist motors, two swing motors, and two drag motors. Typically two to four such sets are installed on a dragline. The common DC bus allows energy sharing among the motors (some motors run in motor mode, some in generating mode).

Usually a regenerative insulated-gate-bipolar-transistor-(IGBT-) based supply unit is used to feed excessive braking energy back into the grid. These units usually have better power factors (~1) and low total harmonic distortion levels. For weak grids, they protect the drive against voltage dips or even short blackouts. Modern units have an adjustable leading or lagging power factor and can be used for reactive power compensation. The duty cycle of dragline motion is complex and demands high drive dynamics, fast controls, and high overload ability on the part of the motors.

Gearless drive systems are currently state of the art for draglines. Gearless technology has emerged from mill and mine hoist applications where gearless ring motors, integrated into a drum, have proven to be reliable and efficient. The AC ring motors, integrated into hoist and drag drums, are powered by MV variable-speed drives.

Primary Crushers

Main crushing motors can be across-the-line starting, soft-starting, or variable-speed, depending on crusher size, type, and application. For hard-rock crushing, typically one motor is MV across-the-line starting. For applications where speed regulation is necessary, VFDs are the best choice, because they can maintain constant-speed seizer rotation regardless of the type or size of material fed into them. Such drives adjust torque on the shaft to maintain speed and a continuous quality of crushed material. The crusher can also be used for mass-flow control, depending on the type of crusher and the motor's type and speed range. Crusher sizes are in the lower megawatt range.

Shovels

Today most shovels have static DC drives with digital field control. Since the 1980s, AC shovels have been used increasingly. Small electric rope shovels with MG sets are still available but are becoming obsolete. Most shovel suppliers now promote AC technology.

Bucket-Wheel and Bucket-Chain Excavators

Bucket-wheel and bucket-chain excavators are used in continuous mining applications where trucks are not used (Figures 9.1-17 through 9.1-19 show three of many possible configurations). Overburden and ore are transported by conveyor belts. Spreaders are used to spread the overburden on the reclamation side of the pit. Continuous mining is efficient for thick, homogeneous seams of ore (generally lignite).

Excavators are complex systems. A bucket-wheel excavator typically has three or four conveyor belts, a propel system with 3 to 12 crawlers, multiple winches for hoisting, a bucket wheel, and swing drives. The supply power of an excavator can reach 16 MVA. The total drive power depends on the size of the excavator and can be ~10 MVA. The supply voltage is usually 4.16, 13.8, 22, or 30 kV. VSDs are used for propel, swing, hoist, belt, bucket-wheel, or bucket-chain drives. Excavators are custom-built according to the geological situation. The drives and power distribution systems for spreaders and reclaimers are similar.

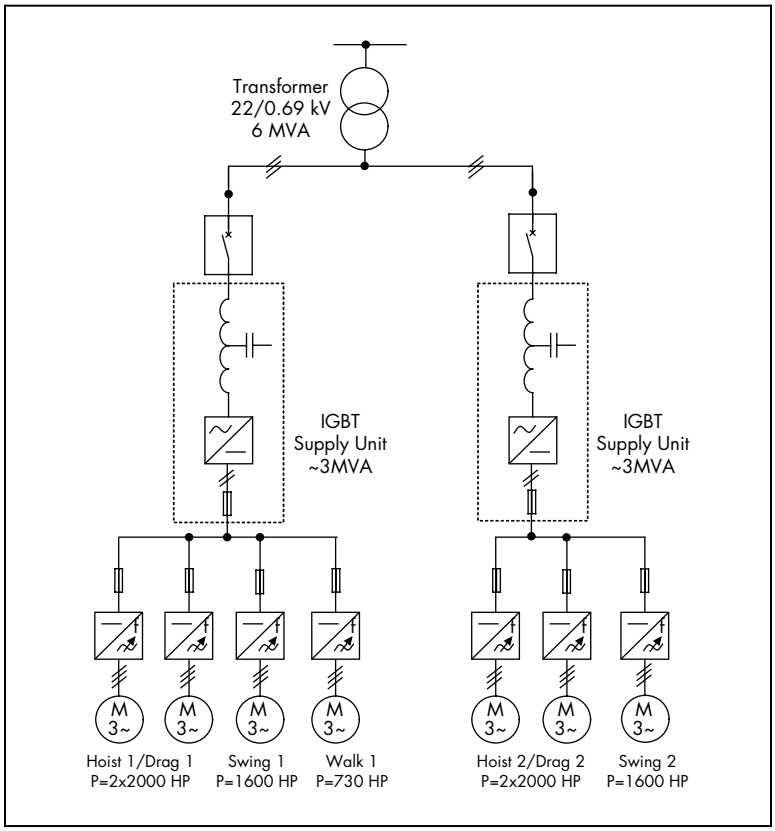


Figure 9.1-16 Multiple-drive AC dragline, single-line configuration

A bucket-chain excavator differs slightly from a bucket-wheel excavator in that it drives on rails, and the propel mechanism usually has about 60 motors. The bucket chain is driven by two large gearless ring motors supplied by a cycloconverter whose supply power is ~2 MVA and motor speed is ~13 rpm. The voltage for the bucket-chain drive is ~1,400 V.

Stackers and reclaimers move on a well-defined path and handle stockpiles of defined size, and so can usually be completely automated for unstaffed operation. Automation is accomplished by means of VFDs, PLC systems, and Global Positioning Systems or laser-positioning systems for collision protection, stockpile scanning, and ore handling. When combined with automated ore-quality detection systems, stackers and reclaimers can significantly increase the overall efficiency of a material-handling plant.

Other Design Considerations

Since the 1990s, plants are often installed at high altitudes in the range 3,000–4,600 m. Installation at high altitude requires special attention in dimensioning and design. In addition, many such places are in active earthquake zones and tend to have high snow loads, low temperatures, high wind speeds, and other harsh environmental conditions. All such factors must be considered in the design of a power distribution system.

Design for High Altitude

Four principal aspects must be considered when dimensioning equipment for high altitude: cooling, insulation, heat radiation, and utilization.

Cooling. At increasing altitude, the density of air decreases and therefore also does its cooling capability. This situation can be remedied by two measures: (1) increase or adjust airflow to obtain the desired cooling effect, or (2) install larger equipment with lower output power.

Insulation. At increasing altitude, the density of air decreases and therefore also does its insulation capability. Factors to use in correcting voltage ratings for reduced insulation capability due to altitude are listed in Table 9.1-7. For example, at 4,000 m, a unit of equipment rated at 690 V can be used at only 500 V (690 V $\times$ 0.73); for MV distribution, a unit of equipment rated at 33 kV can be used only on a 22-kV bus.

Heat radiation. At increasing altitude, copper bars tend to heat up progressively more. Cables and copper bars from the power distribution must be dimensioned accordingly, and this effect must be considered in the design of power distribution.

Utilization. How standard motors can be used depends on altitude. For ambient temperatures in the range 30°–40°C, the maximum design temperature at sea level is reduced by the following amounts:

- Insulation class B motor: 0.8°C for each 100 m above 1,000 m
- Insulation class F motor: 1.0°C for each 100 m above 1,000 m

Design for Earthquakes

Shock waves radiate from fault fracture zones and arrive at the earth's surface as complex multifrequency vibratory ground motion with both horizontal and vertical components. The

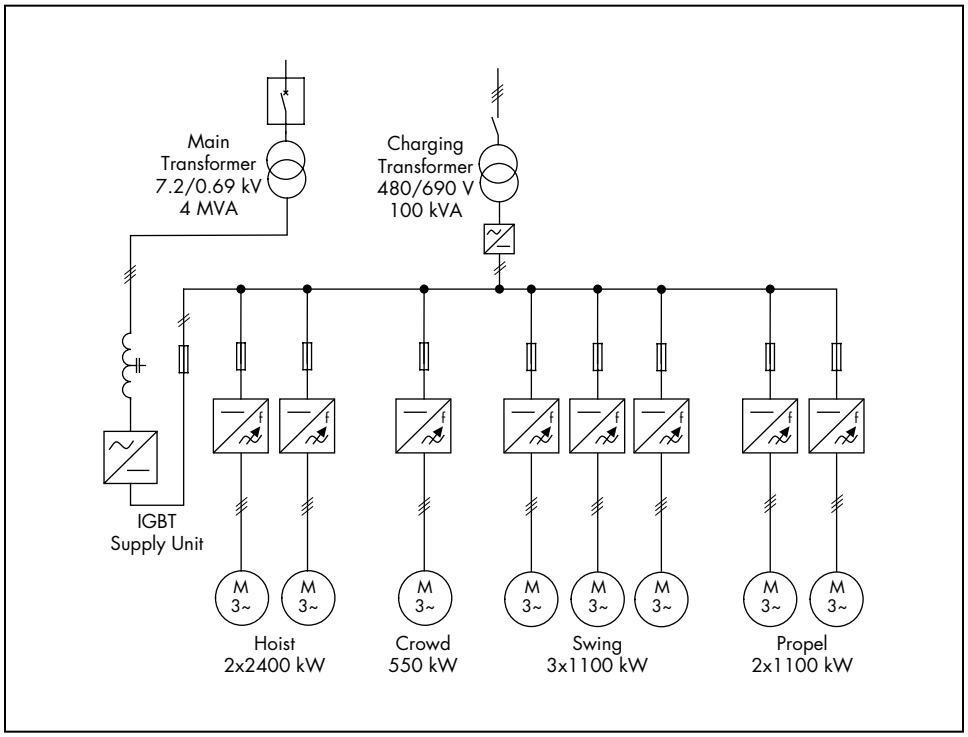


Figure 9.1-17 Bucket-wheel and bucket-chain excavators with spreaders, stackers, and reclaimers

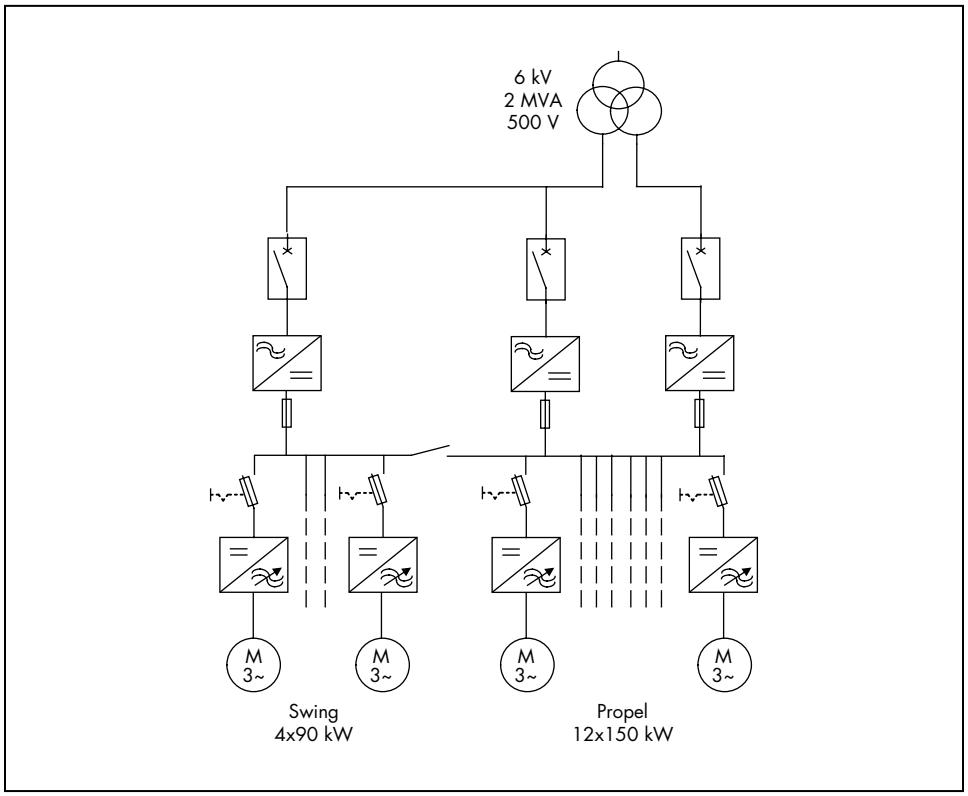


Figure 9.1-18 Bucket-wheel and bucket-chain excavators with stackers and reclaimers (dashed lines)

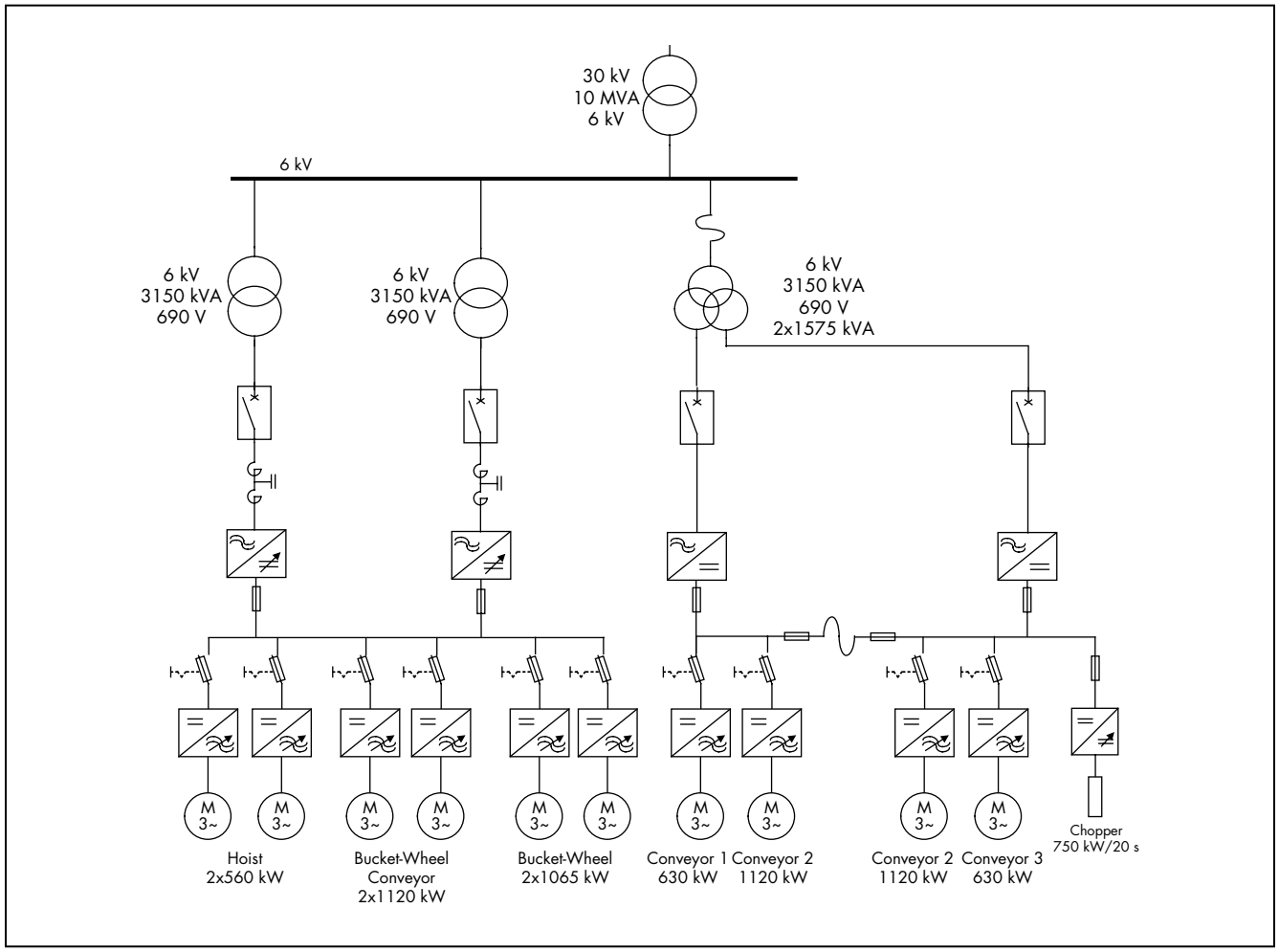


Figure 9.1-19 Bucket-wheel and bucket-chain excavators (multiple-drive) with stackers and reclaimers

response of buildings and mechanical constructions to earthquake ground motion depends on their strength of construction, ductility, and dynamic properties.

The basis for design and construction in areas of intense seismic activity is defined in the standards of the *International Building Code* (ICC 2009). The code standardizes magnitudes of seismic activity and catalogs areas of known activity into defined earthquake zones. For example, most mines in Chile and Peru are situated in IBC zone 4.

Distribution and process equipment such as drives, converter-power-distribution buildings, and switchhouses are required to withstand the expected magnitude of excitation in a defined earthquake zone. Equipment configuration is also relevant. In general, taller and heavier constructions are larger in mass and have lower natural resonance frequencies that can be close to typical earthquake frequencies.

Design for Power Self-Generation

It is generally less expensive and more reliable to purchase electric power from a utility company than to operate one's own generating plant. However, a mine may be located in a remote area, far from a utility transmission or distribution system, making self-generated electric power the only feasible alternative.

Generating plants for mines are typically powered by diesel engines or by coal-, oil-, or gas-fired boilers or even water power. Some mines purchase electric power but also use diesel-powered generators for standby electric power. If the primary source of power is lost, a generator can be started quickly and used to supply standby power to critical equipment such as ventilation fans and personnel or elevator hoists and sensitive process parts. Special attention must be paid to overvoltage (and also undervoltage) conditions that might result from power self-generation.

Network voltage is generally less stable for self-generated power than for interconnected networks. Single-load units can easily use up to 20% of the total installed power. SAG mills can use 20–30 MW, ball mills can use 12–18 MW, mine hoists can use 10 MW, and draglines can also have high power demands in concentrator and mining areas. Not using one or two mills can reduce system load by up to 50%.

A generator does not adapt to new load conditions nearly as quickly or stably as does the utility company. Power can take up to 10 seconds, possibly more, to stabilize, during which time the system voltage and frequency fluctuate with potentially dangerous consequences. Various scenarios must be studied in detail, and the consequences of the scenarios must be considered in system design. Installed equipment

Table 9.1-7 Insulation-capability correction factors for various altitudes

Altitude Above Sea Level, m	Insulation-Capability Correction Factor
1,000	1.0
2,000	0.9
3,000	0.8
4,000	0.73
5,000	0.66

Table 9.1-8 Cable voltage rating required for various distribution voltages

Distribution Voltage	Cable Voltage Rating Required
440 V	600 V or 2 kV
550 V	600 V or 2 kV
4.16 kV	5 kV
7.2 kV	8 kV
12.47 kV	15 kV
13.2 kV	15 kV

must be capable of coping with those conditions. The protection and destruction level of the equipment must be known and designed for accordingly.

CABLES

Cables carry electricity from the substation where power is taken from the utility-company lines to the point of utilization by a mining machine, pump, conveyor belt, or other equipment unit. Many variations in mine distribution are possible, and several types of cables can be put to similar use.

The cable type recommended depends on the application. Some cables remain stationary for years; others are moved frequently. Cables that are connected to mining machines are called *portable cables*. U.S. federal regulations use the term *trailing cables* for the specific type of portable cable that is used in mines (MSHA 1981). Trailing cables are flame-resistant and flexible.

In underground mines, cables that feed the power center or are attached to the high-voltage side must be moved when the power center is moved, which generally is not often. Similarly, in surface mines, cables that feed from switchhouses or unit substations to mobile equipment are moved only occasionally and are not connected directly to a machine. Stationary cables can be feeder or portable cables.

Cable moving is a recurring task both underground and aboveground. Trailing cables can be placed on reels or spools to facilitate moving. Reeled cables are often used on shuttle cars, as are mobile cable reels on surface excavators.

Cable selection is based on a number of parameters including current-carrying ability, voltage rating, and configuration. The basic components of a cable are its conductor, insulation, and jacket; there may also be fillers, binding, shielding, and armor. The conductor is surrounded by insulation and covered by a jacket.

For optimal flexibility, cable conductors are composed of many wires combined into strands, and a number of strands combined to form the conductor. Conductors are either copper or aluminum; the latter is cheaper and lighter but lower in conductivity. The cross-sectional area of a conductor is important for mechanical strength and is closely related to current-carrying capacity.

Insulation is required to withstand stress from heat, voltage, and physical abuse. It must be specially designed not only to protect mine personnel from electric shock but also to separate power and grounding circuits effectively. Excessive heat is particularly destructive to insulating compounds. The main sources of heat are ambient temperature and power loss in cable-conductor resistance.

The maximum, normal, continuous current that a conductor can carry safely is directly related to cable heating, and the term *ampacity* is often used to describe this current level. The ampacity rating is usually based on the maximum rise in conductor temperature, with the temperature limit chosen on the basis of the specified life expectancy of the cable insulation. The *temperature class* (always given in degrees Celsius) describes the maximum allowable sustained conductor temperature at an ambient temperature of 40°C. Cable used in a confined space can overheat with continuous current at the cable's ampacity rating. This is especially true for cable wound on a reel, either for storage or for mobility. The cable's ampacity must be derated in these cases; ampacity and derating-factor tables are available in regulations from various governing bodies.

The most common insulating compounds for cable are neoprene, styrene butadiene rubber (SBR), ethylene propylene rubber (EPR), and cross-linked polyethylene (XLP). SBR is used in 600-V trailing cables. It has good elasticity and flexibility, a 75°C temperature rating, and good resistance to damage by crushing from runovers and rockfalls. EPR has replaced SBR in many trailing cables because, for the same insulation thickness, it has a higher voltage rating (2,000 V) and higher temperature rating (90°C). XLP is also rated at 90°C and is used in HV mine-feeder and portable strip-mining cables. However, it is stiff and therefore not recommended for reeling applications. Typical cable voltage ratings are listed in Table 9.1-8.

The main purpose of the cable jacket is to protect the inner components and hold the assembly in the designed configuration. Common jacket materials are neoprene, nitrile butadiene rubber plus polyvinyl chloride (NBR/PVC), and chlorosulfonated polyethylene (CSPE, also called Hypalon). Armored cables are used in some borehole applications; the heavy metallic jacket affords extra protection to the conductors and insulation.

Flat cables are commonly used on mining machinery such as shuttle cars that have cable-reeling devices. The flat shape allows increased length on a cable reel and is less susceptible to run-over damage. Round cables are typical on all other mining equipment. Low-voltage AC mining machines commonly use unshielded Type G or Type G-GC cables.

Current and voltage regulation are the major concerns in sizing cable power conductors for an application. The effective continuous current through a cable power conductor must be less than the cable ampacity, with correct derating factors applied. The voltage drop across the distribution and utilization systems must be such that voltages at load are within allowable tolerances. For trailing cables to machines, current is often the determining factor, because these cables are almost always relatively short. For feeder cables that serve many loads, however, lengths are often so great that voltage drop becomes a principal concern. Even though cable size may be adequate in terms of ampacity and voltage drop, other factors can affect decisions about conductor size, including tensile

load, weight, and available short-circuit current. Of these, weight can be particularly important, because cable should not be too heavy for miners to handle. The maximum conductor size is usually considered to be 4/0 AWG (American Wire Gauge) for three-conductor cables.

Several methods can be used to determine cable current, including full-load current, effective current demand, and application of a load factor. Regardless of the method used, typical current requirements for mining machinery change continuously over time. The extremely wide variability of mining conditions makes it difficult to define current levels for any part of a given duty cycle with precision. Thus, the use of full-load current is recommended.

The primary concern for voltage conditions is that satisfactory voltage must be at the machine terminals for proper start-up and operation. The allowable voltage tolerance on machine motors is usually ±10% of the rated voltage. Maintaining adequate voltage is one of the more difficult problems in mining and is often the main constraint on mine expansion.

For a thorough voltage study of a mine and its concentrator plant, all impedances and all loads in the power system must be known. A circuit diagram must then be prepared and calculations performed to determine whether voltage levels at the machines are satisfactory. Analyses must be performed not only for normal load conditions but also for start-up of critical motors. If calculated voltages are below those tolerated, system impedance must be reduced. The most convenient way to do so is to increase cable conductor sizes.

Cable couplers are complex plugs and sockets used throughout a mine distribution system to connect mobile machinery to trailing cables, to connect cables with one another, and to connect cables to power centers, switchhouses, and substations. Their complexity is directly related to the mine environment in which they are used; they must resist damage, be sturdy enough to withstand repeated use, prevent electrical hazards, be watertight and dustproof, and withstand heat and cold. Some models are rated explosion-proof. The coupling mechanism must be easy to use yet secure. HV couplers, sized at ~15 kV/500 A, are used to connect switchhouses and mine power centers, to join HV cables, and for HV machines. They accommodate three power conductors, one or more grounding conductors, and one or more ground-check conductors. LV and MV couplers, sized at 225, 400, 600, 800, and 1,200 A, are used primarily to connect mobile equipment to power centers and junction boxes and to connect 600–1,000-V cables. They are sturdy in construction but less complex than are HV couplers.

POWER DISTRIBUTION ARRANGEMENTS

Power distribution arrangements can involve surface overhead-line distributions, back distributions, surface mine distributions, and underground mine distributions.

Surface Overhead-Line Distribution Arrangements

Surface transmission and distribution of electric power is most commonly achieved by means of overhead conductors. The conductors use air space for insulation over most of their length. Their elevation protects them from contact with personnel and equipment.

Overhead conductors are arranged in various configurations to reduce line-to-line contact due to wind, ice loading, or sudden loss of ice load, and may include different combinations of power, neutral, and static conductors. Aluminum conductors with steel reinforcement are commonly used because of their strength and relatively low price, but special applications may call for other materials such as copper.

The types of overhead-line installation used for mining are similar to those used in utility distribution systems. Pole lines are typically used to supply equipment in surface mining and to feed surface facilities related to mining. The lines are normally installed on single wooden poles that can carry up to six conductors, including three for power, one for grounding, one ground check (pilot), and one static. Pole lines can be relatively permanent installations such as those feeding plants, shops, and other surface facilities and long-term pit baselines or ring mains. Temporary poles mounted in portable bases (such as concrete-filled tires) are commonly used in surface mining operations to carry power into the pit.

Basic Distribution Arrangements

Basic distribution systems for industrial applications are radial, secondary-selective, primary-selective, primary-loop, or secondary-spot configurations (IEEE 141-1986). Radial configurations are the most popular in mining, although other configurations are used where special circumstances call for greater system reliability. Surface mines have, of course, greater flexibility than do underground mines and use a wider range of configurations. Secondary-spot configurations, which are the most popular for large facilities in other industries, are uncommon but can be suitable for preparation and milling plants.

A radial distribution system in its simplest form consists of a single power source and substation supplying all equipment. Radial systems are relatively inexpensive to install because equipment is not duplicated. They can be expanded easily by extending the primary feeders.

A secondary-selective distribution system consists of a pair of secondary substations connected through a normally open tie breaker. It is both flexible and reliable; if a primary feeder or substation fails, the bad circuit can be removed from service and the tie breaker can be closed automatically or manually. Maintenance and repair of either primary circuit is possible, without incurring a power outage, by shedding nonessential loads for the period of reduced-capacity operation. If substation requirements are greater than 5,000 kVA, economics often justify this double-ended arrangement.

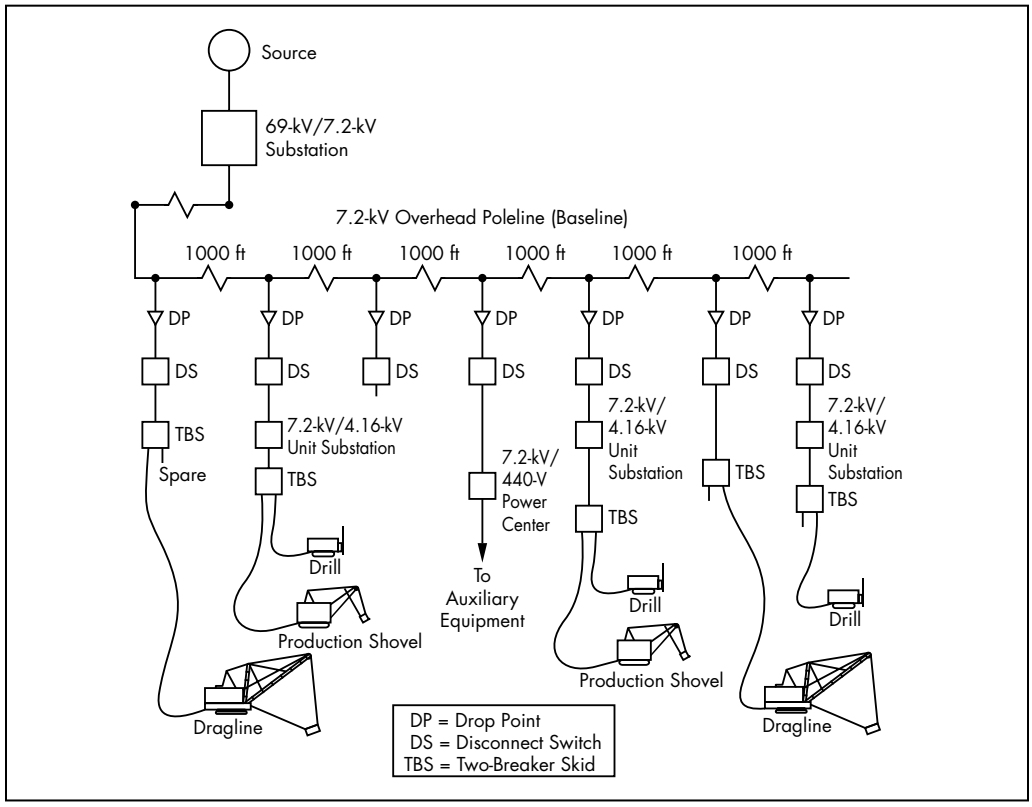
Surface Mine Distribution Arrangements

Mine power systems are of three types: transmission or subtransmission, distribution, and utilization.

Distribution and utilization systems can vary greatly, but in some mines, both functions can be handled on the same system. The location of the mine substation is usually an economic compromise between the cost of running transmission lines and power losses in primary distribution. From the main substation, power is distributed to the various load centers in the operation. Incoming utility transmission should be extended to as close to the load as practical.

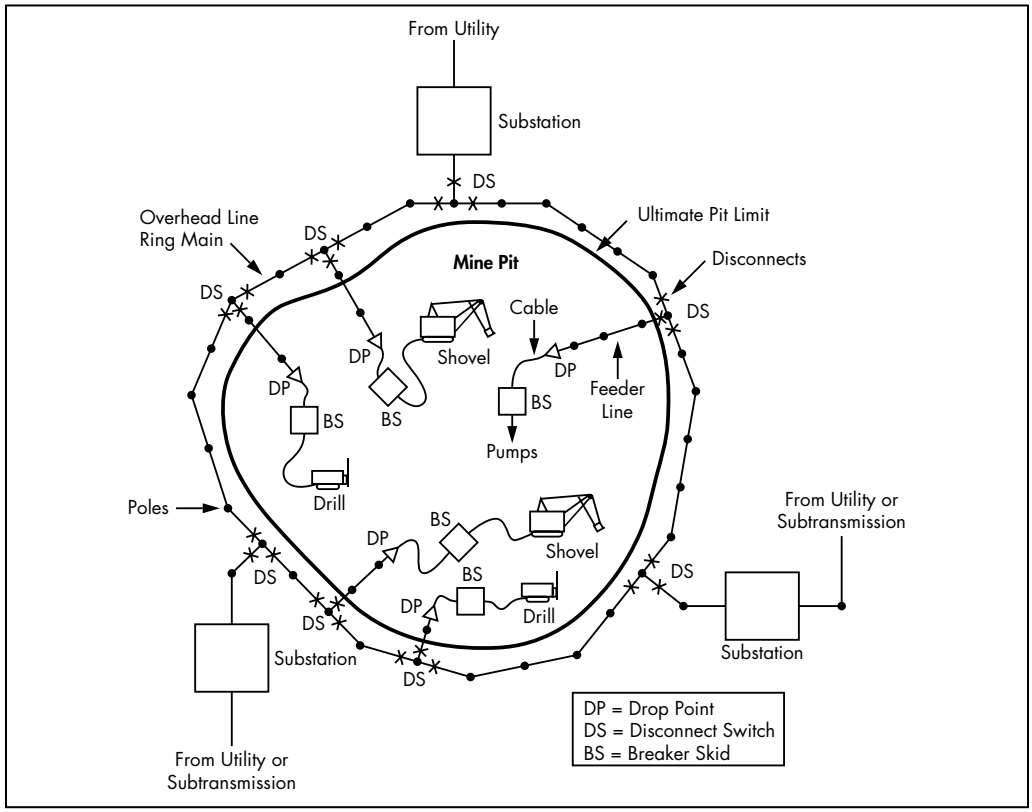
Subtransmission circuits, primary switchyards, and main substations are almost always located in areas unaffected by the mining operation. Surface mine power distribution, in its simplest radial form, consists of a substation, distribution, and a power center feeding the mining equipment (Figures 9.1-20 and 9.1-21). This arrangement is common in small surface operations where the distribution voltage is commonly 15,000 or 4,160 V but can be as low as 2,300 V in older equipment.

Although most strip mines use radial distribution, secondary-selective distribution is also used (Figure 9.1-22).



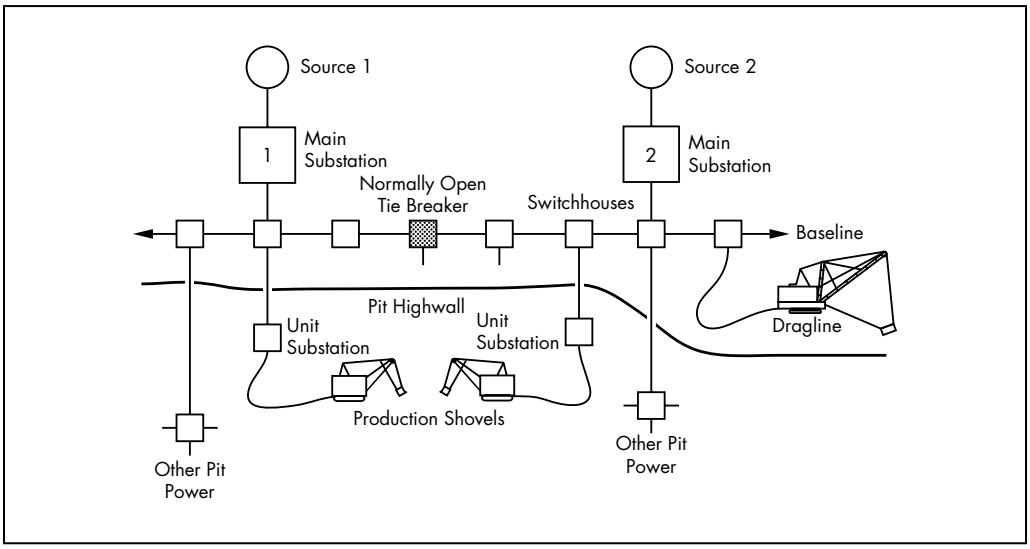
Source: Morley and Novak 1992.

Figure 9.1-20 Strip mine radial distribution system using overhead pole lines



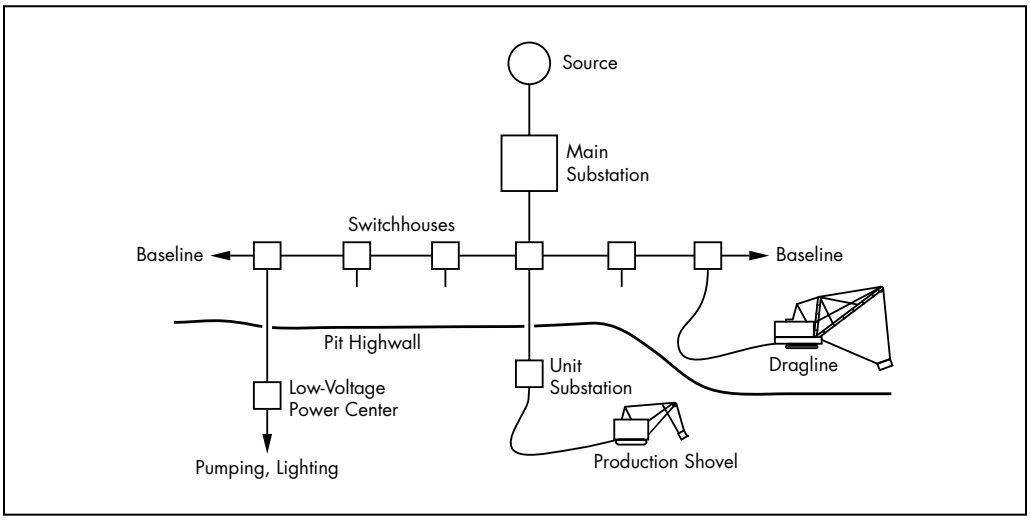
Source: Morley and Novak 1992.

Figure 9.1-21 Open-pit ring mine radial distribution system



Source: Morley and Novak 1992.

Figure 9.1-22 Strip mine all-cable secondary-selective distribution system



Source: Morley and Novak 1992.

Figure 9.1-23 Strip mine all-cable radial distribution system

In this configuration, a portion of the primary distribution is established as a baseline or bus, usually located on the high wall paralleling the pit for the entire length of the cut. It is typically located ahead of the pit and moved as the pit advances. As machines move along the pit, their baseline connections are changed to other convenient locations.

The baseline usually consists of overhead pole lines as shown in Figure 9.1-20. The arrangement of overhead pole lines plus cables is common in older mining operations. Typical spacing between poles, or line span, is 60 m. Drop points, denoted in the figure by triangles, are terminations between the overhead conductors and the cables, mounted about 2.4 m above the ground on poles and spaced at regular intervals in the range ~305–457 m. Cables connected to the drop points deliver power to skid-mounted switchhouses located on the high wall or in the pit. Cable couplers are commonly used for both feeder- and trailing-cable connections. Disconnect and circuit-protection functions are required for each distribution load.

Layouts for all-cable mine distributions (Figures 9.1-22 and 9.1-23) are similar to those just described. In this case; however, the baseline is assembled using cable-interconnected switchhouses. A common approach is to use disconnect skids with three internal switches in the baseline and have separate breaker skids in line with the cables feeding the machinery. Another approach is to combine the single breaker skids into the baseline switchhouses.

Open-pit power systems are similar to those for strip mines with one main exception: distribution typically establishes a ring bus or main that partially or completely encloses the pit. Radial ties to the bus complete the circuit to switchhouses located in the pit, and portable equipment again uses shielded trailing cables.

For secondary-selective distribution (Figure 9.1-22), a normally open tie breaker is placed in the baseline. In some operations, the two substations and the tie breaker are in the same location with two feeders running from the substation area to the baseline. Distribution voltage for the surface mine can be 8, 15, 25, or 34.5 kV, and less commonly 4.16 kV. The primary purpose of any distribution scheme in a surface mine is to provide a flexible, easily moved or modified power source for the highly mobile mining equipment.

Underground Mine Distribution Arrangements

Underground mine power systems are more complicated than surface power systems. Consider, for example, the situation for a coal mine. The nature of the mine and its service requirements dictate that distribution must almost always be radial; the freedom in routing distribution enjoyed by surface mines is not available underground. For increased reliability, secondary-selective main substations are used. Distribution voltage is usually 8 kV, although older 5-kV systems can still be found and 15 kV has increased in popularity in recent years.

Power and mine grounding are fed underground in insulated cables through a shaft, borehole, or intake entry. In coal mines, these cables are required to terminate in disconnect switches within 152 m of the point of power entry into the coal seam. The switches allow total removal of underground power in an emergency. Power is distributed from the disconnect switches, which may be part of a switchhouse, through cables to power centers or rectifiers located as close to the machinery as practical. All cables on HV circuits, usually involving only distribution, have shielding around each power conductor. The prime load concentrations in underground mines are created by the mining sections. Distribution terminates at the section power center, which is a transformer combined with a utilization bus and protective circuitry. From this, several face machines are powered through couplers and trailing cables.

With rail haulage, distribution terminates at rectifiers that contain a transformer and rectifier combination. The rectifiers are located in an entry or crosscut just off the railway. DC power is supplied through circuit breakers to an overhead conductor or trolley wire and to the rail, with additional rectifiers located at regular intervals along the rail system. For further protection, the trolley wire is divided into electrically isolated segments. The typical rectifier supplies the ends of two segments of trolley wire, and each feeder has its own protective circuitry to detect malfunctions. In some mines, DC face equipment and small DC motors are powered from the trolley system through a fused connection (or nip) to the trolley conductor and rail.

All power equipment used underground must be rugged, portable, self-contained, and specifically designed for installation and operation in limited spaces. In addition, all equipment and connecting cables must be protected against failures that could cause electrical hazards to personnel, primarily by protective relaying built into each system part, with redundancy to maximize safety.

ACKNOWLEDGMENTS

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CHAPTER 9.2

Compressed Air

John Hooper

THEORY OF COMPRESSED AIR

Compressed air is used extensively in mining and is often referred to as the "fourth utility," behind water, electricity, and diesel. In essence, compressed air is stored energy and, when controlled, it can be used for production purposes. Energy from compressed air is used to drive pneumatic equipment such as air motors, actuators, instrumentation, and pneumatic tools. It can also be used to cool and clean components or parts and to blow off waste material—a particularly valuable application in mines.

Industry also uses compressed air as an active part of processes that require a supply of clean, dry air, for example, in cleaning the finished product. The effects of compression are a decrease in gas volume, increases in gas pressure and temperature, and an increase in the concentration of contaminants.

HISTORY AND LANDMARKS

Compressed air has been used for as long as people have built fires, as blowing on cinders to make them flare up uses air that has been compressed in the lungs. The first mechanical compressors were hand-operated bellows, with the more-efficient foot bellows coming into use in about 1,500 BC to produce bronze. Bellows and other early blowing machines were primarily used to supply a combustion blast to furnaces and to ventilate underground workings. Use of compressed air as a means of energy transmission began to be discussed in the early 19th century when it was established that the rapid cooling and condensation of steam limited its use to very short distances.

Large-scale application of compressed air to transmit energy began at the Mont Cenis tunnel in the Swiss Alps in 1861 (Peele 1920). Tests with compressed-air-powered drills tripled the daily rate of advance of the tunnel, so the decision was taken to use pneumatic rock drilling and compressors operating at 600 kPa (87 psi). There were problems, including constant repairs, as the materials of the day could not cope with the tremendous stresses. But the tunneling was completed successfully, and it was demonstrated that compressed air could be transmitted over long distances.

The use of compressed air became firmly established with the installation of a compressed air distribution network in the Paris sewer system. The 586-kPa (85-psi) network was immediately successful, and patents were developed for all kinds of equipment, including a clock that kept the correct time thanks to timed pulses from the compressor plant, pneumatic message dispatch systems, passenger lifts, drink dispensers, and air motors of up to 70 kW (95 hp) to operate machine tools, looms, presses, and so on. Air was also used for the pneumatic drive of direct current generators for lighting. Many saw it as the energy distribution method of the future, superior to the emerging electrical distribution system.

Today, the use of compressed air complements that of electricity. Pneumatic tools and apparatus have augmented the power of the human hand, while allowing it to retain the control that is vital for precision work. Air-powered tools are durable and safe, and their light weight avoids operator fatigue. Applications of compressed air also include monitoring, control, and regulation of equipment, often in combination with electric and hydraulic systems. In terms of overall economy, compressed air competes with electric power in many fields and is used in conjunction with electricity in many others.

Compressed air–powered rock drills began to be used in U.S. copper mines in Ducktown, Tennessee, in 1872 and in the copper and iron mines in Upper Michigan in 1878—before then the only method was to drill by hand. Hammer-type rock drills were developed towards the end of the 19th century. At that time, air compressors were primarily steam driven and relatively inefficient. Consequently, mines employing large numbers of drills and other air-powered equipment sought other means of air compression. Several mines located in areas of varying topography installed hydraulic air compressors to solve the problem (Peele 1920). The best-known system was built in 1910 at Ragged Chute on the Montreal River to serve the mines in Cobalt, Ontario, Canada, and is still operating. The development of the electric-powered compressor made hydraulic compressors, which required large vertical shafts to channel potential energy, obsolete by 1939.

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The Swedish method, which was introduced toward the end of the 19th century, was developed to provide a one-person face-drilling operation, whereby the rock drill and "pusher leg" were powered by a new source of energy—compressed air. This accelerated the need for efficient sources of compressed air. By the mid-1940s, the forged tips of drill steels had been replaced by tungsten carbide inserts. Further developments included large crawler drills, which combine a percussive hammer with a separate high-torque rotation device for trouble-free drilling in even difficult rock.

COST AND PRODUCTIVITY COMPARISONS

Compressed air is a form of energy that provides beneficial features such as being safer to use in particular situations, allowing the use of lightweight power tools, being more economical to use and convey, being cleaner than some other forms of energy, and being easier to work with.

Although electricity per kilowatt-hour may cost less than compressed air, it is safer and more convenient to work with compressed air, particularly underground. Compressed air is widely used underground because of the safe nature of pneumatic motors compared to electrically driven tools, especially in areas where there may be flammable gases and sparks that would pose a risk.

A primary use underground tends to be in driving pneumatic tools such as drills. Large quantities of air are also used in removing dust. The air could be supplied by piping networks, or in the case of mobile equipment, by onboard compressors.

The energy savings from the use of compressed-air systems and air compressors could be 20% or more, particularly if variable-speed compressors are used to tailor consumption to current demand.

APPLICATIONS AND AIR-DRIVEN EQUIPMENT

Compressed air is used widely in mines, both in powering equipment and in providing air under pressure for operations such as clearing debris. It is used both in the handheld tools of semimechanized tunneling and with larger equipment for fully mechanized work.

Uses include hammer drills, where the air both powers the hammer and clears away the broken rock. The air may be supplied either by portable compressors or through lines fed from large static compressors at a central location, usually on the surface in the case of an underground mine.

Compressed air can be used at almost every step of a mine's production process. For instance, a copper mine could use it in the extraction, screening, crushing, milling, mechanical sorting, flotation, filtering, and drying operations.

Typical surface applications for air in mining include service air to operate tools, bagging operations, dust removal, and mineral filtration systems to remove buildups. Underground uses include handheld drills, operation of pneumatic tools and water pumps, and power machinery.

The amount of flow required is related to the application. Medium pressure in the range of 350 to 1,400 kPa (50 to 200 psi) will cover 90% of the applications, which vary from pneumatic conveying to pneumatic actuators and tools to auxiliary ventilation fans. The air can be used to operate machinery or pneumatic tools such as the wrenches and tools used in cutting and grinding. Other applications include cleaning.

Processing areas also benefit from dry air and the pressure required is typically in the range of 480 to 690 kPa (70 to 100 psi). Mineral filtration and separation operations on the other hand require higher pressures, typically 1,380 to 1,590 kPa (200 to 230 psi).

A term that is often used is *free air delivery* or *FAD*, which is the volume of air exiting the compressor expressed in terms of the inlet pressure and temperature conditions using the ideal gas law. (*Free air* is defined as air at normal atmospheric conditions [O'Neil 1939].) The difference between the FAD and the actual intake volume represents losses within the compressor. Thus, when comparing models, it is important to be aware that some manufacturers measure the air exiting the compression element rather than the intake volume.

Drilling, cleaning, and ventilation are key applications for compressed air in mining work. Developments in performance and reliability have been very important, as have improvements in the mobility and maneuverability of the equipment. Today's smaller and lighter compressors can be moved more easily to the next location, without the need for additional equipment to carry out the move.

Drilling Using Air-Driven Equipment

Compressed air plays an important role in the drilling of blastholes using techniques that combine the rotary action of drilling with the impact of a hammer. Injected air is used both in driving the hammer that is used to break up the rock and in blowing the debris out of the hole.

The air is used in techniques involving top hammer drilling, where the hammer is mounted at the top of the drill shaft, and in down-the-hole (DTH) drilling, where the intensity of the hammer action is increased by locating it on top of the drill bit rather than at the top of the drill string. Part of the volume of air is used to power the hammer, and the rest is used to flush the hole by removing the material that has been broken apart.

Demand for increased productivity in the benching at open-pit mines has spurred the use of hole diameters larger than 127 mm (5 in.) in diameter.

DTH drills can produce large-diameter holes, with straighter alignments than could be achieved with a top hammer.

Top hammer drilling is, however, a particularly economical method that is ideal for use in good, competent rock for holes of short to medium depth, with little if any deviation and no requirement for large-diameter holes. It gives a good penetration rate for high production capacity and its energy costs are low.

The hydraulic rock drill delivers the impact energy through a string of sectional drill steel to the drill bit. This method is fast, though achieving a straight hole can be more difficult in certain drilling conditions. There is an approximately 10% energy loss in each coupling joint.

The DTH method is also commonly used and comes into its own for drilling holes that go deeper, have large diameters, and where hole straightness has a higher priority than penetration rate. It is also chosen when drilling in extremely broken and fractured rock, where the flushing requirements are very high.

The rock hammer is positioned directly behind the drill bit, ensuring minimal power losses. High-pressure air is used for impact and as exhaust air for flushing the hole.

DTH drilling is a highly reliable method of producing clean, straight holes in a variety of drilling conditions. However, this method gives slower penetration rates than top hammer drilling.

DTH drilling applications include blastholes in production drilling, drop-raise cut holes and pilot holes in raise driving, and ventilation and communications holes for drainage.

Atlas Copco also developed the COPROD system for use where drilling conditions are difficult, where there is broken rock or where higher capacity/productivity is required. It combines the straight-hole drilling accuracy of DTH drilling with a speed superior to top hammer drilling. The system consists of impact rods combined with drill tubes to produce high penetration rates and longer drill steel life. Rotation and percussion are separated, optimizing the drill string for each function. The impact rods are stacked so that they are flush, which transmits percussion without the loss of energy that occurs in the coupling joints of conventional drilling methods.

The main difference between pneumatic and hydraulic drill rigs is simply whether the rock drill's hitting and rotating mechanism is driven by compressed air or by hydraulic oil. The track movement can also be propelled by compressed air.

Increasing Drilling Efficiency

The use of compressed air is vital in the creation of the deep, large-diameter holes that are essential for achieving optimum blast economy in open-pit mines and quarries. The burden between the free face and the drill hole is dependent on the hole diameter, spacing, and depth. Being able to create larger-diameter holes can significantly reduce the number of feet to be drilled, hence, increasing productivity.

Air Flushing

Increasing power output from rock drills leads to higher penetration rates so that efficient flushing of the cuttings becomes increasingly important. The flushing medium is normally air for surface drilling and water for underground drilling. The flushing air comes from a pressure tank supplied by one or more compressors, where the compressor is regulated to maintain the required supply.

Air is blown down through the drill bit, and the cuttings are removed via the annulus surrounding the drill shaft. The cuttings can be bound by a flushing additive added to the working air or can be removed using a dust collector.

Flushing is used to clean the bottom of the hole to prepare it for the next blow and to carry the cuttings out of the hole. The first of these tasks usually gives no trouble, although it is necessary to ensure that the passageways for the cuttings on the sides of the drill bit are sufficiently large and that the flushing holes in the drill bit are directed so that effective cleaning of the bottom of the hole is ensured.

The second task is, however, more demanding. The ability of air to carry the cuttings away will depend on the nature of the particles. To carry cuttings out of the hole, the carrying velocity must be greater than the settling velocity of the material. The settling velocity is calculated using the Newton–Rittinger equation, as the flow of the cuttings will generally be turbulent:

$$v_s = (4gd(\rho_s - \rho_a)/(3C_d\rho_a))^{0.5} \qquad (9.2\text{-}1)$$

where

v_s = settling velocity
g = gravitational acceleration
d = particle diameter
ρ_s = specific gravity of solid particle
ρ_a = specific gravity of air

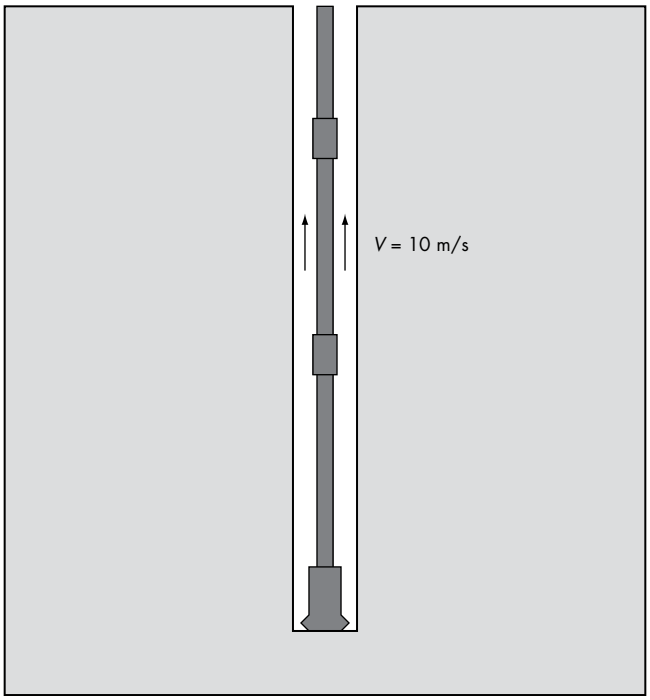


Figure 9.2-1 Flushing velocity in a drill hole

C_d = drag coefficient (usually taken as 0.43)

Equation 9.2-1 shows that the flushing velocity is proportional to the density and diameter of the solid particle. Although not represented in Equation 9.2-1, particle shape also has an effect—it is easier to carry away a particle in the shape of a leaf than one of the same weight that is spherical.

Using Equation 9.2-1, one finds that for a 2-mm- (0.08-in.-) diameter particle with a density of 2.0 g/cm^3 (125 lb/ft^3), the air velocity up in the drill hole must be at least 10 m/s (32 ft/s). A velocity of 10 m/s is considered the minimum required for rock particles of normal density (see Figure 9.2-1).

The flushing air requirements depend on the drill bit size, the rod size, the rock type, and the condition. The necessary air volume can be calculated, as the bit diameter and the rod diameter are known. A 76-mm- (3-in.-) diameter drill bit and 38-mm- (1.5-in.-) diameter rod would require 35 L/s (74 cfm) at an air velocity of 10 m/s (33 ft/s). The same rod in a 102-mm- (4-in.-) diameter hole would require 70 L/s (148.5 cfm) of air.

$$\text{area of annulus} = \pi(76^2\ \text{mm}^2 - 38^2\ \text{mm}^2)/4 = 3{,}400\ \text{mm}^2$$

With an air velocity of 10 m/s, the airflow rate is given by

$$V = 10\ \text{m/s} \times 3{,}400\ \text{mm}^2 = 34\ \text{L/s (72 cfm)}$$

If, on the other hand, one wants to drill holes with a diameter of 102 mm (4 in.) (using FI38 equipment), the annular cross-sectional area is given by

$$\text{area of annulus} = \pi(102^2\ \text{mm}^2 - 38^2\ \text{mm}^2)/4 = 7{,}040\ \text{mm}^2$$

With an air velocity of 10 m/s, the volume is given by

$$V = 10\ \text{m/s} \times 7{,}040\ \text{mm}^2 = 70\ \text{L/s (148 cfm)}$$

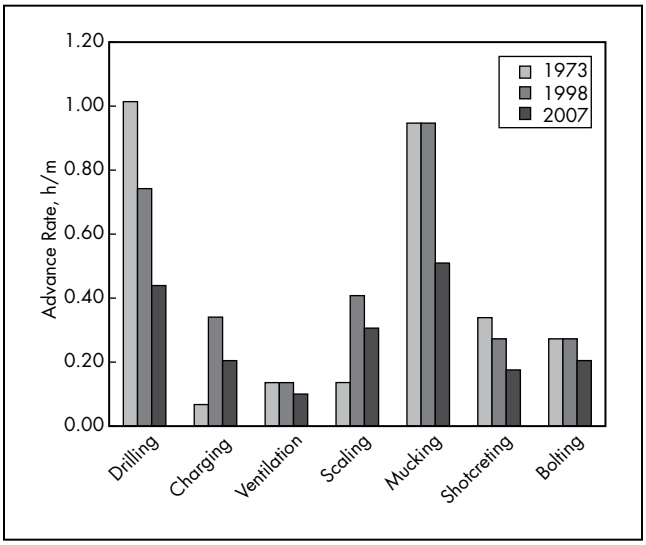


Figure 9.2-2 Relative costs over time of different production activities

Determining Air Pressure and Flows

The air required to clear holes depends on factors including the material to be removed, the drill method, and the hole dimensions.

The amount of air required depends on the equipment and the compressed air pressure chosen. The total air requirement for a typical Atlas Copco 89-mm- (3.5-in.-) DTH hammer at 586 kPa (85 psi) would be 46 L/s (97 cfm). This rises to 200 L/s (425 cfm) at 2,480 kPa (360 psi). For a 150-mm (6-in.) hammer, the airflow required for the hammer would range from 84 L/s (178 cfm) at 85 psi to 480 L/s (1,015 cfm) at 360 psi.

Planning for the total compressed airflow required also requires other aspects to be taken into account. The total air requirement for a drilling task is built up from the demands of the various operations involved, which can include the use of hammers, rotation units, feed motors, dust collectors, and air flushing.

A rig fitted with a 102-mm (4-in.) hammer operating at 2,000 kPa (290 psi) might require 205 L/s (434 cfm) for the hammer itself, but the operation would require twice this amount when the air for the feed motor, rotation unit, and dust collector are taken into account.

Similarly, a 586-kPa (85-psi) operation involving a mid-sized rock drill could require 268 L/s (568 cfm) to operate the drill, but it would require a further 25 L/s (53 cfm) for the feed motor, 70 L/s (148 cfm) for the flushing, and 65 L/s (138 cfm) for the dust collection.

Falling Costs

Time spent per foot of advance at a tunnel face has fallen markedly in recent decades for the sequential operations of drilling, charging, ventilation, scaling, mucking, shotcreting, and bolting (see Figure 9.2-2).

TYPES OF COMPRESSORS

The many types of compressors are divided broadly into two families: positive displacement and dynamic (or centrifugal) types. The most common type of dynamic air compressor is the rotary screw, which is available in oil-lubricated or oil-free versions. Piston compressors have been around since compressors were first invented, but advances continue to be made to create high-pressure models with advantages in efficiency, ease of maintenance, and reduced life-cycle costs.

In an oil-lubricated screw compressor, air passes through the air inlet filter where dust particles are separated. Inside the screw element, air is mixed with the injected oil and compressed. The oil is used to seal the clearance between the rotors and the casing of the element, and it ensures friction of the elements and cools the output air and screw elements. A separator removes the oil from the air, first through centrifugal force and then through filtration.

In oil-free screw compressors, the gap between the rotor elements is minimal. There is no oil injected in the screw element. The male and female rotors receive a special coating, and, as they do not touch, there is no friction and less heat production. The jacket of the element is cooled. The air then passes through an aftercooler, where it is cooled. The condensate is then separated and drained away.

A dryer cools the air through a refrigeration cycle to 14.8°C (5.4°F), and the further condensate that is generated is removed. The air then flows through the integrated dryer heat exchanger, where it is warmed by the incoming warm, humid air coming from the compressor's aftercooler.

The final step on leaving the dryer is to filter the compressed air.

Quality/reliability is generally seen as the most important factor behind the choice of a compressor model.

Portable and Stationary Systems

Having determined the use of rotary, oil-injected air compressors, the mine can decide on the use for an electric-powered stationary installation or diesel-powered portable units.

If a ready and stable electrical supply is available, it might appear that the stationary installation is the preferred option. But it is never that easy, so additional factors to be taken into consideration include

- Cost of diesel versus electricity,
- Stage of development (is the site a greenfield or a brownfield, or is it at the mining setup stage?),
- Distance of the infrastructure and place of work,
- Logistics (complexity of transporting pylons or diesel to the site, plus criteria of storing the diesel on-site), and
- Life expectancy of the mining operation.

Many underground mines today have less need for compressed air circuits than in the past, thanks to the growing use of portable compressors or compressors built into mobile equipment. Portable air-driven equipment such as drills have become increasingly efficient, and they provide great flexibility in their ability to be moved to wherever required.

Another factor has been the advent of load-haul-dump vehicles. Their use has diminished the need for the slushers (double-drum hoists), which were commonly used to move ore and had compressed air motors.

Some underground mines have dispensed completely with a stationary compressed air plant, though most still have a compressed air network. Its uses are varied. It can be used, for instance, to blow away dirt and pipe blockages, apply sprayed concrete, clean tools, clear or cool the air, and power hand tools such as drills. It can also be used as a hazard-warning system if stench gas is added.

Typical compressed air requirements in an underground mine will be for air at 7.8 to 9.2 bar (100 to 120 psig).

Where static compressors are being used, the typical solution will involve using a combination of fixed compressors—which are always either simply loaded or unloaded—and ones with a variable-speed drive (VSD) that cater for the peaks and troughs of the load cycle. Standby compression capacity would also be installed for wherever maintenance is required or in case of failure.

Use of VSDs minimizes the energy required to maintain the required pressure. Precise control of the pressure can be achieved through an external regulating system. This can be preprogrammed with all the required criteria for system operation using rules to ensure optimum efficiency, including factors such as ensuring that operational hours are shared between main and standby machines. This external regulating system is able to obtain substantial energy savings.

Fixed static compressors are generally sited on the surface, with only the piping taken belowground. This ensures sufficient air for the cooling of the compressors, as well as avoiding the harsh underground environment and the difficulties of delivering such large pieces of equipment underground.

Some stationary compressors are available that are built into standard 6-m- (20-ft-) high cube freight containers to allow outdoor use, saving the need for a separate container room.

Applications for small portable compressors, which deliver from 30 to 88 L/s (67 to 187 cfm) in the 8 to 12.5 bar (100 to 170 psig) range, include driving pneumatic tools such as chipping hammers, jackhammers, grinding tools, and rammers. They are also suitable for uses that require high FAD, such as sandblasting. More powerful models provide higher free air delivery while still remaining small enough for ease of transport and maneuvering.

Today's small, light equipment means that the operator can move to the next work point easily, without needing additional equipment to transport the compressor.

The next tier includes heavy-duty models designed for environments such as quarries. These models deliver about 165 to 183 L/s (350 to 388 cfm) at 8 to 12.5 bar (100 to 170 psig). Typical uses include rock drilling.

Where higher pressures between 9.5 and 14.6 bar (125 and 200 psig) are needed, there is a choice of models catering for flow rates between 233 and 400 L/s (494 and 847 cfm). Higher pressures can be achieved through the use of two-stage portable compressors capable of delivering up to 31 bar (440 psig).

The ability to deliver ever-higher pressures has resulted in productivity gains in mines. Changing from a regular-sized 26-bar (365-psig) machine to a 31-bar (440-psig) machine brings considerable benefits. Most machines are built for 26-bar (365-psig) pressures.

Altough it may be possible for the compressor element itself to produce 31 bar (440 psig) or more, other components such as seals and pressure vessels must be able to operate at the higher nominal pressure. Because most components may need changing to achieve such an upgrade, the resulting machine will inevitably be more expensive—the increased capital cost must be balanced against the reduced operating costs due to the productivity gain.

Operations that require a high volume of air include rock drilling. For this application, the compressor needs to be easy to operate as well as easy to relocate for quick start-up in quarries and maximum productivity.

Compressors can also be built into vehicles to suit a customer's needs and provide air power at remote locations—for instance, using a truck that has a power take-off from the drive train. From the power take-off, a belt drive or drive shaft can be used to take power from the engine. The engine is used to power the vehicle when it is driving and to power up the compressor when the vehicle is stationary at the site where the compressor is needed.

Oil-Lubricated Versus Oil-Free Compressors

Stationary compressors come into two broad categories: oil-lubricated and oil-free. In oil-lubricated compressors, the lubricating oil serves a number of purposes:

- It seals the clearance gaps between the screws and the casing.
- It lubricates the drive between the male and female screws.
- It assists in extraction of the heat generated during compression.

After the compression phase, a mixture of compressed air and lubricant (oil) leaves the air end and is passed to an oil separator, where the lubricant is removed. But part of the oil is exhausted from the compressor, usually as an aerosol or in a mist form.

In mining, compressed air is sometimes used to supplement ventilation air through the use of Venturi blowers. The presence of oil could be an issue in these circumstances, although the amount of air supplied in this manner is generally small. The air quality would generally only be an issue if a compressor were being used to supply a majority of the air in long-dead headings with little other air supply.

If oil-free air is required, filters are necessary on oil-lubricated compressors. The costs for maintaining and replacing the filters must be balanced against the smaller capital cost for the compressor compared to an oil-free compressor.

Another aspect in choosing between oil-lubricated and oil-free air compressors is the total power requirements for the compressor. Oil-lubricated compressors are not available beyond 500 kW (680 hp), limited by physics and economics, whereas oil-free compressors can be obtained in megawatt sizes.

Atlas Copco has an installation in an underground Turkish mine where about 8 MW of oil-free compressors are in use. This application would require 16 of the biggest oil-lubricated machines, but significantly fewer oil-free machines. More machines also mean greater maintenance costs. The compressors are installed on the surface, with tubing and pipes to transport the air underground.

Oil-lubricated compressors tend to be chosen when compressors need to be installed underground, as they are less sensitive to the local environment as well as being easier to transport.

Boosters

Today there is a growing demand in mining for much higher pressures that cannot be met by an air compressor alone. Typically, for example, a portable oil-injected screw compressor delivers air at pressures between 7.8 and 31 bar (100 and 440 psig). Compressed air at pressures between 51 and 346 bar (735 and 5,075 psig) requires a booster to increase the pressure.

The outlet of the feed-air compressor is fed into the booster as its inlet. Single-stage booster compressors therefore take in air at perhaps 26 bar (360 psig)—instead of atmospheric

pressure—compressing the air further to achieve the desired pressure up to 70 bar (1,015 psig). Multistage boosters are used to take the pressure up to a maximum pressure of 346 bar (5,070 psig).

Booster applications include minerals exploration drilling using reverse circulation techniques and coal-bed methane, where drilling very deep holes, up to several miles deep, can be demanded. Other applications for boosters include nitrogen injection for mine fire suppression, eliminating the risk of explosion and the risk of oxidation on valves, pipes, and drill bits.

The booster is built on a modified engine block—typically a John Deere or Cummins equipped with booster pistons mounted on modified engine pistons and using different sizes depending on pressure/flow. The capacity of the booster is determined by the inlet pressure and the speed of the driver engine. The higher the revolutions per minute, the more FAD the booster will displace.

ECONOMIES OF DRILLING

In making a purchase decision, people often only consider the purchase price, but it is important to obtain information from suppliers about the actual operating costs of the equipment. This ensures that allowance is made for hidden costs such as preventive maintenance.

Life-Cycle Costs

Manufacturers of compressors continue to carry out development work to optimize the production of air. The aim is to maximize FAD for the lowest power consumption, as this is the main factor in driving down energy costs.

The purchase price is only a small part of the cost of providing compressed air. The quality and service life of the compressor and its components, costs of energy (70%) and maintenance, and the indirect costs of downtime are among other factors to take into account.

Accurate calculation of the total cost of ownership is a complex matter, and some manufacturers produce calculators to help would-be purchasers of pneumatic drilling equipment take into account all the variables, including factors such as annual operating hours for the equipment, depreciation costs, interest on financing, and fuel and operator costs to arrive at total hourly costs.

Two other key elements influencing system operating costs are operating pressure and air leakage. Reductions in either can result in considerable savings. Air leaks have been estimated to waste as much as 30% of many systems' output, and most compressed air systems that are more than 5 years old are estimated to have a leakage rate of about 20% of the total volume consumed. Even a 3-mm (0.12-in.) air leak is estimated to represent almost 42,000 kW·h of wasted energy per year.

As little as 1% moisture in the oil can reduce the life of the compressor's bearings by 40%. Today's compressor designs avoid this through electronic control of the oil-temperature system.

Automatic control systems also play an important role in ensuring efficient operation, for instance, by matching use of VSDs to fluctuations in flow.

Manufacturers develop systems to improve the efficiency of air production and the quality of the compressed air. Recent electronic developments regulate the engine speed and the inlet air to optimize fuel consumption.

The volume flow rate is not a measurement of actual gas volume in a compressed state, but rather of air that has been re-expanded back to inlet conditions. A 47-L/s (100-cfm) compressor takes in 47 L/s (100 cfm) of atmospheric (ambient) air through its intake and may compress it to a gauge pressure of 7.8 bar (100 psig). The original 47 L (100 ft^3) of air has been reduced in volume 7.8 times, and it now only occupies 6 L (12.8 ft^3) in its compressed state.

The compressor used with a top hammer rig will typically be operated for just 45% of the cycle, being operated only when the hole is actually being drilled. For perhaps 35% of the cycle, the compressor will be unloaded but still running and retaining its pressure while actions are carried out such as changing the rods. For the remainder of a cycle, the rig will be in transit, and the pressure vessel can be depressurized.

Appreciation of the times for loading/unloading during the cycle is an important factor in calculating operating costs for the equipment. A typical example is shown in Figure 9.2-3, which indicates times such as 2 to 6 minutes for drilling each rod, up to a minute for adding the next, a few minutes for collecting the rods before moving to the next position, and time for providing a collar to seal the hole.

The deeper the hole, the greater the inlet pressure and flow that will be required to clean it. Maintaining a constant air speed will require increased pressure.

Compressor flow is a function of the pressure at the compressor inlet. It also depends on various factors, including flushing speed, distance, and volumetric efficiency.

Compressors are rated for operation in a particular range of ambient temperatures, which can be an important factor in mining. However, a machine that can work in higher temperatures will cost more, and there may be no need for the extra expense.

The latest advanced machines have been built to perform in a wider range of ambient temperatures—including options to work at up to 50°C (122°F)—and to save energy though the use of features such as automated switching off of the dryer in light load conditions.

Energy Savings and Heat Recovery

Rising fuel costs and environmental issues have increased demand for energy-saving equipment. Manufacturers continue to develop new ways of improving efficiency. For instance, Atlas Copco's compressors feature a fuel-saving system that operates when the compressor is under partial loads. Savings are typically 10% to 20% compared to a similar machine without the system. The investment cost of a system such as this can soon pay for itself in countries where fuel costs are high or where fuel supply is difficult, as in remote locations.

Energy usage is becoming increasingly important, and manufacturers aim to combine the best possible FAD with a reduced specific-energy requirement. The use of advanced computational-fluid-dynamics techniques in designing the equipment helps minimize internal pressure drops and drive down the energy costs. There are sound economic reasons for operating a compressor at the lowest possible discharge pressure.

Economical Compression Through Variable-Speed Drives

The cost of compressed air can represent more than 40% of the total electrical costs. Therefore, energy-efficient solutions can bring substantial savings. One of the key ways of doing this is to use a VSD, which tunes the total compressor

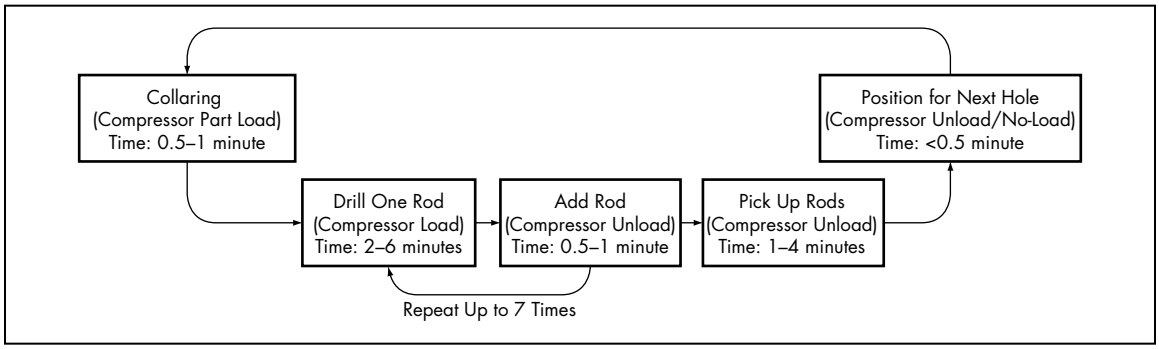


Figure 9.2-3 Loading/unloading times during cycle

capacity from a series of stationary compressors to match the demand for air. Besides providing immediate energy savings, a VSD can reduce overall life-cycle costs of the compressor by an average of 22%. Fixed-speed machines are ideal when they can run at full load for most of their operating time, but where air demand fluctuates, VSDs can achieve savings of up to 35%. The VSD enables provision of constant net pressure, which provides stability for all processes that use compressed air. Peaks in electrical current during start-up are eliminated, and VSD compressors can be started and stopped without limitation.

Another way of saving is to recover energy, through an integrated heat-exchanging module that makes direct use of the heat coming out of the compression element and transforms it into hot water as a form of energy. Uses include space heating and showers.

Preventive Maintenance

Proper maintenance is vital to prevent breakdowns, especially in hostile environments such as underground mines. The ideal is to carry out the maintenance before the machine breaks down, and a growing body of evidence about component life makes it ever easier to schedule maintenance and replace parts at the optimum time.

Manufacturers complain that compressors at mines tend to be undermaintained. The equipment may be underground, exposed to dusty environments, and at risk of a much shorter life than in less-hostile conditions. Many mining companies use their own maintenance crews to service the machines instead of bringing in the manufacturer's expertise. This can be another contributory factor to reduced life if in-house personnel have not received adequate training.

It takes effort to bring the machine up to the surface; consequently, maintenance of bigger underground machines tends to be performed in place.

Atlas Copco points out that in most industries about 50% of its compressors are regularly serviced by its maintenance crews, whereas in mining the proportion is closer to 5%. When machine maintenance is neglected, major problems are likely to occur and operating costs increase. In combination with what can be a hostile environment, lack of maintenance can contribute to an unnecessarily short equipment life in a mine—perhaps 3, 4, or 5 years compared to twice that in other situations.

Special options, such as heavy-duty filtration, are available for dusty environments, and trained technicians would immediately identify where these would be beneficial. The lack of contact with suppliers often means that compressors are ordered without discussing the operating environment by centralized purchasing based on initial costs alone. Specifying extra features such as heavy-duty filtration is particularly important in situations where maintenance is likely to be difficult and sporadic.

New technology brings advanced options for monitoring wear to help plan preventive maintenance, which can help overcome the traditional shortcomings in mine maintenance.

Some types of mining, such as salt mining, are particularly corrosive with a consequent impact on the life of wear parts. In situations such as this, there is a temptation to shun maintenance given that the machine's life will inevitably be short. Nevertheless, proper service would help extend the life of the equipment.

Compressor Rental

Rental, even of major machines, can be useful in cases of short-term additional demand and can be useful from a financing and tax perspective by spreading the costs and dispensing with the need to buy a major plant. Furthermore, with a rental, the supplier is responsible for all maintenance and for supplying a replacement in case of breakdown.

For instance, Atlas Copco has a division dedicated to specialty rental with a focus on industrial applications and the mining sector. The fleet specializes in oil-free and oil-injected high-pressure compressors for mining applications, as well as generating sets. The units deliver air at the rate of up to 755 L/s (1,600 cfm) with pressures up to 45 bar (650 psig). High-pressure compressors can be mounted in a crash frame and are used for deep-drilling application; deeper holes means that higher pressures are required to extract cuttings. Compressors are also available housed in 6 m (20 ft) International Organization for Standardization (ISO) containers that offer ease of transportation (see Figures 9.2-4 and 9.2-5).

Demand for a specialty rental fleet is particularly high in the United States, and rentals are typically from a week to a year, which is ideal for situations where the compressor needs to be constantly relocated. Being diesel driven, there is no need for a generator, making it ideal for use in remote locations. This type of equipment is also brought in where there is a short-term demand for extra air above that available from the permanent installation or in the case of a breakdown. Low-pressure rental units can be used for driving pneumatic tools, breakers, pneumatic pumps, diaphragm pumps, and general cleaning duties. Medium-pressure units, for example, 9.5 bar (125 psig), are used to inflate mine truck tires, to power high-flow pumps, and to supply air to blast furnaces.

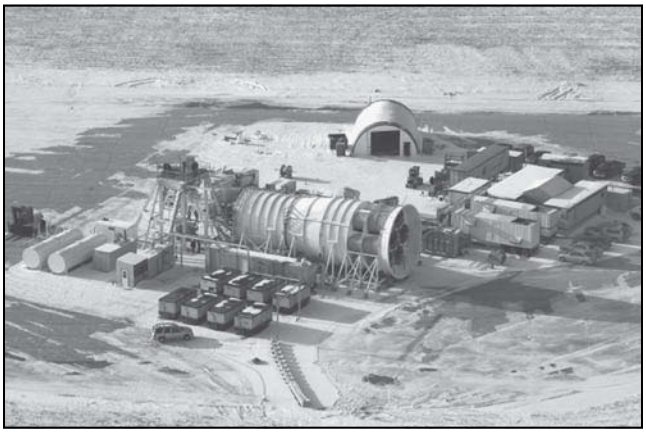


Figure 9.2-4 Compressors in ISO containers at a remote site

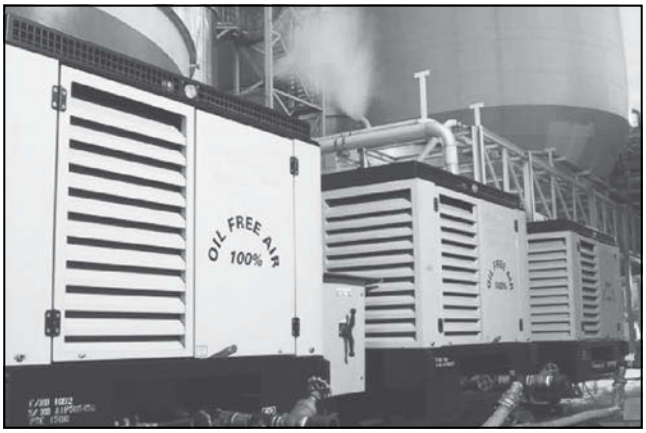


Figure 9.2-5 Compressors in ISO containers

AIR QUALITY

Necessity for Treating Air

Attention has to be paid to minimizing three contaminants that occur in compressed air: solids, moisture, and oil residues. These can otherwise contaminate the compressed air to a considerable degree, leading to corrosion in pipes and resulting in costly repairs. Air treatment solutions can be thought of as insurance for the plant, rather than simply an extra cost. Specification of equipment to remove moisture in particular can be thought of as insurance against corrosion, malfunctions, and production shutdowns. Without it, the costs can be high.

Untreated air may cause extensive damage and lead to serious performance degradation. All plants need basic air treatment to protect equipment, production processes, and end products; more critical applications require enhanced solutions to meet air quality requirements.

Equipment can be specified to improve the quality of the air downstream from the compressor by removing contaminants, including moisture, particles, and oil. The presence of these can be harmful in particular situations. Some form of air treatment is normally required to provide compressed air of the quality required at the point of use. Otherwise, there could be severe effects from the contamination, including damage to tools and valves; corrosion and damage to equipment, piping, and machinery; and contamination of products. Contaminants can cause rust and scale to form within pipelines, leading to the increased wear and maintenance of pneumatic devices, sluggish and inconsistent operation of air valves and cylinders, malfunction and high maintenance of control instruments, and effects such as spotting in paint and other types of spraying.

Many plants need only part of the air treated to a very high quality. (However, if the majority of the air is needed at a high quality, it would make sense to treat all compressed air to the level required by the highest-quality user.) Where compressed air is supplied from a central point, there may be purification equipment in the compressor room to protect the distribution system, with further purification at the point of application. An extensive range of purification equipment is available for both centralized and decentralized compressed air systems. This allows the user to tailor the quality of air for each specific application, from general-purpose ring-main protection through critical clean dry air point-of-use systems.

Purification may also be required for decentralized applications such as operation of pneumatic equipment, machinery and tools, and air instruments. Air treatment that removes moisture and particles increases efficiency—the production of dry, clean air prevents corrosion. Clean, dry air extends the life of equipment in the mine, enhances system reliability, and avoids costly downtime and production delays.

Design Criteria for Air Purity

Whether designing new compressed air systems or reviewing existing systems, the first step is to define the compressed air purity requirements. Air purity requirements vary widely but can all be met with the right equipment. However, unnecessary levels of air purification can significantly increase the associated energy cost.

The quality of air required depends on the application, particularly for oil-free air requirements. To assist in the selection of the necessary air purity, a series of classifications has been developed. Many applications need only part of the air treated to a high quality. In such cases, it is generally more effective to treat all the generated air to a minimum acceptable level and to improve the quality to the desired level close to the usage point. Points to consider for a given application include

- Required air quality,
- Estimated air demand,
- Required working pressure,
- Atmospheric contamination,
- Position and type of compressor intake,
- Temperature of the compressed and cooling air, and
- Condition and configuration of existing piping.

Choice of Treatment Approaches

Air quality requirements vary considerably from application to application. To achieve the levels of cleanliness specified by ISO 8573-1 (2001), careful thought must be given to systems design, commissioning, and operation. Point-of-use purification equipment should be installed as close to the application as possible to remove the maximum amount of rust and pipe scale prior to the application.

Filtration systems can be used with oil-injected compressors to yield air of a satisfactory quality. Different applications have different air quality demands. The ISO 8573-1 standard specifies the quality of the air, in terms of particle content—the quantity of dust particles that the compressed air can contain—as well as the acceptable oil and moisture contents.

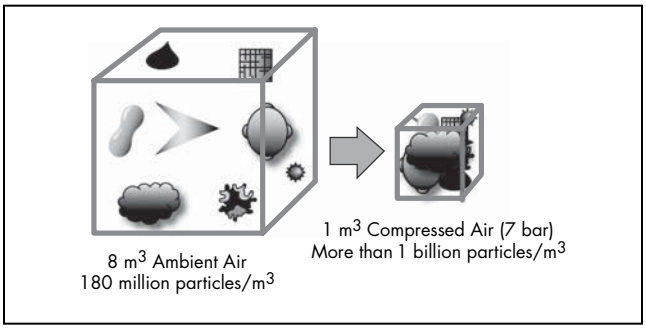


Figure 9.2-6 Increase in dust concentration due to compression of air

A primary compressor can be fitted with different purification systems according to specific categories within ISO 8573-1 and hence be suitable for different applications.

Oil filters are extremely important for ensuring the reliability and availability of compressors. Heavy-duty filters are available that can retain 99% of particles as small as 10 μm, in contrast to conventional filters that retain 50-μm particles. Another advance in reducing component wear is the use of special oil for lubrication of rotating parts, which brings about a significant extension in the life of components. Such changes have resulted in a doubling of the filter life, which is particularly beneficial in remote locations.

Concentrated Contamination

The compressor uses ambient air for compression. Without treatment, any contaminant in the air, such as dust, will also be in the compressed air. Even with filtration, some contaminants will remain, resulting in the introduction of dust into equipment downstream of the compressor, which could cause malfunctions.

Compression reduces the air volume and thus increases the concentrations of airborne contaminants in the intake air. Consequently, compressed air generally requires some form of treatment. These contaminants may also be joined by other substances originating within the compressor or in the piping and hoses. Research indicates that dust concentrations of 173,000 particles/cm^3 are common. Up to 80% of these particles are small enough to pass through a compressor intake filter.

In a typical example, the air volume is reduced by compression to one-seventh of its original. The resulting concentrations of particles such as oil and dust are not compressed, and their presence will be increased by a factor of 7. Thus, as shown in Figure 9.2-6, if there were 180 million particles of contaminant in the original volume, there would be more than a billion after compression.

Preventing Blockages

The presence of solid contaminants is the result of corrosion and rust plus mineral deposits, all of which build up within the aftercooler, storage tank, and piping system. Contributing to this load of solid particulates are solids that bypass the intake filter of the compressor as well as particles resulting from compressor wear. These abrasives range in size from microscopic to large bits and flakes.

Particles can be removed through filters, which are a cost-effective means of eliminating all types of contamination with minimal pressure loss. To protect the compressor from incoming dirt, a prefilter is always provided. In the case of piston, vane, and screw compressors, these filters have a dirt retention capacity of around 5 μm. Centrifugal compressors are more sensitive to incoming dirt, and inlet filtration is normally in two stages, retaining down to 0.2 μm.

Although droplets are visible to the naked eye, particles such as coal dust may only be visible under a microscope, whereas a scanning electron microscope is necessary to identify particles such as paint pigment and oil vapor. Particulate matter includes small solid and liquid particles such as dust, smoke, sand, pollen, mist, fly ash, viruses, and bacteria.

The presence of particles such as dust can block a pipe, increasing resistance to the airflow and, therefore, increasing energy use, decreasing performance, and increasing operating costs. Moisture can add to a blockage by forming a sludge. Moisture also promotes fungal and bacterial growth. Pneumatically controlled valves that control items such as conveying or filtration equipment may also become blocked over time, with a resulting impact on the equipment they control as well as downtime for repairs. With a blockage, a compressor will need to be run at a higher pressure to achieve the desired output pressure, which requires more energy.

Reducing Moisture

Water is probably the most familiar example of a liquid contaminant in compressed air. When the air is compressed, the density of the moisture increases. Water in pressurized air does not present a problem as long as it remains in a vapor state. But condensed water can lead to corrosion in pipes, interference with the lubrication of air tools, and the risk of freezing in pipes and tools.

Moisture is an important pollutant to consider in mining system design. It causes two problems—condensation, which can be troublesome downstream, and corrosion that can develop. Dry air is important for avoiding corrosion of air lines and the subsequent leaks and pressure losses. Replacing a long underground pipe is expensive. Corrosion begins to occur even before condensation is observed. It destroys the interiors of both piping and equipment. Ultimately, it leads to pipe ruptures and equipment malfunctions. Corrosion also introduces rust particles that can damage the equipment using the air. Further costs come from the shutdowns needed to reinstall the pipes and equipment.

Condensation occurs at a relative humidity of 100%, but corrosion starts even at a relative humidity of 50%. When it reaches 60%, there is an exponential growth in the rate of corrosion (Figure 9.2-7).

Moisture can also lead to damage in the bearings of any equipment that uses oil films for lubrication. There can also be resulting losses if air is used to control the filling of containers with final products—valves may stick, resulting in under- or overfilling. The corrosion caused by moisture in the air piping can drastically increase operational costs. Leakage translates into a loss of air capacity. This means increased usage of the compressor as it works to compensate.

A 5-mm (0.2-in.) hole leaks 27 L/s (57 cfm) at 7 bar (90 psig). To compensate for this loss of airflow capacity, 8.3 kW (11 hp) of additional compressor power is required, which can equate to costs of thousands of dollars per year, even from such a relatively small hole.

Even a 3-mm (0.12-in.) hole can cause leakages of up to 10 L/s (22 cfm), which in terms of energy consumption equals an annual cost running into thousands of euros/dollars/

pounds, as shown in Table 9.2-1. This 3-mm hole would leak 100 L/s (212 cfm) at 6 bar (85 psig), giving a power loss at the compressor of 3.1 kW (4.2 hp). For a 10-mm (0.39-in.) hole, the energy loss would be increased tenfold.

Humidity

The maximum water content that the air can hold without condensation depends on the air temperature—more can be held at higher temperatures. At high air temperatures, a relatively large quantity of vapor can be carried in the air, but condensation occurs as the air cools. Moisture condenses out of the air as it encounters lower temperatures. At night, as the air temperature cools and the air's capacity to hold moisture is reduced, dew forms. The air contacts the colder surface and sheds moisture.

Every kilogram of air contains a certain quantity of water in grams, which is the absolute air humidity. Relative humidity is a comparison of how much water vapor is in the air compared to the vapor pressure at the ambient temperature and pressure, which is the vapor capacity. Relative humidity is therefore expressed as a percentage. If the temperature of a volume of air is raised without adding more water vapor, then its relative humidity decreases.

The dew point of air is the temperature at which the water vapor held in the compressed air is equal to the compressed air's capacity to hold water vapor. It is the temperature at which condensation starts to occur when the air is being cooled. If the temperature of the air drops any lower than the dew point, the additional water vapor will condense and drop out in liquid aerosol form.

For instance, inlet air with a temperature of 30°C (86°F) and a relative humidity of 50% will have a dew point of 18°C (64.4°F) (see Figure 9.2-8). After passing through the aftercooler, the air temperature may be (ambient +10) 40°C (104°F), with 100% relative humidity.

Treatment in a refrigeration dryer can reduce the temperature to 20°C (68°F) and the pressure dew point to –15°C (5.4°F), greatly reducing the likelihood of moisture droplets forming.

For most industrial applications, compressed air can be considered dry when there is no liquid water present. Liquid water causes corrosion, contributes to the mineral deposits found in air pipes, and washes away the necessary lubricants required in air valves, cylinders, and motors.

For every 20°C (36°F) increase, the capacity of air to hold water vapor doubles. Conversely, for every 20°C (36°F) decrease, the capacity to hold water vapor is cut in half. Removing the water from compressed air can be accomplished in various ways. An air receiver stores the air after the compressor. Hot compressed air cools in the air receiver, where the condensation water is collected and separated out. This is the least expensive way and though an old method, it also gives the lowest degree of water separation.

Mining applications normally feature an aftercooler. Aftercoolers are used in virtually all stationary compressor installations as a standard feature. If the air goes in at 25°C (77°F), it will typically emerge (before going into the aftercooler) at 90°C (194°F). The aftercooler cools it to perhaps 35° to 40°C (95° to 104°F), removing most of the moisture. The ambient air will generally be cooler than this, resulting in heat exchange between the pipe and the surroundings so that the temperature in the pipeline drops, which leads to condensation. Optional dryers can be used to further reduce the moisture content of the compressed air to very small quantities.

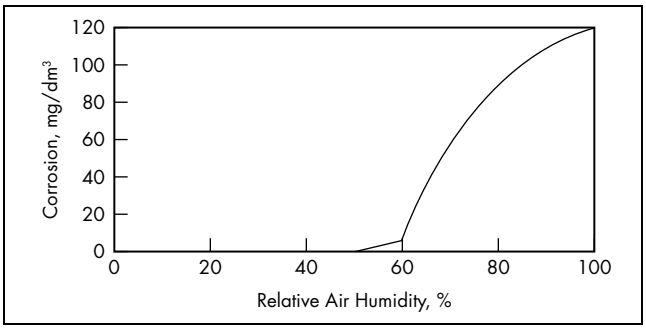


Figure 9.2-7 Corrosion as a function of relative humidity

Table 9.2-1 Compressor leakage losses

	Hole diameter, mm			
Measurement	**1**	**3**	**5**	**10**
Leakage, L/s at 6 bar	1	10	27	105
Power loss at compressor, kW	0.3	3.1	8.3	33
Energy loss at 24/7 kW·h	2,628	27,156	72,708	289,080
Annual costs,* euros, €	131	1,358	3,635	14,454

*Assuming 1 kW·h costs € 0.05.

Air exits a compressor with 100% humidity, and droplets of water form as soon as the temperature falls around the piping. Refrigerant drying is used to condense and separate a large amount of the water. Refrigerant dryers remove moisture from compressed air in even the harshest conditions. Other types of dryers include adsorption dryers, heat-of-compression dryers, and membrane dryers, with the choice depending on the desired dew point.

COMPRESSED AIR NETWORK DISTRIBUTION

Portable compressors can be set up alongside drilling equipment and can supply air via a short length of hose. However, some mines use permanent networks fed by fixed pipelines from banks of stationary compressors. The most important issue in designing a piping network is not the quantity of air but rather achieving the correct pressure to operate the equipment. Air pressure is reduced because of the friction in passing through obstacles such as restrictions, bends, and narrow passages. The pressure drop increases if the air supply chain is long or if the air supply piping is small in diameter.

For the user of compressed air, the airflow rate is of less interest than confidence that the compressor will meet the pressure needs of the work in hand. Machines with a low operating pressure of about 6.5 bar (80 psig) are used in industries such as cement production or coal mining in pneumatic conveying and unloading, or in blowing dust away. This type of application demands huge amounts of air at a relatively low pressure. A higher-pressure of about 8.5 bar (108 psig) is used for materials handling of larger items, and it is a common practice to provide a single air circuit for both uses, even though the result is delays to the higher-pressure use while the air supply is diverted to service a high-volume, low-pressure application. Today, independent lines can readily be set up to supply different pressures for divfferent types of operations, which can result in considerable reductions in operating costs.

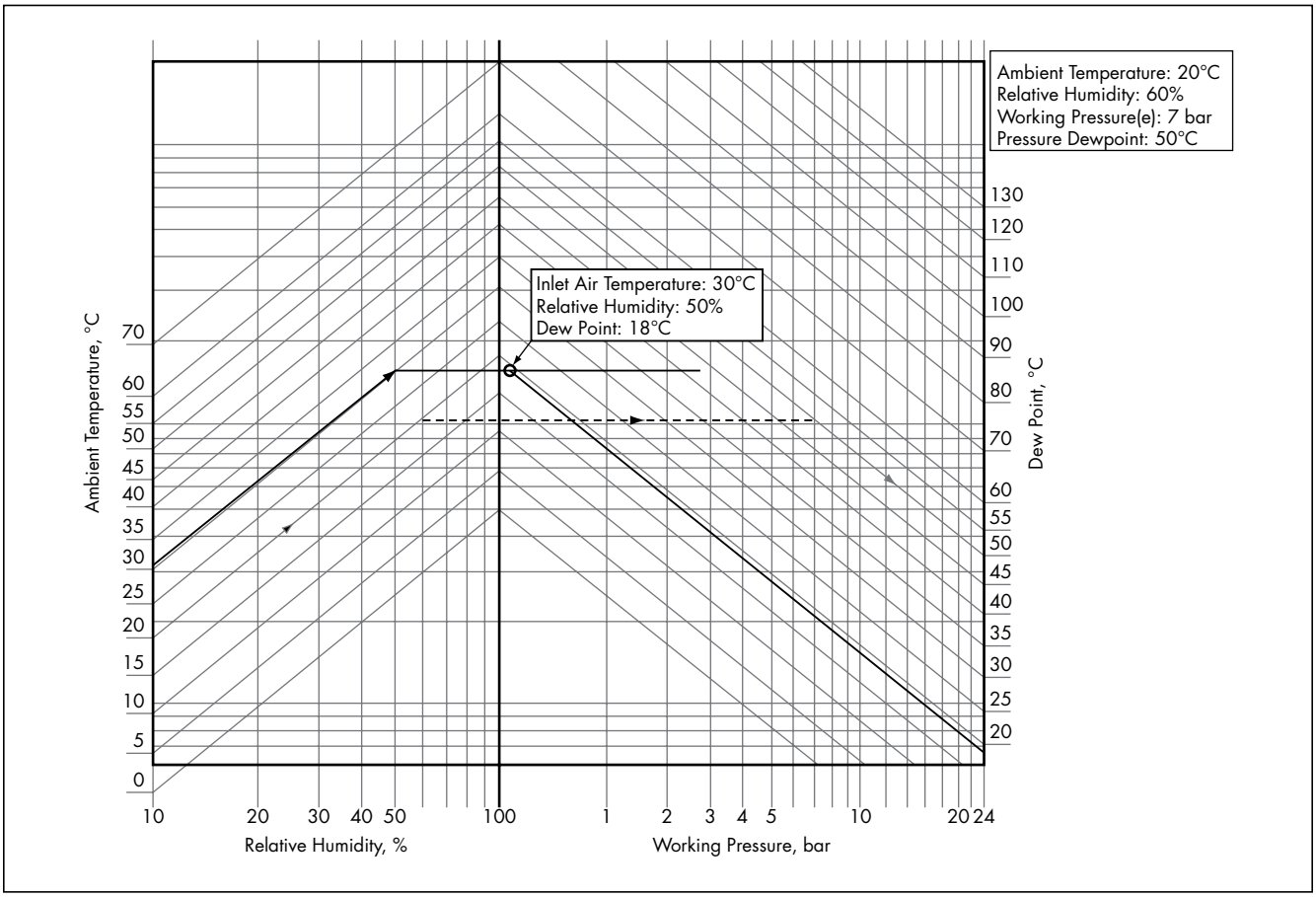


Figure 9.2-8 Dew point as a function of ambient temperature, relative humidity, and working pressure

Air distribution is the critical link between the compressor installation and the machine or tool that is using the air. Distribution requires an effective system of air lines and accessories, particularly in harsh environments such as mines. A well-designed and properly maintained air distribution system should have leakage of no more than 5% of the capacity of the installation. Unfortunately, 15% to 20% leakage is quite common.

The design of the air line and the choice of the fittings are important for minimizing losses. Pressure losses are created by restrictions such as small-size couplings or hoses that are too long or too small in diameter.

Every coupling in an installation creates a pressure loss, but the losses can be exacerbated through the use of incorrect fittings. Specialist fittings are required in particular situations, such as swivels that prevent the hose from getting in the way when working with hand tools and torque arms to reduce fatigue in operators. Ideally, the distance between the tool and the off-take should be no more than 3 to 5 m (10 to 16 ft), though this is often impractical to achieve in major installations such as mines.

The main line distributes the air from the compressor location to the point at which the air will be used. In a large compressed-air system serving several zones, the main line should be arranged so that each branch can be shut off without affecting the rest of the system. This ensures that when working on the piping system, only the branch involved has to be shut off. Leakage may be decreased if nonoperating units are shut off. This approach also makes it possible to ensure that the most important unit is supplied with air if the compressor capacity proves insufficient for the whole system to operate simultaneously.

A ring main distributes the air within the working zone. Excessively long off-takes should be avoided. Use of a ring main means that if an unexpectedly large air usage occurs in any service line, air can be fed from two directions. This reduces the pressure drop and provides level and more-stable air pressure in the entire system.

The off-take or branch line is the final part of the permanent installation and should be carried as close to the workplace as possible. The users in an air distribution system are the air tools together with their air preparation units or any other equipment consuming compressed air. Air-line accessories such as pressure regulators should be of high quality.

Diameters and lengths of a branch line depend on the amount free air to be supplied. The materials used for the pipe and couplings are also important for avoiding leakage and corrosion while minimizing pressure drop along the length to ensure that air at correct volume and pressure is delivered to the work face.

The performance of air power tools is largely dependent on the ability of the system to supply an adequate quantity of air of the correct quality and pressure. Too low an air pressure reduces the performance of most air tools. If the pressure

is lower than the design value, efficiency drops radically and productivity decreases. For a system supplying hand tools that operate at 7.8 bar (100 psig), the pressure should not be less than 7.1 bar (90 psig) at the tool end to ensure acceptable productivity.

Tests on a typical grinder showed that a pressure loss of 1 bar (15 psi) leads to almost 30% less material removed—so the worker has to work longer to do the job. Similarly, tests with a hand drill have shown that every hole that takes 2 seconds at 7.1 bar (90 psig) would take 3.2 seconds at 6.8 bar (85 psig), which would lead to substantial additional operating costs over the course of a year.

EMISSIONS AND NOISE DIRECTIVES

Two widely adopted requirements regarding machine emissions are the Tier 3 standards in the United States and the Euro 3 and 4 standards in Europe. Similarly, noise regulations continue to be enhanced to comply with national and international standards. Equipment from major manufacturers complies with the key international standards, though customers may need to confirm equipment suitability for use in specific countries. Some countries have no specific requirements on either emissions or noise.

Dust can create a considerable nuisance in drilling if not collected, and it is particularly important to control particles smaller than 5 mm (0.2 in.) as effectively as possible. The basic requirement is for the dust collector to operate with a partial vacuum, which means that a suction ejector should be the final piece of equipment in the sequence. Larger drills are fitted with a brake ejector, which dispenses with the need for a seal against the rock and facilitates collaring and allows drilling to be performed with no appreciable drop in production.

REMOTE MONITORING AND CONTROL

Many mines today run a series of independent portable compressors. Those mines that retain a stationary surface compressor room are increasingly adopting centralized remote control to optimize the usage of a bank of multiple compressors. Automatic control ensures that the correct combination of machine size is automatically adopted for maximum efficiency. To achieve this, one or more variable-flow compressors are required in the network. The control ensures that each runs in its optimum range for enhanced efficiency.

Compressor controls can consist of hard-wired digital contacts or a choice of three major field bus industrial interface protocols (Modbus, Profibus, and Ethernet/IP [Industrial Protocol]). This allows the machines to be integrated easily into the control panels so that the operator can start a machine remotely without needing to be in the compressor room.

Today's remote monitoring capabilities are ideal for use in mines, as diagnostics and checks can be performed by experts away from the site. Use is made of the global system for mobile communications (GSM) network to relay information from the compressor to the monitoring system—the system can be used by both portable and stationary compressors. Satellite solutions are also available and are ideal for portable compressors in locations that are outside mobile phone networks.

Different tiers of service are available, starting with a simple notification process when there is a problem. The text message carries details of the problem—for example, a warning status due to the oil temperature. Such notifications can be very useful even where there are technicians on-site, as they receive an instant warning.

Another option offered by manufacturers is local monitoring, where the equipment is connected to a special communications box that displays the status. This requires a physical connection to the box, which may not be ideal in some mining situations.

Portable compressors can be equipped with a special controller that supports both GSM and satellite networks. This box broadcasts the compressor's status and its most important parameters, including its GPS (Global Positioning System) coordinates. This means that it can be used for theft protection—it can be located on maps or satellite photographs. The system will send a text message if it is moved away from a set location. This feature is particularly used by rental companies in checking up on their fleet locations and usage.

Use of such a system ensures that the service operations are well planned—service engineers can check on efficiency, analyze any faults before visiting the site, and plan predictive maintenance. The machines can be monitored and analyzed, with any drop in efficiency highlighted and remedied. Use of this kind of system also helps keep down the costs of servicing, as schedules can be planned for maximum efficiency including making best use of the technician's time.

Internet monitoring is also available for industrial-scale machines. A central password-protected Web site provides access to data about the compressors that are connected to it. This approach supports both Ethernet connections and all types of modems. The data are refreshed every minute, alarms are sent out as required, and service engineers can analyze any problems from thousands of miles away if necessary.

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CHAPTER 9.3

Mine Communications, Monitoring, and Control

Rajive Ganguli, Scott Beer, and Mark Watson

INTRODUCTION

A mine communications system, which includes communication, monitoring, and control, is an integral part of modern mine operations and management. Basically, a communication system is a network of devices that collect, receive, or transmit information. With the passage of the Mine Improvement and New Emergency Response (MINER) Act of 2006 following recent mine disasters, especially those at the Sago mine in West Virginia and the Crandall Canyon mine in Utah (both in the United States), two-way wireless communication systems between surface and underground and electronic miner tracking systems are now required in U.S. mines. However, mine communications systems are not limited to tracking miners in case of emergency. The subject also includes sensing systems used for process control.

BASICS

As shown in Figure 9.3-1, a mine communication system (MCS) has three basic components: the information source or transmitter (such as sensors), the communication pathway or network, and the information receiver. The technological success of an MCS depends on the successful functioning of all three components. However, the receiver may be passive; therefore, the critical components are the source and the network.

The fundamental nature and technology of sensors vary drastically from one to another. A sensor's required accuracy, precision, and sensitivity depend on the need the measurement fills. Thus, a motor stator voltage measurement would be significantly more accurate and precise than, for example, a semiautogenous grinding (SAG) mill ball-loading measurement. However, the precision of measurements is increasingly important, because sensor information is increasingly being used for automatic control, which will be discussed further in a later section.

The part of an MCS currently undergoing rapid change is the communications network. Although Tesla and Marconi first demonstrated wireless radio communication a century ago, mine communication systems, like most industrial communications systems, have been mostly wired. Mines have traditionally used multiple communication networks, each serving a unique purpose. Therefore, there was a telephone network, a sensor network for belt conveyors, and so forth. However, technological breakthroughs in the nonmining world, such as digital radio communications, pressure for mine productivity improvements, and newly legislated mine safety requirements, are driving the industry toward single unified communications systems for entire mines. Communications systems now have options for wireless connectivity or a hybrid of wired and wireless connectivity, although most successful underground networks currently employ cables. Transmission is mostly by means of electricity (such as through phone lines or Ethernet cables), radio signals (such as wireless networks and leaky feeder systems), or light (fiber optics). The mode of transmission is a major determinant of a communication system's ability to meet the needs of a given purpose.

MAJOR TRANSMISSION TECHNOLOGIES

The most popular transmission medium is radio signals because they allow wireless implementation. The various radio transmission technologies are discussed later (Beerbower 2006; MSHA, n.d.; NIOSH, n.d. (a); Schiffbauer and Mowrey 2006).

Low Frequency Technologies

Low frequency (LF) radio systems use the very low frequency (VLF) and ultra low frequency (ULF) bands. The LF band includes frequencies from 30 to 300 kHz, VLF from 3 to 30 kHz, and ULF from 300 Hz to 3 kHz. LF signals are least attenuated when they travel through the earth. In that sense, they are best suited for underground mines. However, their application is limited underground because they have very narrow bandwidth; that is, only minor amounts of information can be transmitted as LF signals and over limited communication distances. A majority of the underground emergency communication technologies that are designed to survive

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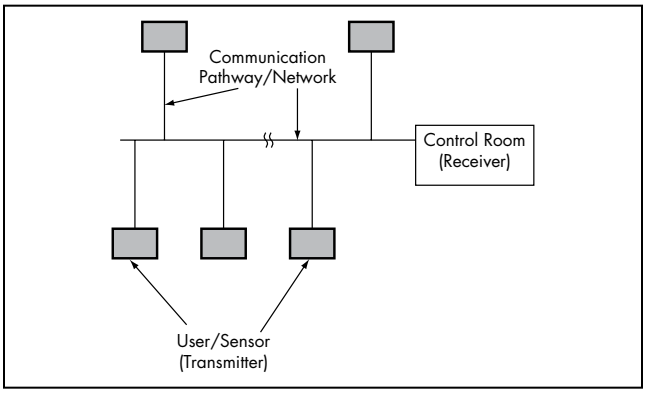


Figure 9.3-1 Simple representation of a communication system

postaccident are LF technologies. Such technologies are commonly implemented as transmission from surface to underground (one-way), though two-way systems exist. Surface antennas are typically large loop antennas laid out over the mine area. Some systems (one-way or two-way) use underground loop antennas.

LF radio systems have a variety of advantages and disadvantages.

Advantages

- Signals travel through the earth. Communication can be in the form of voice or data.
- Communication can be broadcast to all or sent to a single miner.
- When attached to an integral accessory (such as a cap lamp), receivers are always close to the miner.

Disadvantages

- The transmission depth can vary. Most natural and human-made electromagnetic noise is in the lower frequencies. Therefore, LF radio systems are susceptible to interference from noise. Transmission from surface antennas can additionally be affected by foliage and the presence of workings above the current working level. Local noise sources can create shadow zones in a mine that is otherwise covered.
- When surface antennas are required, surface land ownership and terrain can become major issues because of the large area taken up by the antenna. When an underground antenna is needed, the system may not survive a catastrophe. Additionally, underground antennas are not intrinsically safe, so they would have to be turned off during events such as blasting, fires, or fan shutdowns.
- With currently available technologies, if a message is transmitted when the receiver is off, the message is lost. In a one-way system, there is no way to confirm whether a message has been received.
- Message length can severely impact transmission time. Transmission of a 32-character message can take up to 3 minutes.
- Two-way systems may not have portable antennas.

Medium Frequency Technologies

Medium frequency (MF) technologies (300 to 3,000 kHz) combine the benefits of low and high frequencies. External environmental noise is not the major problem that it is with low frequencies. MF signals can also penetrate the earth to some extent. However, because they can couple (parasitic coupling) to existing electrical cables and pipes, their effective range can be very high. Existing mine conductors not only act as conduits but also as antenna (Dobroski and Stolarczyk 1982). Entries in bedded deposits can act as wave guides and extend the range (A.R.F. Products 1986).

MF systems have a number of advantages and disadvantages.

Advantages

- Signals travel through the earth to some extent. Communication can be in the form of voice or data. Parasitic coupling can extend the range of these signals significantly.
- These signals are not severely affected by electromagnetic noise.

Disadvantages

- The bandwidth of these signals is limited.
- Very few commercial products use this technology.
- Antennas can be large, making portability a problem.

High Frequency Technologies

High frequency (HF) systems use the very high frequency (VHF) and ultra high frequency (UHF) bands. The HF band includes frequencies from 3 to 30 MHz, VHF from 30 to 300 MHz, and UHF from 300 MHz to 3 GHz (gigahertz). HF signals are the least suitable for underground applications because they suffer the greatest attenuation when penetrating the earth. Akyildiz and Stuntebeck (2006) report higher attenuation for higher frequencies, with moisture content also deleteriously impacting signal propagation. These limitations are also observed in VHF and UHF devices, such as radios, which are commonly used in mines. For wireless transmissions, higher frequency technologies work best in line-of-sight applications, with VHF working best in wider mine openings and UHF performing well even in smaller openings such as 1.8-m coal seams (Nutter 2007).

HF systems have a variety of advantages and disadvantages.

Advantages

- Many commercial products use this technology. Significant advances have been made in UHF technologies in recent years, many of which are finding application in surface mines (and, to some extent, underground mines). UHF voice radios are a common (industrial) communication tool.
- UHF ranges are extended by mine entries that act as wave guides.
- HF systems are not severely affected by electromagnetic noise.
- Safety-related issues are well understood.
- The high bandwidth of these systems allow for voice and video communications.

Disadvantages

- Signals suffer severe attenuation when going through earth/physical barriers.

Table 9.3-1 Comparison of various communication technologies

Technology	Typical Application	Advantages	Disadvantages
VHF/UHF	Voice, data, tracking	High bandwidth, can allow many users, accommodate high traffic; multiple voice channels; small and wearable antennas; factors impacting safety are known	Best for line-of-sight applications (not good around bends)
MF	Voice, data	Can turn corners; metal infrastructure (pipes, track, wires, etc.) in the mine can help increase range (parasitic propagation)	Limited bandwidth; existing electrical and communications system may cause interference, though not as much as for ULF/VLF/LF
ULF, VLF, and LF	Personal emergency devices	Can go through the earth	Natural phenomena, existing electrical and communications system can cause severe interference; very limited bandwidth; require surface or underground loop antennas
UWB	Tracking, data	Ideal for tracking; no interference from existing electrical and communication systems and multipath effects; very high bandwidth; simple architecture (potential to be inexpensive)	Very short range

Source: Nutter 2007; Chehri et al. 2008; Friedlos 2008; Swedberg 2008; NIOSH, n.d. (a).

- In wireless implementations, multipath effects can make the signals extremely noisy. When an HF signal is transmitted, it reflects, refracts (*multipath*), and interferes with itself on its way to the receiver, thereby reducing the quality of the signal.
- These systems require a lot of infrastructure (such as repeaters, routers, and cables), which makes them susceptible to mine disasters.

Ultra Wideband Technologies

Ultra wideband (UWB) technologies are low-power (short range), high-bandwidth communications spread over a wide spectrum of frequencies. UWB technologies make use of all frequencies, low to high; that is, their short pulses have a bandwidth of 500 GHz or 20% of the center frequency, whichever is less. As a result, they differ from the technologies discussed previously, which were restricted to a single category and therefore had a narrow spectrum. However, because of their higher bandwidth, they have low power on any one frequency.

UWB systems have a variety of advantages and one disadvantage (Chehri et al. 2008; NIOSH, n.d. (b); Intel, n.d.).

Advantages

- UWB systems are not affected by noise from narrowband systems.
- They do not suffer from multipath effects like higher frequencies do because their short pulses are easily identified. The technology can determine the time of travel of the signal. Thus, it is ideal for location/tracking applications.
- They support high data rates, so they can support all communication needs.
- The simple architecture of UWB systems means they can be less expensive than other technologies.

The disadvantage is that UWB systems have low power, so they have an extremely short range (around 10 m).

A comparison of the strengths and weaknesses of the various technologies is provided in Table 9.3-1.

TRANSMISSION MEDIA

Signals to and from sensors and users can be transmitted through wired or wireless media (or a combination of the two). Both media have their own sets of challenges. Wired networks can become logistically and financially challenging. They may not accommodate sensor/user mobility and cables can get damaged. Cable protection through armoring, conduits, or burial is difficult on a massive scale. Creating redundancies in wired networks can also be difficult.

The major challenge for wireless transmission is that the technologies with the greatest bandwidth (VHF/UHF) do not travel through the ground very well and/or have very low range (UWB).

Wired Networks

Wired networks can transmit radio, electrical, or optical signals. The various transmission technologies employ differing types of wiring or cable: twisted, multiple-wire cables for electric signals, coaxial cables for radio transmission signals, and fiber-optic cables for optical transmission signals.

The most common wired system is probably the leaky feeder. Leaky feeders are cables designed to "leak" radio signals (i.e., the cable shield has holes). Signal frequencies can be in the VHF or UHF range. Considerable signal loss means leaky feeder cables require repeaters to continue propagating their signals. Leaky feeder cables can also serve as antennas or conductors for MF communication. Leaky feeder cables maintain a low voltage; therefore, certain low-voltage communication devices need only one cable to power them and to keep them connected to the network. This makes these devices intrinsically safe.

Leaky feeders are usually used in conjunction with two-way voice radios (MF/VHF/UHF), with VHF/UHF being more common because MF antennas can be large and cumbersome. Radio coverage depends on the frequency. Typically, UHF voice radios provide better coverage than VHF radios. To maximize the benefits of all types of radio communication technologies, researchers are attempting to improve the interface between MF radios and UHF/VHF leaky feeders. The intent is to examine whether the parasitic coupling abilities of MF transmission will improve coverage and help create redundancies during emergencies.

Advantages

- Leaky feeders extend the range of the frequencies that have the most bandwidth.
- They can carry voice and data.
- Leaky feeders can power low-voltage devices.
- They can serve as a conductor/antenna for MF communication.

Disadvantages

- The signals that leaky feeders carry do not penetrate the earth.
- They can be expensive; they require skilled labor for maintenance and installation.
- Like all cables, they can be impaired following a mine disaster. Also, they require battery backups for emergencies.
- Repeaters are required to extend signals.
- Redundancy can be complicated to achieve.

The other types of wired communications systems are telephones, pager systems, trolley wire systems, computer networks, or network backbones (such as fiber-optic cables).

Telephones and pagers have so far been the most important communication link in underground mines. They are easy to install and are battery powered. However, they have significant limitations that make them unfit for emergencies or as productivity tools. First, communication is limited to fixed locations. Second, although they are easy to install, they can be difficult to maintain because cables can get easily damaged. Pagers have the added limitation that the communication may not be private. Although telephone lines can be used to transmit data, the bandwidth is limited. Trolley wires are becoming obsolete as rail haulage systems become less common, but they can transmit communication signals on the direct current cable that powers the trolley system. Therefore, this method of communication is noisy. However, because they allow communication on the haulage way, they enhance safety.

Computer networks are a critical component of a modern mine. Networks are usually minewide, though corporation-wide networks are becoming common as well. Corporation-wide networks would probably require a multitude of transmission media. Modern data transfer protocols (such as TCP/IP [Transmission Control Protocol/Internet Protocol]) and networking technology (such as Ethernet) allow most mining equipment, especially in processing plants, to be part of the mine network, making monitoring and data transfer seamless; however, for security and optimal network management, there should be a barrier between office management networks and operation/safety networks (Pamel and Pederson 2007).

Standard Category 5 (Cat 5) twisted pair copper wire Ethernet cables are used to connect devices in a wired Ethernet network. However, they are noisy and subject to energy losses beyond short distances. Fiber-optic Ethernet cables are another (new) option for local networks, though they are commonly used when transmissions occur over long distances (such as for mine communication backbones).

Fiber-optic systems work by sending light signals through a clear core using the phenomenon of total internal reflection by a reflective coating (*cladding*) covering the core.

When the core is very thin, the light travels in a straight line through the core (*single-mode*) so that signal loss is minimized. When the core is thick, the light is reflected and refracted (*multi-mode*), resulting in higher losses. Though multi-mode cables have a smaller range, they can carry many more light frequencies than single-mode cables. In addition to the core and cladding, fiber-optic cables also contain multiple protective layers to add strength.

Advantages

- Fiber-optic systems have very low losses, so the repeater requirement is minimal.
- They have a significantly high bandwidth.
- They are not susceptible to electromagnetic/radio interference.
- Fiber-optic cables are intrinsically safe, though optical transmitters (devices that convert electrical signals to optical signals) may not be. Electrical pulses must be converted to light pulses prior to their transmission through fiber-optic cables.
- Because they do not conduct electricity, fiber-optic cables eliminate ground loop problems (Dubaniewicz and Chilton 1995) and are not impacted by high voltage cables, lightning, and other electrical disturbances.

Disadvantages

- Fiber-optic transmission can be expensive over short distances.
- The inability to carry electricity along with communication signals (as leaky feeders do) can be a major disadvantage because many devices in sensitive areas depend on the communication cable also to provide intrinsically safe power.

Wireless Networks

Wireless networks employ devices (*clients*) that communicate wirelessly using the 802.11X IEEE standard of UHF (2.4 GHz and 5 GHz band) communication. They traditionally required mobile clients to be near a fixed access point that connects directly or routes data back (back haul) to the wired network. Because a mine is constantly changing, fixed access points must be continually added or relocated to maintain network integrity. Access points attached to mobile trailers with battery and solar power are often used for this purpose in a surface mine. This reconfiguration of the network is a continuous process and can be a significant burden. One mine noted that they had more mobile trailers than haul trucks and operated a large crew of technicians to support the daily reconfiguring of the network. Blind spots can also occur regularly, resulting in missed assignments and reduced efficiencies.

Recent advances in wireless technology support greater mobility. These solutions (*wireless mesh networks*) extend message routing capabilities to each mobile node, allowing messages to be routed between mobile nodes back to the closest access point. Each node automatically locates and configures a connection with every other mobile node or access point within its range, resulting in a *mesh* of network connections (*client mesh network*). A small mesh network may be connected through routers to a broader network (see Figure 9.3-2). Message routes are dynamically analyzed and selected to provide optimal throughput. This mesh technology not only extends the flexibility and range of the mobile network but increases the effective throughput by allowing the message load to be distributed across the mesh (load balancing).

High bandwidth wireless mesh networks connecting the underground mining locations to the surface are possible, though they may require that routers be very close to each other. In a test in the Canada Centre for Mineral and Energy

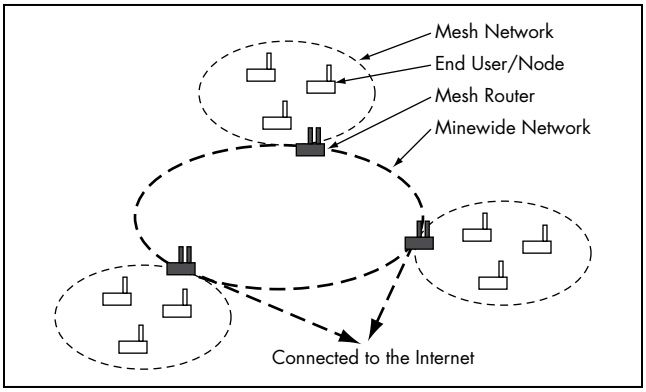


Figure 9.3-2 Wireless mesh network

Technology's (CANMET's) experimental mine, bandwidth dropped by 50% for each hop using TCP (Hakem et al. 2007). TCP is the protocol computers use to talk to each other. The term *hop* is used to describe transmission of signal from one router to the next. Maintaining bandwidth is not the only challenge. Because they are based on UHF radio signals, mesh network systems have the strengths and weaknesses listed previously for VHF/UHF radio systems. Interference from existing electrical and communication systems, the existence of shadow regions, and the magnification of noise when amplifiers are connected in series (Ortega 2007) can still impair such networks. Other issues with wireless networks include reflection/refraction of signals from ground–air interfaces, multipath fading from features such as plants and rocks, and lower propagation velocities in dielectric material.

Advantages

- Given the ease of integrating modern wireless devices with computer networks, wireless devices and networks in mines can usually be easily integrated into a minewide computer network. This enhances safety and productivity.
- At least in a surface mine, wireless mesh networks are easy to establish.
- To a certain extent, bandwidth is not an issue.
- Wireless devices and networks are more prevalent in the nonmining world than in mining, so they are constantly being improved.

Disadvantages

- Wireless signals are UHF radio signals. Therefore, they have all the disadvantages of UHF signals, such as the inability to penetrate the earth and their degradation as a result of multipath effects.
- Wireless devices/networks are not truly wireless because they still must be connected to a power source. Additionally, they need battery backup for emergencies. Powering some of these devices requires explosion-proof (XP) enclosures, which in turn makes them cumbersome. Complicated battery backup systems (such as those used in the northern latitudes where solar systems do not work) can also come with a significant maintenance burden.
- Wireless mesh networks can require a significant amount of infrastructure, especially in an underground mine.
- Very high bandwidth requirements (such as for videos) can be difficult to maintain across multiple hops.

CHARACTERISTICS OF A MINE COMMUNICATIONS NETWORK

An effective mine monitoring, communications, and control system requires a communications network that can support all its demands.

Desired Attributes

A mine communications network should have the following major attributes:

- Ability to accommodate different types of information sources and existing infrastructure
- Ability to accommodate information source mobility
- Safe, robust, and redundant (survivable) operation
- Ability to expand easily
- Ease of maintenance

Ability to Accommodate Information Sources and Infrastructure

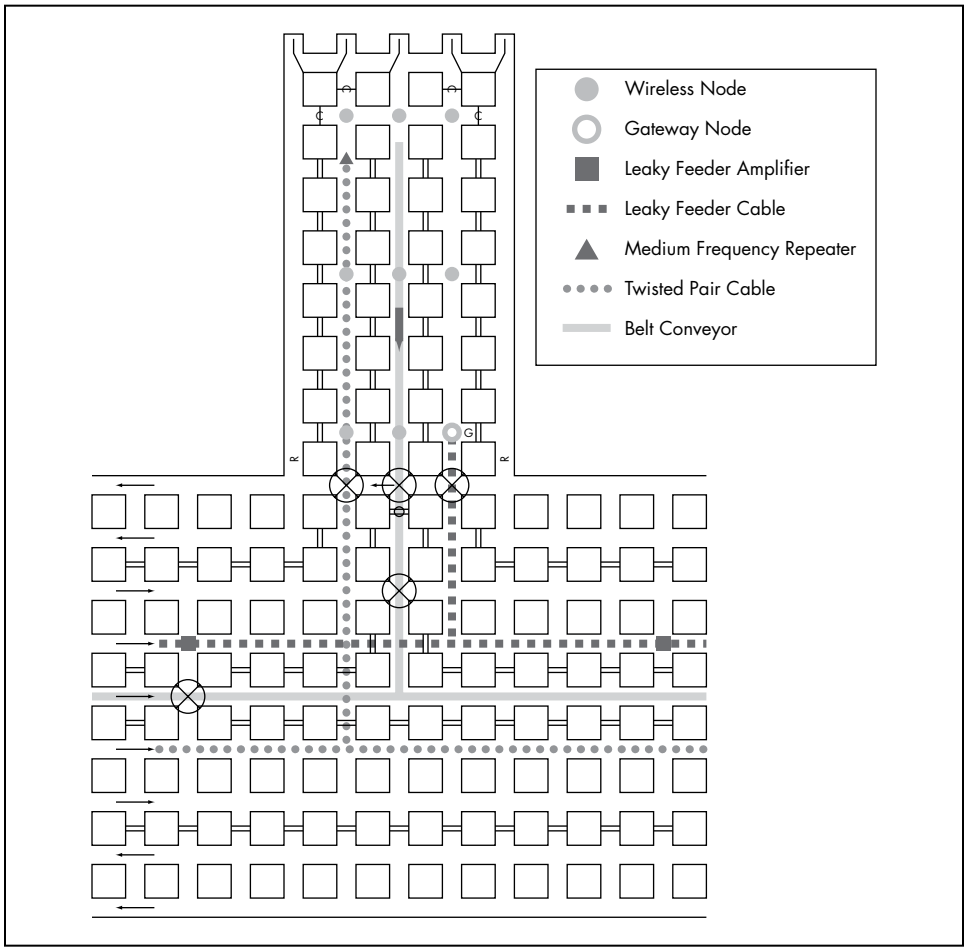
A modern mine has a multitude of information sources. Not only can individual sources differ from each other (e.g., a drill is very different from a methane monitor or a radio), but even within a source, there could be different information sources. For example, a mine truck may be transmitting machine health information (such as engine temperature) and haulage–jolt information (derived from accelerometers and pressure transducers).

Although a mine communications network would ideally have the ability to accommodate different information sources, this ability is often absent because vendors have developed systems optimized for different information sources (atmospheric monitoring, voice communications, tracking, etc.) and different mine types (aboveground vs. underground, coal vs. noncoal). For example, in underground mining, leaky feeders are generally acknowledged to account for more than 90% of all handheld voice communications systems.

A new communications system also may not be compatible with existing infrastructure or systems. The Cayuga mine in New York (United States) had to replace existing fiber-optic backbone and switching hardware before a new system would work (Pamel and Pederson 2007). Mine management at the Bingham Canyon mine in Utah required not only that a new wireless network be compliant with the industry standard but also that the communications vendor ensure compatibility with existing equipment and applications (Beer 2008). Network and data standardization also allows proper data warehousing, which is critical for mine-to-mill type optimization. Indeed, Chen et al. (2007) report on an MCS at a coal mine in China that standardized data generation and transmission with a goal of creating a standardized data warehouse. The industry is certainly headed in the direction of data standardization. For example, International Rock Excavation Data Exchange Standard (IREDES 2008) is creating a standard format for mining machinery.

Ability to Accommodate Information Source Mobility

Humans and other information sources are not stationary but move from one location to another in the course of their work activities. Thus, the MCS should not only be able to accommodate their mobility but should not put any constraints on their movement. In surface mines, wireless mesh networks are making it easy to handle mobility, though the ability to maintain high bandwidth at all times can be a problem. As



Source: Chirdon 2008.

Figure 9.3-3 Two-way hybrid voice communication network in a longwall development section

discussed previously, in an underground mine, mobility is even more of a challenge. Mobility-related problems can be overcome by mixing technologies and making compromises. For example, as shown in Figure 9.3-3, an underground mine could use wireless mesh networks in conjunction with leaky feeder or fiber-optic cables to allow some mobility at the end of wired communication. To ensure postdisaster mobility, it could also have a layer of LF technologies for basic voice/text communications. Communications can also be prioritized to ensure that important communication streams always get transmitted, no matter the bottleneck.

Safe, Robust, and Redundant Operation

Both the sources of information, such as sensors, and the communications network should be able to function safely under typical and atypical mining conditions, including moisture, dust, vibrations, and postdisaster conditions. Sensors and communications equipment that are deployed in areas susceptible to explosion must be intrinsically safe or enclosed in XP boxes. There are other legal requirements as well. The MINER Act (in the United States) requires that underground mines provide means for two-way wireless communication between underground miners and the surface at all times, including after massive disasters. It also requires that miners be tracked electronically at all times and that the communication system be redundant. Redundancy is not legally mandated for productivity purposes (though desirable); it is mandated only for safety purposes. Thus, redundancy requirements have to be met only at the level (bandwidth) necessary for safety-related communication.

One way of achieving redundancy is by means of a self-healing, ring-type communications architecture. *Self-healing* implies that the network will automatically resume transmitting communications after a fault condition occurs. Under normal conditions, communication is one way. When there is a fault in the system, the architecture allows communication to loop back into itself at the location of the fault (Figure 9.3-4), thus enabling two-way communication.

The self-healing, redundant-ring architecture can be achieved in different ways. In a fiber-optic network, the redundant-ring architecture may use two fiber-optic cables for the ring; that is, there are two rings, with each node/user connected to both rings, which carry communication in opposite directions. In the event of a fault, the system connects the two rings on either side of the fault, maintaining communications. In an underground mine using a leaky feeder (or other)

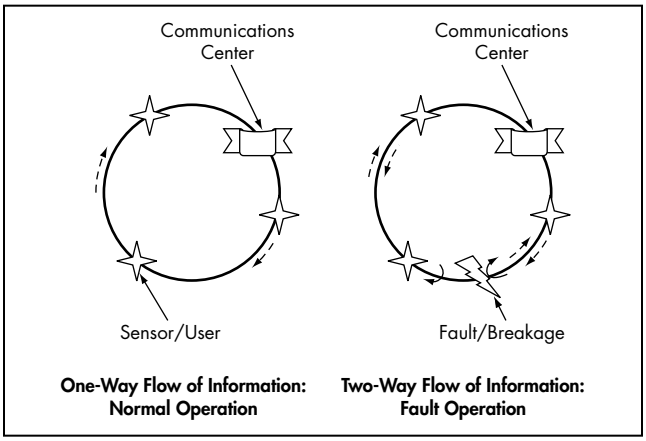


Figure 9.3-4 Redundant communications system

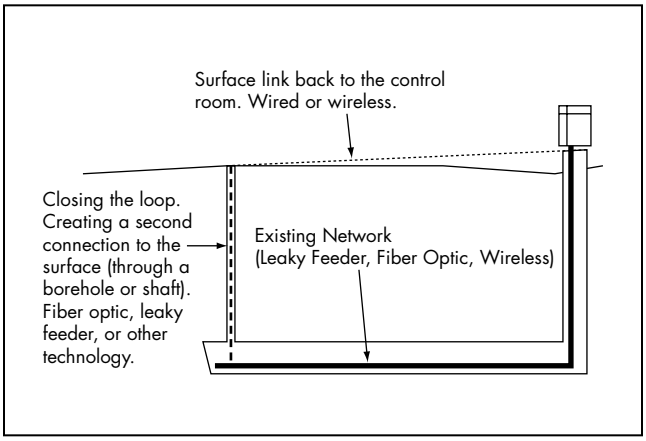


Figure 9.3-5 Creating redundancy in an underground communications system

backbone, redundancy can be achieved by closing the loop, as shown in Figure 9.3-5. The connection to the surface can be through an existing shaft or a borehole. If none exists, a small-diameter borehole could be drilled to extend the leaky feeder back to the surface. The final link to the communications center can be wired or wireless.

An alternative would be to use another communications network, such as an MF radio. The National Institute for Occupational Safety and Health (NIOSH) has recently funded research into an MF radio that may prove useful as a backup or emergency network communications system. Similarly, nonfiber Ethernet TCP/IP communications can be made redundant with a ring topology. Many Ethernet switches are available that provide ring management functionality. This would allow a break in the communication infrastructure to occur; switches would automatically adjust to allow the flow of data traffic around the location of the break. Redundancy can also be achieved by means of wireless mesh networks (especially in surface mines) or hybrid (wired and wireless) networks. An all-purpose (one that meets production and safety requirements at all times), redundant, purely wireless system can be difficult to achieve in an underground mine because of transmission problems through the earth.

Wireless or not, redundancy requires that the components of an MCS remain powered even after a disruption in power supply. Kohler (2008) reports battery backup durations of 8 to 96 hours for redundant systems.

Ability to Expand Easily

Mines are perpetually evolving as the mining face moves constantly. Mines may also increase or decrease their production rate to reflect product demand or change their mining method in response to technological changes. Although it is unrealistic to expect that the communications system will not need significant changes if the mine changes drastically, one can expect a communications system to accommodate minor changes and expansions easily. It is good practice when installing a system to include excess capacity to handle future expansions.

Technology upgrades or changes can be frustrating because new versions of technology do not always work properly with older versions. Proprietary systems can also pose challenges in that they may not integrate properly with other systems. Thus, any additions or changes to communications systems must conform to the existing/broader system. The mechanics of the changes are also important; for example, will the changes require a significant amount of infrastructure to be moved or reinstalled?

Ease of Maintenance

While considering any new technology, the maintenance requirements for that technology should be fully understood. In evaluating maintenance requirements, one must consider the type of mine (aboveground or underground, coal or noncoal, large or small) into which the system will be installed and the level of technical expertise on its staff. One should also contact mines with similar systems and ask them about maintenance issues associated with that type of network.

New Communication System: Before Installation

Prior to installing a new communications network/system, the following checks should be conducted (Schiffbauer and Mowrey 2006):

1. Is the mine's electrical noise environment (surface and underground) understood? That is, have noise sources been identified, quantified, and studied?
2. Have signal propagation tests been conducted?
3. Is the candidate system safe, robust, redundant, easily expandable, easy to maintain, and able to accommodate different, often mobile, information sources? Does the new communication system have any safety issues? Will it set off explosions or affect other monitoring systems?
4. Have other mines that have installed the candidate system been contacted? Any issues? Note that what works in one mine may not work in another. For example, many coal mines in Australia are two-entry mines. Thus, line-of-sight problems can be easily addressed as miners are restricted to only two entries (Beerbower 2006). However, the same line-of-sight-limited solution will not work in a coal mine that employs multiple entries. Therefore, before investing in a new communications system, one must consider all factors as they apply to a specific mine.

However, technical issues are not the only challenges a new system will face. Mine personnel, comfortable and familiar with existing systems, will often resist the introduction of new technology for a variety of reasons. Thus, the new system may not be fully used until the on-site personnel are comfortable with it.

INSTALLATION OF A MINE MONITORING SYSTEM

A wide variety of mine monitoring equipment is available. Although some suppliers may provide solutions for a range of disciplines, no single supplier can deliver an integrated system that spans all disciplines and can address the vast array of mining equipment.

Mine Monitoring Solutions

The design of monitoring systems can vary widely depending on the mining application. For example, monitoring systems for underground mining differ significantly from those for open-pit mining. Generally speaking, mine monitoring systems can be divided into three main categories:

1. Production monitoring
2. Equipment status monitoring
3. Safety and environmental monitoring

Production Monitoring

A number of monitoring systems have been developed to monitor mine production. These systems often concern processes that use expensive equipment. As a result, these systems are almost always aimed at improving asset utilization. For example, instrumentation systems have been specifically developed for drilling, excavating, loading, and hauling. These systems provide monitoring and record parameters such as ore produced, waste removed, commodities consumed, and changes in equipment status. Production output is recorded and reported. Commodity consumption is reported and purchase orders for replenishment made. Equipment status characteristics are analyzed to identify reasons for delays or stops in production so that changes can be implemented to improve efficiency. Some systems provide added control functions such as dispatching that assist in optimizing the routing of equipment through the mine. Mining operations deploying production monitoring solutions may expect increased equipment utilization; however, that may not necessarily occur.

Equipment Status Monitoring

A focus on production efficiency often results in equipment operated in ways that are unsafe and/or result in high maintenance costs. Monitoring systems have recently been developed that focus on improving equipment reliability and availability, as well as on production output. These applications monitor various critical equipment status indicators and alert maintenance personnel to the need for repairs before equipment failure occurs. This facilitates a move toward routine preventive maintenance. These systems provide information that can be used to resolve equipment warranty claims and lead to design changes that improve reliability. The financial benefits of such systems can be significant. A specific case study conducted across a fleet of 65 haul trucks reported maintenance savings of more than $2 million per month and a 57% reduction in the average number of trucks awaiting repair.

Safety and Environmental Monitoring

Personnel safety and protection of the environment are vital to maintaining mining operations. Multiple systems have been developed to track the location of personnel and provide voice communication. Solutions have been developed to track the proximity of equipment and personnel. These systems alert operators when equipment distances violate predetermined safety ranges and can cause equipment to be temporarily disabled or shut down to avoid accidents.

Sensing systems are also available to monitor weather conditions, air quality, ventilation systems, seismic activity, and slope or wall movement. Water handling is often a critical function in mining operations, and functioning effectively requires sensors that continually monitor pump stations and water flow and are interlocked with pumping equipment switchgear. Monitoring systems are used to help maintain process water recycling systems and ensure that water released back into the environment meets regulatory requirements.

Reporting and Interfaces

Effective monitoring systems are always underpinned by effective reporting systems. All monitoring system providers can generally supply a reporting system for their specific application. However, because no single supplier provides the entire range of monitoring systems, multiple reporting systems are the usual result. Consequently, there is a need for communication between monitoring systems.

Reporting solutions often report against their production databases. This can be dangerous if those performing data analysis create such an excessive load on the production system that it disrupts the performance of the monitoring solution. Additionally, in order to keep the real-time systems performing optimally, it will be necessary to have an archiving strategy that keeps the production databases trimmed.

Monitoring systems often use information from other production systems as inputs. In turn, monitoring results need to be reported and communicated to upstream systems. Because of the proprietary nature of monitoring systems, one cannot assume that support systems such as databases will be accessible to outside systems or that interfaces are necessarily readily available. Interfaces are often sold separately and can be costly to have developed. Care must be taken in selecting monitoring solutions to ensure that databases can be accessed directly or through interfaces that will allow the various mining, management, and accounting systems to be integrated. It is critical that the supplier provide clear and accurate documentation. In selecting monitoring systems, preference should be given to vendors who use standard database and interface components.

To address these interface and reporting issues, companies may wish to implement a data warehousing or data historian system that collects and aggregates data from the various mining systems into a single data store that is used for maintaining history, reporting, and analytics, and for providing interfaces to other systems (see Figure 9.3-6). Although there are multiple providers of data warehousing/historian systems, very few have been developed specifically for mining. An understanding of mining operations and processes is necessary to develop an effective reporting system.

Implementation

Multiple approaches to implementing mine monitoring systems can be taken. Prior to selecting an approach, companies should develop a mine monitoring strategy that provides an overall vision and specifies the guiding principles that will be used to select and implement monitoring solutions. Clearly defined business requirements that articulate the scope, goals, objectives, priorities, and timelines will need to be developed for each specialty area. It is vital for this process to involve representatives from each of the various mining departments

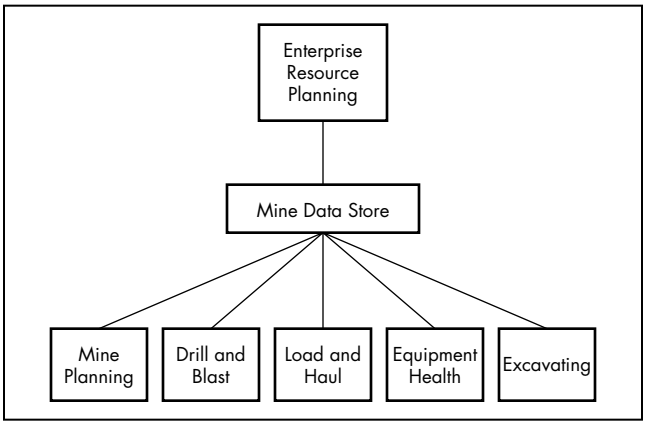


Figure 9.3-6 Potential data warehouse

or disciplines and have senior leadership endorsement and sponsorship.

Strategic Vendor Approach

One commonly used approach to implementing mine monitoring systems is to select a strategic vendor around which to build the mine's monitoring solution landscape. This vendor is selected based on its ability to supply an integrated solution that covers the greatest number of business and technical requirements. This vendor is also used as the system integrator to provide interfaces with other mining solutions that fall outside its specialty area. This approach focuses on delivering a broad, highly integrated solution. It is important that there be wide acceptance within the overall organization for this approach because the needs of specific departments may not be completely met in order to maintain overall simplicity and greater integration between systems. Specific departments may be disappointed with the performance of their specific disciplines or applications because of their awareness that there may be systems available that better suit their purposes.

This approach places a high degree of responsibility on the vendor to deliver the overall solution. It simplifies initial implementation by reducing the number of personnel required for implementation and can also simplify education and training needs for mine personnel. It can reduce the mine's direct labor requirements by allowing support functions to be outsourced to the vendor. This approach depends strongly on the strategic suppliers and requires that a strong business relationship be developed between the mine and the vendor to ensure that the mine's short- and long-term needs are considered and given priority in support and future development.

Best-of-Breed Approach

This approach involves selecting a combination of monitoring solutions, each chosen based on its ability to address the needs of a specific function or discipline. This approach focuses on delivering solutions that can provide both breadth and depth; less emphasis is given to the integration of the entire solution.

MINER TRACKING SYSTEMS

The MINER Act has made manual tracking of underground miners obsolete. Although real-time miner tracking has obvious safety benefits, the benefits extend to production as well. The ability to locate and speak with any miner underground quickly, to call for a part immediately when needed, and to control and track travel efficiently are all important functions of the production cycle in a modern underground mine.

System Components

A typical electronic system for tracking miners is composed of five major components: an identification tag, readers, underground infrastructure, a data storage unit, and a data display. The functions of these components are as follows:

1. The identification tag is a piece of equipment worn by the miner. In the majority of the systems, this is an active (battery-powered) radio-frequency identification (RFID) tag that sends a unique identification signal at a predetermined time interval. There has been considerable debate on the proper location of the tag on the miner's person. This tag can be mounted in a variety of ways—on a lanyard, on the belt, in a radio, on the miner's hat, or in the cap-lamp assembly. Testing has shown that an elevated location works best, away from the clutter of a miner's belt, and high enough to let the signal from the tag radiate outward toward the reader.
2. Readers are the units that receive signals from the tags and transmit them over a network to a storage database. Readers may communicate either by a wired network or wirelessly via radio transmission. All readers must have a wired power source.
3. Underground infrastructure is the equipment required to provide and maintain electrical power to the readers and facilitate data transfer to the data storage unit located aboveground. Underground infrastructure is composed of a wired component to provide power to the reader, a battery if needed, and a cable or radio transmission between readers. Some systems provide power and data over the same cable infrastructure.
4. The data storage unit receives data from the readers, stores that data in a database, and may perform tag location calculations.
5. The data display presents information on tag locations, provides reports on which miners are underground, and may contain other information useful to mine rescue operations, such as emergency contacts. Typically, the display will contain a graphical representation of the mine and the location of the readers. Some systems support multiple data displays for larger mine operations.

System Selection

Mine type, size, seam height, regulatory requirements, numbers of personnel and equipment underground, and support are important factors to consider when selecting a tracking system.

Mine type—metal, nonmetal, coal, and so forth—is an important factor in tracking system selection. Coal mines in particular have intrinsic safety, layout, and regulatory requirements that metal mines do not. A U.S. coal mine must have a system that is either intrinsically safe or XP, can be configured for redundancy in separated entries, and is approved by the Mine Safety and Health Administration (MSHA).

Mine size is also a crucial factor in selecting a system. Large mines must take into account the number of readers that will be installed in a mine and consider the maintenance issues (electrical connections, battery testing and replacement, antenna alignment, etc.) associated with supporting those readers.

Seam height is an important factor if a wireless system is being considered. Because of line-of-sight limitations, reader placement with redundancy in low-seam-height mines could be as frequent as every 20 to 25 m.

Regulatory requirements such as the MINER Act and recent West Virginia (United States) legislation require that tracking systems in coal mines have redundancy, or *survivability*, and may also specify a maximum distance between readers (also referred to as tracking *granularity*).

The numbers of personnel and equipment underground are also important is making a system selection. Systems using low-speed serial communications may have difficulty transmitting tracking data reliably during shift changes in mines with more than two working sections.

System support is crucial for selection. Local distributors with knowledgeable technical staff should be a requirement. A proven track record for the support staff as well as the company is also important.

System Types

The types of tracking systems currently available can be segmented by the type of underground infrastructure used: wired mesh, wireless mesh, leaky feeder, fiber, or serial. The advantages and disadvantages of each type are noted.

Wired Mesh Systems

In a wired mesh system, reader nodes are organized into a mesh topology, providing redundant data and power paths via another single cable. Communications and power are self-healing in the event of a cable problem.

Advantages

- Wired mesh systems are able to achieve redundancy/survivability via interconnections between readers in different entries.
- They can provide power and data over a single cable with no underground power connection or underground batteries in some mine configurations.
- Line of sight between readers is not required, so high density of readers is not required.
- Wired mesh systems scale to any mine size and offer a lower per-node cost than wireless mesh systems.
- These systems are proven in coal mining applications.

One disadvantage is that these systems do not provide a voice communications option at this time.

Wireless Mesh Systems

In a wireless mesh system, radio nodes (generally with integrated reader electronics) are organized into a mesh topology to provide redundant data paths. Although termed *wireless*, power must still be supplied to each node via some type of cable. Each node generally contains a battery to provide backup power in the event of a loss of power.

Advantages

- A wireless mesh system may provide communications and tracking over a single infrastructure.
- This system type may provide 802.11 wireless data transfer.

Disadvantages

- A wireless mesh system requires line of sight between nodes; redundancy requires line of sight between at least three nodes.
- This type of system requires underground power connections and, for survivability, underground batteries.
- This system type involves a much higher level of complexity than competing systems.
- These systems require a higher level of technical competence to maintain.
- These systems have a higher per-node cost than competing systems.
- These systems are unproven in mining applications; there are scalability concerns due to per-node cost and hop-to-hop transmission delays.

RFID Leaky Feeder Infrastructure Systems

These systems use leaky feeder cable as a method to convey transmissions between system components, generally using one of two different methods: a reader can either be directly connected to the leaky feeder cable for power and communications or transmissions may be relayed over the leaky feeder using analog data radio modems with power delivered independently with a separate cable. In both cases, the data from the readers are routed through a device in the leaky feeder "head-end" to the tracking system data server (normally using Ethernet or serial communications).

An advantage is that an RFID leaky feeder infrastructure system may provide communications and tracking over a single infrastructure.

Disadvantages

- An RFID leaky feeder infrastructure system requires underground power connections and, for survivability, underground batteries.
- A system of this type has a higher per-node cost than competing systems.
- These systems are still unproven in mining applications and have associated scalability concerns due to their per-node cost.

Fiber Infrastructure Systems

These systems use standard Ethernet communications to connect fiber-optic cables between tracking system components. Depending on the cable used, distances of 50 km (or more) can be achieved at extremely high data rates (1 gigabyte/s). Although copper Ethernet connections are possible at the terminus of the fiber, no intrinsically safe (as defined by MSHA) components have been developed. Additionally, power must be supplied from intrinsically safe (MSHA-approved) power supplies with battery backup capability. Copper Ethernet can be used if the devices being interconnected are housed in MSHA-approved XP enclosures.

Advantages

- Fiber infrastructure systems provide the potential for communications and tracking over a single infrastructure.
- These systems are high-bandwidth media.
- These systems are able to connect to current underground infrastructure.

- These systems work well in mines without intrinsic safety considerations.

Disadvantages

- Fiber infrastructure systems require underground power connections and, for survivability, underground batteries.
- These systems make it difficult to attain intrinsically safe status for postaccident use in coal mines.
- These systems require XP boxes for postaccident use in coal mines.

Serial Infrastructure Systems

These systems use standard serial communications methods such as RS-232 or RS-485. Generally, standard copper wiring provides a communication path between system components; another pair of wires provides power. Using RS-485 communication can allow multiple devices to be placed on the communications cable and addressed independently. Depending on cable lengths, communication rates up to 115,000 bps (baud/second) can be achieved. For longer cable lengths, rates of 4,800 bps are more common. In order to maintain intrinsic safety, RS-485 requires the use of barriers to isolate areas. Power must be supplied from intrinsically safe (MSHA-approved) power supplies with battery backup capability.

An advantage is that serial infrastructure systems are able to use current underground infrastructure.

Disadvantages

- These systems have low bandwidth.
- Their common infrastructure cannot be used in postaccident implementations.
- XP boxes are required for postaccident use in coal mines.
- No voice communications option is provided at this time.
- The scalability of these systems may be of concern because of bandwidth limitations.

System Layout

Most mines will install tracking readers in the travelway to cover equipment and personnel movement. Coal mines in the United States, following clarification of the MINER Act, will likely also be required to install tracking readers in the primary escapeway, usually the intake entry.

Typically, mines will install readers at intervals of 300 to 1,500 m, with the distance between readers determined by the accuracy of tracking desired, regulations, and cost considerations. The sections delimited by the readers are typically known as tracking zones.

Generally, mines will also install readers or antennas from readers that support multiple antennas at the intersection of travelways to determine the direction of travel. This is typically referred to as gating.

For determining the suitability of a system for a given mine, it is helpful to ask the system vendor to provide a layout showing the placement of readers in the mine and aboveground areas with coverage areas, electrical connections, battery connections, and antenna placement noted.

Redundancy/Survivability Concerns

To achieve redundancy, readers communicating via a wired or wireless infrastructure must be connected to at least two other readers or have a direct connection back to the data storage unit.

Readers communicating via radio transmission are limited to line-of-sight communication and so must be within the line of sight of at least two other readers. In coal mines where ventilation concerns require separate entries for the intake, return, and (possibly) belt, each reader communicating via radio transmission must be connected to at least two other readers in the same entry to achieve redundancy.

For wired mesh systems, readers must be connected to at least two other readers in any entry, as line-of-sight concerns are not present.

Integration Options

Ideally, a tracking system would use a common, redundant, intrinsically safe communications network with the ability to incorporate atmospheric monitoring, belt monitoring and control, maintenance, and voice communications systems, as well as general data transfer. Unfortunately, no current communications network supports all of these capabilities in a cost-effective, redundant, intrinsically safe configuration.

However, ideally, one should select a system that provides the level of redundancy needed, with options for supporting data for some of the other underground systems needed. Some tracking systems support atmospheric monitoring sensors; others support general data transfer.

Summary

Implementation of an electronic tracking system not only improves a mine's safety infrastructure but can also have a tremendously positive impact on production. Many systems are being introduced to the marketplace; the selection for each mine should be carefully researched to find a system that will work best under its particular conditions.

GLOBAL POSITIONING SYSTEM APPLICATIONS

Global Positioning System (GPS) devices rely on spatial and temporal information from a group of satellites for proper functioning. Many major mining activities, such as digging (shovel or dozer), rock transportation, subsidence, and blasthole layout, are typically defined by their movements in space and time. Therefore, there are many benefits to quantifying them accurately and automatically (Flinn and Fileccia 2006): accurate selection of mineralized zones and coal seams by an excavator; reduced need for survey work for laying out drilling patterns or marking dig areas; precise delineation of static hazards; better dozer control, leading to more even benches and ramps; and the automatic creation of as-built digital terrain maps (DTMs).

GPS-based arrangements usually imply a GPS device mounted on a mining machine that provides input to a mine operations system (such as machine guidance or mine management systems). The role of the GPS device is simply to obtain and broadcast its spatial and temporal information in real time. However, this information becomes more valuable when it is used to obtain computed information (such as volumes or updated mine plans) or to guide machine operation. Therefore, it is important for the GPS device to be able to interact with the broader mine operations systems.

Challenges with GPSs

Several common problems are associated with GPSs:

- **Vendor/equipment-specific system.** GPSs are currently vendor or machine specific and because mines use a wide variety of equipment provided by a wide variety of vendors, it is very difficult to integrate the disparate systems. Open-source GPS solutions designed to be vendor and machine independent are, however, under development (Flinn and Fileccia 2006). Such systems will be essential for an integrated mine operations system.
- **Satellite availability.** GPS devices need access to multiple satellites (typically five) for accuracy. This can be a challenge in remote geographical locations or deep mines, under certain atmospheric events, or with certain satellite positions. Also, large equipment can hide satellites from their own GPS devices. Zywiel et al. (2005), however, list the following potential solutions to the satellite availability/shading problem:
  - Use of satellites from other systems, such as the Russian GLONASS (Global Navigation Satellite System). The advantage of this solution is that it can probably be implemented with no additional infrastructure.
  - Use of pseudolites or ground-based transmitters that transmit positional information. An issue with these is that the transmitters would need to be moved as the mine profile changes.
  - Use of advanced tracking sensors. These sensors combine a GPS with a laser and increase availability to up to 90%. These may, however, suffer from line-of-sight limitations.
  - Use of GPSs combined with inertial (or movement) data from odometers. Inertial systems use GPS devices for original spatial information. Subsequent locations are computed or determined from sensors that provide information on direction, speed, and so forth. A disadvantage of this system is that, without guidance data, its accuracy declines with time.
- **Latency.** Latency is the time lag between an object being in a given position and the time when it shows up as such on the operator's screen. Although this may not be an issue for drills, it can be an issue for shovels that use GPS information during digging or for GPS-based proximity warning systems. Latency effects can be minimized by opting for high-speed GPS, data networks, and computers, and/or using inertial systems.
- **Bandwidth.** Lack of bandwidth can also be a problem as high bandwidths are needed to transmit complex information (such as DTMs) between the mine management system and the GPS-based guidance system. However, with wireless-mesh networks, bandwidth is less of a problem.

Improving GPS Accuracies

Seymour (2007) provides suggestions for improving GPS accuracies on excavators. Speed and accuracy can be improved by using two antennas and two receivers (instead of one), and placing them in the back corners of the machinery house. Bucket location can be more accurately determined using tilt sensors and rope drum rotation encoders (in rope shovels). It is essential that these components be robust. Proper maintenance, such as prevention of slack bearings, can also improve accuracies.

PROXIMITY WARNING SYSTEMS

Proximity warning systems (PWSs) alert miners of the presence of static or moving hazards in their immediate vicinities. There are typically two types of PWSs: a system that detects proximity based on spatial locations as determined by GPS devices and a sensor-based system.

GPS-Based Proximity Warning Systems

These are used in surface mines primarily to alert large truck operators to the presence of smaller vehicles in their blind zones. Although a properly placed mirror or video camera can do the job, it may not be satisfactory because of difficulties in judging distances, operator inattention (to the mirror or video screen), or poor-visibility conditions.

GPS-based PWS implementations need very accurate GPSs and a good wireless network. The wireless network is essential because the success of the PWS depends on the ability of each vehicle to transmit its location continually to other nearby vehicles. The system uses this vehicle location information to determine whether another vehicle or object (*hazard*) is within its proximity zone or bubble. The presence of a hazard in the proximity zone sets off an alarm and/or disables movement in a static truck (Miller 2005). Errors in satellite or wireless network transmission or reception caused by antenna placement location, latency, multipath effects, or other effects can seriously reduce the performance of the PWS. Static hazards, such as buildings, are easily detected because their locations can be programmed into the PWS. The systems typically also allow defining the proximity zone or bubble.

One advantage of the GPS-based PWS is that there are very few false alarms. Also, the system can use the existing wireless network. A disadvantage of this system is that unless all moving objects (including humans) are equipped with GPS devices, the system is not completely effective.

Sensor-Based Proximity Warning Systems

These systems consist of source or emitter units and (portable) detection units. The source unit is usually on a large machine, while detection units are on humans or smaller mobile equipment. Source units consist of a loop antenna surrounding the machine or signal sources that are strategically placed on the machine. Examples of sensor technologies that have been used for PWSs include pulsed radar (Ruff 2006), low-power LF magnetic fields (Schiffbauer and Mowrey 2001), and RFID (Ruff and Hession-Kunz 2001).

Radar systems can detect humans and smaller pieces of mobile equipment. However, they are prone to false alarms, especially if they are sufficiently sensitive to ensure absolute detection of humans. Direct sunlight, reflections, and hot equipment can generate false alarms in infrared-based systems. Physical orientation of pedestrians can impact detection in RFID-based systems, though these systems usually have few false alarms.

As with GPS-based PWSs, as the number of persons or objects to be detected increases, more tags/sensors are needed, increasing maintenance requirements.

AUTOMATIC CONTROL

Automated controls play an important role in mining, be it robotic control (such as a telecontrolled or remote-controlled device) or algorithmic control. This section only discusses algorithmic control, where the goal is to control a process parameter within tight limits by manipulating relevant process variables. For example, a SAG mill may be operated to optimize power consumption, by varying parameters such as feed rate and recirculating load. Similarly, the quality (and

quantity) of a pile of coal is controlled by acting on real-time quality of coal on a moving conveyor belt. As communication networks have expanded, so has automatic control and, therefore, they are more important than ever. This section discusses aspects of automatic control that are relevant to mining engineers; some other aspects such as artificial intelligence are covered in Chapter 9.10.

Factors Affecting Control

Simply speaking, *controlling* implies manipulating/controlling a set of process parameters, based on process information, to achieve certain goals. For example, in a particular process for physically segregating coal, a flop gate directs a block of coal either to the wash pile or no-wash pile. The process parameters used for determining whether to open or close the flop gate (the action being controlled) are (1) the quality of the unit block of coal being considered, (2) the average quality and tonnage of the no-wash pile, and (3) the target quality for the no-wash pile. Such a process cannot be controlled using mathematics alone. Accurate real-time data from the on-line ash analyzer is essential if the segregation algorithm is to work as designed.

Sensor Accuracy

Although the need for sensor accuracy is obvious, defining the required accuracy of a sensor is not straightforward, especially for sensors that control processes. It is important to know how often control decisions are made in the process. The decision interval could be defined in units such as time (time between actions being controlled) or throughput (every 1 t [metric ton], 10 t, etc.). Therefore, for the previous example, an on-line ash analysis that is within ±0.25% of the true ash content for every 100 t of coal could result in the gate flopping the wrong way up to ten times for every 100 t if the gate flops every 10 t. It is possible to compensate for such errors by lowering or raising targets, but at that point, optimization, the much-touted benefit of using sensors and automated control, is lost.

Therefore, for a sensor to be useful for automatic control, accuracy standards have to be relevant to the function being controlled. Also, control algorithms should be examined as to the effect of measurement errors (of the process variables that go into the algorithms). Mine management should also ensure that sensors are properly calibrated at all times. Sensors are often neglected after initial installation; unless they are maintained properly, they can hurt rather than help optimization.

Relevancy of Process Inputs

In modeling SAG mill power consumption, Ganguli et al. (2006) examined the usefulness of many seemingly relevant process variables, such as feed rate, density, bearing pressure, revolutions per minute, noise, and recirculating load, as inputs. However, modeling revealed that only revolutions per minute, feed rate, and recirculating load had any noticeable impact on electric current draw. The study demonstrated that the assumption that a process parameter is relevant to the process may be erroneous.

There are two major reasons a seemingly relevant process parameter is found to be irrelevant. The first is simply that the parameter is truly not (sufficiently) relevant. The second and more subtle reason is that the sensor is not appropriate for the process. For example, its accuracy may be at such a broad or coarse level that the short-term information/measurement it provides is simply noise, in which case, any attempt to control the process using the short-term measurement data will be unsuccessful. Note that the inability to control based on a process parameter does not make that parameter irrelevant; it simply means that the *data* being provided by a sensor concerning that parameter are irrelevant. If engineers are confident that a particular process parameter should indeed be relevant, they should invest in a different sensor. Conversely, if the sensor is accurate and is providing appropriate data, then that parameter is truly irrelevant to the process.

Validity of Assumptions

Most algorithms, whether a truck dispatch optimization algorithm or a coal segregation algorithm, make mathematical and informational assumptions. Common mathematical assumptions include data characterization (such as assuming a statistical distribution) and model type (assuming that a particular model still holds). Informational assumptions include cost figures (such as the cost of processing is X-dollars) and values of various constants. Optimization algorithms that run with the wrong assumptions will not be successful. However, assumptions are rarely examined. Many figures, especially cost numbers, are constantly changing. Introduction of new mine monitoring and control systems should, therefore, be preceded by a thorough analysis of the benefits.

CONCLUSIONS

This chapter presented an overview of mine monitoring, communication, and control. Mine communications systems are no longer dictated only by productivity requirements; safety-related communication demands are now an important consideration when setting up a communications network. The desired attributes of a communications system include the ability to accommodate a wide range of applications, users, and bandwidth; it must also be safe, redundant, and robust, as well as able to handle information source/destination mobility. Wireless communications, especially mesh networks, have strengthened surface communication networks. Although high-bandwidth, fully redundant networks are still not possible underground, advances in technology may make such technology possible one day. Real-time tracking of miners underground is sure to push technological limits. Technologies such as GPSs and RFID will continue to make mine-to-mill optimization a reality. Modern communication networks, along with sophisticated control algorithms, certainly allow for that possibility. To optimize sensor systems, sensor accuracies should be properly understood and taken into account.

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CHAPTER 9.4

Mine Surveying

Andrew Jarosz

A mine surveyor provides services from early design and project implementation throughout a mine's life to closure. Surveyors deal in spatially referenced information—that is, information associated with known positions in space. They measure angles and distances to document the positions of mining features and boundaries, and present the information in the form of mine maps. That information is the foundation upon which mineral resources and reserves are defined, mineral and land rights secured, and environmental and mining plans defined, directed, and monitored.

Mine surveying has always been recognized as crucial for the efficient management and safety of mining operations. Because of its impact on safety, the position of the mine surveyor may require authorization, which is issued by regulatory bodies on the basis of education and experience.

Recent technological advances have greatly improved the precision and speed with which spatial information can be gathered, processed, and stored. Mine surveyors must constantly supplement traditional measuring skills with new skills focusing on management, processing, and storage of large amounts of gathered spatial data.

The changing role of the mine surveyor and changes in mine surveying as a discipline are recognized by mine-surveying professional organizations. The International Society for Mine Surveying (ISM) offers the following definitions in the preamble of its statute:

> Mine surveying is a branch of mining science and technology. It includes all measurements, calculations and mapping which serve the purpose of ascertaining and documenting information at all stages from prospecting to exploitation and utilizing mineral deposits both by surface and underground working.
>
> The following are the principal activities of mine surveying:
>
> - The interpretation of the geology of mineral deposits in relation to the economic exploitation thereof.
> - The investigation and negotiation of mineral mining rights. Making and recording, and calculations of mine surveying measurements.
> - Mining cartography.
> - Investigation and prediction of the effects of mine working on the surface and underground strata.
> - Mine planning in the context of the local environment and subsequent rehabilitation.
>
> These activities involve the following:
>
> - The determination of location, structure, configuration, dimensions and characteristics of the mineral deposits and of the adjoining rocks and overlying strata.
> - The assessment of mineral reserves and the economics of their exploitation.
> - The acquisition, sale, lease and management of mineral properties.
> - Providing the basis of the planning, direction and control of mine workings to ensure economical and safe mining operations.
> - The study of rock and ground movements caused by mining operations, their prediction, and the precautions and remedial treatment of subsidence damage.
> - Assisting with planning of rehabilitation of land affected by mineral operations and collaborating with local government planning authorities. (ISM 2009)

Clearly the mine surveyor plays an important role in almost all engineering activities carried out at a mine site throughout the life of a mining project, from exploration to closure.

Mine safety is always a key concern in mine surveying, and deficiencies in survey information can result in direct risks to safety (with the possibility of multiple fatalities), finances, legal status, and reputation. Mine surveyors are responsible for accurately measuring, recording, and informing mine

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management of the position of the mine workings relative to mine design and to geological features, boundaries, hazards, areas of restricted mining, and surfaces and/or adjacent workings that require protection. Most mine-surveying legislation is directed at regulating this function and ensuring that accurate mining records are kept. In the absence of regulations, the United Nations International Labour Organization (ILO) describes mandatory minimum requirements for mine surveying in its codes of practice for mine safety (ILO 1995).

SURVEYING EQUIPMENT AND TECHNIQUES

Generally, a mine surveyor measures geometrical elements—angles and distances—to determine the relative positions of survey control points (benchmarks) as well as terrain and underground features, and presents this information graphically on a map.

The Cartesian coordinate system is commonly used to define the positions of measured objects by means of x- and y-axes on the horizontal plane pointing east and north, respectively, and the z-axis pointing in the zenith direction. This system imposes specific requirements on angular and distance measurements that can be used for the calculation of position:

- Angles must be measured in horizontal or vertical planes.
- Distances must be measured along horizontal or vertical lines. If a distance is measured along a slope, the angle between the slope line and the horizontal plane must also be determined.

For many years, accurate measurement of angles and distances posed a significant technical challenge. Over the last century, however, developments especially in precision mechanics, optics, and electronics have led to the fabrication of surveying instruments that are capable of reliable and highly accurate angular and distance measurements. Today's standard instruments can measure angles with an accuracy of better than 1 arc-second and distances with an accuracy of ±(1 mm + 2 ppm).

Instruments for measuring angles are called theodolites. Instruments for measuring distance include tapes, subtense bars, and electronic distance measurement (EDM) devices. The previous strict division between these types of instrument is now blurring, and these instruments are merging into one universal instrument, the so-called "total station."

The past 20 years have also seen the development of new surveying instruments capable of determining position without reliance on traditional geometrical structures linked to benchmarks on the earth's surface. This new technique uses the Global Navigation Satellite System (GNSS) as a framework of reference points, and space triangulation as a means to determine the location of surveying benchmarks and ground features. However, because this technique uses electromagnetic signals as a carrier, distances between a Global Positioning System (GPS) receiver and satellites can be determined only in open areas on and above the earth's surface. Surveys of tunnels and underground mines must still be performed by means of classical surveying techniques.

Detail surveying has also undergone significant technological change. Recent developments in laser scanners have led to instruments capable of providing reliable and almost instant three-dimensional (3-D) models of terrain and objects. Scanners also have application in underground mine surveying where they can be used remotely to determine the extent and volume of mined cavities without compromising surveyor safety.

SURVEYING AND MAPPING PROCEDURES

In open-pit and underground mines, a mine surveyor is usually responsible for the following tasks:

- Conducting all mine-site surveys
- Producing and maintaining all mine-site and underground level plans
- Managing all survey-related data
- Managing landownership issues and lease tenements
- Monitoring slope stability and subsidence effects of underground mining
- Providing maps and spatial information to other engineering departments

In most western countries, national or regional (state) regulations establish minimal standards for mine surveys and map requirements. Of the extensive list of regulations, the following deserve special mention:

- U.S. Federal Mine Safety and Health Act of 1997, Section 312, amended by the Mine Improvement and New Emergency Response Act of 2006, or MINER Act (MSHA 2006)
- West Virginia (United States) Chapter 22A, Article 2, Miners' Health, Safety and Training—Underground Mines Mining Laws, Rules and Regulations (West Virginia Legislature 2008)
- Government of Western Australia Mines Safety and Inspection Act 1994, Section 87 (Government of Western Australia 1994)
- New South Wales Mine Health and Safety Act 2004, Section 75 (AusLII 2009)
- South African Mine Health and Safety Regulations, 1996, Chapter 17: Surveying, Mapping and Mine Plans (SAGI 1996)

Regional laws and regulations for all types of mines vary and in many cases are more rigorous than are national (or federal) laws and regulations. An example of this phenomenon is the West Virginia code pertaining to requirements for underground maps for deep coal mines (West Virginia Legislature 2008).

Laws and regulations in most states and countries require that mine maps be referred to the national mapping coordinate system and at least two or three permanent survey monuments be established on the mine property. The monuments should be described and shown clearly on the maps so that they can be found quickly and easily in case of emergency.

Mine maps are useful not only for determining relative location but also for determining volumes of ore and rock extracted, so that quantity control can be exercised and royalty payments calculated. Almost all mine engineering work depends on mine maps.

SURVEYING AND MAPPING OF OPEN-PIT MINES

Surveys of open-pit mines combine characteristics of engineering and topographic surveys. The surveyor provides guidance for miners to develop mining operations according to the earlier-established mine plan, then surveys the progress of mining and develops maps and models representing its current state. The maps and models are used for calculating the volumes and tonnages mined and for reconciling mining progress with the mine plan. Surveying of open pits usually involves the following activities: (1) establishment of a mine-survey control network, (2) detailed topographic surveying

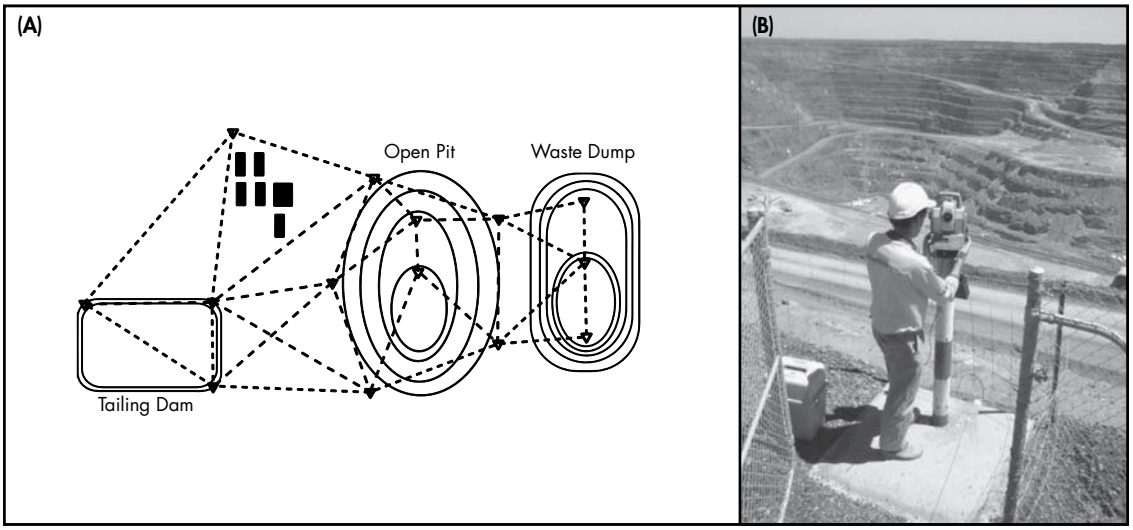


Figure 9.4-1 (A) Mine-survey control network and (B) surveyor using a theodolite on a control pillar

of open-pit and waste dumps, (3) data processing to calculate mined volumes and tonnages, (4) stability control surveys of open-pit and waste dump slopes, and (5) support surveys for earthmoving-machine control systems.

Mine-Survey Control Network

A mine-survey control network for an open-pit mine site is a basic, highly accurate, spatial infrastructure for relative referencing of natural and constructed topographical features, mining roads, benches, buildings, and other structures. It enables surveyors to

- Conduct cyclic surveys for the control and reporting of mining progress,
- Determine and control the elevation and slope of working benches and haul roads,
- Determine the volumes of mined ore and waste, and
- Control and manage the mine dewatering system.

A control network should be linked with the national survey network to enable transfer and use of the national coordinate system.

In most cases the primary control network for a mine is based on triangulation and/or trilateration (Figure 9.4-1A). It is important that control-point positions be (1) clearly visible from the main working areas of the mine site, (2) visible from multiple other control points, and (3) clear of mining activities and other disturbances that could lead to their destruction. Control-point positions are usually determined from angle and distance measurements that can be supported by GPS observation.

As a general guideline, the horizontal accuracy of a primary control network should be 1:20,000 or better. Control benchmarks are usually stabilized permanently by means of concrete-filled steel pillars set on concrete foundations (Figure 9.4-1B).

A primary control network can be densified with secondary and tertiary control networks by means of any combination of micro-triangulation, intersection and resection, traversing, and profile lines. Lower-class control points can be used as stations for detailed topographical and setting-out surveys. The positional accuracy of such points should be no worse than about ±0.2 m. For such accuracy, closed traverses should be ≤2.5 km and open traverses should be ≤1 km.

Topographic Surveys

A detailed topographical survey at an open-pit mine focuses on the locations of bench-slope crests and toes, berms, road edges and gradients, ditches and water dams, waste dumps, power lines, buildings, and other permanent and temporary structures. It includes the collar positions of exploratory and blasting drill holes and information on existing cavities created by previous mining activities. The following survey methods are available: total station survey, aerial and terrestrial photogrammetry survey, GNSS (GPS) survey, and laser-scan survey.

Total Station Survey

A total station survey involves determining the directions and distances to measured points and then simultaneously calculating the horizontal and vertical positions of those points by means of the so-called radiating method. Such a survey is especially useful in open-pit mines where benches, slopes, and roads can differ significantly in elevation. Survey data are collected, stored in instrument memory, downloaded to a computer in the survey office, and converted by application software into a map of the surveyed area.

In the past, to obtain such data, surveyors sighted instruments on reflectors positioned by assistants at various points of interest, and the risk of falling down a slope or being struck by falling rocks was significant. Today's total stations do not require reflectors and can measure distances to almost any object up to several hundred meters away. Some total stations can be combined with GNSS receivers (such as the Leica SmartStation) for rapid station positioning, which is useful when a station cannot be set directly over a control point but should be done only in exceptional circumstances when a rapid survey is required. Obtained results must be checked later or linked back to the mine-survey control network.

Figure 9.4-2 shows a typical total station survey of a mining road and benches.

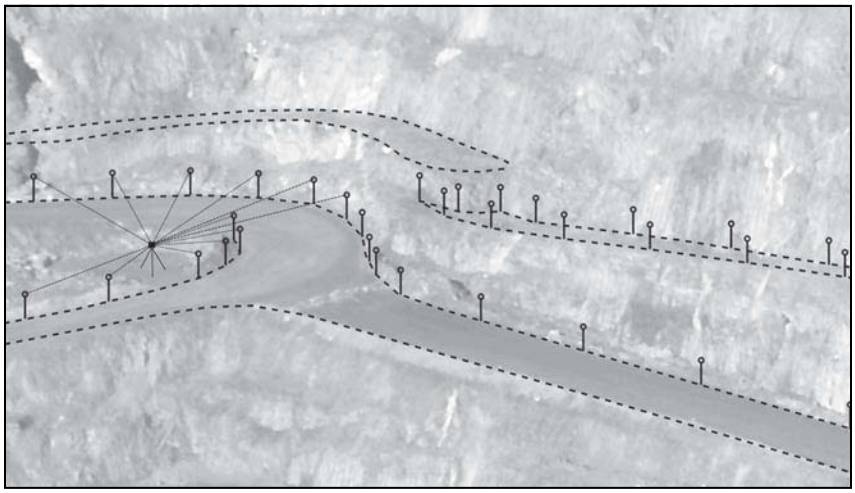


Figure 9.4-2 Typical total station survey of a mining road and benches

Aerial and Terrestrial Photogrammetry Survey

Mapping by aerial photogrammetry is useful for large surface operations, especially those with large differences in elevation, areas that are difficult to access, or areas with large amounts of traffic. The method provides numerous advantages, including (1) real-time registration of details that can be revised at a future date, (2) remote and safe collection of data, and (3) relatively rapid surveying of large areas. Major disadvantages are that final results are not immediately available, but rather require weeks of data processing and that significant advance preparation is required to mark and survey control points, so the method is not economically feasible for smaller mining operations. The positional accuracy of details obtained by this method is in the range ±0.1–0.2 m.

Mapping by terrestrial photogrammetry is useful for high scarps and slopes with rock structures and slope deformations. The method involves taking images from control points of known position with special dedicated metric cameras. From images of an object taken from different locations, a 3-D model can be developed for use in map creation. Some systems use images produced by high-quality commercial digital cameras equipped with nonmetric lenses.

GNSS (GPS) Survey

Current GNSS receivers using GPS real-time kinematic (RTK) technology deliver relatively high positional accuracy (better than ±0.1 m) comparable to that delivered by total stations. GNSS is now the survey method of choice for open-pit mines and is used not only for inventory surveys but also for conducting setting-out surveys, positioning blasthole collars, and locating mining equipment (particularly dozers, draglines, drill rigs, shovels, and tracks). Survey data are collected, stored in instrument memory, downloaded to a computer in the survey office, and converted by application software into 3-D models and maps of the surveyed area. With wireless data-communication systems, maps can be produced in real time.

Laser-Scan Survey

Since 2000, laser scanners and associated processing software have developed into fast, reliable, and accurate long-range surveying devices. Because they do not require the use of reflectors or staff at a mining face, they have made the entire range of surveying operations significantly safer.

Scans are performed from a stationary position, and collected data are transmitted directly to the surveying office. Alternatively, some scanners can be mounted with GNSS receivers on the roofs of four-wheel-drive vehicles.

Figure 9.4-3 shows a laser scan of an open-pit mine.

Data Processing

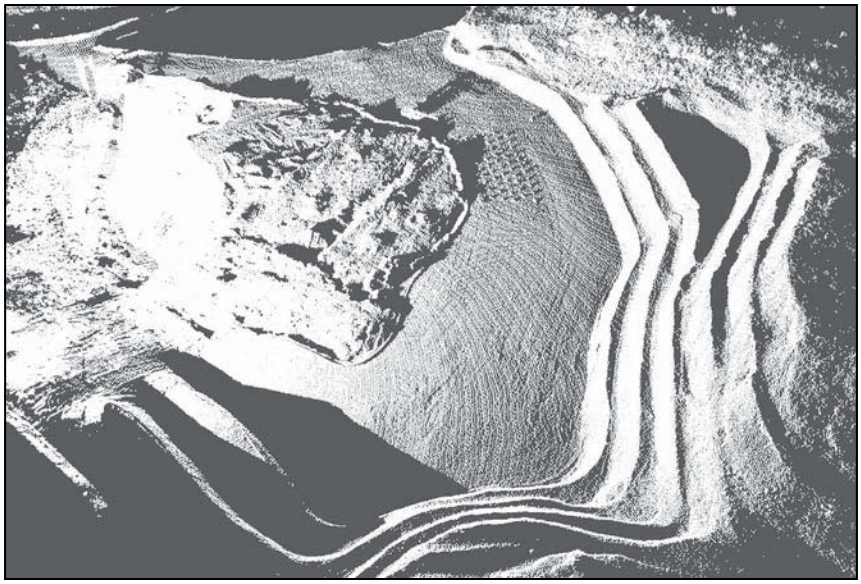
As mentioned previously, survey data are collected, stored in instrument memory, and downloaded to a computer in the survey office where specialized surveying software manipulates the data, performs calculations, and creates 3-D digital models and maps. Mining regulations often require that maps be created to specific standards with respect to, typically, a scale or map coordinate system. In recent years, constantly updated 3-D digital models have replaced classical mine maps.

Among the many surveying software packages available today, most popular are the so-called general mine design packages for creating geological models, mine designs, and plans. The following are particularly popular: Gemcom Surpac (Gemcom Software International), Datamine Studio 3 (Datamine Corporate Limited), Vulcan (Maptek), and Carlson Survey (Carlson Software).

Three-dimensional digital models of mining operations enable relatively easy calculation on a personal computer of the volumes of waste and ore that have been mined. Most of such models use triangulated irregular networks (TINs) to represent the original topography and working levels of a mine. Rock volume between mine levels is calculated by taking the difference between solids, with the base at an arbitrary level (e.g., 0 m) and the top at the mine level represented by the TIN (Figure 9.4-4). Volume can also be calculated from a block model by summing the volumes of individual cells representing these deposits.

Slope-Stability Surveys

Modern open-pit mining operations increase in efficiency as the ore-to-waste ratio increases, usually leading to maximization of pit-wall slopes. However, steeper slopes pose greater risk to personnel and mining equipment due to increased potential of slope failure. To uphold the required safety standards and reduce risk to an acceptable level, many open-pit mines must develop a holistic slope-monitoring program (Cawood and Stacey 2006). Such programs should distinguish



Courtesy of RioTinto Iron Ore Technical Services Group.
Figure 9.4-3 Laser scan of an open-pit mine

the following three slope-monitoring stages: (1) overall monitoring of all pit walls and adjoining areas, (2) focused monitoring of potential instability areas, and (3) detailed monitoring of areas with earlier-detected instability or failure.

Monitoring techniques should also take into account the expected mode of slope failure. There are four major modes of failure in open pits:

1. Circular failure with large rock movements along vertical planes at the top of failure and horizontal movement in the toe zone
2. Planar failure with rock-mass movement along an existing geologic discontinuity, such as a bedding plane, parallel to the slope face
3. Wedge failure when two sets of flat failure surface planes intersect and dip out of the wall; moderate vertical and horizontal movements are expected
4. Toppling failure when vertical or near-vertical structures dip toward the pit wall; large horizontal rock-mass movement is expected at the top of failure

The relative locations of control stations and targets used in a slope-monitoring network should be selected so as to be maximally sensitive to expected rock movement.

Traditionally, specialized geotechnical and surveying methods were used for slope-stability monitoring. However, recent developments in satellite and terrestrial remote-sensing technologies have significantly impacted the methods currently used. Monitoring on a large scale can be done using interferometric radar technologies. If the primary concern is vertical movement (subsidence) of areas adjacent to slope crests or bench areas of large open-pit slopes, satellite-based interferometric synthetic-aperture radar (InSAR) can be used to detect these movements (Jarosz and Wanke 2004). InSAR images can easily cover large areas of the whole open-pit mining operation. Ground-based radar systems, also using interferometric technologies, recently became the tools of choice for continuous monitoring of wall faces. The most successful and popular systems are Slope Stability Radar by GroundProbe (Australia) and Movement and Surveying Radar by Reutech

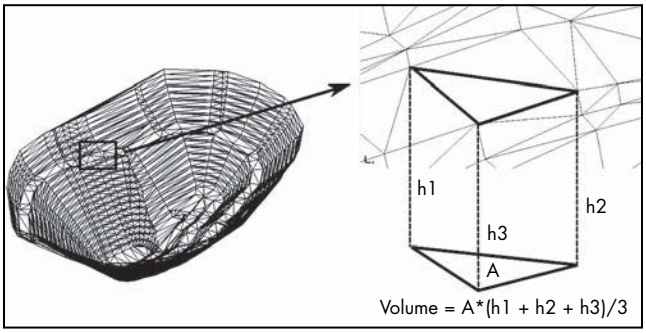


Figure 9.4-4 Triangulated irregular networks used for volume calculations

Mining (South Africa) (Reeves et al. 2001; McHugh et al. 2006). These systems can provide continuous measurements of rock movement across the entire face of a slope wall with submillimeter accuracy. Real-time processing of collected data enables confident tracking of slope movement and management of risk while optimizing safety and productivity.

Monitoring on a smaller scale or of distinct targets can be done using classical surveying techniques. Control stations must be established at stable locations from which targets placed on pit walls are observed. Station stability must be ensured and controlled. Stations should be erected as concrete pillars set into bedrock or into a stable foundation. Stations should be linked by means of a control network with other stations located far from the mining area and considered to be stable. Repetitive surveys, once or twice a year, must be performed to check the stability of the control network. For continuous monitoring of wall targets, automatic high-precision total stations can be used. Such robotic total stations are programmed to automatically measure directions and angles to targets and transmit data to a control computer in the survey office. Data are processed in real time and information regarding target movement is reported out. If movement is greater then the earlier-defined critical value,

an alarm is triggered, enabling timely withdrawal of mining personnel and equipment. Automated slope-monitoring systems are offered by major producers of surveying equipment as well as by independent developers. The most advanced and popular are the Leica Automatic Deformation Monitoring System (GeoMoS) by Leica Geosystems and slope-monitoring software from SoftRock Solutions (Western Australia).

Support Surveys for Equipment and Machine Control

Timely and accurate positioning, as well as monitoring, of mining equipment is increasingly important to efficient mine operation. The following activities are highly influenced by equipment positioning:

- Drilling, where accuracy is critical to efficient blasting operations
- Dragline positioning, where accurate and efficient movement of overburden material is critical for strip mining
- Elevation (gradient) control, where continuous assessment of ground movement must be performed and adjusted in real time
- Grade control, where 3-D mapping of mineral deposits and control of ore digging and loading are required
- Track assignment, where mining haul trucks must be directed to the right place at the right time and each load of ore or waste, depending on its quality and characteristics, must be dispatched to its proper destination

Accurate equipment positioning can be achieved by means of GNSS receivers using RTK technology. However, the vertical accuracy of a standard RTK GPS system is two to three times less than its horizontal accuracy, with vertical standard errors of ±0.01–0.02 m. Such accuracy is sufficient for most mine sites other than finish-graded roads that require tolerances of just a few millimeters. Such greater tolerances can be achieved by combining a rotating laser with a GNSS system. Accurate positioning of a dozer blade can be obtained in the horizontal position by using an RTK system and in the vertical direction by using a laser collimation plane. Lasers with a 10-m vertical swath increase this range even farther, allowing multiple machines, working at distances up to 300 m (linear) and 10 m (vertical) from the base station, to receive grade corrections. Such systems provide the tightest machine controls technically available.

The leading machine-control systems have been developed as collaborations between manufacturers of mining equipment and providers of GPS receivers.

A computer-aided earthmoving system (CAES) combines GPS (the United States's Global Positioning System) and GLONASS (Russia's Global Navigation Satellite System) positioning with an onboard display for the machine operator and wireless IP (Internet Protocol) communications between the machine and the office. Graphical and textual cut/fill information provided to the operator eliminates the need for most survey staking. CAES can be used for haul road and bench construction and maintenance, production dozing, leach-pad construction and maintenance, reclamation, ore-grade control, material identification, and coal load-out terminals. It is designed to be used on scrapers, loaders, dozers, shovels, motor graders, hydraulic excavators, and track-type tractors. Data on the status of each machine are sent wirelessly to the control office for monitoring by mining supervisors (Figure 9.4-5).

SURVEYING AND MAPPING OF UNDERGROUND MINES

In an underground mine, surveys are typically performed in dark, confined, and often wet spaces, and special equipment and methods have had to be developed to comply with underground safety regulations. Mining drifts (drives) and tunnels tend to be long and narrow, requiring the use of traverses as the main survey control. Control points are generally located in the back (roof) rather than in the floor, where they are less subject to heavy traffic and less vulnerable to damage. For use in the dark, instruments must have illuminated cross hairs, targets, and reading systems. Other adverse conditions to which instruments are exposed include dripping water, high-velocity ventilation air, high temperature, dust, strong electromagnetic fields, and mine gases, and therefore only safety-certified equipment can be used (Hart and Parrish 1995). Areas that are unsafe to access may require the use of specialized equipment that can be remotely controlled.

Horizontal and Vertical Control

Deep-mine surveys must be performed in 3-D, and underground and surface surveys must be correlated. If a rescue borehole is required between the surface and a specific underground location, or between two underground locations, survey data and survey marks must provide sufficient reference to enable accurate directional drilling. The location of mine workings must be accurately controlled, especially in relation to mining lease boundaries and other important surface features such as highways, railroads, power lines, gas lines, wells, and historical buildings. Adjacent mine workings must be surveyed and presented in the same coordinate system to eliminate unintended breakthrough that could lead to disastrous gas or water inrushes. Maps of all surrounding mines and other relevant areas must be reviewed and fitted to the master map of the mining operation before the start of any mining project—a job made difficult because, all too often, adjacent mines, highways departments, railroad companies, gas companies, surface landowners, or mineral owners use different coordinate systems.

Chrzanowski and Robinson (1981) recommended that surveys be performed at three levels of accuracy: (1) creation and survey of a primary control network for the mining area and permanent workings, (2) survey of mine headings and development areas, and (3) survey of short traverses necessary to map active mining areas.

In the United States and many other countries, the first level of survey, a primary control network, is generally tied to the second or third order of the U.S. National Geodetic Survey (NGS) network and must adhere to the same accuracy specifications. In many cases the configuration and geometrical characteristics of the mine entrances require use of traverses as an underground control network. Traverses usually have many short (40–50 m) traverse legs and direction is difficult to maintain. It is advisable to perform surveys using quality theodolites or total stations that are maintained and checked regularly. When an underground control network extends over a large area and traverses are long with many legs, the use of gyro-theodolites may significantly improve their directional accuracy. The surveying and mapping of a control network is often performed by a specialized external contractor.

An example of a control network with ties to the NGS network is shown in Figure 9.4-6.

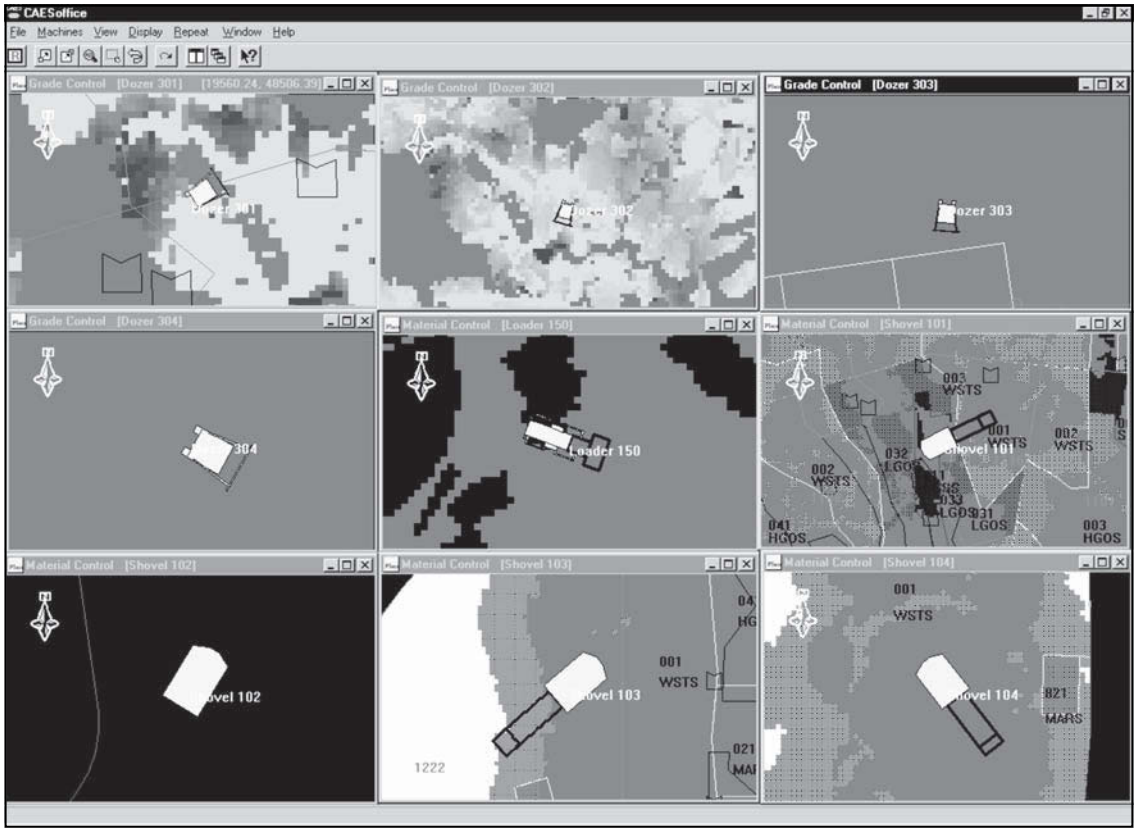


Figure 9.4-5 Computer-aided earthmoving system data on equipment position

The second and third levels of survey are generally performed by resident mine surveyors using typical surveying equipment such as total stations, standard theodolites, EDMs, tapes, and levels.

Traversing with Total Stations in the Backs (Classical Traversing)

Equipment used for surveying in underground traverses is not too different from that used on the surface. Electronic total stations, the primary angle- and distance-measuring instruments, enable automatic reading and storage of directions and distances. Distances are measured electronically with the help of EDM devices that are coaxially mounted in a telescope. For use underground, instruments should be protected from dust and moisture. For use in gassy mines, instruments must be certified for use in such environments; that is, they must be explosion-proof. Most theodolites and total stations today have been properly sealed at the factory and can be used in such environments. If, for safety reasons, EDM devices cannot be used, a steel tape can be used for distance measurements.

Underground survey stations are usually stabilized in the backs of underground drives by the following means to protect them from damage:

- Standard spads driven into wooden plugs set in holes drilled in native rock (spads are hooks for attaching plumb bobs and have defined spots for sighting)
- Power-driven studs driven directly into rock by a stud gun using an explosive charge
- Adhesive spads attached to the rock by adhesive plastic cement or epoxy
- Spads with clamps attached to steel arches or roof bolts

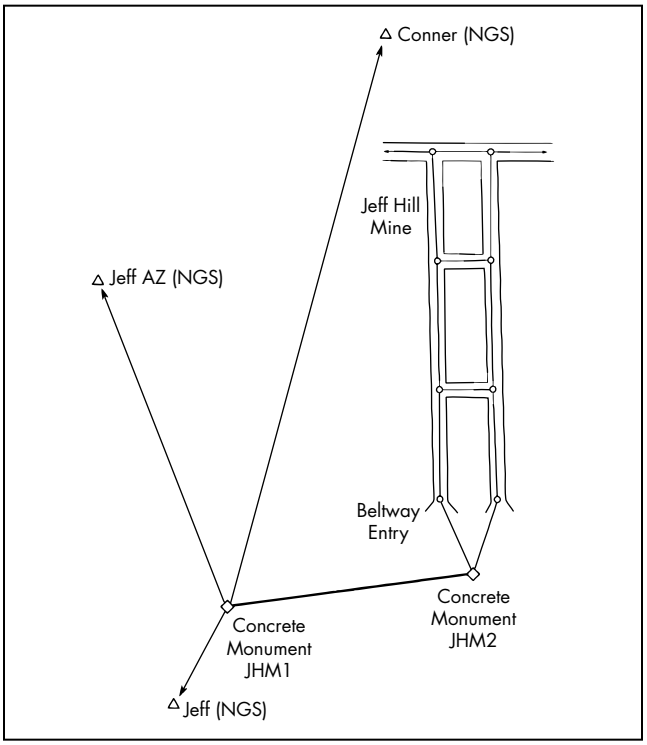
Survey stations should be identified with brass, aluminum, or plastic tags. It is important that tags are permanent and that metal tags not be attached directly to survey stations to prevent corrosion. Identification numbers on the tags can be either (1) sequential, assigned to stations as they are created, regardless of location; or (2) position-coded to identify station location (level, drive, or station number).

Modern theodolites and total stations are equipped with optical or laser plummets for fast, easy centering over or under the control point. If laser plummets are not permitted, standard plumb bobs can be used. When hung from a spad in the roof, the bob can be illuminated by a cap lamp held behind a translucent screen or sheet of paper. Small reflectors and targets can be attached directly to spads and conventional reflectors can be used on forced-centered traverses.

Most underground leveling is performed while traversing, due in part to time constraints, generally by means of the trigonometric leveling technique. Vertical angles can be measured with high accuracy, and thus trigonometric leveling provides similarly accurate results. During traversing and leveling, it is important that instrument height and target or signal height also be measured accurately. Measurements collected at an underground traverse station are shown in Figure 9.4-7.

Traversing with Total Stations in the Walls (Wall-Station Traversing)

Although the back provides a secure location for control points, it presents a number of disadvantages for mine surveying, the most obvious of which is the difficulty of installation



Source: Adapted from Hart and Parrish 1995.
Figure 9.4-6 Control network for an underground mine

and access. Locating control points in backs in a modern high-volume underground mine usually requires lifting apparatus to heights of >5 m. In contrast, locating control points in walls makes installation and access easier, faster, and safer.

Because a theodolite cannot be set under a control point, wall-station traversing requires use of a surveying technique other than that used for classical traversing. Rather, the resection technique (free stationing), with all available distances and horizontal angles measured, is used to determine instrument position. Wall-mounted points are observed from a temporary instrument station located in the drive.

Wall-station traversing has become popular in underground mining operations, particularly in Western Australia, and it is important that surveying professionals have an in-depth understanding of the methodology, accuracy, and limitations of the technique.

The instrument and target are shown in Figure 9.4-8. The surveyor secures a target to the wall by inserting a target stem into a wall sleeve mounted in a drilled hole. The method requires specially designed target prisms that retain a central position regardless of their rotation. Total station instruments must be coaxial; that is, distances and angles must be measured along the same line of sight and software must be able to determine instrument position from resection observations by means of the least-squares best-fit calculation method.

Analyses of wall-station traverse accuracy suggest that, for optimal directional accuracy, a configuration must have acute-triangle geometry (Jarosz and Shepherd 2004). Temporary instrument stations located normal or near normal to wall stations decrease the accuracy of bearing transfer. If acute-triangle geometry is not always possible, the surveyor should take additional observations to the previous and next instrument station to add rigidity to the survey structure and use forced centering when observing consecutive instrument stations, ensuring that the initial triangle geometry is acute (Figure 9.4-9).

Recent experiences suggest that wall-station traversing significantly increases the safety of surveying operations in modern underground mines with large drives. However, it requires the use of modern coaxial theodolite and zero-constant reflectors. Distance measurements must be accurate to ±(1 mm + 2 ppm) and angular measurements must be accurate to ±5″.

Position and Direction Transfer (Surface to Underground)

Determination of an underground baseline with coordinates and orientation (azimuth) to a surface coordinate system involves position and direction transfer. Special care must be taken with direction transfer, because errors significantly impact the correct positioning of underground control points. The standard position error σ_P is linked to the initial orientation error σ_{Az} by the following relationship:

$$\sigma_P = \frac{\sigma_{Az}}{\rho''} \cdot L$$

where

$\rho'' = 206265''$

L = direct distance between baseline and a traverse point

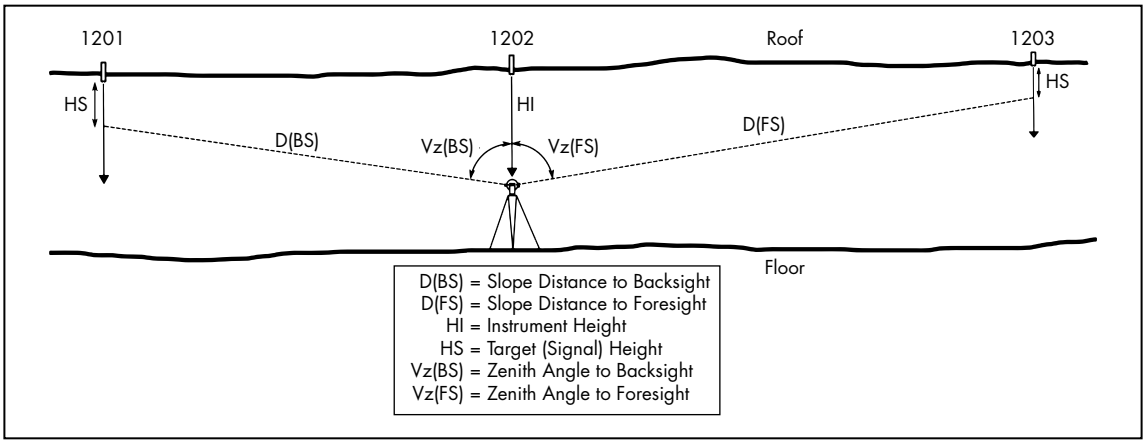
How position error grows as a function of initial orientation error and distance from the baseline is shown in Table 9.4-1. The total position error for a traverse point includes components linked not just to orientation but also to measured angles and measured distances.

The means of underground access determines the method for positional and directional transfer. The following methods are available:

- Traversing directly from surface to underground through a mine adit and decline
- Shaft plumbing using two or more plumb lines in one vertical shaft
- Shaft plumbing using a single plumb line with gyro orientation of the initial azimuth underground
- Shaft plumbing using a single plumb line in two or more separate shafts or rises, followed by so-called fitted-traverse linking of these wires

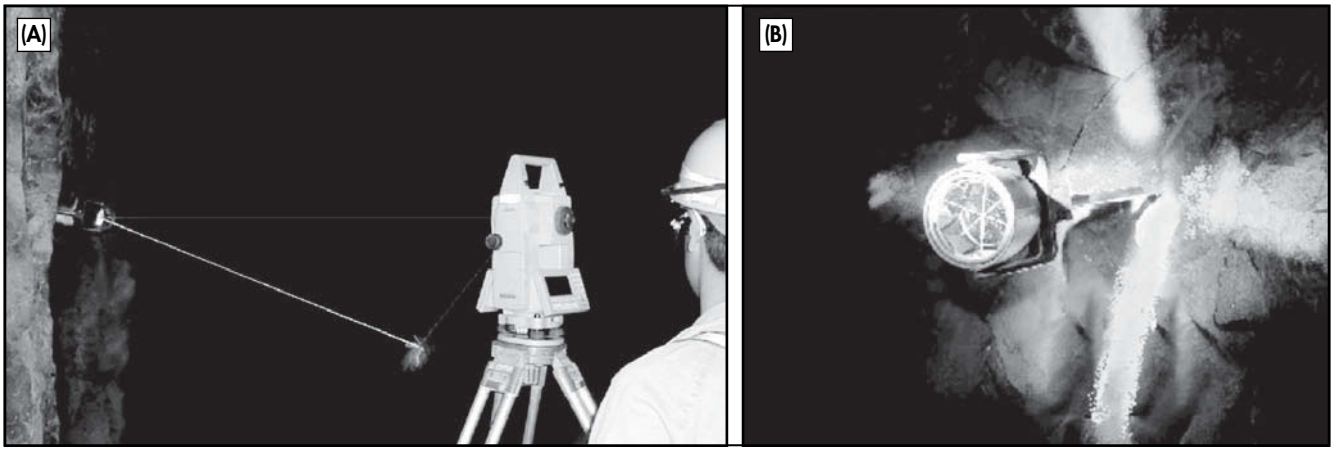
The traversing method involves creating a traverse that starts at permanent monuments on the surface and passes through an access tunnel or adit to the underground workings. After initial traditional traversing, the procedure is as follows: Establish a position on a spad in the roof of the underground drive and measure subsequent traverse legs with the instrument set under the surveying stations. When extending a control network, use double-angle traverses. Measure both left and right (clockwise and counterclockwise) angles at each station and check for closure. Double-center the instrument—that is, re-center it before each measurement. For improved traverse accuracy, force-center the instrument and targets to eliminate centering errors between instrument and targets. Level the theodolite accurately, especially in the steep sections of a decline, for accurate direction transfer.

The shaft-plumbing methods assume that the plumb lines are truly vertical. This assumption is particularly critical when the distance between plumb lines is relatively short (<5 m), as



Source: Adapted from Hart and Parrish 1995.

Figure 9.4-7 Classical traversing



Source: McCormack 2001.

Figure 9.4-8 Wall-station traversing: (A) configuration of instrument and target and (B) reflector inserted into a wall sleeve

when both lines are located in one shaft. Two plumb lines with random deflections e_1 and e_2, separated by distance b, result in a direction error σ_{Az} of

$$\sigma_{Az} = \frac{\rho''}{b}\sqrt{\frac{e_1^2 + e_2^2}{2}}$$

For example, plumb lines with random deflections $e_1 = e_2 = \pm 1$ mm, separated by distance $b = 4$ m, result in a direction error of $\pm 52''$.

A typical single-shaft plumbing setup is shown in Figure 9.4-10. When plumb lines are located in different shafts, the baseline is at least several hundred meters and the resulting errors have significantly less impact on direction transfer.

For the plumb line, piano wire can be used that is small in diameter but strong enough to support applied weight with a safety factor of between 2 and 3. It should be in good condition and stored on winch reels to prevent kinks.

The procedure is as follows: Lower the wire initially with a small weight attached, fix it in position at the upper end, ensure that it hangs freely without touching any shaft fixtures, and attach a heavy bob to the wire. As a general guideline, the weight of the bob in kilograms should be one-third to one-half the shaft depth in meters. To dampen oscillation, immerse the bob in water or oil and eliminate disturbances such as strong air currents from the ventilation system or falling water. Note, however, that oscillation of freely hanging bobs cannot be completely eliminated. Determine the vertical position of the plumb line by taking readings at the extreme left and right of oscillation on a scale placed behind the wire. Take readings from two stations located on two perpendicular lines. Determine the mean position of the line from these readings by fixing (clamping) each plumb line to its mean position using a special apparatus called a Smith plate.

Because a theodolite cannot be set under a plumb line, for direct measurement of orientation angles, the theodolite should be positioned next to the shaft and one of the following special geometrical solutions should be used to determine the orientation of the plane passing through the plumb lines (Figure 9.4-11):

- **Coplaning or aligning.** The instrument is set exactly on the line passing through both plumb lines.
- **Weisbach triangle.** A triangle is used to link both plumb lines and the instrument station. The instrument is

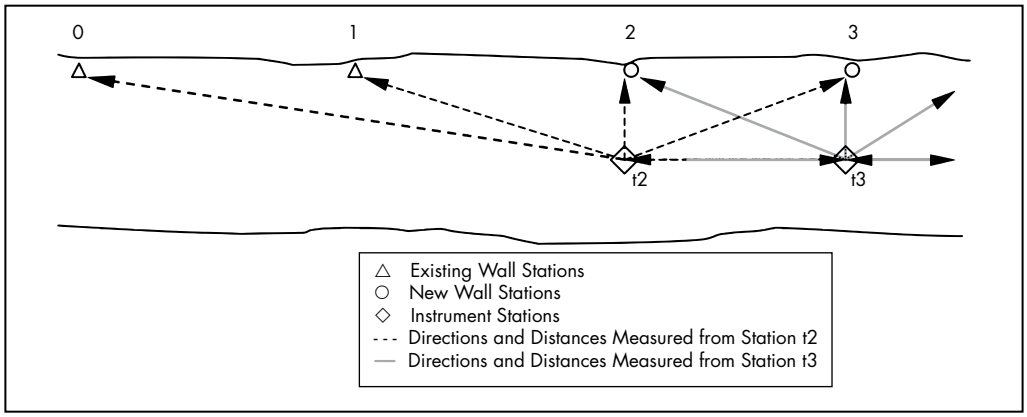


Figure 9.4-9 Wall-station traversing: additional observations between instrument stations

Table 9.4-1 Position error σ_P induced by orientation error

Distance from Baseline L, km	Initial Orientation Error σ_{Az}, m		
	±15″	±20″	±30″
0.5	0.036	0.048	0.073
1.0	0.073	0.097	0.145
2.0	0.145	0.194	0.291
3.0	0.218	0.291	0.436
4.0	0.291	0.388	0.582

positioned close to one of the plumb lines in such a way that the angle at the instrument station is acute (<10') and measured with utmost accuracy.

- **Hause quadrilateral.** When the instrument cannot be set at or close to the line passing through both plumb lines, a quadrilateral can be used to link both plumb lines and two instrument stations.

For optimal orientation accuracy, standard shaft plumbing should be performed in two shafts that are far apart from one another. The procedure is as follows: Conduct a simple surface traverse survey to give the location of each wire in the official map grid system. Then conduct an underground traverse survey to connect the two plumb lines. Initially calculate coordinates of this traverse in a local, arbitrary coordinate system. Then, because the map grid coordinates of the traverse endpoints (plumb lines) are known from surface surveys, recalculate the coordinates of all other points to the surface coordinate system by means of coordinate transformation. Transfer elevations underground by means of long shaft tapes. Calculate corrections to the tape length due to temperature, stretch, and pull, and adjust the measured length accordingly. Measure vertical distances with the help of an EDM device.

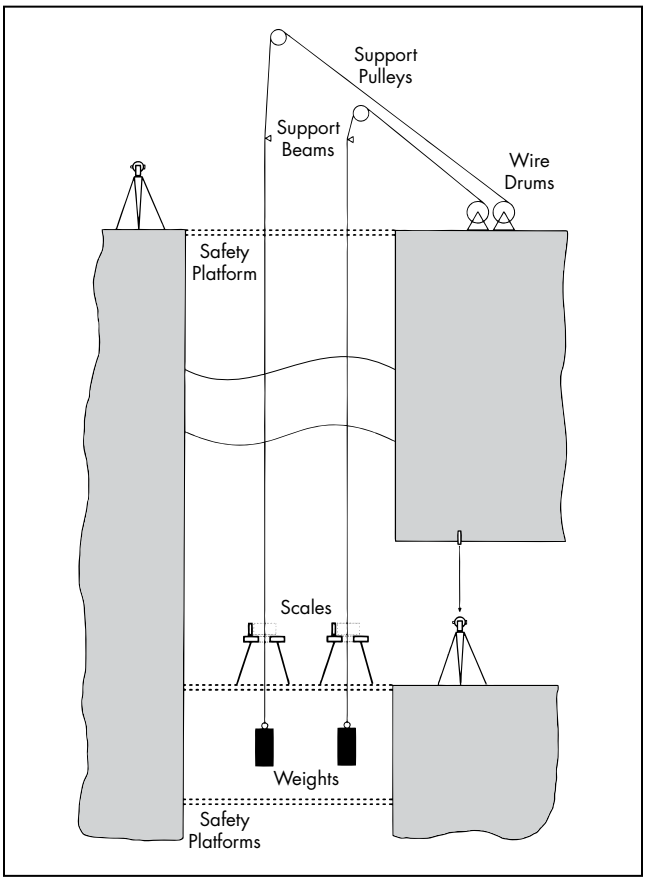


Figure 9.4-10 Single-shaft plumbing setup

Detail Surveys

The details of underground drives, drifts, and tunnels are derived from control stations established in mining structures. In most cases, detail surveys are conducted concurrently with control surveys.

Traditionally, the method of chains and offsets was used to perform detail surveys. The procedure is as follows: Stretch a reference tape between traverse stations. Measure offsets to the right and left walls at regular intervals (2–3 m) with a short tape. Measure vertical distances to the roof and floor with surveying staffs/poles. Note the details, distances, and offsets of existing features. It is sufficient to approximate the perpendicularity of offsets, due to their shortness and the measurement accuracy (±0.05 m). Higher accuracy (±0.01–0.02 m) is required in drives or tunnels with permanent roof and wall supports. Prepare a sketch of the survey in approximate scale. In cases such as when rebuilding a tunnel or drive, prepare a detailed cross-sectional survey. For a reference line, it is traditional to use a plumb line attached to a spad installed in the roof. Measure offsets at different elevations between the plumb line and the walls.

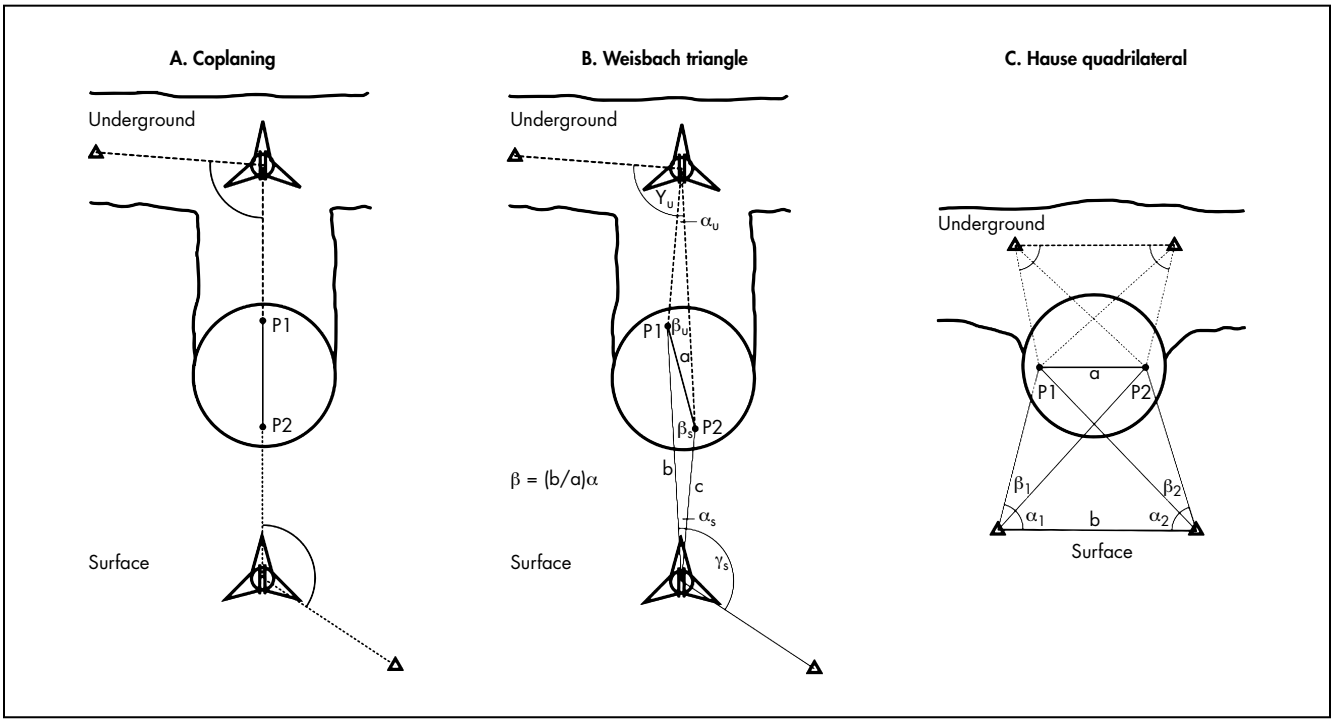


Figure 9.4-11 Geometrical solutions for direct measurement of orientation angles

Today, total stations with reflectorless EDM devices are used almost exclusively to perform detail surveys. The procedure is as follows: Survey a few lines (strings of points) running at different elevations and representing the outline of the structure of interest. Download the collected data from instrument memory to a computer in the survey office for processing, plotting, and, in most cases, creation of a 3-D model of the structure. The newest surveying technologies use laser-scanning devices for fast, continuous collection of detailed and accurate transverse profiles when the device is moved.

Massive Deposits

Mines extracting massive deposits are surveyed almost exclusively in 3-D. Total stations are used to survey access and development drives; laser-based monitoring systems are used to survey cavities and stopes—in the latter case, with the instrument inserted into the stope by means of a boom. The motorized surveying head scans the opening with the help of a laser rangefinder. A total station can determine the position of a device by tying it with positions of existing control stations.

Special inspection and surveying devices are also available to survey inaccessible vertical shafts and orepasses (Jarosz 2008). Collected data are converted to 3-D models with the help of specialized computer software. These models are linked with other models of development and access drives to provide a 3-D model of the whole mine. Such models enable creation of (on-demand) plans and cross sections in any scale or projection.

Tabular Deposits

Relatively thin tabular deposits are traditionally surveyed in two dimensions, with the projection plane parallel to the deposit. Such surveys should be performed in coal mines or mines from which metalliferous reef-type deposits are extracted (as in the gold mines of South Africa), using chain readings and orthogonal offsets to provide detail. The procedure is as follows: For a reference line, use a steel dip tape stretched between strike gullies or drifts. Use orthogonal offsets and triangulation to relate the position of the dip tape to spads installed in the gullies. Use a short tape and a surveying stick to measure offsets between the tape and the stope face. Also measure the width (height) of the face. If the deposit is on an inclined plane (reef), reduce the distances measured along the dip to true horizontal distances. In the survey office, create a horizontal map and use a planimeter to determine the areas between face positions and the volumes extracted.

Recent developments enable 3-D surveying of thin tabular deposits by means of electronic total stations combined with data-collection and display hardware and software. The results of an underground survey can be viewed in real time on a ruggedized laptop computer.

Thick tabular deposits extracted by room-and-pillar mining are surveyed almost exclusively with total stations. The surveying procedure is similar to that used for access and development drifts, as described previously.

Setting-out Surveys

Setting-out surveys provide direction and grade for mine drives, extensions, and tunnels, and are crucial to mine development. The following are common methods for performing this type of survey:

- **Grade sticks.** Two spads are placed at a set distance in the backs, and wooden poles are suspended from the spads. Marks are placed on the poles in such a way that, when lined up, the marks indicate the direction and gradient. The gradient is transferred to the development face.

- **Grade chains.** The method is as above, but chains are used instead of poles and small rings attached to the ends of the suspended chains are used instead of marks.
- **Strings and grade pegs.** Two sets of pegs, spaced 5–10 m apart, are placed in the sidewalls of a drive at calculated levels. Strings are run between the pegs to create an inclined plane with the design gradient. The gradient is transferred to the development face.
- **Direction and grade lasers.** For a short extension of direction and grade, a small laser is positioned in the wall opposite the development heading. The laser is inserted into a sleeve cemented into a hole drilled into the rock in such a way that the laser beam projects the direction and gradient for new development. A drilling pattern is marked at the heading. For a long extension, a larger laser is mounted in a special enclosure attached to the back of a drive. In both cases, a total station is used for laser setup. The direction and grade of the laser beam should be checked regularly. Today, lasers are used almost exclusively to determine direction and grade.

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CHAPTER 9.5

Dewatering Surface Operations

Paul R. Peppers

INTRODUCTION

Hydrology is an imprecise science developed by empirical observations by many people over the course of 3,000 years. Complex nonlinear systems are involved; many of the factors that go into the computations are probability based, time variable, imprecise, or uncertain.

Identifying and quantifying the risks associated with system failure usually sets the level of effort that is justified. More precise calculation methods are evolving, but, considering the uncertainty of the data, approximations are commonly used. The locations, quantities, and detrimental effects associated with water in the operating areas are continually changing as the mine development progresses. This effect renders the precise calculation of the performance characteristics of the pumping system at any particular point of limited operational value.

Computer-based systems hide the complexity of the calculations and allow more rapid solutions of the empirical equations. Complex computational fluid dynamic calculations are usually justified only if the costs and development times are substantial. For example, a manufacturer of a new class of pumps could justify the effort because upfront computer modeling may well reduce the cost and time needed to build prototypes. However, a mine engineer trying to decide if that pump would have sufficient capacity to pump from his pit probably could not justify that level of computation.

Some factors from the mine planning process are predictable with reasonable precision. For example, the hydrostatic lift necessary to pump water from a planned shovel face is predictable because engineers design the elevations of the pit face and pit rim in advance. Changes in these factors are predictable as long as mine operations follow the plans. The size and shape of the drainage basins that contribute to the runoff into the mine area are measurable in advance, but other critical factors affecting the quantity estimation of runoff are difficult to measure accurately. Meteorological factors, climatology, and actual rainfall can vary significantly from the predictions. Changes in land use and development over time reduce the predictive ability of long-term rainfall monitoring and flow-gauging station data. These can show up as drift in the mean runoff levels and intensity over time because removal of ground cover and construction of paving tend to limit infiltration and reduce the concentration time. Some examples of these factors follow:

- Natural basins and channels are nonhomogeneous in their ground cover, slopes, soil type, and moisture content.
- The variability of rainfall intensity over a basin increases with the size of the basin. Larger basins are subject to locally heavier precipitation intensities.
- Most precipitation records are point measurements. Rainfall gauges are small samples of the larger precipitation event. Spatial variability of storm patterns makes even nearby locations difficult to predict without on-site data.
- Streamflow gauges are better indicators than rainfall gauges, where these records are available, because of the integration of the runoff from the upstream basins, as long as the characteristics have not changed over time.

Historic observations used to determine the probabilistic frequency of rainfall events are uneven. In some places, reliable precipitation records may have recorded the variability over several centuries, while in others, only for decades. In the case of some greenfield developments, no historic data may exist at all. In semiarid regions, empirical methods are commonly used to predict rainfall runoff events, which make it difficult to quantify the magnitude of potentially damaging high-return period storms such as 100-year or 500-year events. Coverage is definitely better in the developed world than in remote areas.

MINE DEWATERING TEAM

Few people think of mine dewatering at a surface operation as a major issue when compared to other production tasks such as fragmentation, loading, hauling, crushing, or milling. If neglected, however, no other operations are possible in the mine. Mine dewatering requires applying knowledge from many fields to create practical, cost-effective, and efficient

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systems to manage the water. This chapter discusses practical field methods that provide sufficient levels of accuracy and precision to limit risk. The objective is to put tools in the hands of the engineers and technicians to manage mine water inflow and accomplish production effectively and economically. All of the following specialists have a role to play in managing water, but it is the role of the generalist to understand the big picture and know when to call the experts.

- **Mining engineers** overlap and coordinate with other specialty engineering disciplines to develop a cross-disciplinary engineering approach to the issues involved with prediction, collection, and handling of water in mine workings.
- **Meteorologists** track the data related to historic precipitation patterns, utilizing rainfall-measuring stations, streamflow gauges, radar, satellites, and modern database systems that supply the data to predict probable precipitation values and the associated statistical risks.
- **Civil engineers/hydrologists** predict the runoff recurrence and design the surface drainage controls, storage basins, and conveyance structures.
- **Mechanical/structural engineers** provide the expertise to design pumps, pipelines, and mechanical aspects of pumping stations, including concrete, supports, and pump engineering.
- **Electrical/instrumentation engineers** design the electrical power, distribution, instrumentation, and controls required to run, interlock, and control pump stations and dewatering systems.
- **Environmental engineers** provide the knowledge of federal, state, and local regulations for obtaining permits prior to commencing operations. They assist mines to predict, evaluate, and mitigate pollution from operations.
- **Land, legal, and water rights professionals** manage the legal aspects of obtaining permission to extract and use specific water quantities for industrial purposes. They also help miners manage disputes over historic ownership and water rights law.
- **Government agencies** fund and manage the basic research and development that provide much of the background hydrology data available. In the United States, the National Weather Service and the National Oceanic and Atmospheric Administration (NOAA) collect and distribute much of the precipitation data that are critical for predicting runoff. The U.S. Geological Survey provides a database of streamflow records for drainage basins and groundwater levels in monitoring wells located throughout the country.

REVIEW OF SURFACE HYDRAULIC CALCULATIONS

This sections gives a brief outline of the process of designing a runoff control system for a mine. The process can be broken down into a series of estimates required to scope the problem. Detailed explanations for these estimates are in Chapter 16.4.

Precipitation/Storm Events

To determine the amount of protection necessary for the various mine operations structures, the risk level of the proposed structure can be estimated based on failure consequences. The appropriate design storm point precipitation frequency (PPF) event can be selected from NOAA or other sources to determine precipitation amount and intensity (NOAA 1980).

An example of a design storm selection based on risk follows:

Risk	Design Storm
Low risk	10 years: 6-hour storm
Medium risk	100 years: 24-hour storm
Catastrophic risk	1,000 years: 24-hour storm or probable maximum precipitation

Basin Runoff Calculation

Using the rational method or one of a number of computational runoff-calculating models (U.S. Army 1971):

- Basin characteristics for each basin in the watershed are measured: shape, area, channel length, ground cover, and relief.
- PPF from the design storm for the area is applied: A measure of precipitation (millimeters or inches) that falls within the storm period (e.g., 25 mm or 1 in. of rainfall in 6 hours expected once every 10 years).
- Peak flow rate Q, time of concentration, and total runoff volume are calculated.
- Routing network is designed.

Using the flow information from the individual basins, the runoff protection network can be designed. This may consist of a number of structures to route the water from the source to the discharge point: open channels, culverts, detention and sediment control basins, spillways, weirs, and flumes. The U.S. Army Corps of Engineers Hydraulic Engineering Center (HEC 2009) tools are helpful, as is the American Iron and Steel Institute handbook (Spindler 1971).

SAFETY ASPECTS OF DEWATERING

Water in mine workings presents a number of hazards to the safe operation of the mine. Sudden precipitation events result in rapid, high-volume runoff into surface mines along the road network and over the pit rim. Water allowed to run over highwalls can cause erosion, bringing down debris, talus, and raveling, thereby creating hazardous conditions to mining operations below. Ponding, erosion, and degradation of the haulage surfaces and working benches mean that all operations slow down to maintain control of the equipment. Slides and rockfall risks increase when water percolates into cracks in the highwall.

In cold climates, ice and freeze–thaw cycles increase the risk of raveling and rockfalls as the freezing water swells inside the fractures, wedging the blocks apart. Ice builds up where water seeps out of the highwall. When these overhangs fail and fall, they can cause extensive damage, especially if the ice mass is large. Seasonal thawing causes these ice buildups to break loose and rain rocks and ice into the pit.

Drainage control maintenance should be conducted prior to the arrival of seasonal precipitation in areas where this is predictable. Such control measures include filling surface cracks around the pit perimeter and diverting runoff from the pit rim to limit water infiltration. Regular inspection of ditches, culverts, and drainage channels for damage and cleaning away debris to allow maximum water flow should become standard operating procedure. Scheduling of major construction or modifications should allow for completion during the dry seasons, avoiding critical system downtime on dewatering systems.

High-volume precipitation events cause flash flooding, particularly in mountainous areas and arid climates, such as in the American Southwest. Proper planning of the pit location and associated mine facilities must take into account drainage protection. Significant storm runoff can be controlled by properly designed drainage controls that direct runoff away from the pit to prevent problems before they occur. In-pit haulage roads and benches should be sloped to prevent ponding and direct water through perimeter ditches to sumps for collection. Culverts must be adequately sized to convey water where roads intersect drainage crossings.

Emergency Planning

Contingency plans must exist to protect personnel from entrapment if flooding from sudden inflow of runoff results in loss of the main access route. This can happen if the haul road washes out or slides off the wall because of slope failure. Ensuring that the pit designs include emergency evacuation routes must be part of the mine planning process. Surface mines usually have more than one escape path from the pit. Ramping down to start a new bench in the pit bottom, commonly called drop cutting, can be particularly hazardous during monsoon season. It may be necessary during intense precipitation events to abandon mining equipment in the drop cut and evacuate personnel to a safe location until the runoff subsides. Escape route planning is even more important in underground mines that may be subject to rapid inundation if old, flooded workings exist in the area.

Mine pumping systems cannot keep up with storm runoff in real time due to extremely high inflow rates that are usually at least one to two orders of magnitude higher than economically feasible pumping rates. In severe events, power may be disrupted, and sumps designed to capture the entire runoff volume of a particular design storm can be overwhelmed. Most dewatering systems cannot function during heavy rains and must be shut down to prevent damage from excessive sediment and debris. Floating barge pumps handle these events much better than fixed pump stations as long as their sumps can handle the runoff. After significant precipitation, operators may have to excavate mine sumps in order to recover the original volume lost due to sediments washed in from erosion of the roads and walls above. Pumps may be inundated or buried under sediment and may need to be replaced before pumping from the pit can resume.

Impoundments

An embankment that impounds water at a level above its previous channel elevation creates a potentially hazardous condition. Failure of the impoundment can release a tremendous amount of stored energy, which is then directed at anything below. Embankments, dams, and diversion dikes require higher than normal safety factors if that structure's failure could lead to loss of life or significant property damage. This higher safety factor is based on risk assessments of downstream structures and populations.

Periodic inspections, monitoring, and maintenance of operating impoundments must be carried out to ensure that the structure remains safe and its operation is within its designed limits. The size of the impoundment often determines the frequency and level of inspections required by the permitting agencies. Knowing the applicable regulations can reduce the costs of monitoring and inspections. For example, under the U.S. Mine Safety and Health Administration regulations for coal mining, embankments larger than 6 m (20 ft) high and ones with a capacity exceeding 24,670 m^3 (20 acre-ft) require inspections every 7 days. Impoundments only slightly smaller have less onerous inspection requirements (MSHA 1977).

Dam breaks are now rare because of regulatory vigilance imposed in the wake of significant disasters in the past. The potential damage from a failure can be very high due to the high volumes of water that can be released in a short time and the proximity of population and structures downstream. When impoundments fail, investigations into the causes usually find that the fault lies with improper design, construction, operation, inspection, and/or poor maintenance practices.

Slope Failures into Impoundments

Evaluation of the potential for slope failure should be part of the design criteria for all deep sumps adjacent to highwalls. Backfilling operations into flooded pits create the potential for dump failures to produce large damaging waves. The volume of sumps built under slopes with a history of stability problems should be as small as possible. Shallow sumps with large surface areas are preferred to smaller deep-water sumps.

Miners working near large in-pit ponds and sumps must be extra vigilant if any risk exists of slope failure into the water. Procedures for real-time monitoring of wall stability can mitigate the hazards to provide advance warning if raveling develops. If personnel are required to work near the shoreline of sumps, shallow areas such as old ramps are most hazardous because a wave approaching from deeper water can rise significantly.

An example of slide-induced waves occurred at the Valdez Creek mine in the Alaska Range (United States) during backfilling into a previously mined pit, when a dump failed and created a very large wave. The area, a long, sinuous valley 67 m (220 ft) deep by 457 m (1,500 ft) wide and 1.6 km (1 mi) long, was concurrently collecting tailings and dump material from the stripping operations. The pit had a standing water pond 53 m (175 ft) deep against the toe of the dump. Mine dumping was from the original pit rim elevation in a single lift about 24 m (75 ft) above the water surface.

Tension cracks developed behind the dump crest, and a large section 18 m (60 ft) thick slid into the water. The dump material displaced the water at the toe, resulting in a wave 11 to 12 m (35 to 40 ft) high that hit the wall across the pit. At the far end of the valley 1.6 km (1 mi) away, the wave was still more than 2 m (6 ft) high where it damaged a pump station. Fortunately, no one was hurt, but the potential for severe injury existed for anyone working on the ground near the surface of the pond. As a historic note, in 1958, the largest wave ever recorded, at 533 m (1,750 ft) high, was caused by a slope failure into deep standing water in a fjord in Lituya Bay, Alaska (BBC 2000).

Access Safety and Maintenance Planning

Heavy machinery, deep water, limited access, and people in close proximity create significant safety challenges that must be mitigated when considering floating barge applications. Long-term sumps must have access structures designed into the system. These can include floating gangways complete with handrails, cableways, and pipeline flotation systems. Anyone working around the barge must wear appropriate flotation protective devices. Whenever the barge and pump draft are too deep to allow the barge to be pulled close to shore, docks may need to be built. In sumps with shorter design lives,

the barge may be accessed by boat, providing that its weight limits and stability are not exceeded as a work platform. The following are important factors to be considered:

- Servicing plan for pump/control maintenance
- Weight and center of gravity for the barge, crane reach limitations, and control access requirements
- Occasional operator access for pump maintenance
- Support for pipelines and electric cables to account for water level changes

Moving equipment and machinery on barges creates the possibility of capsizing the barge. To avoid tipping, major shifts in the center of gravity during maintenance or operations need to be carefully planned. Special marine engineering expertise may be required to identify hydrodynamic, stability, and buoyancy issues before construction or major component replacements. Buoyancy control adjustments can be accomplished with air chambers that allow the deck level adjustments. Pipeline supports must include additional buoyancy where the discharge pipe connects to the barge to compensate for the weight where the pipe rises above the water.

Lifting Plans, Weight, and Crane Capacity

Site access must be designed to allow pumps, motors, and other equipment to be successfully lifted from the barge to transport equipment onshore. Pump and barge draft requirements may limit the ability to drag the barge close enough to the bank for service. The slope of the sump bank frequently causes the barge to run aground well before it is close enough to shore to enable convenient access. For small, steep-sided sumps, this is less of a problem due to the higher wall angles.

Large sumps or pit lakes require the development of standard operating procedures that include the weights of the barge, pumps, switchgear, and pipe fittings that will need to be placed both during construction and changed later during maintenance cycles. Some sumps have access limitations that preclude using shore-based lifting equipment. In these specialized cases it may be necessary to employ sky-crane heavy-lift helicopter services to fly the equipment to the operating location.

Generous factors of safety are needed to account for the barge weights that will increase when mud, water-logged floats, and scale accumulate over time. The suction force that has to be added to the weight to break free of the mud if the barge settles to the bottom should not be underestimated. The critical factors in these plans are the crane locations, reach limits, and estimates of the weight that must be lifted. Bank stability is also a concern in sumps with fluctuating water levels.

IN-PIT WATER CONTROL

The primary objective of the in-pit water control network is to expeditiously convey water through the workings to the temporary collection sumps for removal without adversely affecting the mining activity. Runoff sources include the point sources from groundwater infiltration, streams entering the pit, and rainfall collection on the surfaces inside the pit's perimeter.

Factors that water control system designers must consider as engineering trade-offs are the cost/performance designs involving the pump capacity, pipeline diameter, power requirements, system duty cycle, and number of lift stages. Large-capacity pump/pipeline systems with few stages are designed to run intermittently compared with smaller systems with larger sumps that run almost continuously. The large system has a higher factor of safety to handle unusually large inflows but comes at a higher capital cost than the smaller system, which accepts higher risks of being overwhelmed in infrequent precipitation events but handles normal dewatering needs better. Smaller, redundant parallel pumping systems can provide the surge capacity of a larger system while increasing system availability. Increasing sump capacity is one of the least expensive ways of reducing risk without sacrificing the efficiency of the pumping system.

Indirect Costs of Water in the Pit

Water in the rock fractures adversely affects the geotechnical stability of the pit walls. Mining pits at the steepest stable pit highwall angles is the most profitable because it lowers the stripping ratio. Proper water control increases the pit wall stability, thus maximizing the wall angles. Preventing water from infiltrating the fractures in the rock is critical to maintaining maximum stability. Horizontal drain holes drilled to intercept the fractures can successfully reduce the pore pressure.

Water in the mine working areas causes inefficiency in all materials handling operations. Water buildup at the ore-loading face degrades the crushing and conveying efficiencies. Dewatering efforts should move the water to local sumps that follow the loading equipment close enough to prevent digging in wet ore. Fines and water combine to form a highly abrasive grinding medium that destroys moving parts. Excessive moisture can cause backsliding on inclined belts where the feeders drop ore onto a moving belt. The steeper the belt, the worse the problem becomes. When conveying saturated material, water seeps out of the muck, running down the belt until it spills and builds up in the drives and under the rollers. Particularly in cohesive ores with many fines, wet material causes a buildup in chutes, resulting in plugging and costly downtime. Slabs from material buildup on the walls of the chutes can break off, fall onto the belt, and cause adverse wear or torn belts.

Haulage Road Runoff Control

Haulage roads act as arcs in the flow network in the pit because they are continuous and interconnected, and lead from the pit's upper reaches to the bottom working levels, making them ideal channels for water. Haul truck tires are becoming one of the major costs of mining; wet roads, tire slippage, and sharp rocks are primary causes of premature failure. Drivers must reduce their speeds to maintain control under wet road conditions. Maintaining good drainage control prevents visibility degradation and reduces spray from haulage roads. Muddy conditions create poor traction, resulting in increased stopping distances for mobile equipment.

Properly designed haul roads should be crowned or slanted to prevent ponding on the road surface. Crowned roads with at least 2% cross grades have fewer problems (than surfaced roads without drainage grades) with washboard ruts because the road surface retains most of its strength by having the cross grade eliminate standing water. Perimeter ditches concentrate the water and direct it down toward the sumps. On long ramps, the water volume can build up significantly before it reaches the bottom, as smaller tributary streams and rivulets join the main flow. It is desirable to divide the flow before the water accumulation gets deep enough to overflow the ditch

into the haulage lane. Where the ditch is located on the pit side of the road, creating holes in the berm allows part of the runoff to escape the ditch and sheet flow down the wall, reducing the size of the ditch necessary for the remaining portion of the flow. The water flows over the highwall, travels to the next lower area in the pit, and from there to an intermediate sump or another berm relief on its journey toward the bottom.

Switchbacks in the haul road present challenges to handling the water collected in the ditches along the highwall. Haul roads usually have superelevated curves raising the outer edge of the road perimeter. If the straight road section is crowned or sloped back toward the highwall, water runs in a V-ditch between the road and the wall. When approaching the switchback, the ditch elevation increases as the road nears the superelevated curve, and the water escapes the ditch, running across the active traffic lanes to continue downhill. The result is erosion of the travel surface, creation of washboard ruts, and tendency of the water to meander over the entire road surface. Several ways are suggested to mitigate these problems:

- A trench between the highwall and the superelevated curves should be created to allow water to continue past the curve and cascade over the wall. This has the effect of moving the water off the road but does so by directing it to the pit bottom. Most pit designs do not allow enough width to accomplish this because the trench takes up too much room.
- Diagonal swales, grooves, or water bars should be cut into the road surface prior to the curve to force the water to cross the road at a designated location. Aggregate can be selectively placed and compacted to prevent erosion in these designated water crossings. Approach transitions must be gradual enough to ensure no adverse effects on the speed of trucks traversing the swales.
- The most expensive way to move the water is to divert the ditch into a culvert running under the superelevated fill. The culvert can run along the same route that the trench would have taken but without the width "penalty." Alternatively, the culvert can cut the corner under the lanes, thereby diverting the water to the inside edge below the switchback, continuing the V-ditch down the road. The culvert entrance design usually involves a small sump but must prevent water from backing up and flooding over the road. There is a width penalty to construct this, and sufficient fill depth must be available under the road to prevent the weight of a loaded haul truck from crushing the culvert pipe.

Sumps

Sumps capture and temporarily hold runoff to allow for sediment control and for pumping the clarified water. Properly designed water collection networks concentrate the accumulated runoff as high in the pit as possible. Sumps can be located in a pit bottom, a wide spot in the bench along the wall, or at a switchback along a haul road. Obviously, pumping costs increase with the elevation difference between the sump location and the discharge elevation.

Sumps should be large enough to ensure that the active volume will keep the pump running for a reasonable period. Efficient sump design matches the pump capacity to the inflow rate for normal groundwater infiltration, reducing the need for periodically cycling the pump. If a sump has low or infrequent flows, it may not be in the right place. In such instances, it may be necessary to combine the inflow from other ditches to increase the utilization or consider eliminating the sump by rerouting flows elsewhere.

Removal of expected sediment and debris must be part of any sump design. The extent of the cleanout structure depends on the planned life of a sump, the velocity of the water in the approaches, and the presence of sediment traps in channels. Sediment traps have weirs and larger-flow cross sections in the channel, which slow the water flow, allowing the entrained solids to drop out of suspension. Sumps collecting runoff from high-intensity storms can rapidly fill with sediment if the velocities are erosive because of steep channel grades or easily eroded soils. It is easy and advisable to inspect the sumps periodically to ensure that necessary capacity exists.

Slope-stability effects must be part of the design and location of the sump. Geologic structures known to have stability issues should be avoided. Long-term sumps on the highwalls are a source of water infiltration that percolates into the rock, resulting in an increase in the pore pressure along the fractures. This increases the probability of slope-stability problems as mentioned in the safety section.

GROUNDWATER

The "Review of Surface Hydraulic Calculations" section discussed handling runoff water from meteorological sources that originated outside the mine. This section will give a brief outline of the process of identifying, quantifying, and controlling groundwater infiltration into the mine. Groundwater infiltration from aquifers, water-filled cavities, or other sources exists in nearly all mining operations. A more detailed treatment for these issues is found in Loofbourow 1973 and in Chapter 16.4. The process can be broken down into a series of estimates required to scope the problem depending on the status of the project.

Premining feasibility studies:

- The subsurface geologic structure and hydrology of the deposit are mapped and understood.
- Exploration wells are drilled to locate and quantify potential water resources for processing and potential mining hazards.
- The wells are tested to model flow rates, porosity, temperature, and chemistry, looking for possible adverse issues.
- The quality and chemical makeup of the water are assessed in order to design the treatment methodology.
- The interception strategy is chosen for handling the predicted water inflow rates:
  - Interception wells in the aquifer are pumped to dewater outside the mining area.
  - Inflow outside the mine perimeter is blocked by reducing porosity.
  - Inflow into the mine is accepted and handled along with other water sources.

Mine development phase:

- Predictions from the pre-mine feasibility studies are verified.
- The flow outside the mine perimeter is blocked through grouting, freezing, or other methods to reduce porosity of aquifers or fractures identified as high risks.
- Water is intercepted and drawn down using well fields, preventing quality degradation and avoiding treatment issues.

Mine operation:

- Groundwater infiltration is routed into sumps and handled through the in-pit pumping systems.
- Water quality issues are treated along with affected run-on water prior to discharge or reuse.
- Pump system redundancy and capacity are maintained to handle unexpected changes in flow rates. (See the Valdez Creek case study later in this chapter.)
- Interception well fields and blocking structures are maintained and expanded as mining progresses through the reserve.

The presence of groundwater in a mining claim has both beneficial and detrimental effects on the economic viability of the operation. All mining operations and processing plants need water to function, and it must be imported if insufficient supplies exist on the property. However, groundwater intercepted in the pit is usually in the wrong place with inconsistent flow rates and may have quality problems that must be handled before it can be used in the process. The main objective in mine dewatering is to economically capture the water, route it around the operations with minimal disruption, and store it for beneficial use; or, failing that, treat the water prior to discharge.

Wet Drilling and Blasting

Drilling under wet conditions is more expensive and less productive than if the operation is dry. One method of limiting the effects of water on the drill bench is by digging perimeter trenches into the bench. This is particularly useful in deposits that have considerable water present in the rock but relatively low permeability. Perimeter trenching would not be effective if the ore is highly porous because this allows water intercepted by the trench to recharge back into the drill pattern. Even if it is not possible to pump or drain the perimeter trench, it may lower the local water table, sufficiently drawing the water from the bench's upper couple of feet to improve the conditions for starting the hole (collaring). Fracturing from blasting the previous bench is usually sufficient to provide the additional permeability to draw down the local water table from the drill area to the perimeter trench. The permeability in the unfractured, surrounding rock is lower, preventing excessive flow from the walls back into the drilling area.

If the bench conditions are wet, redrilling is often required and may or may not be successful because the same water conditions that caused the initial problems still exist. It is not uncommon to drill a pattern while drop cutting only to find that 90% of the holes are collapsed or plugged by the time the blast is scheduled. After the good holes are loaded and shot to create additional porosity to pull down the water table, redrilling starts again. Under wet conditions like this, it can take a month to develop a 152-m- (500-ft-) long drop cut that might take only a week if the bench were dry.

Although factors such as energy density, detonation velocity, or energy distribution usually dominate the decision about the explosive type, the presence of water in the blastholes prevents using the least expensive blasting agents. Commonly used ammonium nitrate and fuel oil (ANFO) blasting agents will not detonate or will misfire if the powder gets wet. Water-resistant emulsions must be used if the water cannot be removed, but they are nearly twice as expensive to use as dry-hole, ANFO-based products of equivalent energy.

Blasthole dewatering with hydraulic-driven pumps can offset some of the emulsions costs if the water infiltration rates are slow enough. Pumping the holes and using impermeable polyethylene sleeves allows the placement of inexpensive water-sensitive ANFO in areas that otherwise would require emulsions. The cost of the sleeves adds to the blasting costs but significantly offsets the cost of using expensive emulsions.

Case Study: Valdez Creek Groundwater Control

Between 1993 and 1995, Cambior Alaska's Valdez Creek mine in the Alaska Range was reaching the end of the mine's reserves. This large placer gold mine operated year-round above timberline in a remote subarctic stream valley.

This mine operation's primary defining challenge was groundwater infiltration. Flow in Valdez Creek ran year-round across the entire length of the mining claim and continually recharged the subsurface aquifer. Groundwater flowed through the gravel deep in the deposit year-round, although the pit walls froze solid to a depth of 3 to 4.6 m (10 to 15 ft), blocking the flow during the winter. Blasting occasionally breached this frozen wall, resulting in sudden inflows of water and sediment into the pit. The inflow increased during the annual spring breakup with a significant spike in water volume as the walls thawed, draining the water held back during the winter.

Management scheduled closing of the mine by freeze-up in the fall of 1995 due to reserve exhaustion and marginal economics. The company was under pressure to limit spending, particularly on fixed infrastructure that it would have to abandon or scrap at closure. The remote location required more time and logistical effort than most mines to make major improvements. Adding incremental diesel-powered pumps was feasible with some notice (weeks), but adding high-density polyethylene (HDPE) pipeline capacity was difficult, expensive, and required a long lead time (months).

The mine experienced increasing water inflow rates. In 1993, the pit had an approximately 69-L/s (1,100-gpm) inflow with a pumping capacity of 82 L/s (1,300 gpm). By mid-1995, the inflows had increased to 454 L/s (7,200 gpm). Pipeline and pumping capacities were barely adequate for initial conditions. Surges overwhelmed the system, flooded the pit floors for weeks, and interrupted ore production until the inflow subsided. The initial 20-cm- (8-in.-) diameter pipeline was not adequate beyond about 63 L/s (1,000 gpm), requiring two pumps in series to keep up and maxing out pipeline capacity. Because the surge capacity of the system was limited, the mine added a second 20-cm (8-in.) pipeline.

Flows increased in 1993 to nearly 126 L/s (2,000 gpm) by freeze-up, interrupting production more frequently. Management agreed to fund an upgraded 30.5-cm- (12-in.-) diameter pipeline because it was essential to regain control over the water in the pit. This pipeline was in place before breakup in 1994.

The new pipeline worked well for about a year before it was deemed insufficient to keep up with increasing demands. The inflow tripled over this period to 391 L/s (6,200 gpm). The pumps that were on-site or readily available had to be pressed into service even though they were not well matched. The system started with one large diesel-powered pump and then added a large electric slurry pump and finally an additional large diesel pump in series. Two of the pumps operated at poor efficiencies, well beyond their curves due to flows above their designed flow rate. Only one pump was still operating on its curve. Adding the second and third pumps in series to the pipeline resulted in smaller incremental flow increases. Extremely high flow velocity in the pipe, approaching 6 m/s

(21 ft/s), resulted in high friction head losses (3 m [10 ft] of loss per 30.5 m [100 ft] of pipe), causing the hydraulic grade line (HGL) to exceed 10% slope.

Pumping in series adds the pressure of the last pump to the discharge pressure of the previous stages, so the sequential order of the pumps is important. Pumps, like pipelines, have maximum design internal case pressure ratings to avoid overpressuring and cracking the cases. Additionally, the discharge pressure exceeded the recommended working pressure for the HDPE pipeline, which was operating at a reduced safety factor. The operating point for this system was only possible because it pumped cold, near-freezing water, doubling allowable working pressure.

The last incremental 63 L/s (1,000 gpm) required additional pipeline capacity 6 months before closure. Because no additional funds were available, the pump crew salvaged the original 20-cm- (8-in.-) diameter pipeline from the 1993 pit and relocated it to the final pit. Two pumps in the series pushed the incremental flow up the wall, extending production nearly to the end of the mine's life.

Ultimately, the lack of a pump caused abandonment of a portion of the final ore face in the last 3 months. The pit filled with water because of the lack of a 189-L/s (3,000-gpm) capacity cross-pit pump and the availability of a short pipeline to move water from the shovel-loading area. This inundation resulted in the loss of approximately 3,000 to 4,000 oz of gold contained in ore that was uncovered and awaiting haulage to the wash plant.

The mine lost the water battle in late summer when an additional 442-L/s (7,000-gpm) inflow occurred through the pit wall, immediately forcing the operation to abandon production efforts and to commence reclamation. When considering the increasing magnitude of the water inflows over a short period, the mine did an outstanding job managing the pumping problems with limited resources available. It managed to mine more than 95% of the planned tonnage from the final pit even with seriously deteriorating pit water conditions.

SPECIALIZED PIT PUMPING PROBLEMS

Because mines are located where economically recoverable minerals are found, they are frequently in inhospitable locations with extreme climates. Mountainous areas have high relief, topographically controlled rainfall, and steep channels that cause erosive flow velocities. Jungles frequently add extremely high rainfall and seasonal monsoonal patterns. Arctic areas have extremely cold climates, ice, glaciation, and permafrost. Deserts defined by low annual precipitation can be very hot (Sahara), high altitude and cold (Antarctica, Atacama in Chile), or dry with most of the annual precipitation in a few intense storms (Arizona's Sonoran Desert in the United States). Each of these climates has unique challenges that affect the dewatering system designs.

Arctic Conditions and Permafrost

Permafrost is the condition in the soil below the seasonal active layer where the temperature never rises above the freezing point of water. This is not a problem by itself, but if the soil contains moisture and fine-grained soils, it tends to create thick lenses of clear ice. Permafrost that exists in foundation areas must either be kept frozen or removed. If the conditions that formed the permafrost such as ground cover and shading are changed, thawing is possible. Fine-grained soils must be removed from the foundations of the pump stations and replaced with coarser, nonwicking material.

Any source of heat, such as flowing water in pipelines, must be insulated to prevent thawing around the pipe. If the ground thaws, the bearing capacity may be lost, causing collapse of the overlying structures. Buried pipes in frozen ground can melt an annulus around the pipe that changes the permeability of the soil. In the case of an impoundment, sump, or artesian well, this can result in uncontrolled flow outside (and around) the pipeline.

Above about 60° latitude, permafrost may exist in all subsurface areas unless the area has been tested and been found to be clear. Even in areas generally free of widespread permafrost, there may be sporadic permafrost containing clear ice lenses in local areas. Cold climate areas without permafrost have seasonal freeze–thaw cycles that occur in the active layer. Depending on the severity of the climate, these can affect ground support stability as much as 3 m (10 ft) below the surface. Haulage traffic can push the frost line considerably deeper than usual, up to 5 m (18 ft) under the roadway, because of increased compaction, reduced moisture content, and the complete elimination of snow or ground cover.

In permafrost areas, most pipelines are installed aboveground in insulated utilidors for small-diameter installations. Pumps, valves, and other control stations should be enclosed, heated, and insulated to prevent freezing. Steel has a much higher heat-transfer rate and is usually much thinner than HDPE, so the pipes will begin to form ice at the fittings and flanges quicker. The pump crew should be equipped with a propane-fired burner to thaw valves that have frozen in place. Water standing in dead zones will freeze rapidly, preventing the actuation of the valves.

The time that it takes pipes to freeze depends on initial water temperature, pipe-insulating value, heat transfer rate, outside temperature, heat capacity (C_p) of the water, and the latent heat of fusion. Once the water reaches the freezing point, the time required for freezing the pipeline is shown by the first term in Equation 9.5-1. The second term in Equation 9.5-1 is the time required to lower the temperature of the water from an initial temperature T_i to the freezing point. If at some point the pump becomes available before the pipe freezes solid, water could flow and begin thawing the ice by applying the wasted energy from the system inefficiency. Large-diameter (40.6-cm [16-in.]) pipe can be laid on the surface in very cold areas if outages are kept within the heat capacity of the water. Even a small flow will prevent freezing in the pipeline; however, exposed valves and metal fittings may initiate freezing faster. Snow piled on a pipeline acts as additional insulation, further adding to the time available to get the pumping system back in service.

$$\text{time} = \left(\frac{R(\text{mass})H_f}{\text{area}\Delta T}\right) + \left(\frac{R(\text{mass})C_p(T_i - T_f)}{\text{area}(\text{average}\Delta T)}\right) \quad (9.5\text{-}1)$$

where

R = insulation value per inch, 0.17611 ((m^{2}K)/W), ((1 h-ft^2°F)/Btu)
mass = fluid mass per unit length of pipe
H_f = latent heat of fusion
area = outside area of pipe
ΔT = temperature differential between outside air and fluid

C_p = heat capacity of water 4.187 kJ/(kg-K), (1 Btu/°F-lbm [pounds-mass])
$(T_i - T_f)$ = difference between initial temperature and freezing point

Example 1. The following example of Equation 9.5-1 calculates the time to freeze the entire mass of 0°C (32°F) water setting in a pipeline at an ambient temperature of –40°C (–40°F).

Pipe characteristics (derivation not shown) for this example:

- SDR 15.5, 40.6-cm (16-in.) outside diameter pipe, wall thickness 26 mm (1.03 in.)
- Internal diameter, 353 mm (13.9 in.)
- Mass, 29.7 kg (65.7 lbm) water per foot length
- Latent, H_f for water, 334 kJ/kg (144 Btu/lbm)
- Outside area of pipe, 0.39 m^2 (4.18 ft^2/ft length)
- Insulation, R = 0.053 (m^2K)/W (0.3 (h-ft^2°F)/Btu)

Solution. Time to freeze solid

$$= \left(0.3 \frac{\text{h-ft}^2\,^\circ\text{F}}{\text{Btu}}\right)\left(\frac{1 \text{ linear ft}}{4.18 \text{ ft}^2}\right)\left(\frac{65.7 \text{ lbm water}}{1 \text{ linear ft}}\right)\left(\frac{144 \text{ Btu}}{\text{lbm water}}\right)\left(\frac{1}{72^\circ\text{F}}\right)$$
$$= 9.4 \text{ hours}$$

Pipes begin freezing where the sections are most exposed and where internal structures allow ice to anchor, such as elbows, fittings, and valves. As freezing progresses, these areas fill with ice and push water pockets ahead of them. This usually means that the last place liquid water pockets exist is inside walls where the insulation slows down the freezing process and the pipe usually ruptures in the area where it is most difficult to repair.

Desert Pumping

Deserts uniquely challenge pit-dewatering operations in several ways. Extreme swings in seasonal rainfall require system designers to consider both too much and too little water. Water is usually (but not always) in short supply, except when flooding occurs. The sources are frequently deep wells drilled into alluvial valley deposits and may require long overland pipelines to bring the water to the mine site. Because the supply is usually short, the systems design must avoid waste or contamination wherever possible. The temperature range of deserts has extremely wide variations, both seasonal and diurnal. Electrically operated pumps need additional care to ensure that insufficient cooling does not shorten component life. Evaporation and reusing water can cause undesirable concentration of contaminants.

It is necessary to derate the HDPE pipe working pressure for conditions above 22.8°C (73°F). The working pressure ratings must be reduced 50% at 60°C (140°F). Pipe lying in the sun while empty or with the water not flowing can quickly reach this temperature. Pressure surges on start-up can cause the hot pipeline to rupture.

Anchoring pipe along narrow rights-of-way prevents thermal expansion cycles from expanding and deforming into sinusoidal loops (snaking), causing pipe encroachment onto the road. Deserts frequently experience large temperature swings exceeding Δ22.2°C (Δ40°F) daily. Allowing the pipe to lie in the sun with no flow to stabilize the temperature simply exacerbates thermal expansion issues.

Alkalinity or other mineral content of the water in the sumps increases with evaporation. Recycling increases this effect because limited freshwater sources are available for dilution because of arid conditions. To prevent processing issues, the buildup of dissolved minerals must be managed. Bleeding a small split off the main process stream and replacing it with fresh makeup water may prevent undesirable concentration of minerals. If the quality reaches unacceptable levels, it may be necessary to segregate water for less critical uses—for example, haul road dust suppression.

Seasonal monsoon rains can create an excess water volume that overwhelms the sump capacity. Storage reservoirs should be as large as practicable to capture and hold the excess runoff for use during the dry season. If too much water becomes a problem, an alternative to pumping from the pit can be spraying water into the air to enhance evaporation. This sometimes becomes a dilution issue with large collection areas on top of the leach dumps in solvent extraction/electrowinning circuits.

OPEN CHANNEL DESIGN

Open channel designs must include adequate safety factors to carry the required peak flow without eroding the channel and while maintaining sufficient freeboard to prevent overtopping. Manning's equation is the primary tool available to design open channels flowing under gravity at atmospheric pressure. This method covers the design of ditches, rivers, and culverts flowing partly full. Equation 9.5-2 is the basic velocity form. When combined with the flow discharge Equation 9.5-3, it creates the flow quantity Q, Equation 9.5-4, that the channel will carry.

Manning's equation:

$$V = \left(\frac{k}{n}\right) R^{2/3} S^{1/2} \tag{9.5-2}$$

$$Q = AV \tag{9.5-3}$$

$$Q = A\left(\frac{k}{n}\right) R^{2/3} S^{1/2} \tag{9.5-4}$$

where

V = velocity, m/s (ft/s)
k = dimension conversion factor—1 in metric (1.486 English)
n = Manning's friction factor, a unitless characteristic of surface roughness of the channel material normally ranging from 0.010 to 0.050
R = hydraulic radius, defined as the (area/wetted perimeter), m (ft)
S = channel slope, m/m (ft/ft)
Q = flow rate, m^3/s (ft^3/s)
A = flow area, m^2 (ft^2)

From the basin runoff outline given, the normal calculation sequence is to determine the maximum flow rate Q, then design the channel to carry the flow. However, Equation 9.5-4 assumes that the flow depth and channel geometry are known.

Hydraulic radius, depth, and area are interrelated variables and are dependent on the shape of the channel. Depth is the usual unknown value and is a non-isolatable factor in both the hydraulic radius and the area calculations. Because the flow depth cannot be solved directly, the usual methods

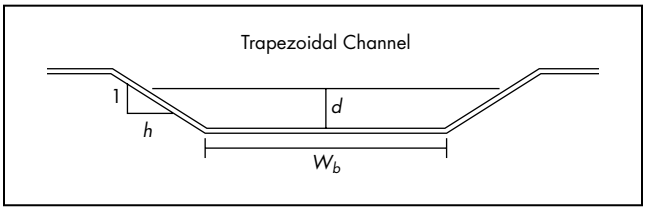


Figure 9.5-1 Trapezoidal cross section

of solution are to assume a channel width and depth, and then iteratively solve the equation.

$$A = W_b d + hd^2 \tag{9.5-5}$$

$$R = \frac{W_d + hd^2}{W_b + 2d(h+1)^{1/2}} \tag{9.5-6}$$

$$Q = (W_b d + hd^2)\frac{k}{n}\left[\frac{W_b d + hd^2}{W_b + 2d(h+1)^{1/2}}\right]^{2/3}(S)^{1/2} \tag{9.5-7}$$

where

W_b = bottom channel width
d = flow depth
h = side slope ratio in the design channel in the form h:1, horizontal to vertical
S = Manning's channel slope

Most channels built for mine drainage use trapezoidal channel cross sections. Figure 9.5-1 shows the general trapezoidal section parameters used to develop Equation 9.5-7. The hydraulic radius (Equation 9.5-6) and area (Equation 9.5-5) calculations are expressed in terms of the channel parameters: bottom width, depth, and side slopes.

If field solutions are necessary, spreadsheet programs with a solver function running on laptop computers or programmable calculators can be set to do these computations. The normal solution starts with Q as a known value from the basin design, and field constraints on W_b, d, S, n, and h.

Example 2. Parameters for a typical trapezoidal channel construction design might be the following:

- Required flow rate Q, 14.15 m^3/s (500 ft^3/s)
- Minimum bottom channel width (equipment width limited), 4.53 m (15 ft)
- Three to one (3:1) side slopes ($h = 3$)
- Channel slope, 0.005 (ft/ft) (0.5%)
- Minimum freeboard above the water surface, 0.61 m (2 ft)
- Maximum flow velocity to minimize erosion of 1.52 m/s (5 ft/s)
- Coarse gravel bottom Manning's roughness coefficient n, 0.045

Solution. From the preceding example, an iterated solution is

W_b = 5.3 m (18.8 ft)
d = 1.05 m (3.46 ft)
V = 1.51 m/s (4.96 ft/s)
Channel depth with freeboard = 1.66 m (5.46 ft)

The use of Manning's equation in the field by technicians or less experienced designers may lead to errors. Nomographs, graphical representations of the underlying relationships, which were widely used prior to the availability of modern computers, are safer. Nomographs for Manning's equation solutions are complex, making assumptions of the specific channels shapes, and are sometimes only valid for particular flow conditions. When using them, it is important to understand the underlying assumptions.

PUMP SELECTION

Most surface mine–dewatering systems use centrifugal pumps because of their wide range of operating characteristics. The required head (H) and flow rate (Q) will determine the pump design scheme. Operating pumps in parallel allows additional flow capacity beyond what is practicable for a single pump and provides system redundancy. Higher head requirements than single-stage pumps develop can be handled by utilizing multistage pumps driven from a single shaft or by using several single-stage pumps in series. Extremely high head applications such as grout pumping or underground dewatering may require positive displacement pumps: piston, diaphragm, variable cavity, or gear pumps. The flow pulsates because of the periodic nature of the pump mechanisms and requires dampening to prevent shock damage to system components (see "Issue: Water Hammer" section later in this chapter). Systems employing positive displacement pumps require more care matching the system's power, flow, and pressure capacity to the required operating point and are less flexible in changing conditions than centrifugal pumps.

Centrifugal Pumps

Commonly available centrifugal pumps have head ranges up to about 107 m (350 ft) in single-stage configurations with flow rates up to 681 m^3/h (3,000 gpm). Pumps are available that have capabilities exceeding these but are less common. Centrifugal pumps can be grouped into the following categories based on their configurations:

- End suction pumps use a single open suction eye from the side of the impellor with the drive motor on the other side. These are the most commonly used single-stage pumps but have high net positive suction head required (NPSHR).
- Double-suction pumps use a manifold to route suction from both sides of the impeller. The drive motor axis passes through both sides of the pump case and can be driven from either side. These are moderate NPSHR pumps.
- Vertical turbines use multiple stages driven from a common shaft and generate very high discharge heads. These usually mount through a horizontal deck with the motor above and require that the suction bell be submerged under the barge or in a deep well. These pumps require the least NPSHR because the suction and the first pump stages are submerged.

Pump Curves

The primary source of pump data is the manufacturer's data sheets. Manufacturers will usually supply a line card showing ranges of each family of pumps they offer, covering a wide range of head/flow requirements. This is the tool most designers use for preliminary screening of the specifications to find a pump with the appropriate capacity for the application. After a general pump family is selected, it is time to get the detailed data specifications from the manufacturer.

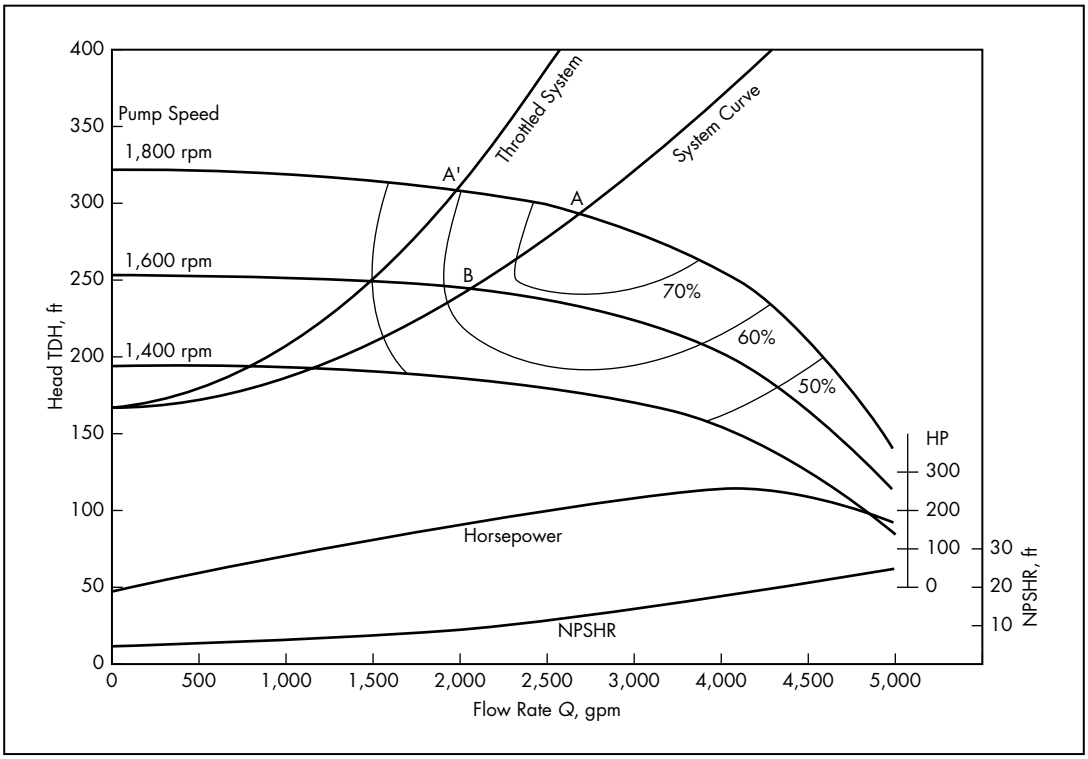


Figure 9.5-2 Pump system head curve

The following are some of the most important specifications that the system designer must determine from the manufacturer's pump operating characteristics curves:

- Total dynamic head (TDH)/flow (Q) range requirements
- Revolutions per minute (rpm)—motor rotational speed
- Driver type, coupling, and efficiency
- Power
- NPSHR

System Operating Point Example

Figure 9.5-2 is a typical system head curve (system curve) showing the pressure–flow characteristic curve for a pumping system. This system consists of a single-stage pump with field-variable rpm. The pipeline system has both a hydrostatic head (elevation lift) of 52 m (170 ft) and friction losses that increase as the flow rate increases. A family of curves that are rpm dependent represents the pump's TDH versus flow rate Q output.

The system curve lines represent the head and flow relationship of the pipeline, valves, and fittings. The system curve intersects the left side of the chart where Q is 0 at 52 m (170 ft) total static head. The pump must develop at least this amount of TDH to overcome the hydrostatic head and begin pumping water. As the system curve moves right with increasing Q, the curve starts to rise as friction losses increase. The difference between the total static head and the system curve is the energy either lost to friction in the system or used accelerating the water column in the pipe (velocity head).

The pump system operating point is the intersection of the system curve and the pump TDH curve: Point A, where the 1,800-rpm pump curve and the system curve intersect at 88 m (290 ft) TDH pumping 170 L/s (2,700 gpm).

This pump works at its best efficiency point (BEP) at a Q of around 177 L/s (2,800 gpm). Ideally, the designer would select a pump for the system that matches the operating point as close to the BEP as possible. Because of internal pump dynamics, the efficiency is not only a function of Q, but there is usually a shift in efficiencies when changing the rpm. Pump efficiency is overlaid on the pump curve and resembles a bull's-eye with the BEP in the center.

Each pump is designed to work within a particular flow range. Centrifugal pumps have a limited ability to continue to pump water past designed upper limits for the pump family. Flow in a pump has a maximum point beyond which TDH drops off rapidly. Above 659 m^3/h (2,900 gpm), the efficiency drops, and, by 1,022 m^3/h (4,500 gpm), the pump's TDH begins to fall rapidly. At zero flow, the TDH rises to the maximum head that the pump can develop (deadhead pressure)—98 m (320 ft). However, the pump is doing no work so the efficiency is zero; all of the input energy is wasted as heat.

Throttling

There are several ways to change the system flow to accommodate variable demand. In a fixed-speed system, such as one driven directly by an electric motor, the most common way to reduce the flow is by installing a control valve in the discharge pipe. The system head curve is modified by partially closing a valve, shifting the operating point from A to A' as the valve creates additional frictional losses, and reducing the flow from 613 to 454 m^3/h (2,700 to 2,000 gpm). The pump is still working hard, but the energy lost across the valve causes the hydraulic efficiency to drop.

In a variable-speed system, similar changes in the operating point require reducing the pump's speed from 1,800 to 1,600 rpm, which would shift the pump curve. This would move the operating point from point A to point B at TDH of 75 m (245 ft)

Table 9.5-1 Pump affinity laws

Characteristics	Speed Ratio	Impeller Diameter Ratio
Flow	$Q_1/Q_2 = (N_1/N_2)$	$Q_1/Q_2 = (D_1/D_2)$
Head	$H_1/H_2 = (N_1/N_2)^2$	$H_1/H_2 = (D_1/D_2)^2$
Power	$P_1/P_2 = (N_1/N_2)^3$	$P_1/P_2 = (D_1/D_2)^3$

and Q slightly less than 613 m^3/h (2,000 gpm). The efficiency would be better than if the system was throttled using the control valve method because less energy is wasted.

When considering a pump curve, the designer must be aware of several limitations that may preclude the system from achieving the desired operating point. Most pump curves show two additional curves that increase with higher Q: the hydraulic power required and the NPSHR (a characteristic of the pump from the manufacturer). Care must be taken when locating the pump to ensure that suction head requirements are met: The net positive suction head available (NPSHA) is above the NPSHR under all operating conditions. The pump driver must be able to supply sufficient power to overcome the pump inefficiency and the electrical losses of the motor and to supply the hydraulic power requirements.

Affinity Laws

Manufacturers design pumps for particular operating ranges, but within these ranges the curves can be shifted. Speed changes are practical methods for modifying the operating characteristics of pumps when the applications change in the field. The effects of changing the speed ratio between the original operating point and the new pump speed (N_1/N_2) are shown in Table 9.5-1. As can be seen, the flow ratio changes are linear with the speed ratio; head is a squared function, and power requirement increases as the cube. Speed ratio changes are most useful when coupled to a variable-speed motor. Keep in mind the old rule of thumb for rotating machinery that if you double the speed, the maintenance costs go up eightfold.

Impeller sizes/trimming ratios (D_1/D_2) have the same effects as changing speed ratios but are more difficult to change after manufacturing. This capability allows adjusting fixed-speed pumps to match the required operating point of a long-term installation without sacrificing the efficiency that a throttling valve would cause. Limited adjustments by reducing the impeller diameters can be done at the mine site machine shops, but this requires that the pumps be disassembled. Increasing impellor diameters is not possible without obtaining a new impeller from the manufacturer. Efficiencies suffer if the impeller diameter is significantly reduced from the original diameter (Casada 2006).

PIPELINES

Pipelines are the hydraulic analog of wires in an electrical system. Like the electrical system, the pipelines have carrying capacity (flow rate Q), voltage capacity (pressure ratings), and resistance to flow (H_f frictional losses).

Bernoulli's equation (Equation 9.5-8) shows the total mechanical energy in a fluid flowing in a pipeline as the HGL. The difference between the HGL and the Z elevation in the profile represents the gauge pressure inside the pipeline. If energy were conserved along the flow in a pipe, the HGL would be constant, and energy could be exchanged among the four components. The first three terms can be reversibly transformed one to another; however, the H_f is usually lost as heat from the system. All of the energy terms in Bernoulli's equation have units of length, allowing modeling the system HGL as a cross-sectional elevation grade line.

$$\text{HGL} = \frac{V^2}{2g} + \frac{P}{\rho g} + Z + H_f \tag{9.5-8}$$

where

V = fluid velocity
g = gravitational acceleration
P = internal gauge pressure
ρ = fluid density
Z = elevation or hydrostatic head
H_f = frictional head losses

The contributions of the individual energy terms (all in length units) are the following:

- HGL
- Velocity head ($V^2/2g$)
- Pressure head ($P/\rho g$)
- Elevation head (Z)
- Friction head (H_f) losses

Practical Graphical Solution for Rapidly Designing Pump Locations

One of the most straightforward ways to lay out a pipeline and locate booster pumps along the route is a graphical cross section. This process works because all factors in the dimensional analysis of the Bernoulli equation reduce to length. The sum of these factors represents the total energy in the system or the HGL.

The topography is drawn in cross section along the actual proposed route (Figure 9.5-3), scaled to compare the pit surface elevations against the HGL in the pipe system. When the HGL intersects the topography, the system has used all of the available energy, and as a result water will no longer flow. By using cross sections in this manner, it is readily apparent where to install booster stations along the pipeline route.

System designs use the high-pressure pipe in the bottom and lower-pressure-rated pipe (less expensive) as the elevation increases. The required pressure ratings drop as the pipe elevation increases (internal pressure decreases) and the H_f slope in the HGL flattens. The first pipe connected to the discharge of Pump A has the highest HGL, is at the lowest elevation, has the highest internal gauge pressure, and thus has the thickest walls required to resist bursting. The highest-pressure-rated pipes have the smallest inside diameters and the greatest frictional head losses (H_f).

Beginning at the sump, the HGL is at the water surface elevation. Entrance and suction side losses reduce the HGL further in the pump inlet piping (NPSHA). The pump adds energy to the system, raising the HGL by the TDH of Pump A. Continued plotting of the HGL in each pipe segment shows H_f as percentage slope related to the pipe friction losses until the HGL reaches the discharge location. Sumps, pumps, and discharge locations are included along the pipeline route.

The point where the topography intersects the HGL is the maximum elevation that the available TDH energy from Pump A can push the water. The booster Pump B is placed at an accessible location along the pipeline below the elevation of the HGL/topography intersection. The additional TDH from Pump B raises the HGL sufficiently to arrive at the tank with no further booster pumps required. The residual pressure

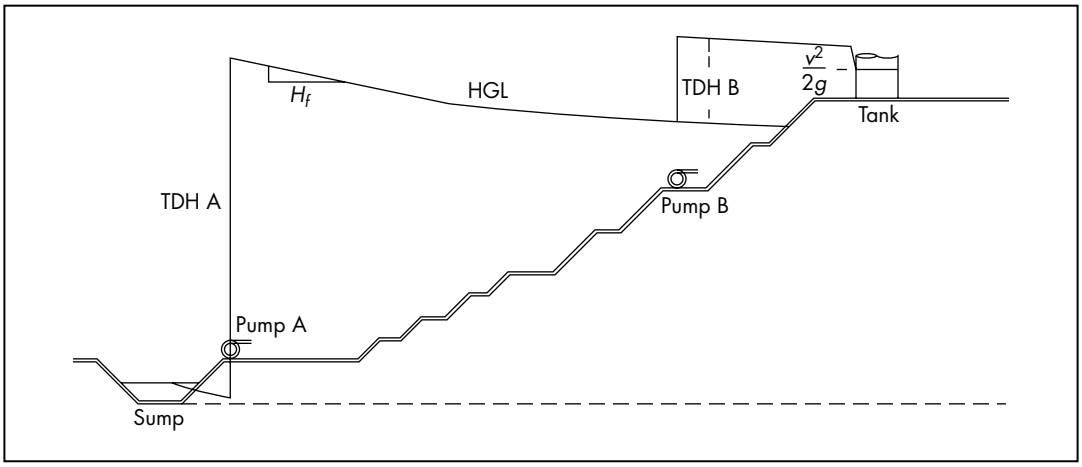


Figure 9.5-3 Pump station design cross section

in the line above the minimum necessary to reach the tank will accelerate the flow from the pipe end. At the discharge, the HGL consists entirely of velocity head as the pressure head drops to atmospheric and delta elevation term falls to zero.

After the pump locations have been determined, it is possible to increase the safety factor by systematically placing the booster pumps at lower elevations than the profile indicates. Note that Pump B is placed on a bench well below the HGL/topographic intersection from Pump A. This accounts for future wear in the pumps and scaling or system degradation, as well as for approximations in estimates of the minor head losses. This method assumes that the entrance, shock, and fitting losses are minor compared with the elevation change in the system. If this is not the case, more precision should be applied to the NPSHA analysis of the pump inlet structures.

Pipeline Material Selection

Selecting the appropriate pipeline materials for pumping projects involves balancing many competing characteristics. The factors that determine which material is best for any given project include the following.

Pipe material characteristics:

- Corrosion resistance
- Electrical conductivity
- Chemical compatibility
- Thermal expansion
- Cost
- Availability, lead time, logistics

Pipe materials:

- Metal
  - Carbon steel
  - Stainless steels
  - Copper
  - Ductile iron bell and spigot
  - Corrugated steel pipe (CSP)
- Plastic pipes
  - Polyvinyl chloride
  - HDPE
- Masonry
  - Vitrified clay
  - Concrete precast
- Composite
  - HDPE-lined carbon steel
  - Grouted steel

HDPE pipe:

- Ease of construction, joining
- Labor skills available
- Weight
- Ductility
- Strength
- Pressure ratings
- Temperature

High-Density Polyethylene

The invention of HDPE pipe is among the most influential products to appear in mining since 1960. Its unmatched combination of low weight, strength, flexibility, ease of installation, and corrosion resistance revolutionized pumping processes in mining. HDPE pipe has many advantages over steel pipe for use in the harsh mine environment. It is much more flexible than steel, allowing installation in areas where it would not be practicable to install steel pipe.

HDPE pipe is resilient and often more suitable in the mine pit environment than are metal or concrete alternatives. Occasionally, heavy equipment can be driven directly over HDPE pipes, although it is necessary to protect the pipe from damage with a thin layer of fine material or conveyor belting. Before driving the equipment across the pipe, the pipeline should be depressurized to limit the transfer of hydraulic pressure to other locations in the system. An HDPE pipe will crush flat and then begin to rebound immediately after the load is removed. Attempting this with any rigid pipe results in permanent deformation or crushing.

HDPE is resistant to most chemicals and is not subject to corrosion—crucial characteristics when dealing with acidic or caustic leach solutions. The main alternative material to HDPE for these applications is expensive stainless steels. Cathodic protection systems are not required for buried installations because the HDPE pipelines are nonconductive and not subject to galvanic corrosion. These characteristics give HDPE pipelines a design life of 50 years' service when buried.

The low density of HDPE makes it an ideal material to connect floating barges to the shore, and its flexibility allows

it to follow the barge as the water level changes. HDPE is slightly less dense than water; trapped air gives it enough buoyancy to float the pipe out to barge pumps.

HDPE pipe is the easiest material to work with in the field because of its low melting point. Joining HDPE requires specialized fusing machines that combine the three operations of the fusing cycle: (1) milling the pipe ends, (2) melting the surfaces, and (3) fusing the ends together. Typically, pipe larger than 6-in. diameter arrives at the job site as a truckload of 12.2- to 15.2-m- (40- to 50-ft-) long segments. The assembly process is considerably faster than welding steel pipes. Semiskilled workers with field-fusing machines can make three to four joints per hour. Because HDPE pipe is flexible and light, the pipe bed preparation requires little work, and small forklifts or backhoes are the largest pieces of equipment necessary to handle the pipe segments.

When it becomes necessary to make repairs, a chain saw is usually sufficient to cut out the damaged section and replace it with a new one, using the fusing machine. When repairs are necessary in areas that are too tight to excavate fully, it is possible to slip an electrofusion coupler over the ends of the broken pipe. This sleeve system allows repairs in inaccessible locations and contains its own internal heating elements. A generator is connected to an electrofusion controller, which melts the collar onto the pipes, sealing the ends.

Several characteristics limit the applicability of HDPE pipe in some situations. HDPE is limited to moderate pressures, with the upper end of its range limited to design pressures less than 2,068 kPa (300 psi) for SDR 6 pipe. The larger the SDR, the lower the pressure rating due to thinner pipe walls. Common working pressure ratings range from 207 kPa (30 psi) for SDR 40 pipe to 1,758 kPa (255 psi) for SDR 7.3 pipe. Steel pipe, on the other hand, has thinner walls and is useful for extremely high pressures. Additionally, the maximum depth of burial for HDPE is shallower than for that of steel.

The pressure ratings of HDPE pipe are temperature dependent, because HDPE is a thermoplastic with a low melting point. The designed pressure rating temperature is for a pipe at 23°C (73°F) and has a service factor of 2. The working pressure needs to be derated 50% for pipes installed at 60°C (140°F) and increased to 200% working pressure rating for installation at 4°C (40°F). As the temperature of the installation diverges from 23°C (73°F), the allowable pressure ratings continue to move toward both extremes. Ultimately, the pipe becomes brittle at very cold temperatures, crystallizing at about –101°C (–150°F), and softens and loses its strength as it approaches its melting point.

Other Pipe Materials

Other materials have characteristics that make them the choice for some circumstances. Composite grouted steel pipelines combine concrete, grout, or other masonry material as a liner inside steel pipe. These are useful in slurry pipelines, tailings, or culverts carrying large amount of debris where the main issue is erosion resistance. Galvanized CSP is one of the major culvert, manhole, and sewer construction materials. Zinc galvanizing offers corrosion protection, allowing long-term installations at competitive costs.

Combining welded steel pipelines with a liner of thin HDPE has the advantages of corrosion resistance of the HDPE and the pressure ratings from the strength of the steel pipe. Pipe joints usually use bolted, raised-face steel flanges, and inner smooth HDPE flanges to prevent the corrosive fluids from contacting the steel. These double-layer pipelines allow for integral leak detection by installing sample ports in the outside steel line.

Anchors, Point Loading, and Thrust Blocks

Anchors need to support the weight of the pipe, water, and live loads developed by the flow in the pipelines. These axial loads can be quite large where pipes hang from supports high on the side of pit walls. The pipe walls must also be strong enough to hold these loads between anchor points; the wall's strength determines the maximum distance between anchors. Pipes need support at pump stations and wherever they are attached to pumps, valves, or other fittings. It is poor engineering practice to allow loads from the pipe's weight to transfer through the flange bolts. Case deformation due to external loads may reduce internal clearances, causing unacceptable wear or binding in valves and pump impellers.

Thrust blocks are designed to help pipes resist the forces developed when water flow changes direction or velocity; for example, the thrust from an uncontrolled fire hose nozzle. Calculations for the thrust block must counter the changes in the momentum of the water stream. The impulse/momentum changes result from the forces applied to the water and resisted by the thrust block.

Confined Stress Due to Temperature Changes

Pipelines subject to changes in temperature will expand or contract proportional to the length of the pipe and the temperature change. HDPE pipe is particularly problematic in this regard, because it has a coefficient of expansion six times higher than steel, and its flexibility makes it less able to resist bending stresses. Warming pipelines can deflect into sinusoidal loops (snaking), and this may result in them moving off the designed right-of-way. Placing piles of earth on the pipe at regular intervals, called point loading, serves to control the lateral movement. If the temperature is high during installation, shrinkage occurs as the pipe cools. This can result in pulling the pipe completely out of fixed pump stations, tanks, or concrete anchors.

As an example of these issues using PlexCalc II, a 41-cm (16-in.) diameter SDR 15.5 pipe can be installed without taking into account thermal expansion. HDPE has a thermal expansion coefficient of 9×10^{-5} in./in./°F (Performance Pipe 2003). (Note: For a 22°C [40°F] temperature swing, a 914-m [3,000-ft] pipe changes length by 3.3 m [10.8 ft]).

Hot installation at 27°C (80°F) cooling to 4°C (40°F) is the short-term temperature change induced by pumping cold water into an empty pipe and will result in a tensile force of 107 kN (24,000 lb-force) trying to pull out the fittings. The change associated with cold installation at 4°C (40°F) warming to 27°C (80°F) will result in a compressive force of 160 kN (36,000 lb-force) if the pipe is constrained and must be guided at 49-m (160-ft) intervals to prevent buckling.

FIELD FLOW MEASUREMENTS

The two necessary measurements that indicate the current operating conditions of a pump are the pressure (head) and the flow rate (Q). The intersection of these two parameters defines the operating point on the pump curve. Many types of pressure gauges are available to obtain the head, but measuring the flow is more difficult. For permanent installations, flow measurements use preinstalled devices in the pipeline at strategic points to monitor flow. These include mechanical rotating

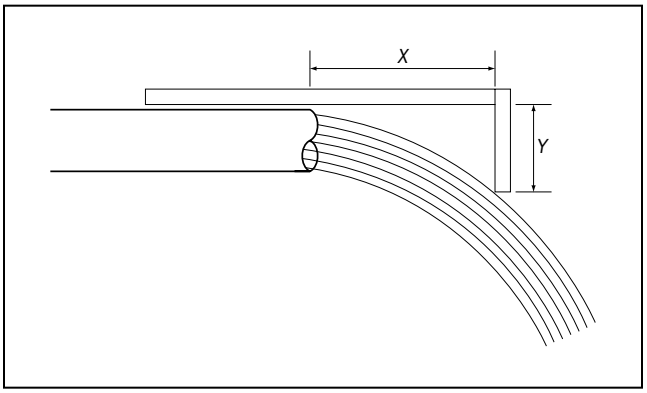


Figure 9.5-4 Pipe discharge drop method

vanes, hot wire transducers, differential pressure manometers with orifice plates, ultrasonic flowmeters, and numerous others. The common factor is that these devices (except the ultrasonic flowmeter) must be inside the flow to function. The ultrasonic flowmeters are portable and use transducers strapped to the outside of the pipe. All of these instruments measure the flow velocity (V): calibrating them for measuring Q requires knowledge of the inside diameter of the pipe to calculate the flow area (A). $Q = VA$ is used to calculate the flow rate.

Pipe Discharge Drop Method

The simplest flow measurement method is the pipe discharge drop, which can be used any place where the pipe protrudes from the bank and the flow discharge drops freely by gravity (Figure 9.5-4). This measurement method uses projectile kinematics to calculate the velocity of the discharge stream. The only instrument necessary is a calibrated stick for measuring the horizontal distance (X) from the mouth of the pipe to the point where the stream drops a fixed distance (Y). A folding carpenter's ruler is a perfect tool for this measurement. The ruler is moved until the distance from the top of pipe to water flow at the pipe end is Y: X, Y, and the flow depth are recorded as a fraction of the pipe diameter. Y is corrected for the wall thickness and compensated for the discharge running at less than full pipe flow to obtain the actual stream drop.

If the operators take measurements at the same locations frequently, it is a simple procedure to precalculate the tables matching the distance dropped because the pipe diameter, wall thickness, and table of partial area flow corrections are constant. Even better is setting up a spreadsheet with the parameters and calculations prebuilt to calculate flows from any pipe discharge, using Equation 9.5-9 to calculate the discharge velocity of the flow. If the flow is not a full pipe discharge, it is necessary to estimate the area of pipe flow. Precalculating a table with this conversion only requires measuring the fraction of the pipe diameter that contains running water.

$$V_o = \frac{X}{\sqrt{\frac{2Y}{g}}} \qquad (9.5\text{-}9)$$

where

V_o = discharge velocity
X = horizontal distance to measurement point
Y = vertical drop of flow stream adjusted for wall thickness and partial flow
g = gravitational acceleration

Open channel flows use similar schemes to determine the velocity of the water flow in natural channels. Various methods include measuring the time that wood chips take to traverse a known distance, allowing the observer to calculate the velocity of the surface flows. Measuring the channel cross section is necessary to estimate the flow area, allowing the flow rate to be calculated. This method neglects the vertical gradient in the velocity profile. Bubble streaming lines can give a visual representation of the velocity variations across the channel.

Weirs and flumes are calibrated structures installed into open channels that are useful for creating fixed gauging stations to measure flow. These use characteristics such as specifically constructed long-throated flumes or broad-crested weirs that correlate the depth of flow to the quantity traveling through the section. Long-throated flumes are replacing the previously used Parshall flumes because their streamlines are more linear and do not have to be calibrated in the laboratory (Swanson and Baldwin 1965). Many computer programs are available for designing flumes for specific flow.

MODELING, INSTRUMENTATION, CONTROLS, AND POWER

Bearing in mind the limitations of the accuracy of the hydraulic predictions previously discussed, an understanding of these highly complex systems requires simulation and modeling. It can be difficult to calculate the complete state of even the simplified system used in the "Simplified Control System Example" section. Given the construction costs tied up in systems that perform poorly, simulation is essential to design an effective pit-dewatering system.

Consider the piping network flow diagram as an interconnected series of arcs and nodes analogous to electrical networks. Many of the computer-based, pipe network modeling systems force designers to break the system down into basic components and assign characteristics to each before running iterative passes until the system operating point is found. Dewatering systems do not operate as disconnected discrete components (Wood and Lingireddy 2010).

Hydraulic networks simulations are based on the same mathematical relationships that govern electrical networks, the hydraulic equivalent of Kirchhoff's laws:

- The sum of flows into and leaving any node equals zero.
- The sum of pressure drops around any closed loop equals zero.
- Hydraulic simulations use specialized boundary nodes that have fixed pressures or act as flow sources or sinks.

Control System Components

Like the electrical network, hydraulic networks contain other components besides pipes (wires) and nodes (connections) that transform the pressures and flows. Components are specialized arcs that connect one or more nodes and transform the pressure, accumulate flow, or perform other specific functions. Table 9.5-2 shows some of the common modeling components in the hydraulic system, their electrical analogs, the functions they perform, and the number of nodes they connect to other components.

Because water is noncompressible under ordinary conditions, communication and feedback are necessary between different parts of the network. The problem is that unless

water accumulation components, such as tanks or reservoirs, exist somewhere in the system, the volume flowing into the system must be the same as that which leaves. To ensure this happens without incident, the components at the end must be able to communicate and control the components at the beginning of the pipeline. In order for this to happen, the system must contain the following components:

- Hydraulic sensors detect fluid levels, pressures, and the presence or absence of flow.
- Sensors detect the mode/state of pumps, valves, control positions, and system demands.
- Actuators, motor controls, and relays control the active components in the system.
- Communication links collect the inputs, transmit them to the command and control component, and send command signals to the actuators.
- The controller (the brains of the system) executes command and control logic, evaluates inputs, and sends appropriate commands to the actuators over established communications links.

Simplified Control System Example

Complex control relationships can be derived from the simplified network shown in Figure 9.5-3, which contains a pump drawing water from a sump and pumping it uphill through a second pump to a tank. For the following description, it can be assumed that pipeline from Pump A discharges directly into a tank feeding Pump Station B.

Hydraulic sensors are necessary to measure the state of the hydraulic network and should be able to detect if the sump contains enough water to prime the pump, fill the pipe, and deliver sufficient water to the tank. The system should have a sensor to detect if the tank needs water or if it is too full and, therefore, about to overflow. When pumping, flow sensors would tell the controller if flow exists downstream from the pumps in order to avoid damage.

In the simplified system, only one actuator is necessary to turn on each pump. The mode sensor to supply the feedback that this has occurred could come from an additional set of contacts to supply the binary state of the starter to the controller as an input, "on" or "off."

Communication links carry the inputs from the sensors and outputs from the controller. The controller is physically connected to these links and to the distant sensors and actuators through wires running to the controller. Other options require that radio links, wireless computer networks, or other telemetry hardware carry the signals. These have the advantage of not requiring physical cables, although they increase the complexity and potentially the cost of the communications network.

The command and control system (controller) is the intelligence behind the pumping system. The controller takes the inputs from the hydraulic and state sensors and the feedback from the actuators and applies the preprogrammed control scheme to create the appropriate action commands to the actuators. A controller can be as simple as a manual switch operated by an employee; when activated, it sends power to the motor and an indicator light, showing the pump status. Remote operations controllers (ROCs) allow the system controller to be located almost anywhere as long as the radio communication link is reliable. ROCs sometimes supply both the programmable controller and the radio hardware in the same box, simplifying system design.

Table 9.5-2 Hydraulic modeling components and electrical analogs

Hydraulic Component	Electrical Analog	Hydraulic Function	Number of Nodes
Pump	Battery	Adds pressure (head) between nodes	2
Valve	Switch	Opens or closes arc to flow	2
Flow control valve—partly opened	Rheostat or potentiometer	Modulates or changes flow by changing arc resistance	2
Tank or reservoir	Capacitor	Accumulates and releases water volume based on conditions (may also have fixed HGL elevation if used as boundary node)	1 or 2
Check valve	Diode	Allows flow in only one direction	2

Most of the modern controllers use some type of a programmable computer-based system. Programmable logic controller (PLC) systems are the most commonly used type of industrial controller. Programmers program a PLC using a simplified language called "ladder logic" that uses simple constructs to simulate signals that enable contacts, timers, and sequences to create the appropriate output for each system operation. Systems that are more complex run on larger computers incorporating powerful programming languages that can create internal simulation models capable of controlling entire water-handling networks with many components and interrelationships.

When the pump network has more than one stage in a series, it becomes necessary to interlock the pump stations and holding tanks. Intermediate tanks receiving water from below and supplying water to a local pump pushing up to the next stage must balance the flows to avoid overflowing the system. Normally, tanks in this sort of configuration use three level sensors to feed information to the controller about the tank level state:

1. High tank level sensor shuts down the lower stage pump—the tank is full.
2. Low tank level sensor shuts down the next stage pump—the tank is empty.
3. Mid-tank level sensor starts the next stage pump in the series when the tank has enough water to pump without quick cycling. This allows both stages to pump simultaneously for part of the cycle, improving the system's efficiency.

PUMP STATION DESIGNS

Design of pump stations must balance a number of competing constraints in the pumping system, especially the availability of electric power. The design life of the pump station at a particular location will determine the effort that goes into its layout and construction. The access quality and space availability for the station may also limit the selection.

Deciding on the type of motor to drive the pump is among the primary design constraints. Selecting the appropriate drive system depends on several competing characteristics. Under normal circumstances, diesel is generally the more expensive power source than electricity. However, at remote sites where

utility company electric power is not available, it may be less expensive to run the pumps from diesel engines than to use the same fuel to generate electricity for electric-driven pumps.

Diesel-Driven Pumps

Diesel-driven pumps are convenient to use for self-contained trailer-mounted pumps that operators can deploy quickly. This makes diesel pumps the choice for pumping water out of the shovel-loading faces or other in-pit situations where the sump must move every time the pit advances. The most effective pit-dewatering crews have a number of these pumps covering a wide performance range. The variable-speed operation inherent in the design of the diesel power plants allows considerable flexibility in the varying performance of the head and flow rates at the operating point of the attached pump.

The main disadvantage of diesel engines is that they require operator access for daily fueling and engine servicing. This requires maintaining an access road capable of carrying fuel trucks to the pump station as well as a turnaround area. The energy conversion efficiency of the best diesel engines falls far short of comparable electric drives. High-efficiency, low-speed diesel engines have thermal conversion efficiencies of more than 50% compared with electric motor efficiencies in the mid- to upper-90% range.

When using diesel engines, automation is not as viable as when using electric motors. Some diesel engines can operate remotely with sensors controlling the starting and stopping, but these features do not account for the fine-tuning that diesel starting and shutdown procedures usually require. The engine speed must usually be set manually, although pneumatic dashpot controls may allow automatic operation requiring a complex feedback system if the operating point is not constant.

Electrical Motor-Driven Pumps

One of the most critical parameters that determines pump performance is the rotational speed at which the pump operates. As discussed in the "Affinity Laws" section, changes in the pump's discharge head are proportional to the speed ratio squared. Maintenance costs for rotating machinery are proportional to the speed ratio cubed; doubling the speed raises the maintenance costs eightfold. The lowest rpm that meets the required operating point is the most reliable. Low-speed pumps usually cost more and are proportionally heavier than the higher-speed pumps to achieve the same performance, so the initial costs versus operating costs need to account for these factors.

Motors for common pumping applications are usually three-phase induction motors with power ratings ranging from 4 to 224 kW (5 to 300 hp). The rotational direction of the motors must match the designed rotation because centrifugal pumps can only run in one direction. When using a three-phase powered system, reversing any two phases to the motor changes the motor's rotation direction.

The number of poles, p, in the stator windings and power line frequency, f, determine the motor's synchronous speed:

$$\text{synchronous speed} = 120\, f/p \qquad (9.5\text{-}10)$$

Induction motors produce torque at any speed less than synchronous, so the motor's armature runs slower. Synchronous motor rpm ratings appear odd because of the armature lag; for example, a four-pole motor whose synchronous speed is 1,800 rpm may run at 1,750 rpm.

In synchronous motors, the more the motor speed lags behind the synchronous rate, the more current it consumes and the more torque it produces. This effect is highest during start-up when the motor's current can be up to 10 times larger than the normal running current. Starting requires special attention in order to prevent excessive current causing undesirable voltage fluctuations in the incoming power grid. Soft-start motor controllers are necessary on the larger motors, particularly if the installation is at the end of a long power line. Soft starters can also control the system's ramp-up speed, thereby effectively acting to control the dynamic shock effects to the system (see the "Issue: Water Hammer" section).

Electric-driven pumps are more suited for fixed installations that move infrequently, such as booster stations or floating barges. The selection of the electric pump operating point requires more engineering than does the variable-speed diesel-driven pump. The highest-efficiency installations couple the pump directly to its drive motor so the electric motor's speed must match the pump at the desired operating point. Field speed adjustments are limited to changing drive pulley ratios if the pump is belt driven. Otherwise, adjustments to the pump operating point involve throttling valves in the discharge pipe network with a commensurate loss of efficiency.

Ratios not obtainable by changing the number of poles are possible by using belt- or geared-drive systems. If variable speeds are necessary, variable-frequency drives (VFDs) can be economic for smaller pumps up to about 56 kW (75 hp). VFDs change motor drive speeds by electronically converting the line power frequency supplied to the motor, thus making a wide range of output speeds possible. However, this comes at a significant cost, both in acquisition and efficiency.

Automatic operation is easy to design into these pumps because numerous types of level switches or instruments for switching electric signals are available. Start-up against an empty pipeline may present problems keeping the pump "on the curve" because there is little backpressure until the water rises enough to prevent cavitation (a damaging condition caused when internal pressure falls below the vaporization point of the fluid (see the "Issue: Cavitation" section). VFD motor controllers are available that can control the start-up speed profile, gradually increasing the pump output pressure as the pipeline fills.

Pump stations with electric drives do not have to maintain regular access for daily operations, which makes electric pumps suitable for installation in difficult terrain and confined areas. The major constraint is that power lines or flexible trailing cables have to be run to the pump's location from the motor control panel. The starter panels and other controls must be located where operators have access, but these can be some distance from the pump.

Booster Stations

Pumps capable of handling large variations in the suction lifts are well suited for pumping from sumps but often do not have high head discharge pressures. This is where the pump system frequently combines booster pumps in a series with sump pumps, fully utilizing the best characteristics of both. Fixed booster pumps maintain constant discharge heads because the hydrostatic lift to the discharge location does not change. Fixed boosters stations may be in-line or tank fed with level controls that cycle the pumps to match inflows. Tank-fed booster pumps are normally designed to maintain the water

level above the suction eye of the pump, ensuring a positive suction head.

Floating Barges

The decision to use a floating barge system is usually a good fit when pumping the water from a long-term sump with fluctuating levels due to irregular or seasonal inflows. Water elevation changes result in changing pump static lift conditions that can affect the pump capacity. This can happen in very deep ponds where the surface elevation drops significantly during the life of the system to the point where the static head exceeds the dynamic head capacity of the pump. Booster stations may be required but can be added later if the barge pump's initial capacity is adequate.

When properly designed, barge pump installations do not suffer from suction head issues (NPSHA) because the pumps maintain a fixed distance between the pump inlet and the water surface. This is a dramatic improvement over trailer-mounted pumps that have to be moved down the ramp each time the water surface drops by between 3 and 4.6 m (10 and 15 ft). Increasing sump levels float the barge, whereas the trailer pump would risk inundation. Barge pumps are much less labor intensive to operate, but their installation and maintenance costs are likely to be higher.

Minimum water depth must be maintained in the sump to account for the pump and barge draft requirements. To avoid grounding the pump or sucking debris and rocks into the system, minimum clearance must be maintained between the suction inlet and the sump bottom. The suction line must be located deep enough to avoid the formation of vortices that suck air into the inlet screen or pipe. This is detrimental to performance and can cause the pump to lose prime. In horizontal centrifugal pumps, the flexible suction lines must be weighted to prevent the screen from floating too close to the surface, but they must also be supported to keep it out of the bottom sediments.

Bottom clearance issues can be a problem with vertical turbine-style pumps, because the intake depth is fixed by the design distance between the pump intake and the barge deck. This style of pump is more likely to ingest debris if the sump's water depth drops to the minimum necessary to float the barge. With proper barge design, the pump intake can be kept at a safe distance from the bottom by keeping the pump intake pipe shorter than the bottom supports of the barge, thereby leaving sufficient clearance.

The use of a vortex prevention collar around the suction intake can allow the suction to be closer to the surface than the flow would otherwise allow. This collar is essentially a fin that prevents the streamlines that are approaching the suction line from rotating. The depth required above the suction hose to prevent the formations of vortices is dependent on the pumping rate (McNally n.d.).

TROUBLESHOOTING PUMPING SYSTEMS

Because pumps are simple machines that are good at pushing water, a system that suddenly fails is most likely because of operator error. Upon pump failure, mechanics need to (1) check for improperly set control valves or broken valve shafts, (2) ensure that the source has water and the motor has power before blaming the pump, and (3) determine whether the discharge pressures are as expected and that the discharge pipelines have had time to fill with water. Depending on the diameter and length of the pipe, it takes a surprisingly long time for an empty pipe to fill enough to begin discharging (Equation 9.5-11). A 1.6-km (1-mi) pipe with a cross-sectional area of 0.1 m^2 (1 ft^2) will take almost 40 minutes to fill from a pump pushing 227 m^3/h (1,000 gpm).

$$\text{time to fill pipe, min} = \left(\frac{\text{area}(\text{length})k}{Q}\right) \qquad (9.5\text{-}11)$$

where

area = pipeline cross-sectional area, m^2 (ft^2)
length = pipeline length, m (ft)
k = unit conversion factor, 60 min/h (7.48 gal/ft^3)
Q = flow rate, m^3/h (gpm)

If the causes are not immediately apparent after checking the obvious, the second thing to do is look for changes in the system that may or may not have been intended. Changes in piping during normal maintenance are a common reason that pumping networks that have been running successfully for some time suddenly appear to fail. These include changes in length, pipe diameter, new valves, or other physical changes that could result in the pump capacity being no longer sufficient. Maintenance on the pipe may have created an air-locked condition in the pipeline or the pipe may be empty.

Issue: Air Locking Pipelines

Air in the pipeline can consume all available pump energy trying to push the air against the hydrostatic lift. An example is the force needed to push a balloon down to the bottom of a swimming pool. The air's buoyancy resists the flow in the pipe when flowing water tries to push the entrained air bubbles down from a local topographic high to a lower elevation. Under less extreme air conditions, the fluid in the pipe can flow under a relatively stable air pocket. Even in the flat-lying areas of the pipe, air bubbles restrict the cross-sectional area, increasing the flow velocity and friction losses in the pipe. Eliminating air from all overland pipelines prevents loss of pumping efficiency. Turbulence increases the friction losses, and gas compressibility robs the system of energy, converting it to heat.

Air release/vacuum breaker combination valves should be used at all significant topographic high points on the pipeline profile. These valves are usually large ball-check valves with a floating ball and must be installed vertically. Air release function allows the air bubbles out of the pipe where they naturally accumulate. The ball falls from the seat while air is present, allowing the trapped air to escape from the pipeline. When the water rises into the valve body, the ball moves up into the seat to seal the valve. If the flow conditions drop the internal gauge pressure into vacuum, the ball check falls back, allowing air into the pipe and preventing suction collapse. This is necessary to prevent vacuum conditions where the flow is intermittent or where the pipeline profile significantly declines in elevation, particularly when using HDPE or other flexible pipelines.

Pumping over a ridge frequently results in the hydraulic flow regime changing from pressurized flow (Hazen–Williams equation) to open channel flow in a partly filled pipe (Manning's equation). The pipeline flows with the pipe full until reaching the maximum elevation of the section. Depending on the flow rate, either the pipeline will develop a steady open channel flow or will begin cycling the trapped air bubble, as previously described. An air release/vacuum breaker valve will allow air into the pipe, thus smoothing the

flow, and should create a stable open channel flow. The pipeline may convert back to full pipe pressure flow near the bottom of the hill. Computations of the parameters in this sort of mixed-flow regime are complex. It is usually sufficient to be aware of the potential issues when evaluating a pipeline in rolling topography.

Issue: Cavitation

Cavitation occurs wherever the local flow conditions reduce the absolute pressure below the fluid's vapor pressure. Rapidly moving water creates low-pressure areas, particularly when passing through structures that result in flow area reduction, resulting in formation of bubbles. Velocity increases at flow-control valves, restrictions in the cross-sectional area, or around the ends of rapidly moving components as they reduce the pressure, causing the fluid to begin vaporizing. The bubbles themselves are not damaging, but, after the restriction, the fluid slows and the pressure rises above the vapor pressure. This causes the bubbles to collapse suddenly, resulting in localized shock waves and pitting.

Cavitation occurs most frequently at the suction eye of the pump impeller, behind the impeller tip, and on the downstream side of valves where this collapse zone exists. It occurs on pumps when NPSHA is less than NPSHR due to the design of the inlet piping failing to maintain a proper positive suction head. Fittings and elbows too close to the pump inlet can cause the restrictions and turbulence, leading to excessive friction losses. Cavitating pumps frequently sound as if they are pumping gravel even though no sediment is present in the fluid.

Issue: Priming

Priming the pump is the operation that removes the air from the pump body and fills it with fluid. Centrifugal pumps cannot handle much air in the pump chamber; below 0.5% is usually acceptable, but above 6% by volume, the pump's capacity is seriously degraded. Removing air from the pump chamber is necessary to allow positive suction lift to develop. Self-priming pumps usually have a vacuum pump to remove air from the pump case, allowing external atmospheric pressure to push the fluid into the intake. Foot valves (check valves at the end of the suction pipe) hold the prime by keeping water from draining back into the sump when the pump is turned off.

Issue: Insufficient Suction Head

Many pumping problems occur when the pump is located too far above the water surface of the feed source. The elevation difference between the suction eye of the pump and the water surface varies as the sump level changes. The pressure of the fluid must be higher than the vapor pressure at the suction eye of the pump at the operating temperature, or cavitation can result.

Pumps are efficient at pushing water along pipes, which is the main reason why they are useful in mining. There is a common misconception that pumps suck water out of sumps with equal efficiency, but this is not the case. Pumps must maintain a positive pressure at the inlet of the pump in order for water to flow (Equation 9.5-13). A pump's NPSHA is limited by available pressure P_o, the vapor pressure of the fluid P_v, the static head assisting (resisting), and minor losses h_L (Figure 9.5-5). The gauge pressure inside a pressurized tank, P_o would be equal to atmospheric pressure, P_a on an open sump. When calculating minor losses with Equation 9.5-12,

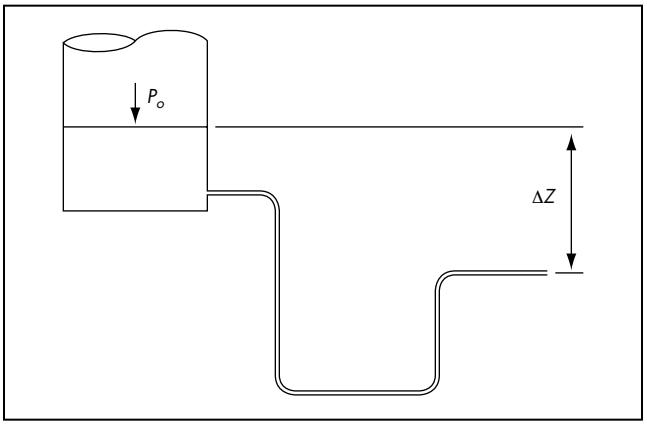


Figure 9.5-5 Simplified net positive suction head

the loss coefficient (k) must be used; this is a characteristic of the particular transition geometry.

$$h_L = k\left(\frac{V^2}{2g}\right) \tag{9.5-12}$$

The pump's inlet is frequently located above the water surface, requiring additional lift. The maximum NPSHA from an open sump is limited to the elevation-adjusted atmospheric pressure (P_a) less the vapor pressure of the fluid less the elevation head below the suction eye. The pump manufacturer determines the NPSHR while testing the pump during design and manufacturing. The operating point of the pump affects the NPSHR because minor losses in the entrance piping increase with higher velocity heads. The NPSHA must be higher than the NPSHR throughout the entire operating point range.

$$\text{NPSH} = \frac{P_o + P_v}{pg} + \Delta Z - h_L \tag{9.5-13}$$

where

NPSH = net positive suction head
P_o = pressure on the water surface (P_a if tank is not pressurized)
P_v = vapor pressure of fluid
pg = specific weight and gravitational acceleration
ΔZ = elevation head (lift if negative)
h_L = head losses due to transitions and fittings

Methods to solve problems on the suction side of the pump include the following:

- Lowering the pump closer to the water surface of the sump, reducing the static lift.
- Ensuring that the pipe is clear and the suction screen is not blocked.
- Reducing the length of the suction hose and increasing the diameter of the suction piping—both of which reduce friction losses in the intake piping.
- Ensuring that the suction side fittings are not leaking; air ingestion can break the priming.
- Ensuring that the pump is not cavitating; cavitation can have the same effects as air ingestion, limiting the pump efficiency.
- Ensuring that the foot valve is holding to maintain prime.

- Reducing the number of fittings and keeping them as far from the pump as practicable to limit the friction losses in the suction line.
- Ensuring that the pump is operating near the BEP.
- Splitting the load by using several smaller pumps in parallel instead of a single large pump.

Issue: Minimum Flow

Centrifugal pumps must have sufficient fluid flow to avoid overheating. Pumps convert the mechanical energy from the drive shaft into various forms, including

- Hydrostatic pressure (head),
- Fluid kinetic energy (velocity head), and
- Heat (friction losses).

Fluid flowing through the pump carries the heat from the mechanical and hydraulic losses in the system. If the system is prevented from flowing by closed valves, insufficient suction head, or insufficient discharge head pressure to overcome the system's static head, most of these losses are converted into heat, eventually causing the water inside the pump casing to boil. Explosions are possible if the pressure due to vapor expansion exceeds the maximum casing pressure rating.

Significant damage will occur to the pump impeller from cavitation at temperatures well below the normal boiling point. As the temperature increases, the vapor pressure of the fluid increases, resulting in more cavitation. This has the same effect of operating the pump with insufficient NPSH by increasing the necessary static pressure required to prevent vaporization.

Issue: Sediment, Debris, and Screens

The primary source of debris entering the pumping system is through inadequately maintained intake piping. The first line of defense is to install an appropriate screen over the suction pipe to prevent large solids from entering the pump suction. The suction hose should be kept above the sump bottom to prevent the flow from sucking debris onto the screen.

During construction, pipe segments must be inspected and cleaned before they are joined. When dragging pipes into place, the ends act as a scoop and can pick up a surprising amount of rocks and debris, which require flushing before closing the section. Large rocks will migrate along the flow and jam against valves, preventing them from properly actuating. Pipe that has been stored outside should be inspected prior to construction for animals that have moved in. In-line screens, traps, or cleanout fittings should be installed periodically in the pipe and located to protect valves, pumps, regulators, or other sensitive fittings. Drains and cleanout fittings should be placed where the discharge during cleaning does not cause damage.

When the pipeline runs through hilly country, it is important to ensure that the pipeline has sufficient drain valves installed at logical topographic low points to allow flushing of the system. This allows removal of sediment buildup in pipe systems before blockages occur and is particularly necessary if the pump system handles turbid runoff or slurry, because the solids will pass through the normal intake screen but may settle in the pipe.

Issue: Pressure Bleed Through Check Valves

In a high-lift multistage system, pressure can bleed through check valves, causing excessive hydrostatic pressure in the lower stages and resulting in pipeline ruptures when the pump system starts. Systems can experience problems with pipe segments splitting in the bottom stage of a series of pumps because of starting pumps against excessive hydrostatic head. This can be a problem when using pumps in a series without an atmospheric pressure break between stages.

Check valves are designed to allow flow in only one direction. When holding pressure against the upstream face, the valve closes, preventing backflow and holding the differential pressure across the valve. Pitted faces or improperly sealed valves allow small water seeps that can equalize pressure on both sides. Because water is incompressible, this can happen with little seepage volume over time in systems that have almost no trapped air.

The lower stages of the pipeline are the most likely to experience pipe rupture because the pressure cascades through multiple stages above. The dynamic head from starting the lower pump is added to an already overpressurized pipe segment. Indications of this problem show up in HDPE pipe sections that have been repaired numerous times. The solution is to design multistage pumping systems with transfer tanks between the stages at atmospheric pressure. More area is necessary for the pump stations, and the system will be less efficient, but the starting sequence and dynamic interactions are easier to control. An alternative would be to install accumulator tanks to allow expansion against a compressible air bladder.

Issue: Water Hammer

Water hammer is a dynamic shock/pressure wave that travels through the fluid in a pipeline. The primary cause is a shock wave created when the flow's velocity or direction changes rapidly. Water hammer shock waves can travel at the speed of sound in the fluid, not at the fluid flow velocity. The speed of sound in water can exceed 1,219 m/s (4,000 ft/s).

Equation 9.5-14 is a simplified peak pressure formula empirically relating pipeline pressure spikes and the transitions caused by rapidly closing the valves (Plast-O-Matic Valves 2002).

$$\Delta P = \left[\frac{(0.07)VL}{t}\right] + P \tag{9.5-14}$$

where

ΔP = pressure spike from water hammer
V = flow velocity in ft/s
L = pipeline length in feet
t = time in seconds
P = pipeline pressure in psi

For example, consider the magnitude of the pressure spike from an ordinary action: An operator closes a butterfly valve in a 4.8-km- (3-mi-) long pipeline flowing at 1.5 m/s (5 ft/s), thereby stopping the flow. The operator takes 5 seconds to close the valve on a pipe that is carrying 690 kPa (100 psi). What is the pressure spike from this action?

$$\Delta P = \frac{\left[(0.07)5\frac{\text{ft}}{\text{s}}\left(5{,}280\frac{\text{ft}}{\text{mi}}\right)3\text{ mi}\right]}{5\text{ seconds}} + 100\text{ psi}$$
$$= 8{,}270\text{ kPa}\,(1{,}200\text{ psi}))$$

Added to the hydrostatic pressure, spikes of this magnitude can overstress pipes, valves, and pumps, causing significant structural failures. Water hammer can cause oscillations in

both positive and negative (limited by vapor pressure and kinetics) excursions in a system, resulting in damage.

Valves should operate by controlling opening and closing times using automated valve actuators at programmable rates. Pump stations should have flywheels on the pumps to slow the velocity change rate, *dV/dt* during start-up, and unplanned power interruptions. Electrical motor soft starters should be used to ramp pump motor speeds when starting and stopping pumps. In some installations, the time constants for flow changes may be as long as an hour to limit system pressure damage. Water has inertia and carries considerable momentum when moving. The longer the pipeline and the higher the flow rate, the larger the mass that is moving. Check valves that slap closed rapidly when flow direction reverses are a common cause of water hammer.

Long pipelines with high flow rates need to have shock-absorbing structures designed into the system. Hydropneumatic dampeners, accumulator tanks, air traps, or air bladder tanks absorb and dampen shocks by using compressible gas and must be sufficiently large to absorb the shock energy expected from sudden transients in the pipe. Air release/vacuum breaker valves can be useful for allowing air into the pipe to cushion the shock of a water slug moving away from a closing valve. The air acts as a shock absorber, thereby limiting damage.

COSTS

Due to the cost of designing and building pit-dewatering systems, efforts to keep water outside the pit are usually a good investment. Intercepting and diverting water at the pit rim is less expensive than pumping from the bottom of the pit. The direct pumping costs include power, infrastructure, and required space designated for the pump stations and pipeline rights-of-way. Pit water may also require treatment before discharging off-site. Indirect costs can be significantly higher because of the effects that water has on productivity, reducing highwall stability, and safety hazards that the operation must mitigate.

Direct Cost Example for Pit Pumping

If a mine allows water to flow into the pit because of the lack of an effective diversion, at an average inflow rate of 6.3 L/s (100 gpm), this accumulates 200,000 m^3 (52.6 million gal) annually. Water has to be pumped vertically 305 m (1,000 ft) from the pit floor. If the pumping rate is 63 L/s (1,000 gpm), the pumps would need to run 10% of the time at 60% hydraulic efficiency of the pump and a power cost of \$0.05/kW·h. The annual costs in terms of power and the infrastructure to set up the dewatering system can be estimated using the applicable power equations in Equations 9.5-15 through 9.5-17:

$$\text{hydraulic hp} = [(H)(Q)(SG)]/3{,}960 \quad (9.5\text{-}15)$$

$$\text{hydraulic kW} = [9.81(H)Q(SG)]/1{,}000 \quad (9.5\text{-}16)$$

$$\text{brake hp} = \frac{\text{hydraulic hp}}{\text{pump efficiency}} \quad (9.5\text{-}17)$$

where

H = total dynamic head, m (ft)
Q = flow rate, L/s (gpm)
SG = fluid specific gravity

Results:

- Motor power required = 314 kW (421 hp) (without pipe friction losses)
- Annual power cost = 8,760 h/yr (10% duty cycle and 314 kW at 0.05 \$/kW·h)
- Annual power cost = \$13,770/yr in electricity

Infrastructure cost estimate:

- \$250,000 for 3,048 m (10,000 ft) of 8-in. pipe at \$25/ft installed (along 10% road)
- \$75,000 for three pumps with head capacities of 107 m (350 ft) TDH per stage, including motors at \$25,000 each

$$\frac{\$200{,}000 \text{ power to pit floor}}{\$525{,}000 \text{ total infrastructure}}$$

The annual cost for power in this example just to pump the water was \$13,770, or \$2,180 per L/s (\$138/gpm) allowed into the pit. These costs are avoidable (or at least reduced) by intercepting the water higher up the wall or diverting it completely.

Direct Cost Example of Hauling Wet Ore

Increasing moisture content increases the density of the material, adding to the costs of haulage and conveying. If the dewatering crew can lower the moisture content of the ore from 4% to 2% by improving water control at the shovel-loading face, a mine processing 89,290 t (100,000 st) per day saves 1,786 t (2,000 st) of excess water previously hauled to the crusher daily. This is the equivalent of pumping 21 L/s (333 gpm) of water out of the pit using the most inefficient of pumping systems—the haul truck. If haulage costs \$0.75/t, on an annualized basis, this reduces annual operating costs by \$548,000.

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CHAPTER 9.6

Dewatering Underground Operations

Peter L. McCarthy and Malcolm G. Dorricott

The inflow of water can affect the cost and progress of underground mining, limiting the mining methods used and presenting hazards. Mine dewatering has effects both on the groundwater table, which is usually a shared resource, and on the environment receiving any discharge. Although many mines have been worked successfully beneath perched underground reservoirs, streams, lakes, rivers, and seas, there are also many examples of catastrophic inrushes. Experts such as hydrogeologists, hydrologists, and pumping engineers can contribute to the success of mining projects and should be consulted when appropriate.

The general objective in the control of water in underground mining is to permit safe, efficient work with acceptable consequences. The most common method of control is pumping. Where an impervious cover above a mineral deposit can be maintained, it may not be necessary to draw down the groundwater aquifer, provided accesses, including shafts, can be developed by grouting, ground freezing, or casing. Many salt, potash, and coal mines are of this type. Pregrouting with cement slurries can reduce the hazards of serious water inrushes and minimize delays during shaft boring or sinking, whereas cover grouting using chemical grouts can reduce inflows to underground development.

As part of any mine plan, an overall water balance is essential to ensure that enough water is available and that surplus water is managed and disposed of in an acceptable manner.

SOURCES OF WATER

The main sources of water encountered in underground mining are

- Inflows of surface water through natural (geological) conduits, mine openings, and boreholes;
- Inflows of groundwater through the natural permeability of the rock and from secondary (fracture) permeability;
- Mine service water, such as water used for drilling and dust control sprays; and
- Drainage from hydraulic backfill.

Although prediction of the volume of mine service water is straightforward, the magnitude of natural inflows sometimes can be difficult or impossible to predict until mining takes place. Nevertheless, experienced engineers, geologists, and hydrologists are able to make satisfactory predictions in many cases, so that appropriate control measures can be put in place. Sudden inrushes of water or mud can arise from

- Heavy rainfall events via mine openings or surface subsidence and caving zones;
- Unplanned connections to the sea, lakes, rivers, swamps, clay deposits, wet cover, tailings dams, and water dams;
- Connections to water pockets such as caverns in carbonate rocks and fault conduits;
- Magmatic water, if mining near a volcanic caldera; and
- Connections to adjacent flooded mines.

IMPACT OF WATER ON OPERATIONS

The presence of water in underground mines impacts virtually all operations and activities. Its main beneficial effect is in the control of dust in drilling and during rock transportation. It also serves as a transport medium for hydraulically placed backfills and sometimes, as at the McArthur River uranium mine in Saskatchewan, Canada, as a medium for transporting ore.

The negative effects of water include the following:

- Increased heat transfer from the rock in hot mines
- Freezing in very cold mines
- Increased humidity, causing less comfortable working conditions and reduced labor productivity
- Reduced ground stability due to water pressure
- Corrosion of plant, equipment, and ground support elements
- Requirement for more expensive explosives
- Spillage from trucks and conveyors
- Water and mud rushes from orepasses and other sources
- Erosion of roadways, particularly ramps
- The need for special precautions for mining in the presence of water under pressure
- Increased power consumption for pumping and ventilation

All of these impacts add to the cost of mining, and in extreme cases may be the difference between profit and loss.

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PREDICTING WATER INFLOWS

Prediction of water inflows requires an understanding of the climatic and hydrogeological factors plus any existing human-caused conditions, such as abandoned boreholes and nearby flooded mines, and the effects of planned mining work. Many sites present no significant water problem, but this should never be assumed. It is difficult to predict groundwater flows into underground mines, as they are dominated by structural controls that usually cannot be identified from exploration drilling. Hydrogeologists have access to sophisticated numerical modeling tools, but experience shows that these models often fail to accurately predict inflows to an underground mine because of the lack of data, particularly in the early stages of a project.

To maximize the reliability of predictions, data should be collected at all stages of a project or mine development. Each mine should have an efficient data management system, which covers the collection, storage, and distribution of the data to the relevant people. The data to be collected at each stage of a project or mine should be determined by a group of competent persons prior to work commencing. For example, the following is a checklist for the collection of information required for the prefeasibility stage of a mine project:

- Topographical information
- Rainfall quantity and intensity data
- River flow and flood level history
- Tidal levels and flows
- Earthquake frequency and strengths
- Climatic conditions including temperature, humidity, and evaporation rates
- Surface water runoff characteristics
- Groundwater characteristics including levels, salinity, temperature, recharge, discharge, and general flow directions
- Hydrogeological conditions, including geological units and their permeabilities and storage, and general geological features (faults)
- Information on soil and rock types, their characteristics, depth of weathering, and so forth

Precipitation and Runoff

To predict the likely inflow from a single precipitation event is quite straightforward and can be calculated from the following equation (rational method):

$$V = C \times i \times t \times K$$

where

V = volume of potential inflow
C = catchment area
i = intensity of precipitation
t = duration of event
K = runoff coefficient

It may be useful to consider the rate ($Q = C \times i \times A$) as i is a time-dependent variable (and A = area). The catchment area can be estimated from topographic data and an understanding of the regional hydrogeological conditions. It may be feasible to reduce the catchment area by diversion of water courses or containment.

Intensity is derived from historical records. However, a choice must be made as to which event should be the basis for design. Typically, a 1-in-100-year event is considered reasonable, assuming such records exist for the location, but the expected life of the operation needs to be taken into consideration. A short-duration mine is less likely to receive such an event than one that has operated for decades, but many mines have much longer lives than initially expected. The decision should be based on a risk assessment and a realistic consideration of the consequences of the scenario selected.

The runoff coefficient must be selected carefully, because high-intensity events are more likely to occur when the ground is already wet from normal precipitation, thus initial abstraction is small.

There is also a trade-off between storage capacity and pumping rate. Once the inflow volume of the worst-case scenario has been estimated, the option exists to divert the inflow to available storage and pump it out slowly after the event or to cope by matching the pumping rate to the expected inflow rate. The best option is usually somewhere between these two extremes. Of course, storage capacity must be empty if it is to be relied on in an emergency.

An example of such a calculation is provided for the Mount Lyell mine in Tasmania, Australia (Atkinson 1982). At the time, the open-pit mine, located in an area of high rainfall, was transitioning to an underground mine, with breakthroughs from the underground mine to the base of the pit. The emergency storage capacity required for a 1-in-50-year storm event was calculated from the following equation:

$$\text{required storage capacity } V = C \times i \times t \times K - (\text{pumped volume} - \text{other inflows})$$

$$V = 45 \times 10^4 \times (1–0.25) \times 1.1 \times e^{(3.2 - 0.58 \text{ Ln } t)} \times K - (600 \times 0.75 - 60) \times 3{,}600 \times t \times 10^{-6}$$

where

- The catchment area C is 45 ha or 45×10^4 m^2, and the proportion of runoff intercepted and diverted away from the mine is 25%
- 10% of the rainwater flow returns to the mine from the mud displacement pump system
- The 1-in-50-year rainfall intensity-duration relationship is approximated by Ln i = 3.2 – 0.58 Ln t (Ln = log normal)
- The runoff coefficient (K) increases from 0.5 to 0.9 during the event
- Pumping capacity is 600 L/s with an availability of 75%
- The water inflow from mining operations and groundwater is 60 L/s
- The event duration (t) ranges from 1 to 22 hours

The results of this analysis (Table 9.6-1) show that the maximum storage required is 7.5 ML (megaliters) after 10 hours. A lower pumping rate would require more storage and vice versa. A cost–benefit analysis is required to determine the optimal balance, taking into account the level of risk.

Hydrological Studies and Modeling

Hydrogeologists, petroleum engineers, and civil engineers all start with Darcy's 1856 observation that velocity of laminar fluid flow through sand is directly proportional to the permeability of the medium and the hydraulic gradient. Hydrogeologists find pumping tests useful in predicting the performance of water wells. Petroleum (reservoir) engineers

Table 9.6-1 Emergency storage required for a 50-year storm

Duration, h	Runoff Coefficient	Intensity, mm/h	Storage, ML
1	0.5	24.5	3.1
2	0.6	16.4	4.5
4	0.7	11.0	5.8
8	0.8	7.3	6.2
10	0.9	6.5	7.5
12	0.9	5.8	6.4
16	0.9	4.9	3.8
22	0.9	4.1	0

test within much deeper wells and on cores taken from them to forecast production of oil and gas. Civil engineers concentrate on soil moisture and the effects of pore pressure on stability. Efforts to calculate flow to tunnels, shafts, and underground mines are made difficult by complex geology and lack of data.

In simplest form, the water table, below which the ground is saturated, is nearly a horizontal plane. In a succession of layers of diverse permeability, there may be multiple water levels. Artesian water is trapped under a layer of comparatively low permeability so that water rises above the ground surface. Where permeability is irregular or discontinuous, the water level is similarly erratic. Where water feeds into the ground, the water table is higher, and groundwater flows from such points to lower points of discharge. The slope of most natural water tables is gentle, which usually mimics land surface, and the motion of flow is slow.

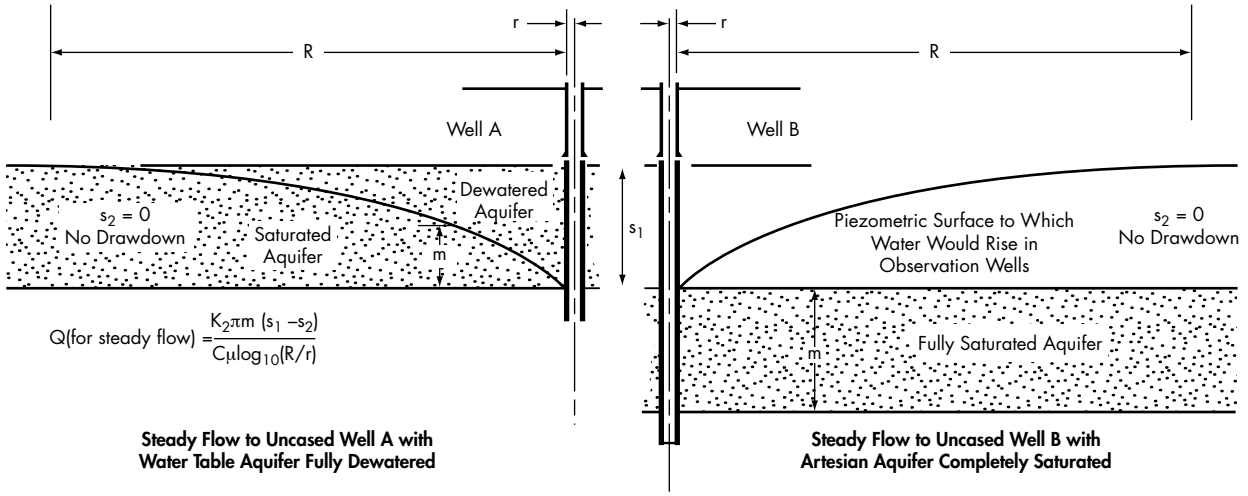
The water table, or piezometric surface, is depressed by pumping a well. The resulting gradient causes flow toward the well. The simplest analysis, such as the Theis solution (Theis 1935), is based on an ideal aquifer—homogeneous, isotropic, and horizontally infinite—and a perfect well, which is open to the full thickness of the aquifer or is very long compared to its diameter and thus receives water by horizontal flow. The depression is an inverted cone, symmetrical around the well. Its slope decreases logarithmically to the radius of effect where the depression is not measurable. If permeability is not uniform, the cone is distorted. Where water is only in fractures, it is discontinuous.

As a well is pumped, the drawdown increases until the well is dewatered or until, because of the steepening gradient, the rate of inflow balances the pumping, and flow becomes steady (Figure 9.6-1).

As the cone expands, its shape may be changed by masses of higher or lower permeability or by recharge or barrier. Schematic profiles in Figure 9.6-2A show the extreme effect of a barrier surrounding a shaft being deepened. Those in Figure 9.6-2B show the effect of an irreducible recharge (e.g., on a shaft being sunk in the center of an island). The cone around such a shaft expands normally in Stages 1 and 2, but its shape is changed by a barrier or continuing recharge.

The void space within a soil or rock is referred to as its porosity. The amount of water that can be stored within the porosity and subsequently released is referred to as *effective porosity* or sometimes *coefficient of storage*. The total porosity of shales and clays is very high, generally greater than 25% and in some cases 50%, but the water is absorbed in the crystal structure and cannot be removed easily, whereas the intergranular spaces are typically so small that movement of water under usual conditions is negligible unless huge areas are considered. Most of the porosity of coarse sandstones is effective as storage. Close-spaced open fracturing can give dense rock important storage capacity, but it is rare. Solution cavities can provide storage in otherwise dense limestone and dolomite. Some lava contains ash, erosion surfaces, and tubes with high permeability and more or less storage. Weathered surfaces of most strong rocks are likely to carry water. Induration (hardening) closes pores and fractures.

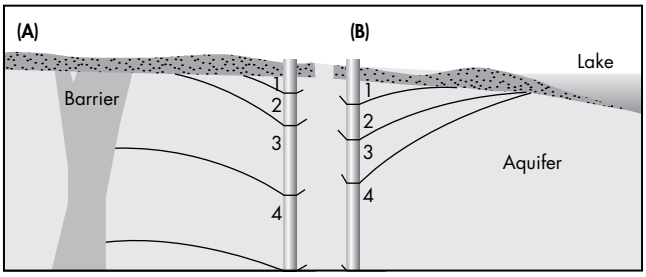
The amount of water that can actually flow through and from the rock mass and into the mine depends on the



Note: In Well A, 100% dewatering is, in fact, impossible; if water is to continue flowing to the well, there must be some saturated wall through which it passes.

Source: Lucas and Adler 1973.

Figure 9.6-1 Comparison of flow to wells in dewatered and saturated aquifers



Source: Adapted from Lucas and Adler 1973.

Figure 9.6-2 Extreme effects of a barrier and continuing recharge

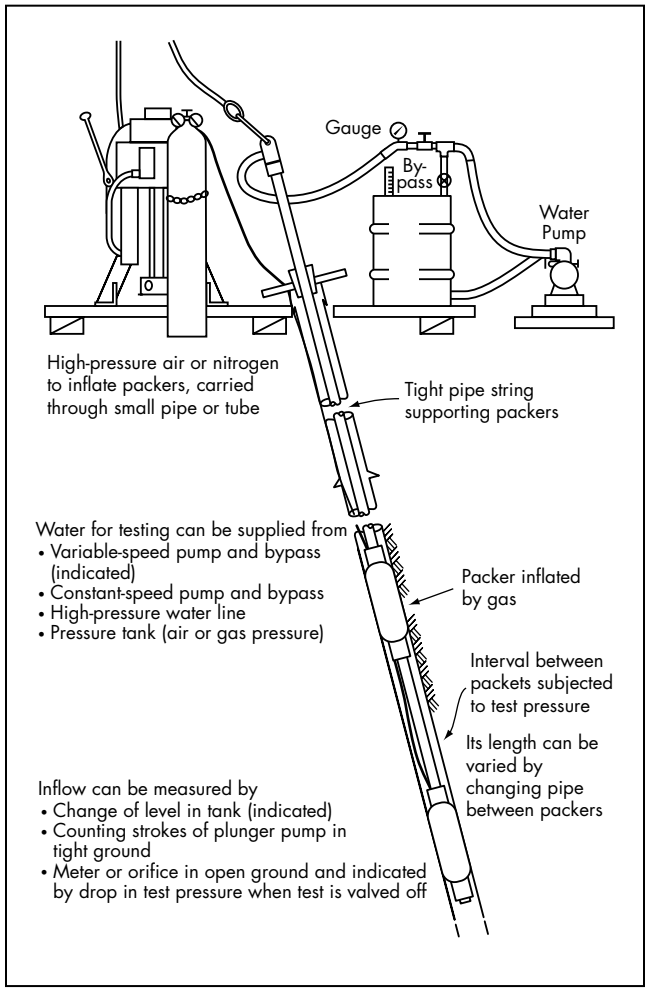
permeability of the rock mass and, more importantly, the presence and permeability of flow paths such as faults, shears, and bedding plane breaks. These are generally not measureable before mining, hence the limited value of modeling. Pumping tests can sometimes yield limited guidance.

Pumping tests are made by pumping water from a well at a sufficient rate long enough to lower the water level measurably in several observation wells, which should be at prescribed locations with respect to the pumped well. The pumped well should penetrate the entire aquifer and, in usual tests, receive water freely from all of it. In thick aquifers this may not be possible, but useful conductivity data could still be obtained. The time and cost of pumping tests generally limit the number that can be made on a project. Hence, it is difficult or impossible to determine properties of more than one part of a complex rock mass, indicate uniformity of any part of the mass, or establish averages that can be used with confidence. Where these points can be determined in other ways and where the test can be set up to provide the information needed, there is no better source of data.

Pressure tests with packers are water injection tests, in some ways the reverse of pumping tests, but they can be made in holes of usual diamond-drilling size, and the packers can be set to test any interval of open hole and can be reset repeatedly (Figure 9.6-3). Generally the water pressure at the section under test can be between 1.1 and 2.5× the hydrostatic head without danger of opening fractures. Where permeability is high, there is an advantage in testing at a low pressure difference to limit pipe friction and the likelihood of turbulence. Where it is low, a higher pressure difference provides a measurable inflow more quickly.

Packers may fail to seat where erosion of weak rock has enlarged the hole and water can leak around them through fractures or connected pores. Such leakage can be reduced by using long packers, by making successive tests below a single packer near the bottom of the hole as it is deepened, and by testing below double packers. A series of tests can be checked by a single test of the same section of the hole. Leakage around packers or through pipe joints leads to overestimating inflow. However, any mud or grease caked on the walls of the hole would have the opposite effect.

Borehole flowmeter surveys may be used to measure water flow patterns within a well. The flow log reveals zones of water entry and exit and allows flow contributions from individual zones to be measured. Spinner flowmeter logs are used during well pump tests to measure hydraulic conductivity. With very low flow rates, the spinner flowmeter may be insufficiently sensitive. The heat-pulse flowmeter may be used



Source: Lucas and Adler 1973.

Figure 9.6-3 Pressure tests with pneumatic packers in straddle configuration

in a stationary mode at selected depths to detect linear flows down to 1 mm/s.

Holes can be cleaned by swabbing (i.e., pumping by repeatedly lifting a column of water above plastic cups on a wire line inside a string of casing or tubing which may extend 100 m or more below the water level in the hole). Measurement of the rate of water level recovery after swabbing is a negative-pressure test. The swab can be run inside tubing on which packers are set to determine inflow from the interval between them and to sample the inflow.

A series of tests between packers efficiently expanded and contracted from the surface can be made at the rate of one to three tests per hour. Packer tests take time for lowering and recovering the packers and setting up for each test. Drill-stem tests are made with a special tool lowered into the hole on a string of drill pipe. Above the tool is a packer, which can be expanded to close off the bottom of the hole. The tool can be placed between two packers to test the section enclosed. A pressure transducer and data logger can record the pressure increase as the tool is lowered, the shut-in pressure in the test section, pressure changes during the test, and pressure decrease as the tool is being removed. To start the test,

a valve is opened, permitting fluid to flow from the ground through the tool into the empty pipe, and a sample is recovered. Potential production can be calculated from the rate of flow and the recorded bottom hole pressures. Each normal drill-stem test calls for a trip into and out of the hole with a string of tight pipe.

Use of Tracers

Tracers are put into a groundwater system at some point and used to indicate the direction of water movement and, in some cases, its approximate rate, by being recognized at a point or points downstream. None of the many tracers is ideal under all conditions. A tracer should

- Be recognizable after dilution, generally with portable equipment, in some cases in test holes;
- Be unimpaired by physical, chemical, or bacteriological reaction and adsorption with the water being tested or rock in contact with it—at least until recognized;
- Move with the water;
- Be convenient to use, reasonably available at moderate or low cost, easily soluble in water, and require no elaborate equipment or procedure; and
- Present no environmental hazard or cause anxiety to anyone.

Fluorescein, an extraordinarily intense coal-tar dye, usually is purchased as a red-orange powder. When dissolved in water it is a brilliant green. One part in 40 million ordinarily is recognized by sight. One part of good quality fluorescein in 5 to 10 billion parts of clear water can be recognized in a colorless tube about 1 cm or less in diameter by 1 m long, with a black rubber stopper or other black bottom. A number of tubes can be mounted side by side in a rack. Examination should be made by good white light in front of a white reflecting surface.

Tubes containing 0 to 0.002 ppm of fluorescein can be used comparatively. Fluorescein is not affected by carbonic acid but is made colorless by contact with peat, acetic acid, and mineral acids. It is unaffected even by long contact with limestone, sand, silt, montmorillonite, and other common clays. Its vivid color gives fluorescein a special advantage where it is desired to make results evident to all observers. Other dyes—fast crimson, congo red, methylene blue, and so forth—may be used similarly.

Chloride ion or salt is recognized in test holes by decreased resistance to electric current or chemically, providing dilution is not too great. Dense solution may be trapped in low spots. Otherwise, salt solution seems to move at the same rate as the water. It is obscured by any natural brine and changes the permeability of some clays. Bromide, nitrates, and other ions are also used.

Dextrose, recognized chemically, is not adsorbed and moves at the same rate as water but is attacked by soil bacteria and is more easily lost in dilution than radioactive isotopes. Radioactive isotopes are said to be recognizable in concentrations of 10^{-18}, but are generally precluded because of public sensitivity to their use.

The normal application is to place the tracer in a possible water source and watch for it to appear in the mine. For example, a tracer may be added to mine water discharge to show whether it is returning into the mine. This may be impossible if the suspected source is a very large body of water. If the mine is flooded, the tracer could be put in the mine and water pumped behind it to cause the tracer to be observed at the surface, although the results could be inconclusive.

CONTROL AND COLLECTION OF WATER

Planning for the control and collection of water must consider all possible sources under normal conditions as well as the possibility of extreme events.

Control of Surface Water

The following measures can contribute to management of inflows to an underground mine:

- Shaft collars and ramp portals can be built on elevated ground or surrounded by earth or rock bunds. Bunds should have a uniform level with no low points that could lead to failure.
- Where a portal is built within an open pit, it should be elevated above the floor of the pit.
- Surface earthworks should not unintentionally impede or divert natural stream and sheet flows.
- Shafts should be pregrouted and imperviously lined.
- Rivers can be diverted, lakes and swamps drained, and streams cleared and straightened to reduce direct inflow and recharge.
- Intakes (including stream beds) can be covered with rolled clay or concrete.
- Water can be intercepted in shallow wells or caught in water rings in shafts.
- Slopes should be cleared and drains built.
- Trees should be planted in low, flat areas to increase evapotranspiration.

The mine should have a procedure that will be followed to manage the effects of extreme weather events. For example, with normal rain, work should be continued but awareness maintained. If cyclonic rains are in the area, the situation should be monitored continuously on weather radar and crews withdrawn from high-risk areas. If there is flooding potential, work should be stopped, all crews withdrawn, and the situation monitored until danger has passed.

Management of Drilling

In a situation that could pose a threat to future mining operations (e.g., holes from a dry lake bed or frozen lake), all drill holes should be fully or partially grouted. During exploration, hydrogeology data should be recorded for future planning needs. When working underground in areas suspected of containing high-pressure underground water, the hole should be drilled using appropriate equipment and procedures.

Exploration drill holes must be cemented in some areas to prevent migration of water from different aquifers. If the hole is left unsupported, any soil or weak rock soon caves, partly plugging the hole and hiding it, after which treatment from the surface can be difficult. If an underground working is connected to a hole to which water has access, it may enter, perhaps with gas, at such volume and pressure that sealing from below is difficult.

The practice at one salt mine is to leave a 45,000-t pillar centered on the mapped location of each hole. The capacity of high-pressure water to ravel weak rock and, if it carries grit, to erode strong rock and metal must not be overlooked. An extreme example of the problem was the inundation and loss of the Diamond Crystal salt mine at Lake Peigneur in Louisiana (United States) in 1980, when an exploration drill hole broke into the workings. Even without being connected to workings, wet holes can decrease stability by permitting water to saturate and weaken clay shales and raise pore pressure.

Some holes, if kept open, could be used for observation of water levels, treatment, geological testing, telephone or electric lines, driving raises, and so forth. Exploration holes generally should be plugged securely before abandonment, or, if needed for communication or observation, pipe or casing should be set through all weak or wet ground.

It is sometimes useful to know the maximum flow expected if a borehole is encountered, so that emergency drainage and pumping capacity can be provided. Water freely supplied to vertical boreholes falls through them with head loss of 100%, meaning that the head loss per length of hole is unity. Flow can be estimated from the empirical Hazen–Williams equation, which in metric units can be expressed (Larock et al. 1999) as

$$S = (10.67\ Q^{1.85})/(C^{1.85} d^{4.87})$$

where

S = head loss (in meters of water) per meter of pipeline or borehole
Q = volumetric flow rate, m^3/s
C = roughness coefficient
d = inside diameter, m

Flow through a clean diamond drill hole can be estimated by substituting $S = 1$ and a typical value of $C = 115$ into the previous equation, then rearranging and solving. For an AQ drill hole (1.89 in. diameter), $d = 0.048$ m, and hence $Q = 0.0108\ m^3/s$.

Flows through clean holes of standard diamond drill sizes are shown in Table 9.6-2.

Table 9.6-2 Flows through holes of standard diamond drill sizes

Size	Diameter, in.	Flow, L/s
AQ	1.89	11
BQ	2.36	19
NQ	2.98	36
HQ	3.783	67
PQ	4.828	128

Prevention of Inrushes

A risk assessment is required to identify the necessary precautions to manage potential sources of inrush. Prior to mining beneath surface water, near underground water sources or old workings, water should be drained or pumped to below the current working level. If this is not practical, a minimum safe approach distance should be determined and maintained.

The maximum flood levels should be determined for all mining areas. Access to an underground working should be constructed above the expected maximum manageable flood level in that area. In flat land, sheet flooding to a depth of only 100–200 mm can rapidly erode a channel into an underground mine and flood it.

Surface water should not be allowed to pond or accumulate near mines and should be drained or pumped away. Where this is not possible, adequately designed and constructed bunding must be in place and the freeboard monitored. The migration of water under the bund should also be monitored.

No structures, dams, tailings dams, storage facilities, mine infrastructure, roads, or rock dumps should be built over the footprint of any underground workings if any potential exists for caving to the surface. Adequate crown and shaft pillars should be provided and monitored. Notably, rock dumps can be a significant water source for deep infiltration.

Underground dams, water storage facilities, slimes dams, or any other storage facility that might be damaged and cause an inrush should be properly designed and located so they are not influenced by mining-induced ground movement.

Where mining activity is close to the base of oxidation or the surface, the area should be fenced or barricaded and bunded. Allowance should be made for the angle of repose of the possible breakthrough area.

If tailings or surface water storage facilities overlie an underground mine, a risk minimization program should emphasize water diversion, groundwater drainage, and sound slope-stability practice. Monitoring should include checking the stability of structures on a regular basis, maintaining adequate storm freeboard, and having diversion structures to deal with an overflow.

Tailings should not be used to fill surface subsidence above working mines because of the potential for liquefaction and inrush through a relatively small subsidence opening. This was the cause of the 1970 inrush at Mufulira mine in Zambia, which caused the death of 89 miners.

Backfilling

Backfill is used with many mining methods and may be used to reduce the risk of subsidence and some forms of inrushes. Planning should ensure that

- Stope voids are sufficiently filled so that caving is minimized;
- Controls are in place to prevent the fill material from liquefying during placement or remobilizing due to subsequent water inflows and ground vibration;
- Fill strength is adequate to ensure stability during adjacent mining or pillar extraction;
- Fill should be drained so that it cannot remobilize; and
- Filling schedule limits the accumulated void space in the mine.

Design of backfill may include tests for particle size, water retention characteristics, field capacity, volume, abrasiveness, transportation systems, fluidity, the possibility of liquefaction, and the need for additives such as fly ash, cement, and gypsum.

During hydraulic filling of voids, monitoring should include pressures behind the barricades/bulkheads, ponding on the fill surface, the barricade/bulkhead integrity and drainage rates, and vertical filling rate.

Draining Underground Water

In some cases, making initial openings in tight ground to drain conduits or masses of wet rock has advantages, but the procedure may be complicated by one or more of the following conditions:

- High-pressure water, hot water, or dissolved gas
- Weak ground, unstable in contact with water flowing under high pressure
- Mud and grit causing or increasing erosion
- Uncertain location of wet ground or conduits
- Uncertain rate of inflow

To indicate the water's location, pressure, and perhaps something of the rate of flow, pilot holes usually are drilled ahead of development openings. Cover drilling, which is drilling in advance and to the sides of the heading with multiple holes, may be necessary for safety.

Tapping can be accomplished by developing an opening into the water source. Where water occurs in small conduits in a rock strong enough to resist erosion, the heading can be continued until the desired inflow is obtained. If water is carried by a well-defined clean conduit, this may still be a good approach, but drilling and blasting the last round is tricky, and, unless a pressure door has been built, only friction in the conduit and in the heading restrain the inflow.

With more time and cost, tapping can be controlled by driving to a safe distance from the wet ground, cutting a drill station, and drilling many radiating holes. The safe approach distance depends on the geology, water pressure, and knowledge of location. Where high pressure is expected, and especially where it could erode the collars of the holes, work must be protected by drilling an oversize collar hole and cementing a collar pipe with a bypass tee and a full-opening valve. Drilling is completed through this valve. Water is drawn from many points throughout a sizable rock mass, which may be important to minimize erosion. Because these holes are short, head losses of 10 to 40 m per 100 m may be acceptable. In this range, each 50-mm-diameter hole may produce 4 to 8 L/s, and each 75-mm-diameter hole, 12 to 24 L/s.

It may be worthwhile to utilize the pressure of tapped water to reduce the pump head. This can be done by piping water under pressure directly or through a settling tank, or *kettle*, to the pump suctions. If the concentration of sand and solids in the water is significant, then effective settling and regular cleaning of the pressurized settling tank can be a problem.

Grouting

Grouting may be used to reduce the water flow into workings, but flow reduction is usually accompanied by a pressure buildup behind the grout, which can be dangerous. In drive and tunnel construction it is good practice to grout and then drill release holes after tunneling passes the area, arranging for the water to be drained toward sumps.

Grouting can be used to

- Control water in shaft sinking and tunneling, rarely stoping, in weak or fissured wet ground;
- Reduce or stop flow past underground plugs and bulkheads;
- Reduce leakage from reservoirs, especially under dams;
- Consolidate and strengthen ground;
- Plug a conduit through which work has been flooded; and
- Make concrete for plugs and so forth with or without preplaced stone.

Limitations are notable. The grout operator controls the nature of grout and the rate at which it is injected into the prepared hole. The operator can limit the pressure and, in some cases, influence the movement of water in the spaces to be filled. Generally, the operator's picture of these voids and what happens in them is vague. After the grout has left the pipe into which it is pumped, it goes where it wants to, unseen. Although the existence of clay in voids interferes with cement grouting, appreciable water movement interferes with all kinds. Although new grout materials have increased the range of conditions in which grouting can be used, a degree of uncertainty persists.

The usual procedure is to drill the ground through casing anchored sufficiently to withstand the pressure to be used, test permeability, and inject a grout that should be chosen and used in accordance with the conditions and objectives. Conduits below the water table should be grouted before a heading reaches them. Thereafter, water movement is much more difficult to control. Deep holes usually are grouted in stages (i.e., the hole is drilled until a degree of permeability is found, then grouted). After the grout has set, the hole is drilled out, deepened, and again grouted. A method of stage grouting from the bottom up has been developed for grouting alluvium.

Where walls are to be built, grouting should precede walling, if at all possible. If not, grout pressure must be kept as low as possible. Pressure against the ground must exceed hydrostatic; ordinarily, it is not allowed to exceed the calculated vertical stress. In special work it may be desirable to open fractures by pressure exceeding the in-situ stress. After grout pumping is begun, it usually is continued without interruption until the planned sealing-off pressure is reached.

Dyes can be mixed with grout to tag various stages of injection. Grouting effectiveness is tested by drilling new holes between or near holes that have been grouted. The degree of confidence in the result depends partly on the uniformity of the ground and partly on the nature of subsequent work.

Cement grouts do not enter the smallest fractures or pores finer than those of coarse sand. Prior treatment of the ground and admixture of sodium silicate and bentonite improve penetration, but, even with this help, pores of medium sand are a limit. Bentonite decreases strength but improves the pumpability of cement slurries and gives body. It also acts as a dispersant, reducing or preventing *bleed*, or separation of water. Neat cement slurries do not set unless the cement particles are brought together at a specific gravity of about 1.5.

Setting time is reduced by use of high early-strength cement, by the addition of calcium chloride, and somewhat more by the use of special fast-set additives. High pressures and temperatures also shorten setting time. Sawdust, shredded plastic, and similar materials can be added to help plug large openings. As far as the delivery system and the size of the openings permit, fly ash, sand, and fine gravel can be added to reduce cost without sacrificing strength. Powdered aluminum reduces or counteracts shrinkage, but where the grout sets under high pressure its effectiveness is questioned.

The water–cement ratio is important in controlling the behavior of cement slurries. In grouting deep holes that cut openings of various widths, the usual practice is to start with a thin slurry (e.g., 5:1 water to cement or even 10:1 by weight) in the expectation this will get into the smallest possible openings. Average slurry on one series of shaft pregrouts was 4:1, and water was reduced to 1:1 where possible. Cement slurries of about 0.5:1 can be pumped. Slurries can be made to stand under water at 20° to 30° from horizontal. Trial runs should be made with unusual mixtures.

Acceptance of a large quantity of grout without pressure increase generally indicates that grout is running through a sizable conduit out of the ground that was intended to be grouted. Remedies may include thickening the slurry, adding bridging materials such as sand or chopped plastic, reducing the rate of pumping, and letting the hole stand for several hours.

Where large quantities of cement grouts are to be used for an extended time, labor may be saved and slurry quality improved by mixing at a central plant with bins, weighing devices, water meters, agitators, and pumps. With pressure, slurry can be pumped through around 1.5 km of 25-to-50-mm-diameter pipe while the plant and the grout operator are connected by telephone. For short jobs, a batching plant and truck mixers are useful.

Reciprocating simplex or duplex grout-fitted slush pumps are usual. Compressed air drive is convenient because of its flexibility. For low pressure, centrifugal pumps can be used, which usually are fed by gravity from agitators.

Ground Freezing

Ground freezing has been used extensively for groundwater control and excavation support in mining and underground construction since the late 19th century. The ground is frozen using coolant pipes to convert soil water into ice. The resulting frozen material is strong enough to allow a shaft to be sunk through deep saturated soils.

The coolant, which flows through pipes installed in boreholes, can be brought to temperatures well below –150°C. Once the ground has been frozen, much less power is required to maintain it in a frozen condition.

Drainage System Design

Mud and silt from blasthole drilling, fill decant water, road traffic in wet areas, and flow through caved or weak ground are problems in many mines, even where the natural permeability of rock is low. The drainage system must cope with the suspended solids load of the water. Drains alongside rail development at gradients of 0.5%–1.0% require frequent local sumps and associated ongoing cleaning, particularly if hydraulic fill is used. Rubber-tired equipment causes more mud to enter the drainage system and requires gradients of at least 2%, favoring a free-flowing system designed to minimize settlement before the main mine sump.

Drainage should be via a dedicated system of side or center drains, boreholes, pipes, and designated storage areas. It should not be directed into old stopes, passes, or other workings unless a risk analysis has been conducted for the potential of an inrush.

The benefits of controlling mud and water may include easier maintenance of track ballast or road surfaces; substantially improved tire life; neater, cleaner, and safer traveling ways; less water percolating to any lower work; and lower cost, less congestion, and less loss of time for cleaning.

Pumping stations should be located to ensure sufficient emergency storage below the station, which gives time to carry out emergency pumping repairs at maximum expected flow rates. There should be a contingency plan for inrushes due to pump failure, including emergency evacuation procedures.

Rising mains should be placed so that they can be regularly inspected to ensure no premature failures. The mine drainage system should be recorded on a suitable plan, such as a long section or isometric plan of the mine, so that it can be understood as a system.

Contingency Planning

Each site should have a contingency plan, which is enacted when a lead indicator exceeds its threshold value. The contingency plan, which is the responsibility of the mine or project manager, is a response plan that defines

- Levels in the lead indicator that would trigger escalating responses, including possible damage reduction;
- Those responses, including evacuation procedures;
- Ongoing monitoring requirements; and
- Types and sources of emergency support available.

In addition, the plan should be reviewed annually, when there is a change of scope in the mine, or following any major incident when the plan has been enacted.

CLARIFICATION AND SETTLING SYSTEMS

Sumps are (1) accumulators to partially equalize the continuously changing rate of inflow and increase the regularity of pump operation, (2) reserve storage in which water can be held during short power interruptions and periods of suddenly increased inflow, and (3) settlers in which some suspended solids can be removed. Where pumping is substantial, the higher efficiency of separate settlers or clarifiers is advantageous.

Clarification

Removal of suspended solids from mine water may allow the use of more efficient clean water pumps, which is especially important in deep mines with large volumes of water to be removed. All clarification systems utilize gravity, sometimes assisted by inertial and/or mechanical means, to separate the solids from the liquid. Neutralization of acidic water can also be done at the settler. The most commonly used settling systems are described in the following paragraphs.

Horizontal Sumps

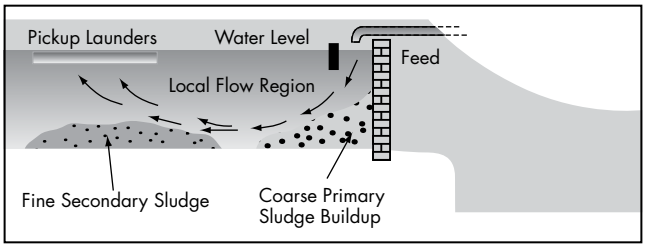
Long development openings are used with horizontal sumps, relying on laminar flow and retention time to allow the solids to settle to the bottom, with clean water discharging over a wall at the end or into a launder (Figure 9.6-4). The sumps must be periodically cleaned, so multiple sumps in parallel are the usual layout, with one sump cleaned at a time, either by pumping out as a slurry or removal by a loader. Baffles have been used to control velocities and improve settling, but their value is questionable and cleaning is more difficult.

The main advantage of these sumps is their simplicity and the absence of any mechanical devices. They are essentially labor free, except for cleaning. Their main disadvantage is low efficiency, requiring large excavation volumes, which makes them unsuitable for high water flows. To enable cleaning with mobile equipment, often the wall is destroyed and then rebuilt.

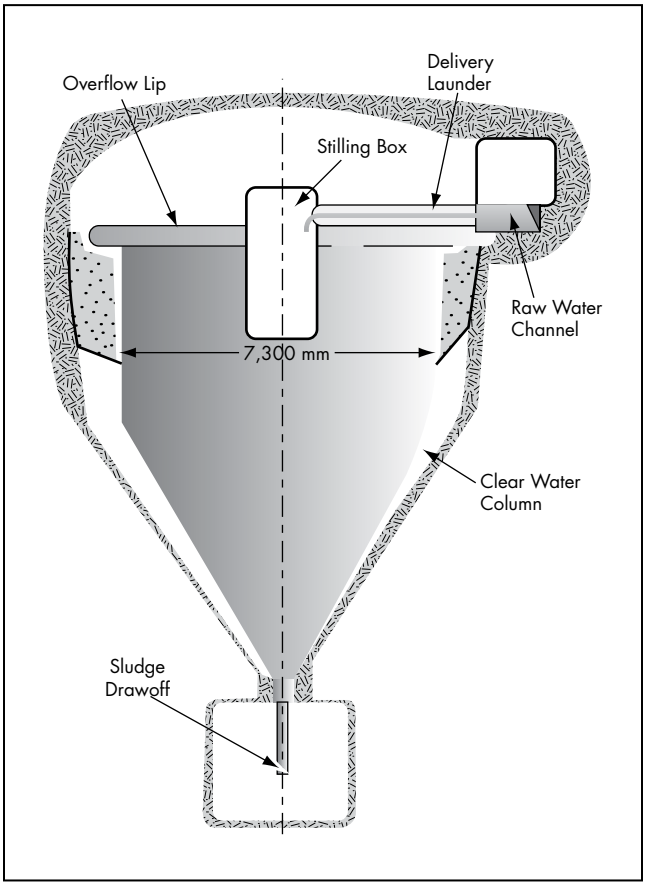
Vertical Sumps

Vertical sumps also rely on gravity and retention time to settle out the solids. They are circular in shape, sometimes with a cone at the bottom to facilitate mud removal (Figure 9.6-5). Their height is typically the distance between two sublevels. The dirty water flows in via a central pipe, which discharges at about one-third of the cylinder's depth, and clean water rises for removal via a ring launder around the top. Flocculants may be added to improve the settling characteristics of fine solids. Mud collects at the bottom and is removed at regular intervals, allowing continuous operation.

These settlers are more compact than horizontal sumps, are much easier to clean, and do not have to be removed from service for cleaning. The large-diameter (5–10 m) vertical openings require reasonable ground conditions and are typically lined with Fibercrete or concrete. Multiple units can be accommodated in close proximity to the pumping station, and the settled mud can be gravity fed to the mud removal system.



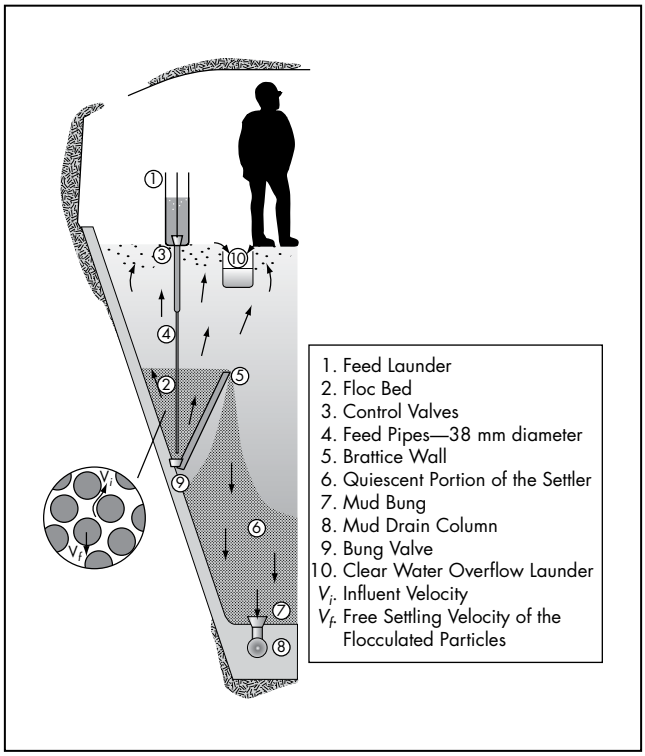
Source: Adapted from Vutukuri and Singh 1993.
Figure 9.6-4 Horizontal settler



Source: Adapted from Hunter and Emere 1977.
Figure 9.6-5 Cylindroconical settler

Vee Settlers

Vee settlers evolved to maximize the benefit from addition of flocculants to achieve high throughputs in relatively small excavations. The settler has three internal compartments—two for initial settling and one for mud storage and thickening (Figure 9.6-6). The dirty water is introduced under a floc bed in a V-shaped compartment. The clear water rises and is removed via a launder, while the finer solids are captured by the floc bed, which overflows into the central compartment where it thickens for eventual removal. Coarser solids settle in the vee compartments and are periodically released into the central compartment via drain cocks, which may be automatically operated. The sludge in the central compartment thickens and is drawn off for separate disposal.



Source: Adapted from Atkinson 1982.
Figure 9.6-6 Cross section of double-vee settler

Maintenance of a stable floc bed is the key to successful performance of these settlers. This is facilitated by dividing the long settlers into shorter compartments. The overall capacity can be adjusted by varying the number of compartments in operation.

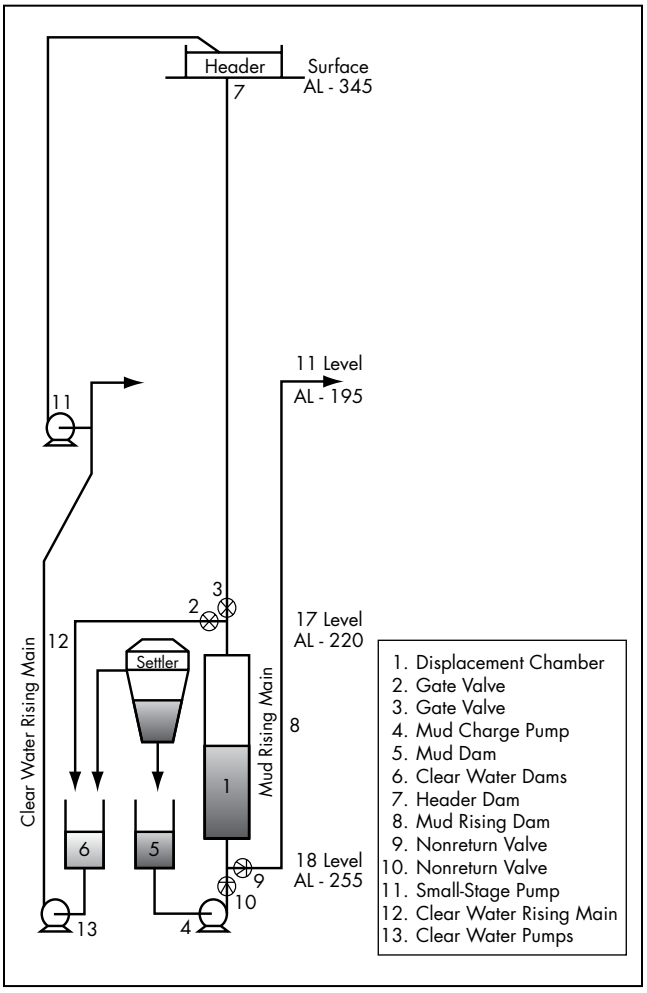
Lamella Plate Settlers

In lamella plate settlers, the dirty water flows across a stack of parallel inclined plates. As the particles settle onto the plates, they slide down and out of the stream. Because the vertical distance between the plates is small, even small particles can settle out relatively quickly, provided the flow velocity is low. These devices are more commonly used in processing operations where space is at a premium.

Mud-Handling Systems

Thickened solids from settlers can be removed from the mine in a variety of ways. This material (sludge) often contains valuable minerals, particularly in a gold mine with coarse free gold, and should be delivered into the processing stream. Typical removal systems include

- **Mixing with other rock:** The sludge can simply be mixed with either ore or waste (depending on its value) and removed by the normal materials-handling system. This can be quite messy but may be satisfactory if the quantity of material is small.
- **Filtering:** The sludge may be dewatered to produce a moist cake by a filter press or vacuum filter for removal by the normal materials-handling system or separately, if desired.
- **Injection:** The sludge may be injected into the rising main, such that the clear water pumps remove the sludge



Source: Adapted from Atkinson 1982.

Figure 9.6-7 Mud displacement system at Mount Lyell, Tasmania, Australia

without being damaged by it. The velocity in the rising main must be sufficient to prevent settling of the coarsest particles.

- **Pumping:** If the solids are sufficiently fine, they may be pumped out by suitable slurry pumps.
- **Hydraulic displacement:** This method uses the head of a column of clear water, supplemented by some additional pump pressure if required, to displace the thickened slurry to the surface. The system was developed in South Africa and is used at Mount Lyell (Atkinson 1982), where the discharge tunnel is sufficiently lower than the shaft collar to eliminate the need for pumping, except for the flushing cycle (Figure 9.6-7).

PUMPING SYSTEMS

The design of pumping systems of even moderate capacity presents complications that require specialists but should be guided by those who are fully acquainted with broader plans and objectives of the work. The need to pump any inflow of water with complete reliability means at least one capable pumping unit plus one equal spare. In a three-unit system, any two should be able to pump the maximum inflow. The greater the number of units, the less the percentage of spare capacity, but beyond some point a larger number of smaller units will cost more and will have lower efficiency. Where only brief pumping interruptions can be permitted, duplicates of all essential parts must be provided, including power supply, pipes, and valves.

Positive displacement pumps are capable of handling dirty water and can greatly simplify the underground handling of water before it is pumped. In general, pumping for shaft-access rail mines is optimized with centrifugal pumps. The steeper gradients and greater volume of mud generated by mobile equipment in trackless mines favor the use of positive displacement dirty water pumps.

High-speed centrifugal pumps are capable of high efficiency provided close tolerances are maintained. They are vulnerable to wear by grit and essentially are clear water pumps. Therefore, means to remove suspended solids before pumping are an important consideration.

Unlike most positive displacement pumps, centrifugal pumps can be rotated without fluid flow. Because the power demand is least when no water is pumped, large pumps generally are started with discharge valves closed. Flow also is limited or stopped if (1) discharge pressure is not sufficient to force water into discharge pipe (as from extreme impeller wear), (2) passages anywhere in the pump system are plugged, or (3) pressure in the suction is insufficient.

Cavitation develops in the intake of impellers if pressure there drops below the vapor pressure of the fluid so that the fluid boils. The formation of bubbles reduces the capacity and efficiency of the pump, and their collapse damages it. To determine the potential for cavitation, it is necessary to calculate the difference between the total head on the suction side of the pump (close to the impeller) and the liquid vapor pressure at the operating temperature.

The net positive suction head (NPSH) can be determined from

$$\text{NPSH} = p_s/\gamma + v^2/2g - p_v/\gamma$$

where

p_s = static pressure in the fluid close to the impeller
γ = specific weight of the fluid
v = velocity of fluid
g = acceleration due to gravity
p_v = vapor pressure

The pump manufacturer will provide the required NPSH for a particular pump based on test work. The system should be designed so that the available NPSH is significantly higher than the required NPSH to allow for head losses in the suction pipe and in the pump casing, for local velocity accelerations, and for pressure variations.

Typical relations of suction head, total dynamic head, and specific speed (revolutions per minute) for single- and double-entry single-stage centrifugal pumps handling clear water at sea level are shown in Figure 9.6-8A. Figure 9.6-8B shows the various power losses typical of centrifugal pumps in relation to specific speed.

The affinity laws for centrifugal pumps can be used to relate flow rate, head (pressure), and power consumption to changes in rotational speed or impeller diameter.

The flow rate relationship is

$$q_1/q_2 = (n_1/n_2)(d_1/d_2)$$

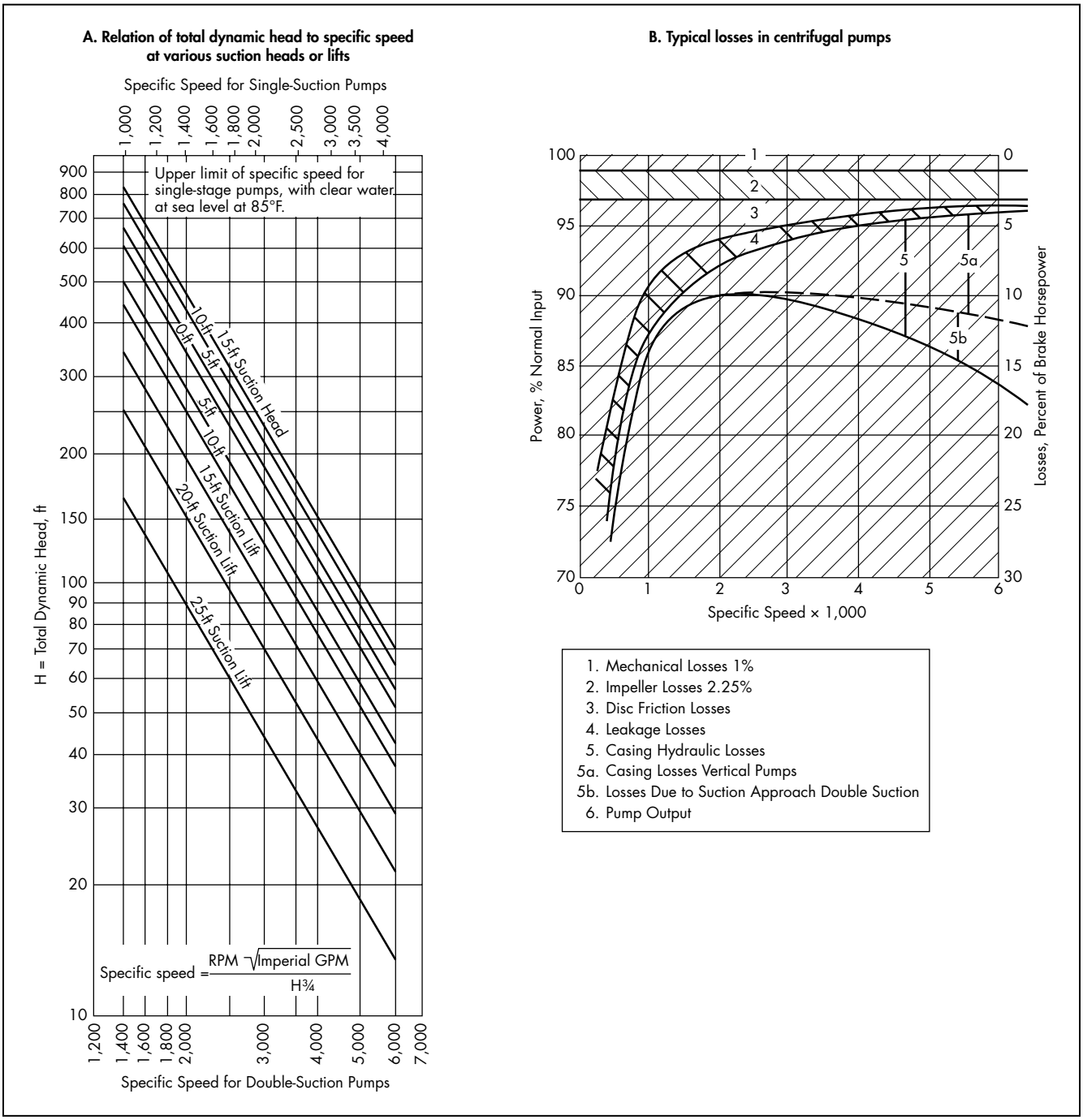


Figure 9.6-8 Suction and losses, single-stage centrifugal pumps

where

q = volume flow rate, m^3/s
n = impeller velocity, rpm
d = impeller diameter

The head or pressure relationship is

$$p_1/p_2 = (n_1/n_2)^2 (d_1/d_2)^2$$

where p is the discharge head or pressure (m). The power relationship is

$$P_1/P_2 = (n_1/n_2)^3 (d_1/d_2)^3$$

where P is power (kW). In other words, the capacity or rate of discharge varies with peripheral speed at exit of impeller, the maximum discharge pressure varies with the square of peripheral speed, and power input varies with cube of peripheral speed.

Design of Pump Stations

Objectives for designing a pump station usually include the following:

- Enough units of all essential components should be provided so that any one can be repaired while others handle greatest expected flow. Ordinarily, this requires spare units with appropriate valves and crossovers and a crane capable of moving the heaviest.
- Pumps and all electrical equipment should be protected from being flooded.
- Sufficient storage should be provided in sumps or otherwise.
- Vertical or submersible pumps should be used to take water from sumps.
- In severe conditions, the pump room should be protected with an adequate pressure door and a raise for access and ventilation.
- Capacity should be added if needed.
- Sufficient controls, usually at least automatic start-and-stop, alarms for high water, and protection for pumps and motors, should be provided.
- Positive suction head should be provided, if reasonably possible, by pumping from sump with vertical pumps or locating pump room lower than sump.
- High-tension items should be protected from accidental jets and splashes. Floor should be sloped and a drain provided.
- Ventilation should be used, as needed, for normal and emergency operation, discharging hot air to the mine or returning it to the surface or, if both are objectionable, cooling it mechanically, adding to the temperature of water discharged.

System Design

The first decision required in the design of a pumping system is whether to pump dirty water or to remove the solids to enable the use of clear water pumps. This will depend on a number of factors, including the volume of water to be pumped, the properties of the water, the depth of the workings, the expected life of the mine, and demands and constraints on disposal and/or reuse of the water.

The second decision is to consider whether to pump in stages or in a single lift from the bottom of the mine. Some of the considerations include whether the water will be clarified or not before pumping, the depth and vertical and lateral extent of the workings, the expected life of the mine, the type of access available (shaft and/or decline), whether the overall mining sequence is top down or bottom up, and whether development of the mine is progressive or largely completed prior to production.

Clearly, these two decisions are interdependent, and the overall system selection will consider all of these factors. Some general principles are listed here:

- Deep mines with large water volumes and long life will favor clarification and clear water pumping (high capital cost but lower operating cost).
- Shallow mines with low water volumes and shorter life will favor dirty water pumping.
- A top-down mining sequence with progressive mine development will favor a staged pumping system.
- A mine with large vertical and lateral extents will favor staged pumping.

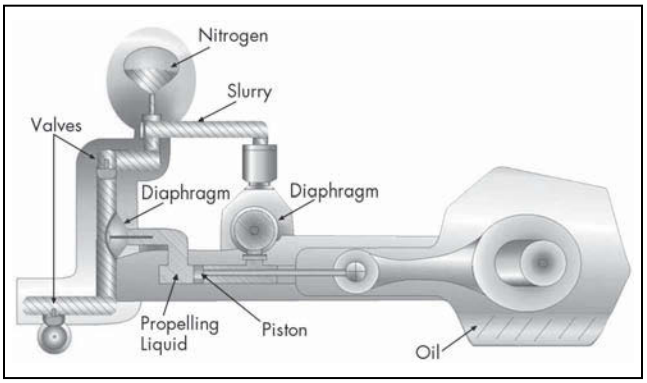


Figure 9.6-9 Crankshaft-driven piston diaphragm pump

- A shallow mine with limited vertical extent will favor single-lift pumping.

Pump Selection

Some of the types of pumps used in underground mining are described in this section.

Small Portable or Semiportable Pumps

Sump pumps may be powered by compressed air or electricity. They are usually vertical centrifugal pumps with open impellers and abrasion resistance or, less commonly, diaphragm and displacement pumps. They are moved to and from sumps where they normally work submerged but can run dry without damage and must be self-priming. Compressed air pumps are limited to heads of about 100 m at flows less than 1 L/s but can deliver up to 10 L/s at low heads. Electric pumps exceed these limits and can be used to clear small inflows from shaft bottoms, clean sumps, and so forth, where high portability is desirable.

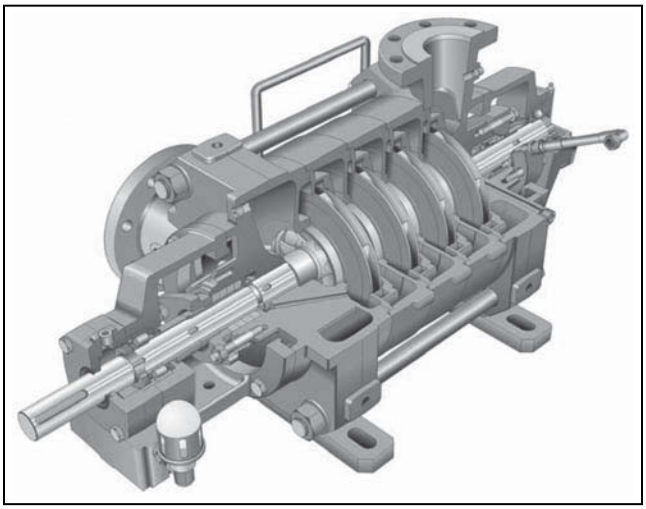
Reciprocating Positive Displacement Pumps

These include dirty water pumps, mud pumps, slurry pumps, and grout pumps powered by compressed air or electric motors through belts or gears. Plunger pumps and diaphragm pumps have high abrasion resistance and low to moderate capacity but are capable of high discharge pressure. Valves are accessible for cleaning.

Some positive displacement (PD) pumps displace the fluid directly, either via a piston working in a cylinder or a plunger working through a stuffing box. A supply of clear, high-pressure water may be required to lubricate glands and minimize wear. A diaphragm pump has a flexible diaphragm between the fluid being pumped and a chamber containing oil that is displaced by a piston, thus preventing any wear on the piston side (Figure 9.6-9). In large sizes, PD pumps are used together in main pump stations to deliver 100 L/s or more of dirty water directly to the surface from depths exceeding 1,000 m.

Single-Stage Horizontally Split Centrifugal Pumps

These are usually directly connected to electric motors for compactness, dependability, and ease of control. Discharge sizes are up to about 300 mm. Generally furnished with close clearances for clear water, these pumps can work with high efficiency to capacities of about 200 L/s and heads to about 150 m. Pumps are made for larger capacity at somewhat lower



Courtesy of Sundyne Corporation © 2010.
Figure 9.6-10 Multistage centrifugal pump

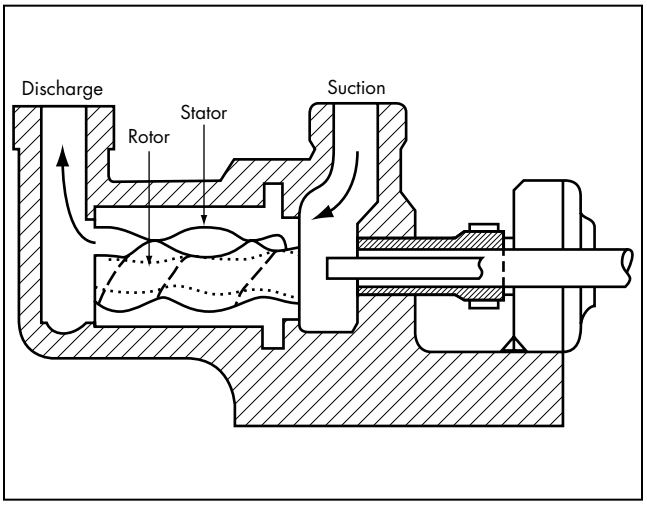


Figure 9.6-11 Progressive cavity pump

heads. Installation and maintenance are simpler than for multistage pumps.

Horizontal Multistage Centrifugal Pumps
These are usually direct-connected to electric motors for compactness, dependability, and ease of control (Figure 9.6-10). Discharge sizes are from 75 to 250 mm with 2 to 10 stages. Although total efficiency is likely to be less than that of single-stage units, efficiency per stage may be higher. Heads typically range from about 150 to 700 m.

Although the first cost of these pumps generally is higher than for single-stage pumps of similar power, their use may save the capital and operating costs of duplicate facilities. The close tolerances require clean water to prevent excessive wear, so the cost of building and maintaining settling dams is a consideration.

Vertical Turbine (Deep Well) Pumps
These are essentially vertical centrifugal pumps made in comparatively small diameters to work in water wells and similar narrow but high spaces. Water ends are essentially similar whether close coupled to a vertical electric motor, connected by a long shaft to a motor at the top of a well casing, or connected to a submerged motor in the well. Vertical turbines are made with as many as 20 or more bowls (stages) for heads of 5 to 30 m per stage. Pumps are made in diameters from about 150 to 1,000 mm or more.

Because pump intakes normally are submerged, priming is not a problem. Motors of close-coupled pumps can be well above normal water levels. Impellers can be removed if a pump is to work at less head, or more bowls and impellers can be added for greater head.

Vertical Centrifugal Pumps
These are made in one and two stages with proportions like those of horizontal centrifugals. They are direct-connected below vertical motors by short shafts. Because the pump always works submerged, priming is not a problem. They have the simplicity and high capacity of horizontal centrifugals and in many applications approach or equal the convenience of vertical turbines.

Vertical centrifugal pumps are made for capacities up to 950 L/s and heads to 250 m with motors as large as 1,000 kW. They are commonly used to raise water from a lower to an upper sump, from which it can flow with positive suction head to horizontal centrifugals. They also can pump to emergency storage.

Air Lifts
The efficiency of an air lift is low at best, yet there is no simpler pumping system. Typically two strings of concentric pipe are submerged, and compressed air is introduced through the internal pipe. Anything that can get into and through the larger pipe is pumped without doing damage. They are especially good in dewatering partially blocked shafts but require a minimum submergence of around 33% and so cannot completely dewater a shaft.

Progressive Cavity Pumps
With abrasion-resistant rotors in rubber stators, progressive cavity pumps will pump any mud that can be drawn into them. They are widely used in mine dewatering (Figure 9.6-11). Multistage units are capable of heads up to 720 m at up to 60 L/s, or flows up to 150 L/s at lower heads.

Testing Performance of Pumps
Well-equipped manufacturers have extensive facilities for testing large pumps with precision. Comparable test facilities are uncommon in deep mines, yet records of performance are essential to evaluate the results of pumping and to guide maintenance. Modern pump stations are often well instrumented, but serviceable data can be obtained by such means as the following:

- The total output by a weir on the surface can be measured, integrated, and recorded. In some cases, it is possible to measure rate by pumping into or out of a large sump or tank but usually with less accuracy. Inflow can be estimated by adding the water removed with ventilating air and subtracting any water piped into the mine for drilling and washing, and decant water from hydraulic fill.
- The pumping rate of each pump can be measured by any of several means, including magnetic flowmeters,

venturis, orifices, weirs, or pumping from or into a sump. Provision of these measuring devices in the discharge of each pump makes it possible to check each periodically with little labor and loss of pumping time.
- Individual discharge and suction heads can be measured with calibrated Bourdon-type gauges or manometers placed appropriately.
- Mechanical input can be determined from a calibrated watt-hour meter, subtracting the motor loss as taken from the manufacturer's test data. Various torque dynamometers can be used for a more fundamental determination of power input.
- Pump speed can be measured by a calibrated tachometer and stopwatch, and this is generally adequate.

The rate of discharge of a centrifugal pump and its discharge pressure can decrease with time because of scale formed in the discharge or other change increasing the dynamic head; decrease in suction head due to scale, obstruction of intakes, and so forth; decrease in effective diameter of the impeller or other severe pump wear; blocking or other obstruction of the impeller passages; and wear causing excessive leakage between stages of multistage pumps and across wear rings in single-stage.

All pumps discharging into a single line should be kept balanced to operate together efficiently. If unbalanced, the head of one pump could decrease until flow is reduced to the degree that the pump bearings seize.

Cost of Pumps

Usually other factors are likely to be more significant than the first cost of the pump. The cost of standard water ends of horizontal centrifugal mine pumps generally is 30% to 50% of the total cost of the pump-motor-starter unit. It may be only 5% to 15% of the cost of the pumping plant, including sumps, clarification, power supply, ventilation, suction, discharge, and required excavations. The construction of the pump influences the cost of the station (e.g., vertical pumps require less floor space and no suction line). Efficiency is important because power ordinarily is the largest part of operating cost. Where planning makes it practicable, a single high-head pumping installation can be more efficient, as well as simpler, than a series of pumping stations each working at a lower head.

Initial duty calculations can be made assuming efficiency of 60% (motor $\times$ pump) for small units, 75% for larger pumps, and up to 80% for units of large capacity. Large units, in which both motor and pump are excellently matched to the pumping duty, may go higher (e.g., 90% pump $\times$ 93% motor).

Pipework

Normal working pressures may be increased considerably by the pulsations of reciprocating pumps or by surging and water hammer. Special steels or extra wall thickness may compensate for pressure or corrosion but with added cost or weight.

Where significant ground movement is likely, the discharge pipe should be able to survive some misalignment. If it is likely that individual lengths or sections of a discharge line must be removed to clean scale or repair accidental damage, pipe should be coupled and supported accordingly. In some places, repair by cutting and welding is difficult.

In some cases, reasons exist for not treating water underground, and protection of the outside of the steel pipe is likely to be incomplete. The angle of the pipe, its exposure to accidental damage, working room, and service facilities also are factors in selection of pipe and couplings and how it is supported. Because many of these factors are difficult to evaluate, continuous dependable service is important, and repair can be highly inconvenient, innovation in high-pressure design pipe is uncommon.

Preference for seamless steel pipe for high-pressure lines has decreased with better quality control in welding. High-pressure lines usually are no larger than 250 mm (10 in.) to limit the weight of long, steep columns, facilitate placement and repair in limited shaft compartments, reduce the number of pumps discharging to the same line, and make it easier to provide a spare line. Wall thickness needed for any pressure is calculated from the minimum ultimate tensile stress with a good factor of safety; 5 is frequently used in shaft rising mains, whereas 2 may be acceptable in more accessible areas. There is much more latitude in choice of pipe for lower pressures, such as discharge from single-stage pumps. Where volumes are large, 500-mm (20-in.) pipe may be used.

On high-pressure lines, the practice is to join pipe with flanges. Simple flanges generally are preferred, but gasket material must resist extrusion by strength or by retention. These long strings generally are assembled from the bottom up by adding one or several lengths. With the use of a cage with extensible crawls to handle the pipe and decks where crews work, more than 100 m of pipe can be placed in a shift.

Weight usually is carried on steel bearer beams or brackets concreted into the shaft wall at intervals of 120 m or less. The provision of an expansion joint below each bearer assists in equalizing weight, adds vertical flexibility, and makes it unnecessary to raise an entire string to replace a length.

Continuous rising mains can be added in 20-m lengths, with each joint welded as it is added at the surface like well casing. Alternatively, screwed well casing may be used. Guides prevent side movement, but all weight bears either on a concrete bridge at the elevation of the pumps with the line in compression or is suspended from a bridge at the surface with the line in tension.

Gate and check valves are placed in the discharge of each centrifugal pump for use in starting and for protection during the repair of a pump connected to an active discharge line.

BULKHEADS AND PLUGS

Deep gold mines in South Africa have developed outstanding practice with plugs, demonstrated in tests and in both routine and emergency construction. Generally, these plugs were built in hard, strong quartzite, impermeable except on fractures. Culmination of this development came with the four plugs placed in emergency conditions in the West Driefontein mine in November 1968 at depths below 1,000 m. They were completed 20 days after the inrush, in 3.0 $\times$ 3.7 m crosscuts. Work was undertaken with water flowing more than 1 m deep on each level. Valves to stop bypassing water were closed on the 23rd day, and the lower plug withstood a head of 1,116 m of water on the 26th day (Cousens and Garrett 1969).

Several conclusions are reached from South African practice (Garrett and Campbell Pitt 1961). Generally, it is more difficult to stop leakage past a bulkhead than to make it strong enough to resist thrust. Passage of water through rock fractures is related to the pressure gradient, which should be moderate.

Many plugs are neither hitched, tapered, nor reinforced. Reliance is placed in the strength of the concrete bearing against the usual irregularities of the rock surface. Such

plugs have withstood pressures of more than 6.9 MPa, one at 15.5 MPa. No indication of structural failure resulting from thrust was noted in the examples reported.

Leakage is likely along the floor and roof, even at low pressure, where mud and honeycomb, laitance (a weak layer due to excess water), and air pockets commonly weaken the rock–concrete contact. These leakages sometimes are sealed acceptably by one stage of grouting.

At higher pressures, water is likely to break through rock fractures. This appears to result from rock movement induced by pressure on rock surfaces. Part of the water entering the fractures may not appear outside the bulkhead. Much of this leakage can be sealed by several stages of cement grouting at pressures to at least 2.5× the hydrostatic head, in holes drilled as far as 10 m into the rock. The effect of each stage of grouting seems to be to fill the fractures, perhaps poststressing the ground around the bulkhead, and increasing its resistance to the entrance of water.

The loss of several bulkheads subjected to more than 6.9 MPa is attributed to failure of gaskets, threaded plugs, and other fittings. The possibility that even the smallest leakage through fractures may be enlarged by high-pressure erosion should not be underestimated.

Preferred construction is by injecting cement–sand grout into clean, strong, angular rock previously packed between timber forms. This generally results in better concrete, easier logistics, and, in some circumstances, less time and cost than direct concrete placement. Ordinary portland cement usually is used but rapid-set cement can be required by urgency. Concrete should reach at least 17.2 MPa in 28 days. The four plugs at West Driefontein were made of cement–sand slurry only. Slurry can be mixed under good control at a central plant and pumped more than 1,000 m through small pipe. Horizontal cold joints should be avoided by all means.

Recommendations from this work include seeking sites in tight sound rock. In good ground, at least, keyways are unnecessary, but note the length next recommended. The plug should be long enough that the pressure gradient is moderate. In one test, a gradient of 9 MPa/m was reached after several stages of rock grouting, but for working bulkheads, designed gradients of 900 to 1,400 kPa/m have proved effective (i.e., for each 6.9 MPa, allow between 8 and 12 m of plug).

All mud and loose rock should be removed, the water flow across the floor should be stopped, and a ventilation hole provided for high spots in the roof. Pipe, gaskets, valves, and fittings should be tested at a pressure somewhat greater than that to be withstood. Several stages of grouting to reduce leaks should be planned. The first, through pipes at concrete–rock contacts, might be at around 2,000 KPa, but later stages through holes drilled successively deeper into the rock can be at successively higher pressures, up to at least 2.5× the expected hydrostatic head.

Comparable experience in other rock is unknown, but several inferences seem to deserve consideration. In any strong rock that can be grouted effectively, similar practice seems applicable. In weaker rock, however, or one with seams that do not take grout well, it appears prudent to work to lower pressures and lower pressure gradients to reduce the risk of uncontrollable leakage through erosion and enlargement of rock permeability or defects.

No large excavation should be subjected to pressure greater than the maximum hydrostatic head unless after full study and evaluation of rock stresses. Lastly, the use of expanding cement, using aluminum powder or similar, seems promising.

INRUSHES

An inrush is a low-probability but high-impact event, usually leading to major disruptions to production and multiple fatalities. For example, Vutukuri and Singh (1993) tabulate 33 mine inundations in England, Scotland, India, and the United States having an average of 31 and a maximum of 375 fatalities (at Chasnala colliery, Jharia, India, in 1975).

One definition of an *inrush* is "the uncontrolled mass movement of material," whereas *subsidence* is "a sudden collapse of a large volume of rock," often referred to as *plug* or *pipe* subsidence (McCarthy and Harvey 1998). These definitions recognize that flows of mud, tailings, wet ore, or saturated surface material may also become inrushes, and that sudden subsidence, although a hazard in itself, can create unexpected connections to these materials.

Causes of Inrushes

A review of 43 examples of inrushes, mainly in noncoal mines, shows the following causes (McCarthy and Harvey 1998); a further 18 examples gave insufficient details for classification:

- Surface flooding enters mine: 33%
- Mining broke into old workings: 28%
- Strata water enters mine: 19%
- Accidental connection made with sea, river, lake, etc.: 14%
- Failure of a dam, seal, borehole, etc.: 5%
- Earthquake: 1%

It is interesting to compare this list with the next, which was prepared by Job (1987a, 1987b) based on 208 incidents in British collieries during the period 1851 to 1970. The much greater frequency of contact with abandoned workings is to be expected in intensively exploited bedded deposits. The lack of surface flooding may reflect a greater predictability and awareness of flood levels than has been the case in arid environments. This list does not include fill bulkhead failures, which are common but rarely become public knowledge unless a fatality is involved:

- Contact with abandoned old workings: 78%
- Clearing old shafts or shaft sinking: 9%
- Failure of an underground dam or seal, or leakage of a borehole: 4%
- Contact with surface water—pond, river, canal, or stream: 4%
- Contact with surface unconsolidated deposits—glacial or organic: 4%
- Strata water entering working: 1%

Frequency and Location of Inrushes

A survey of mining periodicals for the years 1980–1996 identified 33 inrush incidents involving fatalities or loss of production. This is a rate of about two incidents per year worldwide, not including unreported incidents in China and the former USSR. These locations are for inrushes from 1980 to 1996:

- Africa: 9
- North America: 6
- Southeast Asia: 5
- Australasia: 5

- Europe: 5
- India: 2
- South America: 1

The occurrence of inrushes seems to generally correlate with the level of underground mining activity. Without new preventive measures, a country such as Australia should experience one incident every 3.4 years. As about 70 significant underground mines exist in Australia, the *return period* for an inrush incident is 238 years, which means that personal experience and awareness of the hazard are likely to be low. The nature of low-probability, high-impact events means that a severe incident could happen at any time and that special management procedures are required to maintain awareness.

More recent instances, particularly in coal mines in North America and China, suggest that this problem remains a significant one, is a serious threat to employee safety, and has a serious impact on the public perception of the mining industry.

SURFACE WATER DISPOSAL

All of the water and associated sediments removed from the mine must be disposed on the surface or recycled. Some water is disposed into the atmosphere via the exhaust ventilation and evaporation from wet rock dumps and water storages. In many mines, the water from underground is an important source for processing operations and other surface operations.

Today, even if the mine water is of better quality than the water in local streams, direct discharge into surface waterways is often not permitted or requires extensive and expensive purification. The presence of heavy metals or acidity will exacerbate this situation.

Mines with superfluous water may construct evaporation ponds for water disposal. This is not particularly effective in cold climates with low evaporation rates. In some cases, properly treated mine water can be an important water source for the surrounding community. Any water discharged into the natural environment should meet internationally accepted water quality standards.

ACKNOWLEDGMENTS

This chapter was developed from "Ground Water and Ground-Water Control" by R.L. Loofbourow, which appeared in the first edition of this handbook (Lucas and Adler 1973). The authors acknowledge the many contributors to that chapter.

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CHAPTER 9.7

Physical Asset Management

John Woodhouse

INTRODUCTION

This chapter introduces and illustrates the whole picture of "joined-up" asset management. The term *joined up* encompasses acquisition, operation, maintenance, and logistics support as well as ultimate replacement, decommissioning, and/or disposal. In addition to focusing on physical mining and processing equipment, the chapter describes application of asset management principles to higher levels of asset definition, such as the ore body, mine, or whole mining enterprise (in the same way that asset management became the core business model for North Sea oil and gas operations). In particular, the chapter aims to differentiate between asset care (i.e., just equipment maintenance) and truly optimized whole life-cycle management of an ore body, the physical equipment, and mine systems. Illustrations of component methods and typical problems encountered draw on mining examples wherever possible, but some cases deliberately reference other industry practices to make the point that cross-sector learning is part of good asset management practice.

WHAT IS PHYSICAL ASSET MANAGEMENT?

A mining organization is critically dependent on mineral resources and a wide range of physical equipment, infrastructure, and operating systems. The economic performance of the enterprise is closely linked to the correct investments, extraction strategy, system design and selection, appropriate use, high performance, reliability, safety, and maintenance of such physical assets. The optimal management of these assets is therefore at the heart of good mining business management.

The term *asset management*, however, has been applied to a variety of asset types and activities over the years, so it is important to recognize and distinguish between the current usages of the term to focus on appropriate business priorities and value.

The financial services sector, for example, has long used the phrase to describe the management of a stock or investment portfolio—trying to find the best mix of capital security or growth and interest rates or yield. This vision of asset management has close parallels to the physical systems version, insofar as it involves juggling investments, risks, performance, and sustainability. However, there are big differences between financial assets and physical ones and between the options for their management.

Some engineering and industrial maintenance departments (and certain software vendors) have relabeled their functions as asset management to boost their professional standing and credibility in the organization. This, however, is a restrictive and ultimately ineffective view that misses many of the biggest opportunities. Asset care (maintenance) is just one part of asset management. Some of the biggest prizes in effective asset management, for example, involve optimizing the relationship between asset utilization ("sweating the assets") and the needs for maintenance or renewal. This is only possible if a cross-functional view is taken of (whole life) asset management.

A more holistic view of assets and their management has emerged since about 1990, and this concept of asset management is the focus of this chapter. Such a model encompasses all aspects of investing in the right assets in the first place, exploiting them appropriately, caring for them (maintenance), and ultimately replacing or disposing of them. Various levels of granularity exist in this context, depending on the definition of the asset. Some organizations limit the term to the management of physical equipment items or process units, while others take a more holistic view and define their assets at the level of the whole ore body, site, or mining enterprise. In all such cases, however, the challenges of optimizing investment, operations/utilization, maintenance, and whole life value are similar—and are addressed in a publicly available specification, PAS 55:2008. This standard defines terms and requirements for "optimal management of physical assets."

PAS 55 defines asset management as "systematic and coordinated activities and practices through which an organization optimally and sustainably manages its assets and asset systems, their associated performance, risks and expenditures over their life cycles for the purpose of achieving its organizational strategic plan."

Other wordings and definitions are in use in various industries, but the most robust ones all include the key elements

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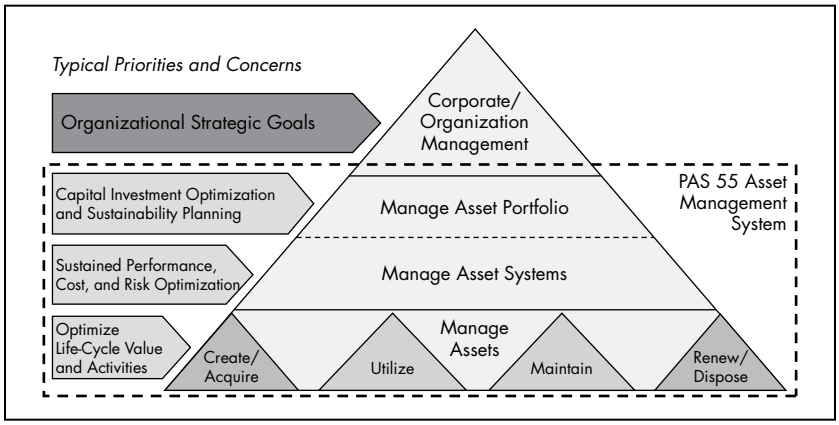


Figure 9.7-1 Integrated view of assets and their integrated management

of this PAS 55 definition: They identify the coordination and systematic approach, the need for optimization, and the combined consideration of expenditures, asset performance, and risk exposures. They recognize the need for a whole life-cycle or sustainability view (i.e., the long term) and for asset management to be directed toward a clear corporate goal. For many organizations, such aspirations represent a significant challenge, yet the prize for getting it right is substantial.

Benefits of Optimized Whole Life-Cycle Asset Management

Many organizations are tackling the integration and optimization challenges of asset management in all sorts of industries and business environments, and they are coming to remarkably consistent conclusions. A cross-disciplinary, whole life-cycle view of asset management yields significant tangible and sustainable benefits compared to the "silo" behaviors of individual functional departments that have often historically been focused on conflicting and/or short-term goals. Typical quoted benefits include

- Thirty percent reduction in total cost of ownership (Nuon electricity network, the Netherlands),
- Thirty percent reduction in mining equipment maintenance costs with 3%–10% increased system availability (copper mining, Chile),
- Seventeen percent increased output at 50% lower operating cost (Shell North Sea oil platforms),
- Twenty-eight percent reduction in planned system downtime (U.K. National Grid),
- Budget savings of A$11 million per year (New South Wales government), and
- Twenty-nine percent increased output at no extra cost (Baltimore, Maryland, United States, power generation).

Further and maybe even greater long-term benefits are found through improved credibility in the eyes of customers, enterprise investors, and other stakeholders. Good asset management also results in much greater engagement and remotivation of the work force and in more sustainable, continual improvement business processes.

It is not easy, however, to create such an integrated system in the first place, or to resolve the inevitable trade-offs that exist between short-term and long-term effects or among costs, performance, and risks. It usually involves a fundamental review of stakeholder expectations and business drivers, and a quantified or consistently scaled method for determining the criticality and value of assets and activities. Sophisticated tools and processes are also usually needed to optimize the mix of capital investment, operating costs, performance, maintenance requirements, risks, and achievable economic life spans. And education, communication, motivation, and behavioral changes have consistently proven to be absolutely critical success factors. Modern optimized asset management therefore involves an integrated business model that bridges departmental contributions to deliver net value for money on both short and long time frames, in line with explicit business goals.

What Are the Assets?

One of the key messages of PAS 55 is that joined-up asset management needs to occur at multiple levels in every organization. The characteristics and criticality of different assets and asset types vary, and the options for investment, exploitation, care (maintenance), and ultimate disposal or renewal differ, but the requirement to deliver sustainable, total value for money remains a consistent underlying objective. The PAS 55 standard for asset management illustrates this stratified view of assets and their management (see Figure 9.7-1).

In a typical mining enterprise, therefore, a hierarchy of assets would correspond to the following categories:

- Whole mining enterprise, including all business functions and locations
- Specific ore bodies and/or geographical regions of the enterprise
- Specific mine and associated extraction/processing facilities
- The different extraction and/or processing systems
- Discrete equipment items

At the corporate or enterprise level, asset management simply means good business management. However, at lower levels, asset-specific characteristics and divisions of responsibility (departments and functions) make it increasingly difficult to see the whole picture and optimize the whole life-cycle costs and value. This is the area in which a robust asset management strategy helps to ensure coherence between the organization's goals and the day-to-day realities of who does what, when, where, and why.

Emerging Standards in Asset Management

Since 1990, two very different business environments have made very similar discoveries about what is needed and what

is achievable through a more integrated approach to whole life-cycle asset management.

The North Sea oil and gas sector faced massive wake-up calls in the late 1980s: the *Piper Alpha* disaster, an oil price crash, the Cullen Report (1990) recommendations on risk/safety management, market globalization, and so on. These forced a fundamental reappraisal of underlying business models and the recognition that big companies, while holding several strategic advantages and economies of scale, were losing the joined-up thinking and operational efficiency or agility that smaller organizations naturally enjoy (and need in order to survive). BP and Shell experimented with the creation of a small-business culture within a large organization with its attendant leverage. Asset-centered production units emerged, employing multidisciplinary teams with significant budget and performance freedoms to find the best combination of investments, exploitation, and care for each reservoir or production platform. The results were spectacular, reducing total costs of ownership and raising productivity nearly tenfold, while simultaneously raising safety and environmental performance. Recent high-profile events in BP's U.S. operations (Texas City refinery and the Gulf of Mexico leak) have, of course, implied otherwise, but the underlying trend has been profound and significant; fatalities, injuries, and environmental incidents have fallen dramatically since the 1980s.

At the same time, public utilities and municipalities in Australia and New Zealand—facing a sustained period of degrading services and escalating costs combined with some highly publicized system failures—realized that a different approach to sustainability and value for money was needed. The whole asset life cycle needed to be considered, and capital investment planning began to take on a cross-functional view to find the best value-for-money options. Again, the results have been spectacular, with significant and sustained service improvements, cost reductions, and reestablished public confidence.

Now the lessons are being adapted and applied to mining, road and rail transport, water, electric and gas utilities, process and manufacturing, health, education, and other industries in both private and public sectors. There is increasing recognition, therefore, that many of the good practices in managing assets are generic, and there are many opportunities for learning and improving by exploring how other industries are addressing similar problems in different environments.

In 2002, the Institute of Asset Management (IAM) initiated a project with the British Standards Institution to document these observations in the form of a publicly available specification. PAS 55 was first published in 2004 with the backing of about 25 organizations. In the next 4 years it became a mandated requirement for the regulated U.K. electric and gas utility sectors and was picked up internationally in railways, mining, water, and manufacturing industries. The most recent update of the standard, published in 2008 (PAS 55:2008), involved more than 50 organizations from 15 different industries in 10 countries. It is currently going forward to form the basis of a full International Organization for Standardization (ISO) standard with the support of the national standards organizations of more than 20 countries. It is safe to say that asset management has thus evolved to become mainstream—an expected competency in any organization that is heavily dependent on physical assets, resources, or infrastructure. The understanding of what this involves, however, is still evolving: Best practices—or even what should be regarded as good practices—are constantly moving forward. This is why a management system is necessary: to coordinate the many functional contributions of an organization with a common understanding and clear purpose, optimally managing the inevitable (and inherently conflicting) subgoals and doing this sustainably while still retaining enough flexibility to cope with future unknowns.

Basic Terminology

Before delving into the specifics of what needs to be done, why, and how, it is necessary to be very clear on terminology. A common problem in this field is one of poor communication or misunderstanding of meanings. Some terms in asset management are used and interpreted very differently depending on role or context, while others are simply misused or confusing. PAS 55 tries to address this with a definitive glossary of 35 common terms, the most important of which are reproduced here. Other useful guidance is to be found in complementary standards such as ISO 15663:2000 for whole life costing, BS 3811:1993 for maintenance and reliability language, and ISO 31000:2009 for risk management.

- **Asset:** At the simplest level, an asset is anything of value (tangible or intangible) that is owned by the organization. In this context, however, it refers particularly to plant, machinery, property, natural resources, buildings, vehicles, and other physical items or infrastructure. However, the principles and practices of good asset management also apply to other asset types (data or information, reputation, financial resources, licenses or entitlements, natural resources, etc). Note also the relationships between individual assets within functional asset systems and portfolios of assets. See Figure 9.7-1.
- **Asset life cycle:** An asset life cycle is the time interval that commences with the identification of the need for an asset and terminates with the decommissioning of the asset or any associated liabilities. The principal stages of an asset's life cycle typically include creation/acquisition, use, maintenance, and renewal/disposal.
- **Asset management:** Asset management comprises the systematic and coordinated activities and practices through which an organization optimally and sustainably manages its assets and asset systems, their associated performance, risks, and expenditures over their life cycles for the purpose of achieving its organizational strategic plan.
- **Asset management system:** The asset management system comprises the organization's asset management policy, strategy, objectives, and plans and the activities, processes, and organizational structures necessary for their development, implementation, and continual improvement. A management system also includes the organizational structure, roles and responsibilities, planning activities, standards, information systems, practices, processes, procedures, and resources.
- **Asset management information system:** The information system, usually computerized, is for the storage, processing, and transmission of asset management information such as asset registers; drawings; contracts; licenses; legal, regulatory and statutory documents; policies; standards; guidance notes; technical instructions; procedures; operating criteria; work plans; asset performance; and condition data or audit results. *S*uch systems

are sometimes referred to as enterprise asset management systems. Their role, however, is largely that of a computerized maintenance management system, and they should not be confused with the larger scope of the life-cycle business management of physical assets (an asset management system) as described in this chapter.

- **Asset management policy:** The asset management policy is a specific mandate that states the principles, commitments, obligations, and control framework for the development and implementation of the asset management strategy and plans. It must be clearly derived from the organization's corporate values and strategic plan and must be complementary to (not conflicting with) other company policies, such as those for safety, health and environmental commitments.
- **Asset management strategy:** The asset management strategy is a vital, centralized, and cross-disciplinary view of the long-term optimized approach to the management of the assets. It needs to reflect the top-down requirements and goals of the business and the bottom-up realities of the assets, including their condition, performance, capabilities, and opportunities. It also needs to provide enough coordinated clarity and detail that specific asset management plans (who will do what, when, where, and why) can be derived and optimized.
- **Asset management objectives:** Two levels of asset management objectives should be considered. Asset management objectives need to be defined for
  - Specific and measurable outcomes or achievements required of the physical assets and asset systems (e.g., how assets themselves need to perform), and
  - Specific and measurable outcomes or achievements required of the overall asset management system (e.g., how asset management should perform). This reflects the need for a scheme of asset management performance monitoring in the context of other corporate business priorities.
- **Asset management plan:** An asset management plan is a document that specifies activities, resources, responsibilities, and time scales for implementing the asset management strategy and delivering the asset management objectives. Such a plan needs to be optimized for cost, risk, performance, and sustainability (in line with the planning horizons for the asset life cycles and the company business plan).
- **Optimization:** This is the process required to achieve the best-value compromise between conflicting factors (e.g., performance, costs, and risks) within any nonnegotiable constraints. Optimization requirements exist in many areas of asset management, such as trade-offs between short-term and long-term goals, or between capital investment and operating costs, or between costs and risks. The processes of optimization are explained and illustrated later in this chapter.

STRUCTURE OF AN ASSET MANAGEMENT SYSTEM

An integrated system for managing assets is primarily a coordination mechanism for breaking down the silos of individual departmental roles and activities in favor of delivering the mining enterprise's core objectives. Like any other management system, however, it is a set of connected processes, controls, and enablers that ensure the right things are done for the right reasons in the right way to deliver the objectives and enable ongoing flexibility and continual improvement.

Care should be taken not to confuse an asset management system with either of these systems:

- Asset system (the functions of the physical equipment and infrastructure in production systems)
- Asset information management system (asset registers, work planning, and history-collection systems)

An asset management system is not a software tool, nor is it the hardware comprising the production or processing plant and infrastructure itself. The management system for assets, rather, is in the same category as a quality management system, a safety management system, or an environmental management system. It includes statements of policy; some clear strategies, objectives, and plans; and a range of delivery mechanisms, along with performance monitoring, review, and improvement processes.

An asset management system is particularly focused on the coordination and integration of what takes place—to break down the barriers between departmental contributions and their localized self-interests or short-term goals. This requirement emphasizes the optimization of the combined effects and value for money resulting from investments, asset usage, and asset care over the whole life cycle. It also highlights the importance of clarity, communication, and alignment between asset management activities and the overall purpose of the organization—the need to understand where the organization is going and why—before determination is made of what is therefore worth doing, when, and how. For example, a mining enterprise may have the goal of maximizing the total lifetime yield and margin from an ore body, so the development, use, and maintenance of facilities developed to support extraction should be planned with this horizon and total value in mind—not just year-by-year budgeting or the selection of cheapest equipment options (or, at the other extreme, the gold-plating of technical solutions when the incremental expenditures are not worthwhile).

Figure 9.7-2 is a high-level view of the most important attributes of a joined-up management system for physical assets. The boundary of such a system can be seen as sitting just beneath the organizational strategic plan (the company's overall business plan). This is the interface at which the language of business drivers and stakeholder expectations (e.g., safety, profit, reputation, quality) must be translated into the practical and specific implications or requirements for creation, use, care, and renewal/disposal of the physical assets or systems required to deliver these objectives.

Within an asset management system that is compliant with PAS 55, it is necessary to develop and demonstrate a logical and transparent top-down chain of connectivity from the overall business objectives to a clear definition of policy (mandated obligations and principles) and from preferred or optimal strategies and objectives to the specifics of what needs to be done to which assets, why, when, and how (i.e., asset management plans). This sounds obviously desirable and logical, but it often proves very hard in practice. The language used in corporate initiatives and priorities is often very different from day-to-day operational language; priorities and expectations change and short-term objectives conflict with long-term goals. A very important bottom-up influence must also be established and exploited—the asset portfolio's diverse characteristics, needs, risks, and opportunities should

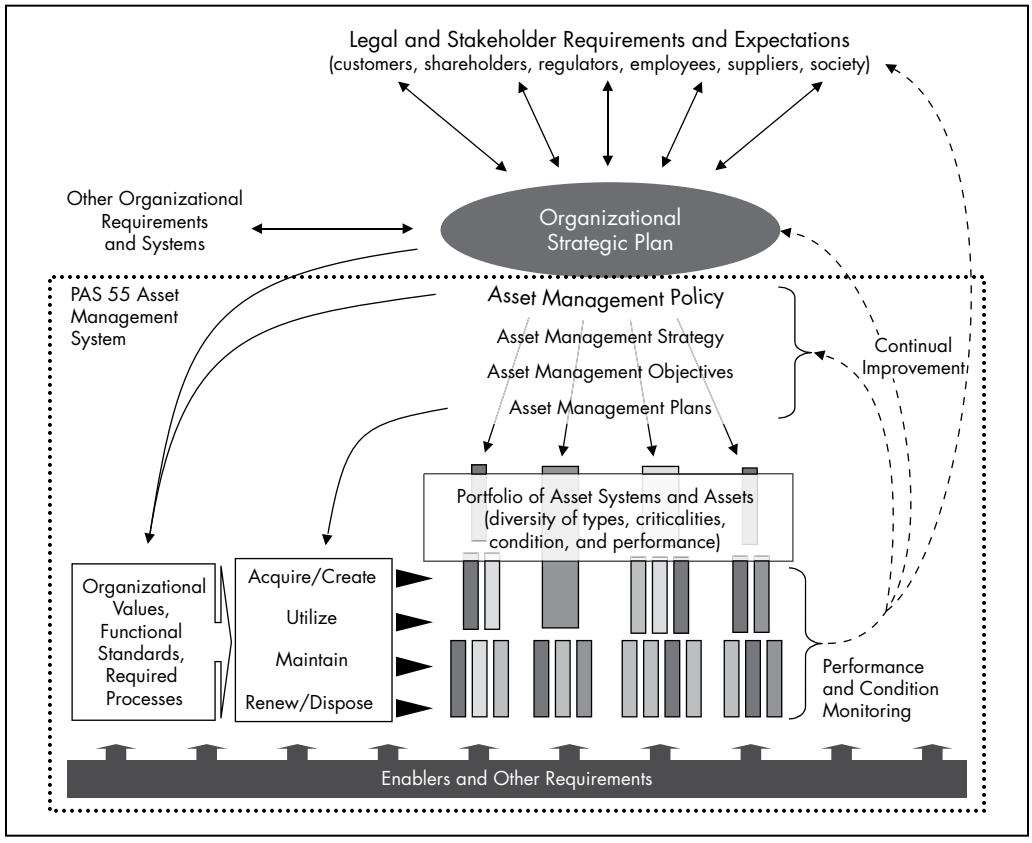


Figure 9.7-2 An integrated management system for assets

be influencing the organization's strategies and even its strategic plan. Later in this chapter the practical steps needed to establish and maintain such alignment are discussed.

Of course, asset systems and assets are diverse in type, role, importance, capability, performance, and health. So whatever the overall strategy, there is a vital optimization to be done in determining what is the best mix of investment, exploitation, maintenance, and renewals to deliver the most value for money. *Optimization* is a much overused and misused word, but it is correctly applied here—the requirement is to find the best blend of competing factors (such as cost versus performance versus risk, or capital cost versus operating costs, or short-term versus long-term results). Such optimization and decision making is at the core of good asset management, and there are many significant developments in this area—these are also summarized later in this chapter.

The right side of Figure 9.7-2 addresses the need for continual improvement, using the reality of assets and operating/maintenance experience to modify plans, strategies, the whole organization's ambitions, and even the expectations of the stakeholders.

In addition, the underlying "enablers" or lubricants of the whole management system must not be forgotten. These profoundly influence the combined performance and sustainable nature of the asset management system. Ever since the industrial revolution, businesses have increasingly specialized their activities and job functions in the search for greater efficiencies and performance. One result is the increasing difficulty for any individual or functional department to see the whole picture and understand the effects of doing something for apparently good local reasons (e.g., saving 5% in the purchasing costs for a major project) that has unanticipated performance, cost, or risk consequences on other parts of the business (e.g., performance, maintenance, or spares impact). To minimize these false economies and ensure that net value for money overrides any local, vested, or departmental interests, an asset management system must include a range of important enablers such as good communications, clear roles and responsibilities, competencies, risk management, and so on. These are discussed in detail later in this chapter.

Asset Management Processes

Business processes that are more specific can be identified in the life-cycle management of physical assets—these correspond to many of the daily functional activities in any mining enterprise. However, their integration, coordination, and optimization represent the greatest challenge and opportunity for improved efficiency and effectiveness. Figure 9.7-3 shows a simplified view of these main functions.

Seven Key Features of a Good Asset Management System

To precede an expanded discussion of the components and functions of an effective asset management system, it is necessary to recognize some underlying generic requirements. These are characteristics of the whole system and provide an important frame of thinking within which the component activities, methods, tools, responsibilities, and performance measures should be considered. As part of the development of PAS 55, the participating organizations identified seven key

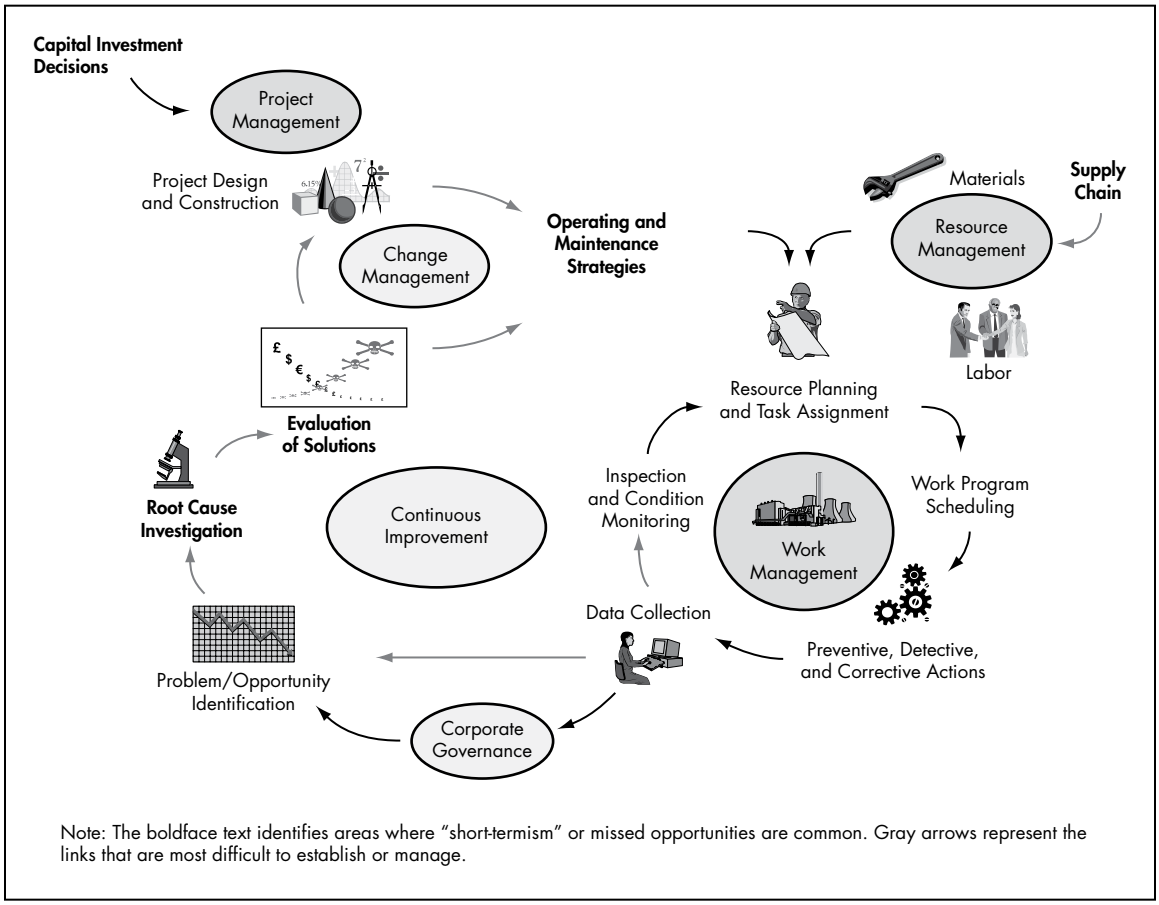


Figure 9.7-3 Typical asset management business processes

features that characterize good and effective asset management systems (see Figure 9.7-4).

1. **Holistic:** cross-disciplinary and focused on the total picture and total value.
2. **Systematic:** rigorously applied in a structured management system.
3. **Systemic:** looking at assets in their own systems context, seeking net total value for money rather than component or localized goals.
4. **Risk-based:** incorporating risk consciously and appropriately into all decision making and planning.
5. **Optimal:** seeking the best net compromise between conflicting objectives, such as costs versus performance versus risks, or short-term versus long-term benefits.
6. **Sustainable:** delivering optimal whole asset life cycles rather than artificial short-term results at the expense of long-term consequences. The management system itself also needs to be agile and sustainable in the face of future uncertainties and inevitable change.
7. **Integrated:** the importance of being joined up and collaborative. This is at the heart of good asset management; the total jigsaw puzzle needs to work, to be more than the sum of its parts.

These generic principles apply to all asset management activities—they should be evident features of investment planning, resourcing, contract management, organization structures, materials management, work scheduling, information capture and use, and a host of other business functions. They form a central feature of a maturity scale for asset management and are good indicators of the likely performance, robustness, flexibility, and sustainability of the company's core functions (if it is heavily dependent on physical assets). The IAM has, for example, developed a five-level maturity scale against all the PAS 55 requirements that is widely used for gap analysis, benchmarking, improvement planning, and so on (IAM 2002).

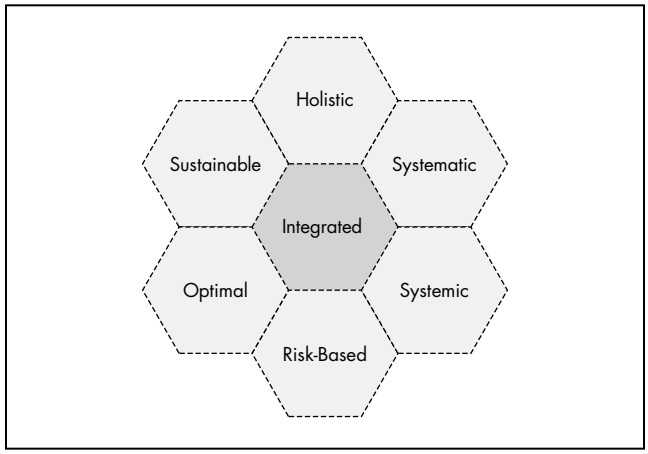


Figure 9.7-4 Seven key attributes of good practices in asset management

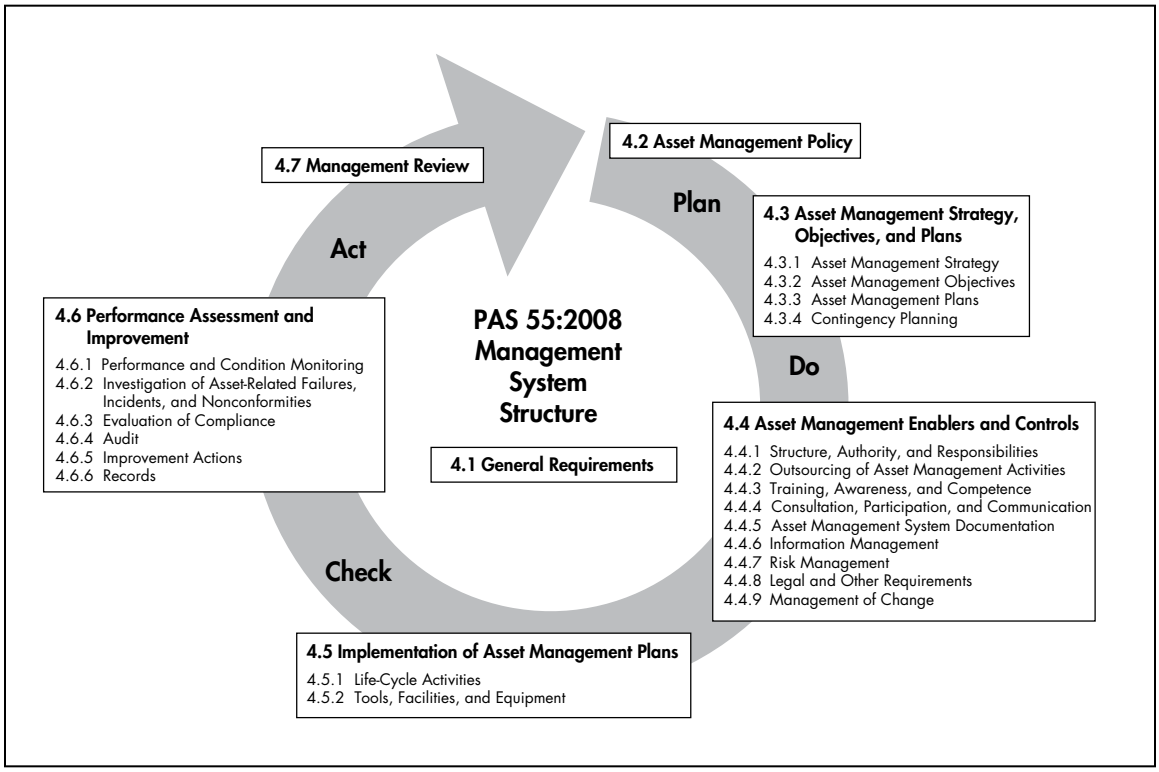


Figure 9.7-5 Closing the loop in an asset management system

SPECIFIC REQUIREMENTS FOR GOOD ASSET MANAGEMENT

PAS 55:2008 identifies a range of more specific requirements for competent asset management. These are must-do items that represent the cumulative experience of good practice, irrespective of what the assets are or what the organization is aiming to achieve. PAS 55 is not a definition of best practice, nor is it a specification for how to manage assets. What is best and what is appropriate in methods or techniques are very dependent on each organization's circumstances, such as the regulatory, cultural, technology, or business environment.

As seen in Figure 9.7-5, the requirements are grouped into the Deming cycle of plan-do-check-act. This is because good asset management is closely aligned with good quality management and the need for continual improvement. In fact, PAS 55 and ISO 9001:2008 management systems are highly complementary: A guide to this alignment (and to ISO 14000:2007 and OHSAS 18000:2007) can be found in the back of PAS 55 documents. This enables efficient, coordinated monitoring of compliance with all such requirements, with minimal duplication.

The "plan" area in Figure 9.7-5 addresses the requirement for explicit asset management policy and the derivation of corresponding strategies, objectives, and plans (including contingency plans). In the "do" area are the necessary asset management enablers and controls covering such things as human factors, risk management, and information systems, whose impact is felt in the overall effectiveness, efficiency, and sustainability of the system. Relatively little detail is needed in the area of implementing asset management plans, as this deals with the day job of developing and delivering projects, operating and maintaining assets, and so on. This reduced level of detail in PAS 55 reflects the fact that, in such areas of activity, existing capabilities and performance are often quite good already—in contrast to the bigger challenges (and opportunities) that often lie in either determining what is worth doing in the first place or in learning from experience ("check") and moving to "act" on that learning (continual improvement).

This section offers more detailed exploration of the requirements and an illustration of good practices that use the associated tools, techniques, and processes that have been applied successfully to improve the total picture.

General Requirements

An asset management system must include all the processes and documentation that define how an organization acquires or constructs, uses, maintains, replaces and/or disposes of its physical assets, including strategic planning, investments, operations, performance and risk management, and continual improvement. Top management is expected to periodically review the asset management system.

Asset management is the primary method by which an asset-dependent company delivers its main business goals—so a visible cascade of connected logic is expected to show how the strategic business plan is distilled into asset management policy, strategy, objectives, and specific plans. Figure 9.7-6 shows some of the linkage steps needed to translate corporate requirements into these specifics. Correspondingly, and shown in Figure 9.7-2 earlier, there will also be a bottom-up influence—based on the operational realities of the assets and their condition and performance—on what is desirable and achievable.

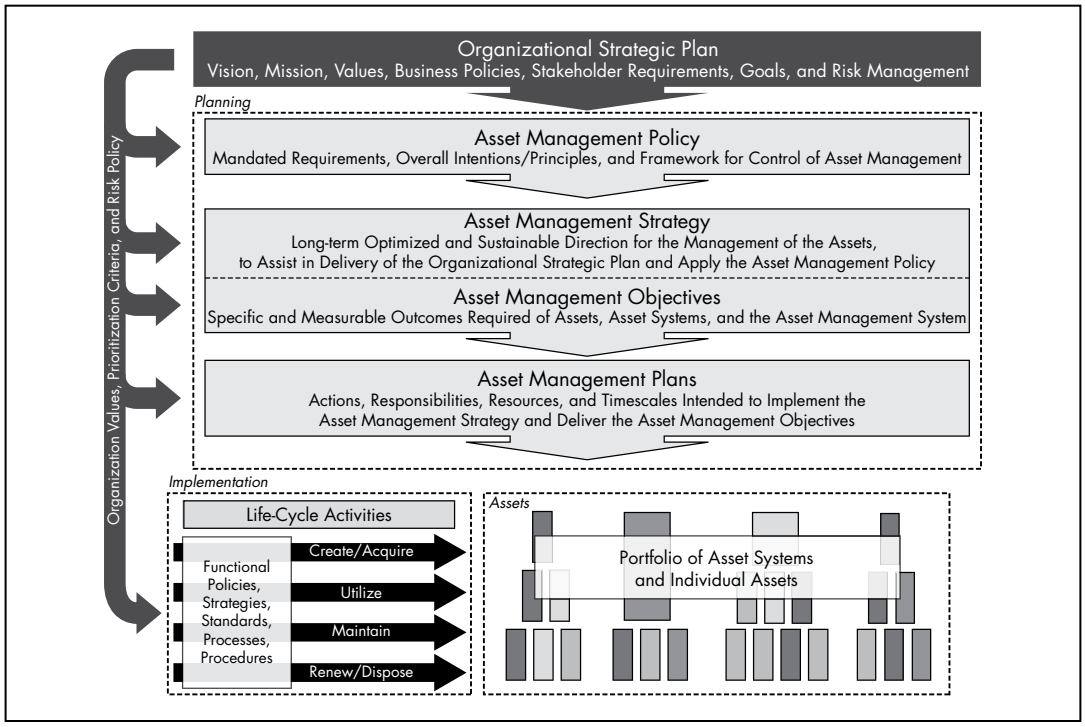


Figure 9.7-6 Transparent, logical connections between organization goals and asset management plans

Asset Management Policy

An asset management policy is a high-level document that sets out the principles, commitments, and framework of control that the company undertakes to apply in managing its assets. The policy is therefore similar in nature to a safety policy or environmental policy and is highly visible throughout the organization. An asset management policy

- Must be derived from an organizational strategic plan;
- Provides a framework for developing strategy, objectives, targets, and plans;
- Must be consistent with the risk management framework;
- Must include a commitment to continual improvement;
- Must include a commitment to legal and regulatory compliance;
- Needs to be authorized and visibly endorsed by top management; and
- Must be documented, communicated, implemented, and reviewed/maintained.

In a mining operation, for example, the asset management policy might include phrases such as these:

- A life-cycle cost basis will be used in selecting and acquiring all critical mining and primary processing equipment.
- Processing systems will be maintained using risk- and condition-based techniques in line with the organization's absolute commitments to safety and environmental standards.

Asset Management Strategy and Plans

An asset management strategy is a chosen approach for turning the asset management policy into practical reality (i.e., the preferred style, methodologies, overall long-term campaign/plan). There are differing interpretations of where and how such a strategy develops into asset plans that are more concrete and specific. However, an underlying requirement is that an adequate rationale and overall approach must be clear, from which more detailed action plans can logically be derived. A strategy should therefore have the following characteristics:

- Be long term—at least as long as the organization's business plan, and in many cases longer still (e.g., for long-life infrastructure development and whole mine life/ore-body exploitation horizon)
- Be consistent with policy, business plans, and other organization strategies
- State the required functions, performance, and condition of major assets or asset systems
- Take account of risks and asset criticality
- Contain high-level overall action plans: approximate time scales and the responsibilities/mechanisms for development of more specific asset management objectives and action plans
- Be optimized for value for money over whole asset life cycles and economic mine/ore-body life
- Be periodically reviewed and updated

Typical asset management strategies in the mining complex might include the following:

- Consolidation and updating of the asset information systems over the next 3 years
- Transition from time-based maintenance to condition-based methods for critical rotating equipment over the next 2 years using the reliability centered maintenance methodology

As these overall strategies get crystallized into more detailed asset management plans, many critical decisions have to be faced. Indeed, the backbone of good asset management

(and a source of problems in poor asset management) is usually said to be the decision making about what is worth doing, where, and when. Robust, optimized, and defensible plans will accomplish the following:

- Cover the whole asset life cycle (both in time horizon and in consideration of short-term/long-term functional requirements, costs, risks, asset performance, etc.). The term *life cycle* should normally refer to the economic production horizon of the mine/ore body or to the equipment itself if multiple equipment life cycles are likely within the mine's economic life span.
- Include capital investment, operations, inspection and maintenance, refurbishments, spares, skills, and other requirements. Some organizations only consider asset management plans to cover asset care activities (maintenance)—this misses important opportunities to optimize value by making capital investments or changing operations to reduce needs for maintenance.
- Be optimized for total value for money over the whole asset life cycle. This is a key feature in identifying the right mix of activities and in demonstrating the business justification for the costs and resources involved.
- Contain specific action plans, resource plans, timescales, and responsibilities including, in most cases, performance objectives. Longer-term plans and assumptions will inevitably be more uncertain or approximate, but visibility of these future intentions (even if they are qualified or provisional) is important to reinforce the view of the total picture. The hierarchy of plans from long-term whole life-cycle programs down to short-term tactical work programs should be transparently connected and refreshed at appropriate intervals.

One further manifestation of good asset management planning is the need to have a road map for the improvement of the asset management system itself (i.e., how to get progressively and continually better at managing the assets). Not to be confused with the asset plans described previously, this nevertheless represents an important part of the planning outputs. It requires the mining enterprise to be self-aware about current asset management capabilities and performance and to be both aspirational and flexible in recognizing the need for continual improvement. The typical development of such a road map is covered later in this chapter (see the "Asset Reference Plan" section).

Asset Management Enablers

Several factors prove particularly important in the integration, optimization, and sustainable nature of good asset management. These can be seen as generic enablers that have a facilitation effect to asset management processes. They include three principal areas of consideration:

1. Risk management
2. Information management
3. People management

All three are very big subjects in their own right, but this chapter has space only for a look at the most important features and must-do elements.

Asset Risk Management

The organization's approach to asset risk management is a strong and easily recognized differentiator between good and poor asset management competency. At the weaker end, it may be a superficial or cosmetic recognition of equipment reliability or project delivery confidence (often using purely relative terms such as high, medium, and low).

In more mature asset management models, risk management is quantitative and disciplined, and it permeates through all asset management decision making and activities. Clearly the level of detail, the appropriate tools worth using, and the business significance of asset-related risk management vary widely with the issues being considered. (Bigger risks warrant more detailed assessment, analysis, and evaluation of controls and/or mitigation.) At the very least, a competent asset management system should include an asset risk register that contains assumptions about potential failures or adverse events, their likelihood, potential consequences, appropriate controls or mitigation strategies, and accepted residual risk levels. A matrix is also commonly used to identify any cases of unacceptable risk, in which a control or mitigation action is required whatever the cost, or cases in which the cost/risk trade-off can be considered to reduce risk to as low as reasonably practicable (ALARP; see Figure 9.7-7). The scales of probability (or event frequency) and consequences vary considerably among organizations, business sensitivities, and regulatory/cultural environments, as do the chosen boundaries for unacceptable/ALARP/negligible risk.

Closely linked to the subject of asset-related risk management is the concept of asset (and activity) criticality. This, too, is a large topic, as there are usually several manifestations of importance and criticality to the business. A good asset criticality system will define the scales of importance in various dimensions, several of which may involve elements of risk (e.g., safety, environment, reputation, profit/cost), to enable a logical and auditable ranking of issues, opportunities, and activities. It will also align directly to the choice and use of key performance indicators: measures of asset management success that demonstrate achievement of the important business values and goals. The shamrock diagram in Figure 9.7-8 shows some of the complexity of considering all the interconnected and conflicting factors: The outer ring represents typical business drivers, while the inner five elements identify the available methods of scaling or quantifying the significance and contributions of those business drivers.

Asset Information Management

As mentioned in the introduction, asset information systems are sometimes confusingly called asset management systems. Even when this terminology is sorted out, there are some significant challenges and opportunities in correctly specifying, implementing, and using systems for managing asset information. Many organizations invest millions of dollars in such systems with the aim of improving efficiency, yet a large proportion of such companies do not then exploit the technologies beyond the basics of work planning and control, inventory management, and (mostly financial) reporting. Most such systems are modular and aim to support the following functions:

- Asset register and technical details storage
- Maintenance and inspection task specifications
- Work planning and scheduling
- Spare parts and inventory control, often interfaced to purchasing systems
- Maintenance and reliability history records

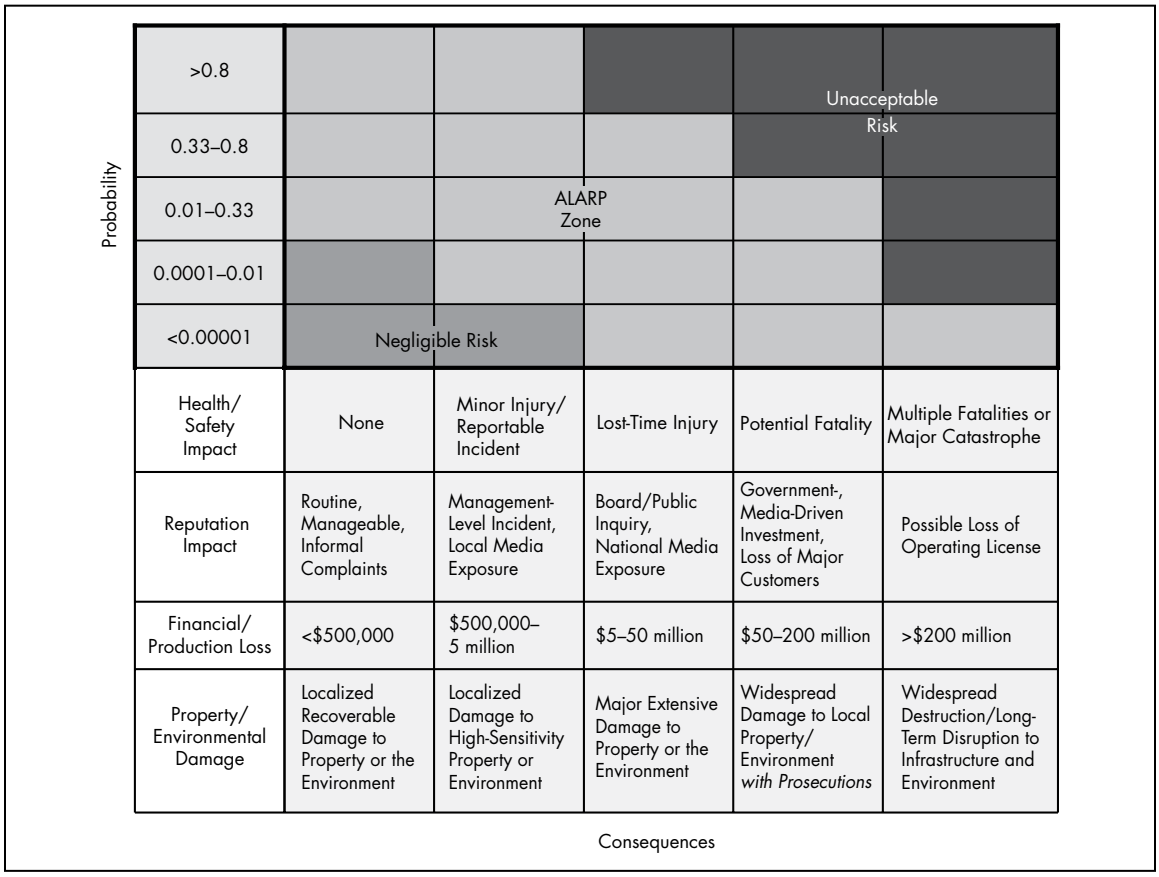


Figure 9.7-7 Risk matrix for asset-related failures and their potential consequences

- Reporting and performance analysis, often interfaced to financial systems

Asset identification and asset registers comprise the minimum starting point for such systems, and even here, great debates occur about appropriate coding and data models—for tracking individual equipment items that may perform different functions in different locations, or may be mobile, or may be embedded parts of complex systems. The key requirement is usually to identify assets by two attributes:

1. What type of asset it is (a parent/child hierarchy of equipment families, equipment types, models, components, and subcomponents)
2. What functional location it fulfills (a systems engineering view of the function provided by the asset)

The next important issue is that of granularity: Information should normally be collected and assigned at the lowest component level of an asset or asset system that requires discrete replacement or maintenance actions. For example, if the mine's maintenance strategy was always to run a particular motor to failure and then replace it, there would be little point in recording information (and procedures) for replacing the individual bearings within the motor. The normal level of detail required, therefore, is at the level of the maintainable unit. Sometimes this is referred to as a maintenance-significant item, and it is also the basis for equipment identification or tagging. However, information collected and recorded at a lower component detail may be worthwhile for the identification of failure modes and the diagnosis of failure root causes.

Information relevant to asset management normally includes the following:

- Unique asset identification numbers, often combining asset type and functional role codes
- Description of assets
- Functional and geographical locations, possibly obtained by using geographical information systems, and consideration of their static or moveable characteristics
- The criticality of assets to the organization
- Details of ownership and operations/maintenance responsibility (especially where assets interface across a system or network)
- Engineering data, design parameters, and engineering drawings
- Details of asset dependencies and interdependencies
- Vendor data (details about the organization that supplied the asset)
- Commissioning dates and data
- The condition and duty of assets, including any operating limits or constraints
- Condition and performance targets or standards, including key performance indicators
- Asset-specific standards, processes, and procedures
- Life-cycle asset management strategies, covering operations maintenance, spares, life expectancy, and obsolescence/renewal/decommissioning

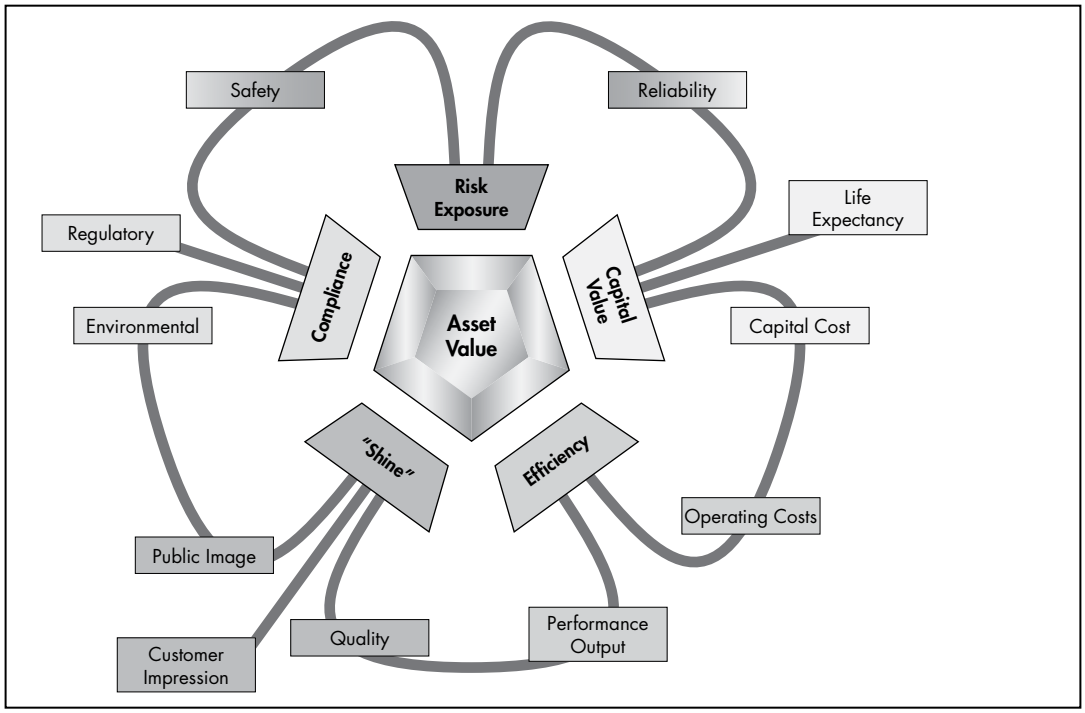


Figure 9.7-8 Shamrock diagram of conflicting business drivers and dimensions of asset value/criticality

Human Factors: Roles, Responsibilities, Competence, Communication, and Teamwork

One of the most important lessons in achieving good management of physical assets is the realization that focusing on the people who design, build, operate, and maintain such systems yields the most sustainable, biggest improvements. If people are taken care of properly, they tend to want to take care of the physical assets properly.

Work-force understanding, motivation, collaboration, communication, and good leadership are thus core and critical to optimizing physical asset management. For example, whereas a traditional competencies program ensures that a mine worker is able to perform required activities safely, productivity and engagement is increased greatly if that individual understands why the task is needed in the first place. Leading asset management organizations clearly demonstrate that ideas for improvement mostly stem from the work force (see Figure 9.7-9), so education and consultation, communication of purpose, and cross-departmental teamwork yield disproportionate benefits. This includes relationships with contractors and service providers.

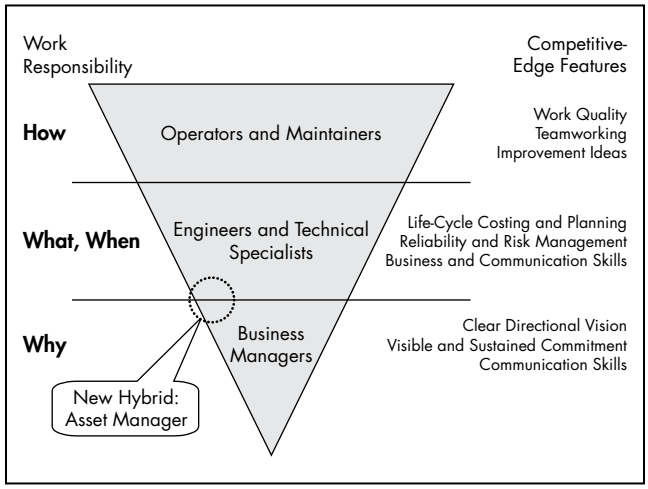


Figure 9.7-9 Support structure for ensuring the right contributions to optimal asset management

Implementation of Asset Management Plans

The requirements identified in PAS 55 for implementing asset management plans are fairly brief but they are clearly crucial—there is no point in developing sophisticated systems and optimized work plans if the delivery is poor or out of control. Yet despite the cumulative experience gained from decades of doing the day job, mining operations still suffer from operational performance and efficiency frustrations. In many cases, these occur through insufficient coordination and communication or underinvestment in competencies. Conflicting performance measures also add to the problem by reinforcing departmental silos, in turn causing inefficiencies and waste in some organizations. Each of these issues is encountered in all of the life-cycle stages of assets—from design to procurement or construction, operations, maintenance and renewal, decommissioning, or abandonment. In such cases the solutions often lie in better attention to the asset management enablers.

Monitoring and Continual Improvement

Repetitive errors and inefficiencies are also encountered through inadequate closing of the loop in capturing information and actively learning from experience. Engineering-led organizations such as mining enterprises can be very conservative or even arrogant about existing practices and ways of working. One of the challenges in asset management, therefore, is to establish and sustain the habit of looking for and evaluating improvement opportunities and acting on

them—that is, closing the loop. This represents a requirement for continual change and it exists at several levels:

- The individual assets and their life-cycle costs, reliability, efficiency, and longevity
- Mining, treatment, and other production systems—new methods, profitability, safety/environmental impact, sustainability, etc.
- The asset management system itself (processes, resources, and enablers)

The primary checklist for ensuring that such learning is taking place covers the following areas:

1. **Performance and condition monitoring** of the asset management system, the assets, and the asset systems. These should
   - Be criticality-targeted against the key business drivers (the shamrock diagram in Figure 9.7-8 provides a good check that these have been considered);
   - Be SMART (simple, measurable, available, relevant, and timely);
   - Cover leading and lagging measures (warning signs and post-event learning);
   - Look internally (e.g., asset condition and performance measures) and externally (e.g., identifying new-technology opportunities); and
   - Capture information sufficient for all legal compliances, problem identification, and root cause diagnosis.
2. **Identification of failures, concerns, trends, nonconformances, and improvement opportunities.** Formalized root cause analysis methods are usual for major incidents; however, there is also significant benefit to be had from developing a more general culture of inquisitiveness and learning at the day-to-day level: asking "Why?" to any observed problem rather than simply continuing to accept and cope with it. This style is not, however, compatible with a blame culture in which employees are driven by accountability and fear of failure. It takes time and visible behavior change by management to build trust and establish an environment of innovation and creativity.
3. **Identification of appropriate solutions or improvement actions.** There are various methods for encouraging lateral thinking in finding solutions; for example, reliability centered maintenance (see the "Reliability Centered Maintenance and Failure Modes and Effects Analysis" section) uses seven key questions to determine what type of maintenance activity is appropriate to particular equipment failure modes. One further and essential step in this process is the objective, risk-based evaluation of the potential improvement(s) in value-for-money terms. One-off changes (e.g., design modification) require different cost/benefit/risk appraisal than cyclic activities such as changed maintenance intervals or timing of asset replacement. Business skills are often a problem in technical staff, so special care is needed to ensure that processes and tools are appropriate for such evaluations and optimizations (see the "Cost/Risk/Performance Decision Making and Optimization" section; also see Chapter 2.4).
4. **Implementation of corrective or preventive action.** Even when the right changes or improvements have been identified and justified, their implementation is, of course, essential. Both the immediate actions and the confirmation of their effectiveness form part of the closing of the loop. Verification of assumptions and benefits is vital, both as ammunition to demonstrate that such innovation/change is worthwhile and as a source of further learning. For example, what proportion of anticipated improvements is actually obtained—and if not 100+%, why not?
5. **Record-keeping and documentation updates.** A sample of engineering drawings and other technical documentation often reveals that many are out of date and have not been maintained in line with the asset or management system changes. No continual improvement activity should be deemed complete until such documentation has been corrected. This is not just a matter of system integrity but is increasingly a matter for legal culpability in some jurisdictions: Out-of-date documentation can be used in a court of law as evidence of management's lack of control.

Auditing and Management Review

One advantage of an integrated asset management system is the forcing of a holistic view. Another is the requirement for regular reviews and refinement. The three-yearly certification cycle and annual surveillance requirements of PAS 55, for example, stimulate a regular practice of standing back and reflecting on the total picture. Indeed, the standard requires a formal periodic management review or audit of the whole system to ensure that it is still fit for purpose, efficient, and effective. In addition, some of the PAS 55 requirements inevitably overlap with other audit requirements, such as ISO 9001, ISO 14000, and OHSAS 18000. So it makes sense to coordinate the reviews of these systems to avoid duplication of effort.

Top management is explicitly required to participate in such periodic reviews of the overall system—this will involve senior mine management and any corporate board/parent organization. If physical assets are core to the performance of the company, then their optimal development, exploitation, and care are also critical to the success of that organization.

ASSET MANAGEMENT BUSINESS MODEL

A truly asset-centric approach to realizing business value requires an operating model that identifies the core sources of such value (the assets) and then builds activities, resources, and plans to maximize this value. The first step, therefore, is to be clear about what the real assets are.

Defining the Mining Assets

To run a successful mining business, an ore body must be found or acquired and developed, and mining and extraction facilities must be constructed, operated, maintained, and ultimately decommissioned. A wide range of ancillary services must be obtained (e.g., transport, catering, information technology [IT], telecoms, materials purchasing, recruitment, and training); these too require physical infrastructure; a wide mix of personnel, data, and knowledge; licenses; supply chain relationships; and logistics systems.

As with the North Sea oil production assets, a mining operation has to be managed as a viable business contributor or profit center. It has to include many specialist elements and yet deliver the best combination of performance and value across the whole system life cycle. To do this, the oil sector discovered that all costs and resources must be clearly measurable in relation to the resulting net outputs; the business unit (or asset) boundaries therefore needed to be as clearly defined as possible. This usually corresponded to the treatment of each

discrete hydrocarbon reservoir, with its associated production infrastructure and operating staff, as an asset business unit—albeit a very complex one, with lots of very different component subsystems and equipment types. Similarly, a typical mining operation really includes the following:

- An extraction operation
- A mineral treatment facility
- A by-product treatment facility
- A water supply and wastewater treatment plant
- A compressed air and pumping facility
- A power station or substation
- A smelter or mineral export facility
- A control and communications center
- An airport and transportation service
- A hotel or accommodation facility
- Workshops and stores
- A clinic and medical treatment facility
- An emergency control center

Such diverse component asset and system types, operational processes, people, and skill requirements comprise the dedicated asset resources, whose costs are fully attributed to the asset as a business unit. Similarly, it is the net total output of all these systems and processes working together that represents the asset's performance. This challenges and breaks down the old (strongly departmental) culture and goals, whose output measures can reinforce silo behaviors and generate internal conflicts and missed opportunities (e.g., production versus engineering/maintenance or inventory control versus maintenance/reliability).

Intermittently required resources and common services across multiple-asset business units (such as corporate finance, human resources, IT functions) should be treated as shared service providers. Some will be positive enablers, adding value to each asset at a shared cost (such as recruitment, training, and information systems support)—and these functions must be able to demonstrate competitive advantage over the alternative of using external service providers. Others may need to exert control and alignment obligations (such as common budgeting and performance reporting systems). Natural tensions do therefore exist between central command-and-control activities and individual asset needs, opportunities, and sense of local ownership. However, with clarity about the asset hierarchy and appropriate levels of delegated authority and performance measurement, the total can yield more than the sum of the parts.

Asset Manager

In mining, each asset business unit requires a specific individual with personal accountability for all costs and resulting outputs over the economic life span of the asset. Such a single point of accountability has proven to be a vital part of the business model: It involves equipping a specific person with the freedom to optimize the mix of capital investment, operating expenditures, short- and long-term production benefits, and compliance/risk management—in return for assuming accountability for the resulting net value. Clearly, such a critical role requires not only detailed asset knowledge (of all the systems and subsystems) but also commercial acumen and leadership, communication, and risk management skills. Given the strong engineering/technology and financial/management polarization of traditional recruitment, education, and career paths, however, the required cross section of skills in one person is uncommon. BP, for example, identified 20 natural asset units in the U.K. continental shelf portfolio but only 13 individuals with the requisite range of skills to become the mini-business CEOs (asset managers) to lead them. Accordingly, smaller assets were bundled together to create just 13 asset business units until such time as additional multiskilled asset managers were developed. In a mining enterprise, the corresponding role is usually fulfilled by the mine manager—with similar critical dependency on leadership, business and communication skills, and intimate asset knowledge.

Asset Management Team

As described previously, the team leader or asset manager must have clear, single-point accountability for the capital and operating expenditures and for the performance results for the asset. He or she holds the formal, signed mandate that defines the decision freedoms and constraints, asset management objectives, and responsibilities for managing the asset as a mini-business.

The asset manager is supported by the heads and/or nominated representatives of each function or department. For a typical mine, this would include the following:

- Production
- Mineral resource management
- Exploration
- Engineering and maintenance
- Human resources
- Logistics
- Finance
- Other focal-point personnel who are responsible for critical service providers

The asset management team is thus a cross-disciplinary group of managers who share responsibility for assisting the asset manager in optimization and delivery of the asset reference plan. They work with an overall asset "dashboard" or similar performance scorecard and have an obligation to seek the best overall value or asset performance irrespective of localized or departmental vested interests.

Asset Reference Plan

The asset reference plan (ARP), also called the whole life asset management plan, is a cross-functional, integrated picture that considers long-term and short-term issues and opportunities and how they affect performance, cash flow, resources, and profitability of the asset and its support activities and stakeholders. Strengths, weaknesses, opportunities, and threats (SWOT) that can affect the asset during its operational life are systematically considered, and management plans are documented. The ARP provides a shared and consistent basis for quantifying and optimizing activities for best net value to the parent organization (asset owner).

The ARP contains key information such as the asset strategy, base-case plan, and life-of-mine estimates (based on the asset strategy). Most importantly, it contains references to sources of more detailed information such as production forecasts, resource requirement assumptions, predicted dependency on critical services, SWOT analyses, and sensitivities to possible changes. The ARP also documents the main assumptions and uncertainties, and it summarizes the growth and development opportunities for an asset business unit. In short, it is the single source of truth that represents the collective understanding of what the asset is, what should be done to

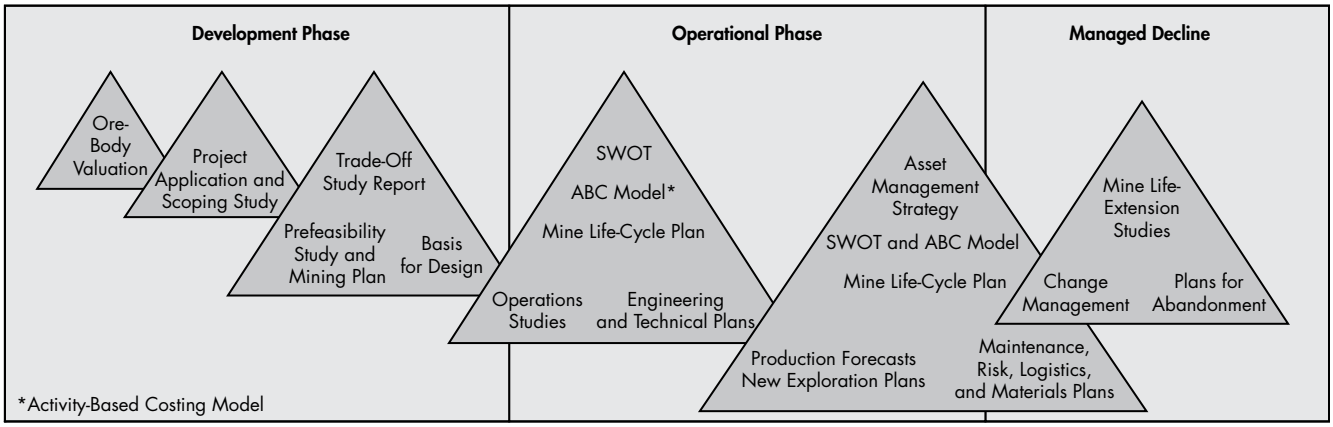


Figure 9.7-10 Evolution of an asset reference plan during asset life-cycle stages

maximize its whole life value for money, and how this maximized and sustained value will be ensured.

An ARP is the binding document that translates corporate objectives into asset-specific realities, opportunities, and plans. It has the following elements:

- Clarifying the nature, boundaries, and characteristics of the asset
- Developing and defining the whole life asset management strategy
- Planning for long-term exploitation and production (the asset utilization view)
- Integrating planned actions, responsibilities, assumptions, and uncertainties (including the asset investment and asset care/maintenance views)
- Identifying growth opportunities and risks and how they will be managed (projects, innovations, contingency planning, and alternative scenarios)
- Understanding and optimizing the life-cycle costs and production values associated with these activities and opportunities (the net value-for-money view)

The ARP is used to

- Define the boundaries of the asset at which cash flows and all aspects of performance are measured;
- Provide a clear management mandate (the responsibilities, ownership, and limits of decision making within the asset);
- Professionalize asset planning, subsurface studies, facilities design, and equipment selection by providing for better cost estimation, forecasting, and what-if scenarios;
- Integrate all process and functional strategies into a single picture, including key external dependences on service providers;
- Integrate the management team and functional contributions—all commit to a single shared purpose rather than often conflicting departmental goals; and
- Continuously assess key event dates and critical decision points in the project-development, operation, and abandonment stages of the asset life cycle (the manifestation of real continual improvement, including a record of why things are being done).

The content of an ARP evolves over time (see Figure 9.7-10) from initial design stage with its many assumptions and uncertainties to infrastructure construction and commissioning, steady-state operations, and ultimate decline (including any life extensions, reuse, and/or abandonment). Nevertheless, this living document acts as a maintained central register of the knowledge, assumptions, and plans for the asset, pooling the contributions of all relevant parties and functional disciplines. Success in any one area of activity is only measured in terms of the net impact on the total picture.

Questions That an ARP Needs to Answer

The ARP must contain a clear description of the asset and its characteristics (including SWOT); the strategic development and production plan; the key events, risks, and assumptions or uncertainties; and the component resources and functional activity plans. It is therefore the single source of truth for asset management. It should answer the following vital questions about the asset:

- What is it, and what condition is it in, today?
- Where are we going with it? (Commitments, strategies, objectives, challenges)
- Why? (Corporate business drivers and policies, asset SWOT)
- What needs to be done, by whom, when? (Life-cycle plans)
- What value will this generate? (Economic and other stakeholder views such as safety, customer satisfaction, compliance, reputation, etc.)
- What are the alternative options, risks, and contingency plans?

These elements map very closely into the requirements that have been incorporated into PAS 55, and they also align with the elements of asset management documentation that have since emerged in other industry sectors.

What Goes into an ARP?

One of the important lessons of the oil and gas asset management experience is that creation of the ARP is itself a valuable catalyst that leads an organization toward the necessary changes of focus and behavior. For example, long-term consideration of resourcing, investments, and cost-effectiveness is forced by the need to construct a whole life production (asset utilization) and a view of the asset's financial and economic life. All relevant activities, assumptions, and cost elements (e.g., engineering design, operating scope, financial

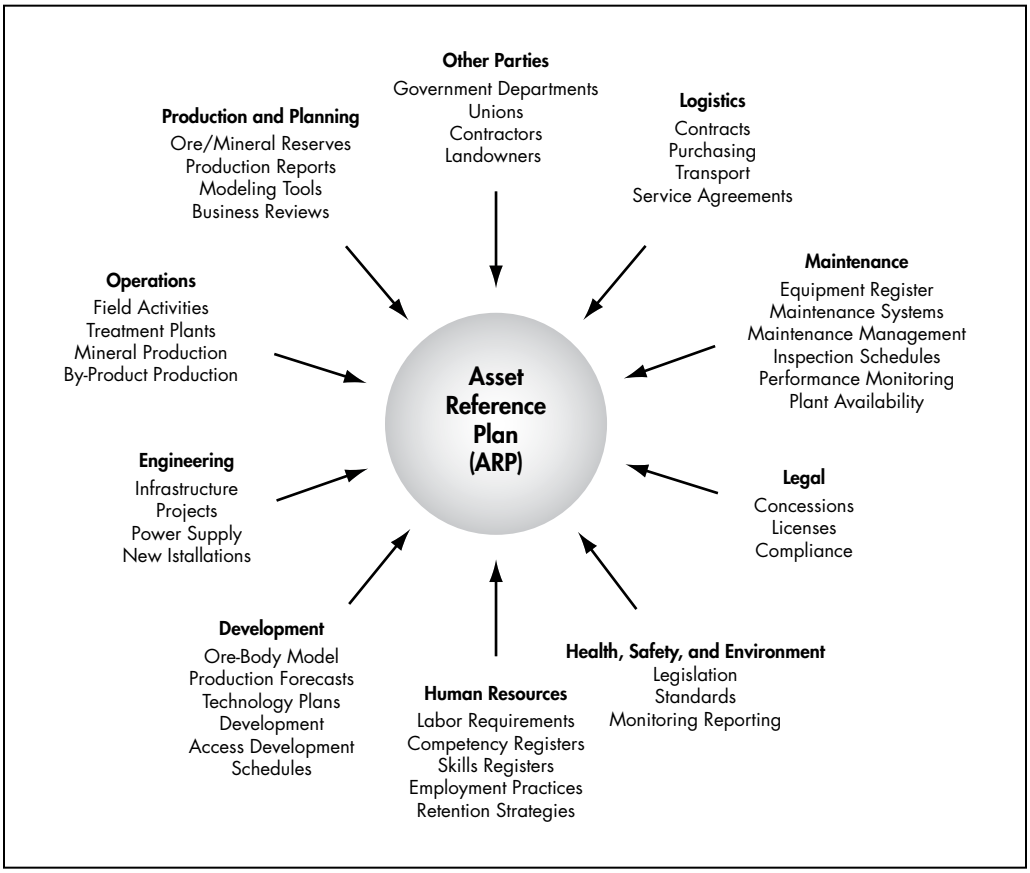


Figure 9.7-11 Inputs to the asset reference plan come from nearly all areas of the business

values, maintenance requirements, logistics, contractors, and assumptions about materials) are needed to construct such a jigsaw puzzle, and all parties can then see the whole picture (see Figure 9.7-11).

After such a full picture is assembled (one that includes recognition of uncertainties and risk elements), both the immediate priorities and the long-term plans can be made to ensure best-value delivery. So, for example:

- Capital investment requirements can be prioritized and justified with greater credibility and relevance.
- Human resource requirements can be met with adequate development (recruitment and/or training) lead times.
- Physical infrastructure, equipment, and facilities can be developed, operated, and maintained in line with the whole life business values.
- Future technology developments can be identified, explored, and exploited more flexibly, including initiation of more focused research and development.
- New legal requirements, concession/license changes, and other constraints can be identified and mitigated more readily.
- Information and data requirements can be specified more appropriately in relation to identified needs (e.g., performance criteria or key uncertainties/vulnerabilities).

The ARP provides a consolidated view of the current status of (and future plans for) the asset at a specific moment in time. It cannot hold, of course, all the detail about all the activities—if it did, it would become totally unwieldy and unreadable and would be constantly out of date. This means that a distilled summary of each contribution is needed at the total-picture level, creating a good and useful discipline for each contributor, to encourage clear explanation of what is needed in each area, why and when, with what likely or possible outcomes (and risks). Again, the collective creation of an ARP acts as a valuable catalyst to improve individual competencies and teamwork, encouraging all to think and present their proposals in more businesslike terms and to communicate in a language others can understand.

Because the information in each area is being continuously updated, the most recent ARP still needs to be frozen at a specified point so that corresponding assumptions can be made; net value/logic can be checked; and resource implications, costs, and plans can be developed or adjusted. For example, the production potential for a new mineral or mine block, including 10%/50%/90% confidence bands, must be agreed to before the facilities requirements, exploration and access development plans, and human resources can be developed. Subsequent emergence of a new drilling technology or changing environmental constraints would change these assumptions, so the ARP content will need to be reconsidered and updated—possibly in all respects. For an oil and gas company, this involves a review of the ARP at least annually plus, of course, following any significant external event or change in circumstances. Other sectors would need to update their asset management strategies and plans on different cycles or triggers, depending on the stability of the embedded assumptions and on external business influences.

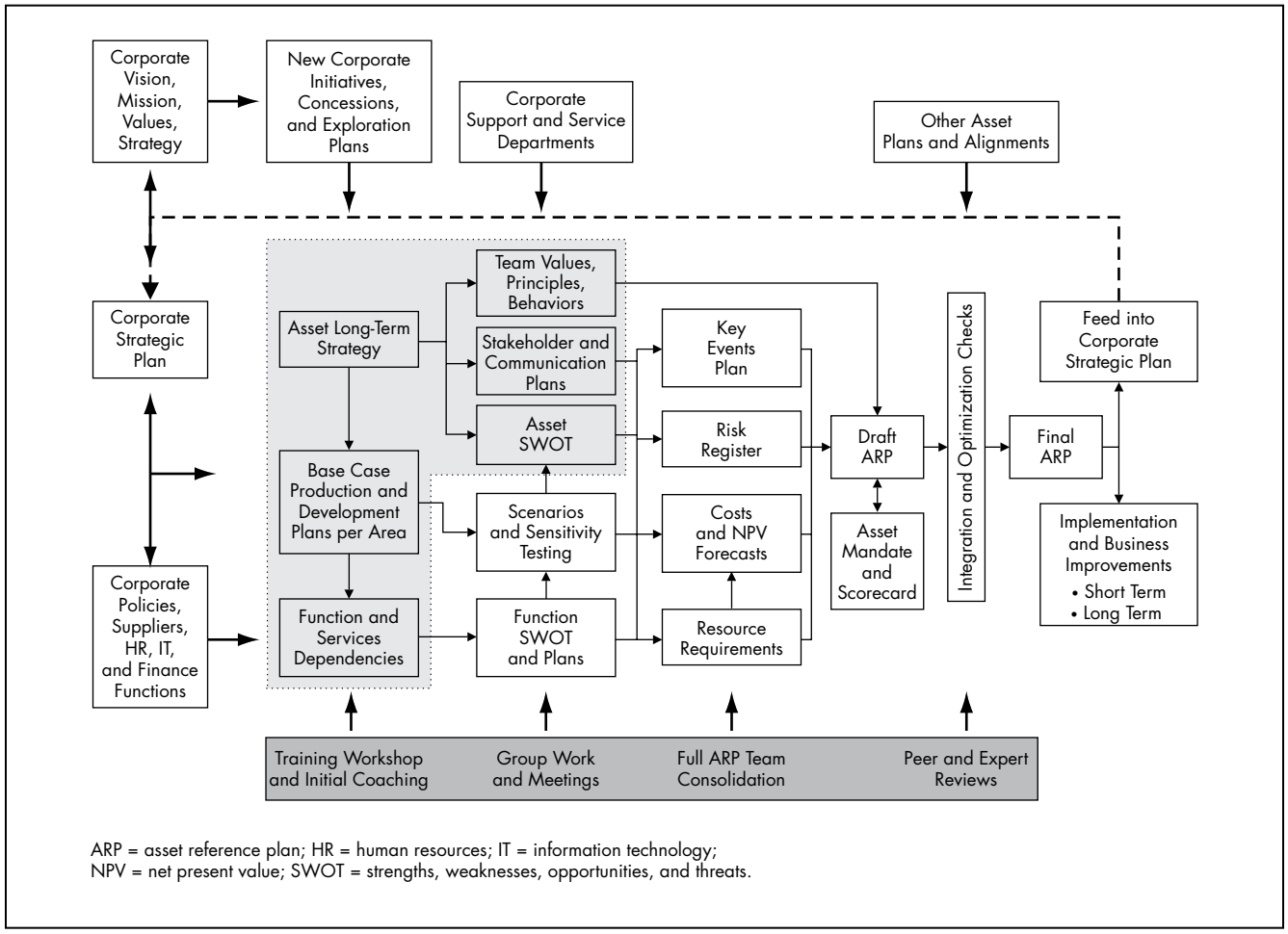


Figure 9.7-12 Development steps in building an asset reference plan

ARP Development Process

The development of a first ARP is very challenging, particularly in an organization used to working in rigid departmental silos. (These may be functional silos such as operation, maintenance, and inspection, or asset-type silos such as electrical, mechanical, or civil engineering). In such environments, staff members are often not familiar with cross-functional working, long-term thinking, or the language of business value. In the oil and gas sector and in other industries subsequently, it has been found that resistance can be reduced, and confidence built, by breaking down the development of the ARP into carefully planned simple steps characterized by good program management and leadership and allowing enough time for the development of understanding and ownership among the contributors. The first phase in particular—usually an intensive training workshop and coaching period; see Figure 9.7-12)—should involve all the key personnel who will be involved in developing the ARP and working together in the future to optimize the whole life performance and value of the asset. Thereafter, the work comprises a mix of parallel activities (each area assembling, filtering, and clarifying its contribution to the total picture) and team-based mutual explanation, integration, alignment, and iterative what-if consideration of alternatives, barriers, implications, and contingency plans.

ARP Most Likely Case and Risks/Uncertainties

The ARP most likely case, or base case, should describe the current state of the asset and future most likely development plans, including only the production and development/cost assumptions that have reasonable expectations of success or occurrence. (In oil and gas terms, this is described as P50, representing a 50% probability of achievement.) This is the baseline against which future developments, innovations, and improvement options can be tested for incremental value. The combination of such potential "stretch targets" comprises the optimistic case (e.g., P10 in Figure 9.7-13—the highest predicted performance, but with only a 10% chance of attainment). Conversely, any threats, vulnerabilities, or uncertainties about the achievement of the baseline plan are regarded as risks that need to be managed (including prevention, mitigation, and contingency plans). The least-predicted production profile is represented by the P90 scenario in which, even in the worst case, there is a 90% confidence level of extracting this performance, assuming such risks and problems are encountered. Figure 9.7-13 shows a typical production forecast example. Similar graphics might be used for the capital investment program, various operating expenditures, human resource requirements, and other dimensions of the ARP—provided that the assumptions are pooled and aligned (i.e., that the base-case expenditures, resources, and production/performance assumptions work together).

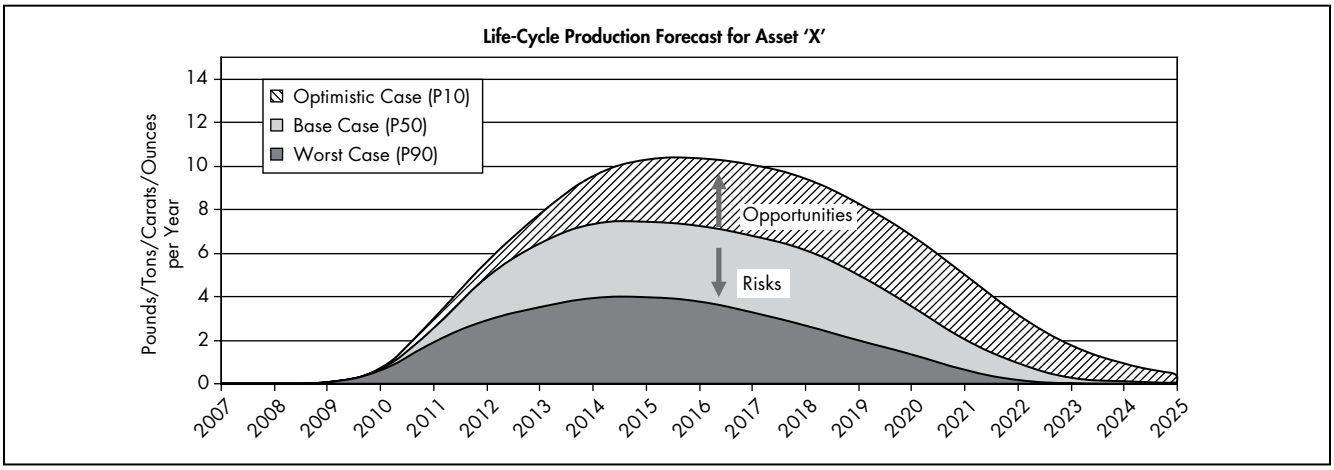


Figure 9.7-13 Performance forecasts, asset life cycle

Similar logic applies to the scope of improvement opportunities and to the identified risks.

Finally, based on many cycles of developing such a cohesive, asset-centric view of strategic plans, a key requirement of the ARP development process has proven to be the understanding and agreement of what is included in a credible base case. It is imperative that, in compiling the most likely case, only realistic and approved development assumptions are included and most likely costs estimated. ARP teams should not include immature proposals that have a low probability of success, or resource requirements/budgets that include contingencies and spare capacity. Such project proposals or budget contingencies should always be handled as possible variants from the most likely case (see Figure 9.7-13), either as improvement opportunities (with their individual cost/benefit/risk appraisals) or as identified risks (with appropriate probability and severity consideration, and prevention/mitigation/contingency justification).

METHODOLOGIES AND TECHNIQUES FOR EQUIPMENT ASSET MANAGEMENT

At the level of managing individual equipment assets and asset systems, several methodologies and good practices have evolved—many translated across from other industry sectors. The following is a short summary of the most common ones, with a brief comment about their usage and limits in the mining industry.

Life-Cycle Planning

Equipment life-cycle plans (see Figure 9.7-14) are a good way of centralizing assumptions about specific technical, management, and business issues. In many cases, these plans will start out as asset-type strategies—such as generic maintenance plans for trucks, drilling equipment, or electrical systems. The same equipment, however, can exhibit very different performance and deterioration characteristics and require very different care/maintenance if it has different business criticality or is used in different ways or in different operating environments. This is why an understanding of criticality and risk management is essential to the optimized asset management of physical assets across their life spans.

Life-cycle planning and evaluation are very different processes in the design or purchasing phase than they are in subsequent reviews and decision making. In the acquisition phase, the necessary assumptions about the downstream operating performance, costs, and achievable life spans are inevitably uncertain. Furthermore, such equipment acquisition decisions often have big, immediate cost and cash-flow implications—and the false economy of selecting the cheapest initial cost solution can result in negative consequences over the long term in the areas of performance, operating costs, or repeat investment. This is why life-cycle costing (LCC) methods are so important in this phase—to select the best value-for-money option.

Downstream refinement of life-cycle plans are, by comparison, mere fine-tuning of the equations having to do with reliability and risk, performance, cost, and life span. Only as deterioration reaches the point at which normal maintenance is insufficient, or as technology obsolescence looms, will the big price decisions need to be reconsidered. Again, LCC methods should be used to evaluate major refurbishments and life-extension options, asset replacement, upgrades, or decommissioning. Understanding of the disciplines and decision-making methods for optimal management of aging assets is still limited, but some exciting developments are on the horizon for incorporating business rigor, risks, and commercial optimization in determining what is worth spending on what and when (see the "Managing Aging Assets" section and SALVO Project [2009]).

Life-Cycle Costing

The principles of asset LCC or whole life costing are widely documented but often poorly applied, despite good intentions and the evident benefits in terms of value for money. The most important such principles are these:

- Life-cycle costs should include all capital and operating expenditures over the full economic life of the asset. They should also include all identifiable risk exposures, liabilities, and lost-opportunity costs associated with asset performance, utilization, or ownership.
- The assumed asset life should be determined based on optimal renewal timing where practicable. Calculations should be based on maximum time the asset will be required or 40 years, whichever is shorter.
- Future cash flows, risks, and performance impact should be discounted to present-day economic values using

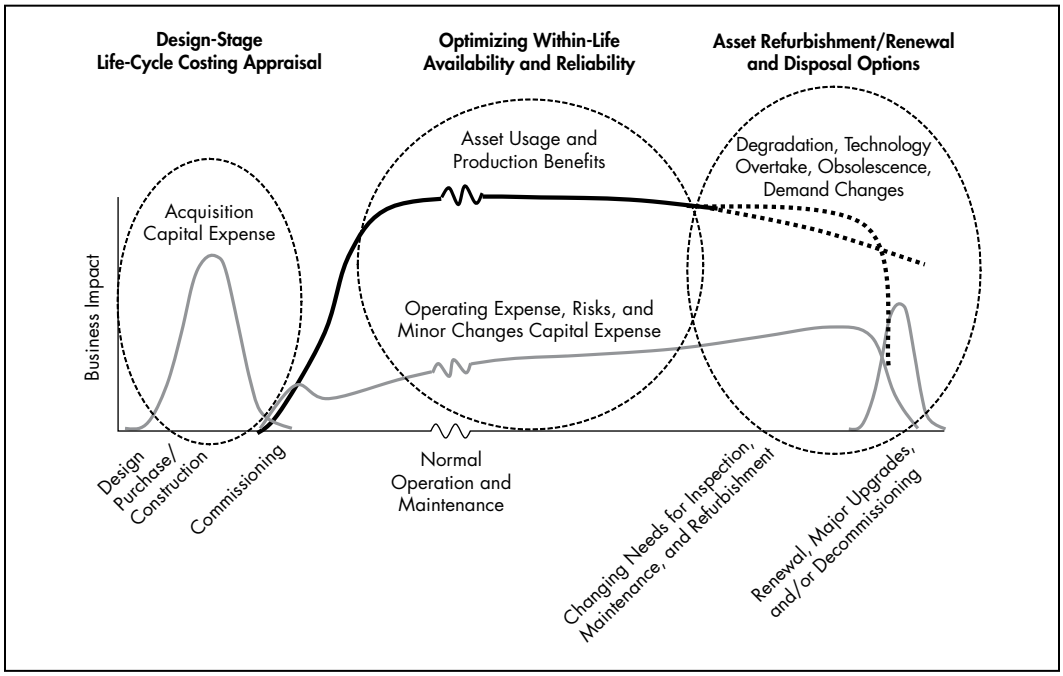


Figure 9.7-14 Life-cycle phases for equipment

standard financial accounting methods such as net present value, internal rate of return, equivalent annual cost (EAC), and process and instrumentation.
- Uncertainty in assumptions or data quality should be explored for potential impact on the asset management decision (sensitivity analysis).
- Analysis rigor, and methods or tools used for analysis, should be proportionate to the criticality and complexity of the decision being faced (see Figure 9.7-15).

Risk and Reliability Engineering

Designed-in reliability offers a valuable opportunity to eliminate maintenance and operating costs and unnecessary downtime and to achieve good economic asset life expectancy. However, the reality of many purchasing and design-stage decisions is distorted by affordability, time pressures, and "siloed" budgets. Despite this, an understanding of asset risk and reliability engineering can significantly improve the exploitable life-cycle value of physical equipment by at least guaranteeing that the right questions are asked about potential sources of unreliability, unavailability, and risk. In critical cases, it may also be worth constructing a reliability, availability, and maintainability model of a proposed system. Such a model can be used to run simulations that identify vulnerabilities in proposed designs or configuration options.

Downstream, during equipment operating life, there are opportunities to observe actual behavior and to investigate sources of unreliability based on hard evidence. The understanding of failure mechanisms and patterns is also vital at that time. The most basic of these concepts is that of the "bathtub curve"; the combination of risks associated with "infant mortality," random causes (not usage or time related), and degradation mechanisms (see Figure 9.7-16).

In practice, neat, symmetrical bathtubs are exceedingly rare (it would by very surprising to find equipment whose early-life risks exhibited the same level of threat as the old-age degradation rates). The concepts are valuable, therefore, but the quantitative treatment of such patterns needs to be considered with great care. Many academics and commercial service providers are guilty of overselling particular analytical methods for equipment reliability and maintenance data (e.g., Weibull analysis), but the lack of data volume and/or the noise in the system from other failure modes often invalidates the significance of the diagnosed patterns. In particular, looking at just one failure mode at a time in the desire for simplified analysis and understanding can result in distorted conclusions; the existence of other failure modes or planned maintenance affects the observed cases in complex ways. A few examples illustrate this problem:

- Many failure modes and effects analysis (FMEA) and reliability centered maintenance (RCM) studies—both discussed further in the "Reliability Centered Maintenance and Failure Modes and Effects Analysis" section—consider each failure mode one by one. The chosen prediction, prevention, correction, or mitigation action itself, however, may introduce other failure modes (such as maintenance or recommissioning errors). System reliability and total maintenance requirements are not just the sum of the parts.
- Analysis of the reliability and life cycles of, for instance, pump seals might involve collecting time-to-failure data for those seals. However, the existence of other pump failure modes (such as lubrication problems, overloading, damage by a foreign object, etc.) will mean that observed seal failures are only the cases that did not encounter these other problems. The degree of such censoring (or of resetting the clock through other repairs) and their implications for net reliability are complexities that render many reliability engineering studies tenuous or even invalid.

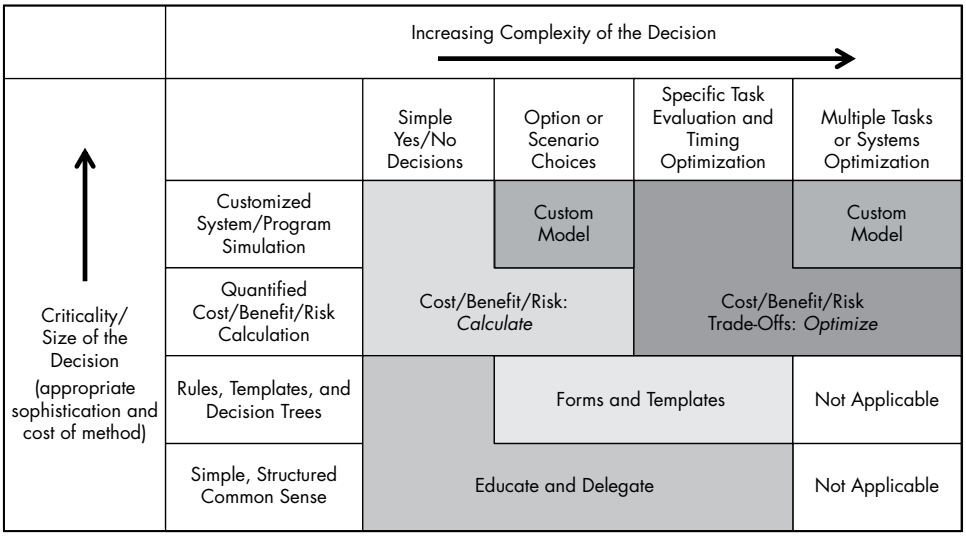


Figure 9.7-15 Level of rigor proportionate to criticality and complexity of decisions

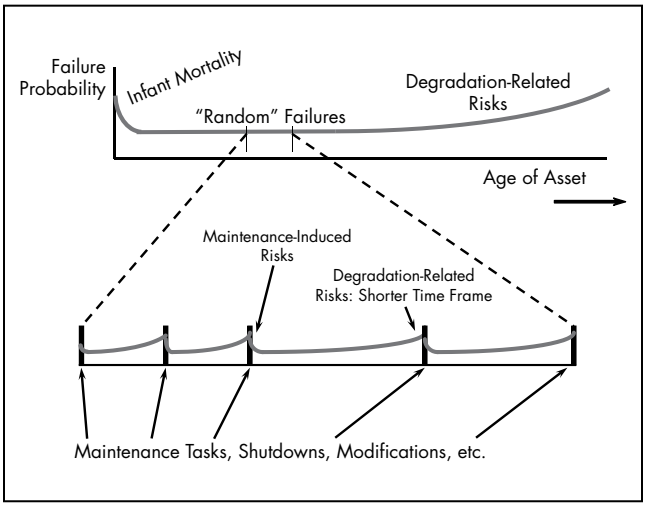


Figure 9.7-16 Bathtub curve and subsidiary hand-basin curves

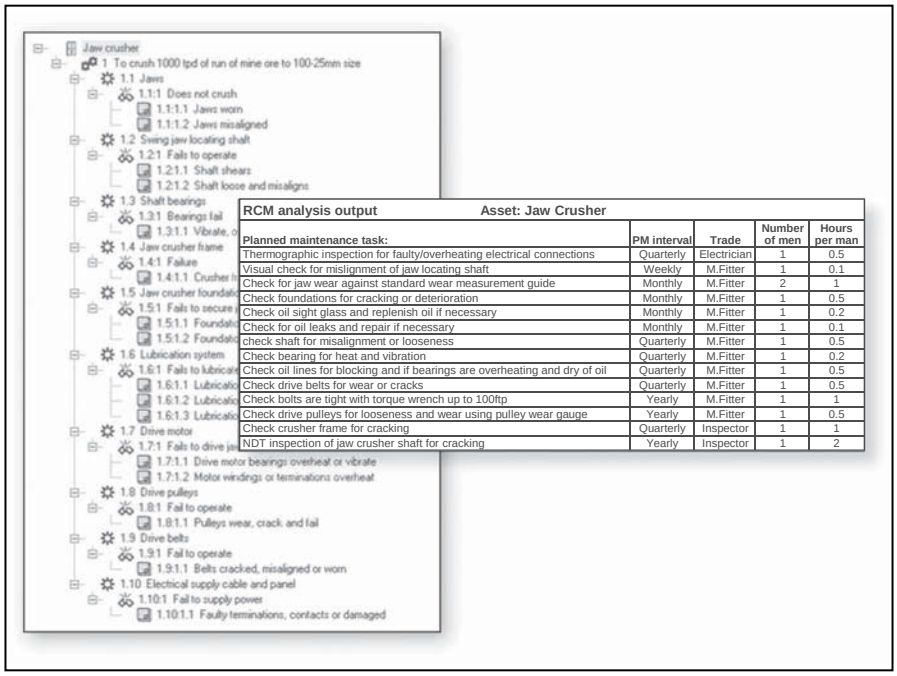
A further need for care lies in the hierarchy of components, assemblies, equipment units, and systems. What is an unrecoverable failure at one level (requiring a maintenance or renewal resetting of the clock) may be just a patch-and-continue event in the next higher level of the system hierarchy. This can lead to more confusion in the consistent recording, coding, and analysis of maintenance and reliability data. Unfortunately, few practical and robust guidelines exist for navigating this subject (an exception is www.macroproject .org [MACRO 2000]), despite the large volume of academic research and niche examples that exist. So it is important to obtain access to high-caliber reliability engineering expertise when addressing this topic.

Reliability Centered Maintenance and Failure Modes and Effects Analysis

RCM was originally developed in the civil airline industry to force early consideration of how equipment and components might fail and so the appropriate predictive, preventive, corrective, or contingency planning options could be identified. Using seven mandatory questions, it gives members of a multidisciplinary review team some auditable logic rules for determining what type of maintenance is appropriate, based on the possible failure mechanisms and consequences. This is particularly valuable when managing a complex mining or processing plant, where there are invariably many possible failure modes (see Figure 9.7-17): it provides a consistent navigation path with logical pigeonholes for predictive, preventive, detective (failure-finding), and mitigation actions.

FMEA is an older and simpler format for recording how equipment components fail, and with what effects, and for capturing the expert opinion about options to prevent, predict, or mitigate such events. In many cases, organizations use RCM for the most critical systems and complex equipment (typically about 40% of mining and processing equipment) to ensure added rigor in the process. They use a shallower, more informal version such as FMEA for less critical cases for which an RCM-level study would result in "paralysis by analysis."

However, both RCM and FMEA have their limitations—they only consider each failure mode individually and may miss some important combinational effects. They are also *reliability* centered methods, so they miss the tasks that might extend operational life (such as painting) or raise operational efficiency (e.g., catalyst changes or replacement of mill balls). Furthermore, there is plenty of debate on various flavors and derivatives of the basic techniques, such as the need for criticality prioritization, the viability of streamlining, and the need to analyze *all* equipment and *all* failure modes. The techniques also fail to determine and demonstrate that the identified solution is the most cost-effective option (sometimes an 80% solution at just 50% of the cost is better than a total solution at greater expense). Also, the majority of RCM studies during the early days of application in the mining industry were not fully implemented or sustained, usually due to lack of criticality targeting (selective usage) or inadequate appreciation of the significant training and worker hours involved. It seems that RCM programs need some aspects of total productive maintenance (TPM) to survive and deliver their full potential.



RCM analysis output **Asset: Jaw Crusher**

Planned maintenance task:	PM interval	Trade	Number of men	Hours per man
Thermographic inspection for faulty/overheating electrical connections	Quarterly	Electrician	1	0.5
Visual check for mislignment of jaw locating shaft	Weekly	M.Fitter	1	0.1
Check for jaw wear against standard wear measurement guide	Monthly	M.Fitter	2	1
Check foundations for cracking or deterioration	Monthly	M.Fitter	1	0.5
Check oil sight glass and replenish oil if necessary	Monthly	M.Fitter	1	0.2
Check for oil leaks and repair if necessary	Monthly	M.Fitter	1	0.1
check shaft for misalignment or looseness	Quarterly	M.Fitter	1	0.5
Check bearing for heat and vibration	Quarterly	M.Fitter	1	0.2
Check oil lines for blocking and if bearings are overheating and dry of oil	Quarterly	M.Fitter	1	0.5
Check drive belts for wear or cracks	Quarterly	M.Fitter	1	0.5
Check bolts are tight with torque wrench up to 100ftp	Yearly	M.Fitter	1	1
Check drive pulleys for looseness and wear using pulley wear gauge	Yearly	M.Fitter	1	0.5
Check crusher frame for cracking	Quarterly	Inspector	1	1
NDT inspection of jaw crusher shaft for cracking	Yearly	Inspector	1	2

Figure 9.7-17 Example failure modes analysis and resulting maintenance task list

Risk-Based Inspection

Risk-based inspection (RBI) provides a systematic criticality assessment of static equipment and provides for the choice of appropriate condition monitoring or inspection methods. Developed by a consortium of oil refineries into the American Petroleum Institute's Recommended Practice RP 580:2009, it is heavily focused on hydrocarbon processing (corrosion and other deterioration mechanisms, vessel and pipe materials characteristics, etc.), but cross-industry variants are already appearing. Its strengths lie in the systematic nature of the necessary survey of pressurized or containment assets, such as pipes and vessels, the "probability × consequence" view of the risks they represent, and the mass of technical data available on corrosion rates, materials properties, and inspection methods. It is notably weak, however, in determining how much to spend on the inspections or condition monitoring (e.g., inspection intervals requiring consideration of cost/benefit/risk optimization methods) and in pointing to alternative risk-treatment options such as periodic replacement or contingency planning. RBI complements RCM, therefore, by providing an expanded level of detail in the methods and criticality of condition monitoring for static equipment such as pipes and pressure vessels. Both methods, however, fail to address the business case (financial cost/risk justification) for the proposed actions—for which cost/risk optimization methods are needed (see the "Cost/Risk/Performance Decision Making and Optimization" section).

Total Productive Maintenance

TPM encourages attention to detail, shared responsibility between operators and maintainers, and a holistic overall equipment effectiveness (OEE) measure of success.

$$\text{OEE}(\%) = \text{availability}\% \times \text{utilization}\% \times \text{quality}\%$$

Emerging from the Japanese automotive industry, TPM has largely transformed the asset-responsibility culture of the manufacturing sector. Stimulating a culture of "right first time," cleanliness, and autonomous maintenance (getting the operators to do the obvious diagnosis and first-line maintenance actions), it changes attitudes and delivers many quick wins. It falters, however, on the specific tools needed to determine which tasks are worth doing in the first place (where RCM is strong), in some aspects of risk management (low-probability, big-consequence events slip through the net of OEE calculations), and in consideration of sustainability or equipment life expectancy (OEE focuses only on the short term). TPM (and the OEE measure) focuses predominantly on these main sources of loss and inefficiency:

- Downtime
  - Equipment failures
  - Setup and adjustment time
- Speed losses
  - Idling and minor stoppages
  - Reduced speed and underutilized throughput capacity
- Defects and quality losses
  - Scrap or rework
  - Reduced yield rates

Total Quality Management and Six Sigma

Total quality management (TQM) and Six Sigma (6σ) are proven and thoroughly respected bundles of continual-improvement techniques. From Deming via Japan and from Motorola, respectively, TQM and 6σ represent the push for quality in processes, for fact-based continual improvement, and for greater client focus and teamwork. They work through multi-disciplinary quality circles and improvement activities, and they are excellent catalysts for communication, clearly focused objectives, and analytical decision making. They

bring with them a host of data-centric analytical tools for diagnosing patterns, root causes, and trends. However, they are most suited to high data volumes and repetitive activities to provide the raw material for analysis. They also lack some of the teeth of the RCM and RBI methods (to link the diagnosis of a problem to the best solution, and the right amount of that solution). Again, like TPM, they are good for revealing some quick-win opportunities and, if supplemented by appropriate complementary tools, they can form a valuable framework for sustainable continual improvement.

Root Cause Analysis

Root cause analysis covers a family of methodologies (many of which are also embedded in TQM and 6σ) for investigating anything from major single incidents to repetitive equipment failures. Most companies have procedures for the former, but few have really established a "why?" culture in which employees routinely drill down to the underlying causes of observed problems to solve them properly. The basic methodologies are excellent for the fact-based consideration of why critical events occurred and for learning how to prevent reoccurrence. They force a multi-disciplinary, open consideration of the evidence and all possible interpretations or conclusions before potential improvement actions are selected. However, the sustained *habit* is difficult to establish and maintain without the enablers of motivation and recognition and a continual-improvement culture.

Example: Root Cause Analysis Used in Improvement Program

A vanadium pentoxide mine consists of three main asset systems: the open cast mine; the crushing, screening, and milling plant; and the kiln. Since commissioning 5 years previously, this particular mine had struggled to produce a profit. Major reliability problems were still being encountered, despite an RCM study that had moved the mine from a mainly on-failure (corrective) maintenance strategy to a preventive and predictive program.

Step 1. Diagnosis of the problems (root cause analysis)

Input, process, and output (IPO) mapping. The IPO method was used to identify weak links, bottlenecks, and vulnerabilities in the mining and treatment processes. Four IPO studies were performed, with the following results:

- Theoretically, the whole mining operation should have been able to produce profits and have a sustainable output.
- The kiln was the bottleneck in the whole process.
- The crushing, screening, and milling operation had more capacity than the kiln. Because most of the equipment was in series, breakdowns frequently interrupted kiln production. Planned shutdowns to overhaul the troublesome plant feeding the kiln also interrupted kiln production.
- The open cast mine could boost output by hiring loaders and trucks at short notice, if necessary, to make up any lost output and build the 48-hour buffer stock once more.
- A vulnerability to flooding also needed to be investigated for appropriate risk management.

Review of current maintenance strategies. RCM had introduced improved condition-monitoring techniques and preventive maintenance overhauls of the jaw crusher, screens, cone crushers, and ball mill. This resulted in an improvement in uptime, but the workload for fitters, riggers, and electricians required additional staffing, and contractors were engaged to fill the resource gap. However, poor workmanship from inexperienced contractors then resulted in some recommissioning failures shortly after each start-up. The net result was that the improved output was still insufficient to secure sustainable profitability of the plant.

Review of operational flexibility (interprocess buffer stock). (1) Ball mill to kiln: The stock level of milled vanadium-bearing ore in the two existing silos between the ball mill and the kiln averaged 8 hours of kiln production. (2) Mine to jaw crusher: The stock level of mine ore between the mine and the jaw crusher was 48 hours.

Step 2. Improvement options

Maintenance strategy. Further improvement in maintenance strategy (preventive and predictive) could possibly produce further marginal benefits, but this would take 2 to 3 years to materialize.

Buffer stock. (1) Ball mill to kiln: An increase of up to 2 weeks in process stock was identified as possible by installing a bypass conveyor belt in an adjacent unused area. This could be added to the kiln feed system at short notice with a rented front-end loader to maintain feed to the kiln. Savings from this improvement could be validated and investment in a new loader considered in the future. (2) Mine to jaw crusher: An increase of 1 week in an area outside and adjacent to the pit was identified as possible.

Step 3. Implementation

Ball mill to kiln. Improvements to the buffer-stock facility did not require capital investment and were soon implemented.

Mine to jaw crusher. The additional stock of mined ore was only necessary over a period of 3 months in the rainy season. The improvement was organized to be implemented prior to the next rainy season, as a heavy rain could put the mine out of operation for a week (this occurs approximately once a year).

The net result was that the mine produced its first clear profit within the first year of implementation.

Cost/Risk/Performance Decision Making and Optimization

Good decision making lies at the heart of good asset management. Yet the priority concern of many organizations has instead often been efficient delivery—trying to do the same thing more quickly and/or more cheaply. Investment projects, for example, are pressed to deliver on time, on budget, irrespective of downstream consequences for performance, value, risk, or sustainability. Cost/performance/risk optimization, particularly over different time horizons, is a crucial part of asset management (see Figure 9.7-18). This is invariably a cross-disciplinary process, involving compromise between budget holders and beneficiaries, between technical excellence and affordability, between the interests of the customer (or public) and the asset owner. The optimization process can be complex and dynamic, often involving data uncertainties and convoluted mathematical relationships. Nevertheless, this area yields some of the biggest fruit of all—by remixing the priorities for capital investment or maintenance activity and challenging what is worth doing in the first place (i.e., doing the right things) before ensuring that such activities are delivered as cost-effectively as possible.

Optimization opportunities exist in almost all aspects of asset acquisition, development, operations, maintenance, modifications, spares, upgrades, replacements, and decommissioning. The process requires multi-disciplinary inputs of any relevant data, tacit knowledge and assumptions, math

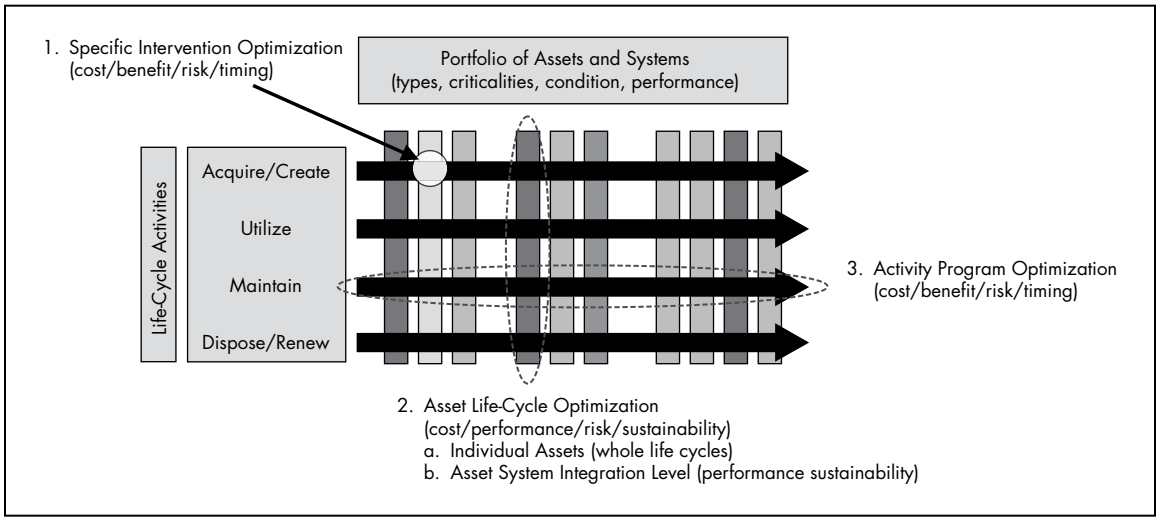


Figure 9.7-18 PAS 55 requirements for optimizing asset management activities

calculation of the business impact for different options, and the selection of the best combination (minimum total business impact; see Figure 9.7-19).

Two practical challenges are encountered in the effort to identify this optimal compromise—the lack (or poor quality) of data available and the mathematical complexity of determining how much risk (or performance, or asset life) will result from what level of expenditure. The first of these problems can be handled to a large degree by estimating ranges and testing for sensitivity: This reveals the key assumptions that have the greatest influence on the decision and the areas where future data collection should therefore be focused. Meanwhile, the correct mathematical treatment of assumptions, in determining the total business impact of different options, is increasingly viable and practical with the availability of modern software tools. For example, the MACRO project (MACRO 2000), a 20-company European research and development program carried out in the 1990s, yielded seven bundles of analysis calculators to tackle 42 types of common asset management decisions, ranging from "How many spares should I hold?" to optimal inspection/maintenance intervals, shutdown strategies, capital investment options, and optimal asset replacement timing (see www.decisionsupporttools.com).

The examples shown in Figures 9.7-20 and 9.7-21, for instance, involved optimizing major overhauls of an air compressor and determining critical spares requirements. Compared with the vendor's recommendations, this revealed a 25% longer interval between overhauls and a 15% reduction in costs and risks. Although not yet common in the mining sector, such techniques are now being successfully applied in oil and gas, petrochemical, electrical and water utilities, rail and air transport, defense, and manufacturing industries. Typical results range from 25%–40% reductions in maintenance costs, 3%–15% increases in production systems' uptime, and up to 30% savings in asset whole life costs. Furthermore, such methods are explicitly designed to work with incomplete information, using range estimates and exploring sensitivity to different assumptions. Thus they also provide a clear indication of the robustness of the decision and a quantified value for money, allowing better data to be obtained in the future.

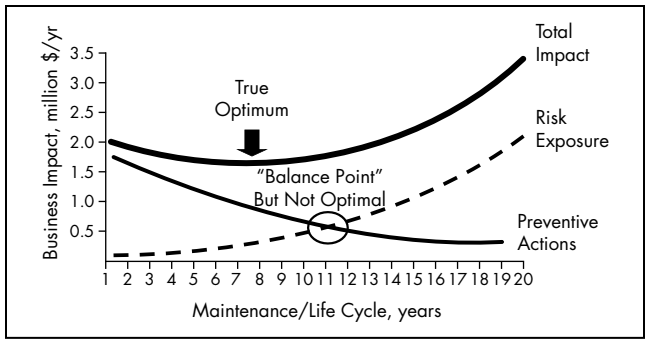


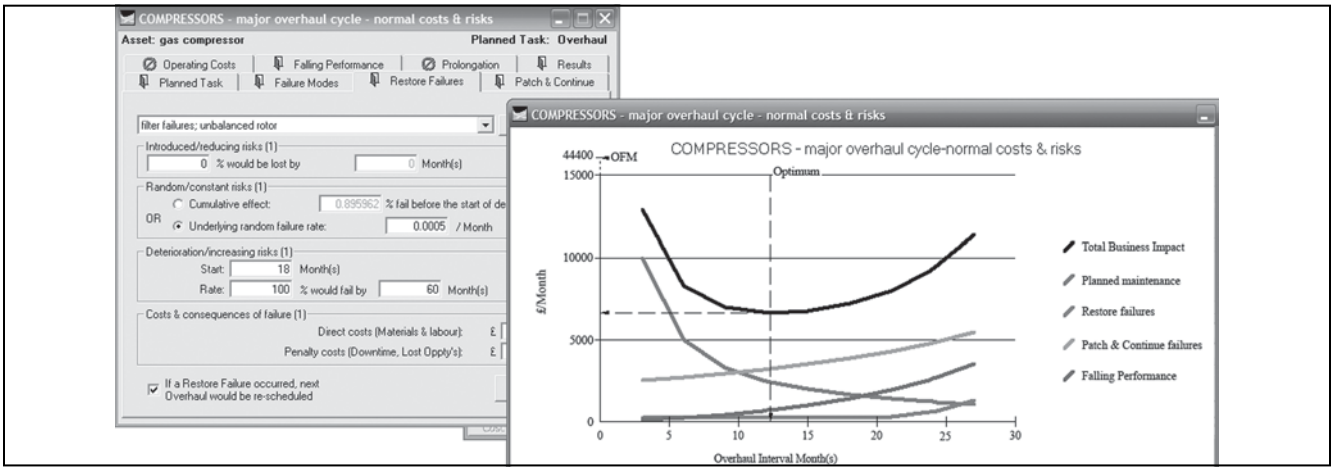
Figure 9.7-19 The true objective of cost/risk optimization

Managing Aging Assets

A great deal of attention is rightly paid to initial investment decisions, major capital projects, and capacity improvement opportunities during the life of a mine or asset. The "managed decline" phase, however, can also yield significant opportunities to influence mine performance and longevity. Secondary production and development opportunities can obviously extend a mine's profitable life span, but so too will an appropriate program of reducing operating and maintenance expenditures as the end of mine life approaches (see Figure 9.7-22). This is sometimes possible as the criticality of production systems lessens (i.e., as failure consequences are declining). Changes to maintenance intervals, nonreplacement of spares, and decisions about optimal timing for facilities decommissioning can provide significant scope for savings, thereby deferring the inevitable ultimate point of economic mine abandonment.

Similarly, as the production or processing equipment ages or deteriorates with use, changes to maintenance regimes, spares policies, refurbishment options, and equipment replacement considerations must be considered (see Figure 9.7-23).

Equipment renewal is, however, often mishandled in terms of objective total business impact: there is a tendency to push equipment to the limit, often into areas of degradation and risk well beyond the economic refurbishment or replacement point.



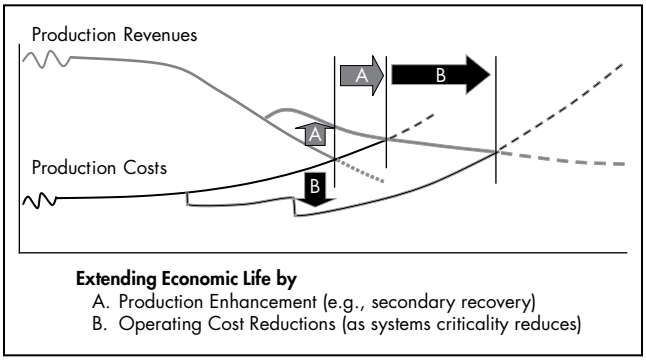
Courtesy of Decision Support Tools Ltd.

Figure 9.7-20 Optimizing maintenance intervals



Courtesy of Decision Support Tools Ltd.

Figure 9.7-21 Optimizing slow-moving spares



Extending Economic Life by

A. Production Enhancement (e.g., secondary recovery)
B. Operating Cost Reductions (as systems criticality reduces)

Figure 9.7-22 Extending mine life by increasing production and/or reducing costs

This is partly due to the perceived effect on cash flow; renewal can be a major capital expense, whereas periodic patch-and-continue repairs are individually tolerable. There are no natural trigger points that signal the need to review the cumulative cost (and projected increases) of such repairs, to review the financial value of growing risks or of degrading performance, or to calculate the total business impact for different intervention timings. The correct asset management decision, on the other hand, should evaluate alternatives: doing nothing (continued use/maintenance), refurbishing (life extension), opting for like-for-like and design change renewal (if possible), and choosing from among risk/impact mitigation options. The best choice, and optimal timing, will be identified by minimum total business impact (with appropriate discounted cash-flow considerations for intervention and expenditure timings—EAC is the most flexible financial method for doing this). Such evaluations and business-case justifications for appropriate action should also demonstrate the net impact of delay, or premature intervention, thereby quantifying the urgency or timing sensitivity for the activity (see Figure 9.7-24).

CONCLUSION

Asset management has clearly grown to become a lot more than just looking after the hardware (maintenance). During the years since 1990, it has emerged to become a core business framework for industries such as the mining sector that have a preponderance of physical assets. At a corporate level,

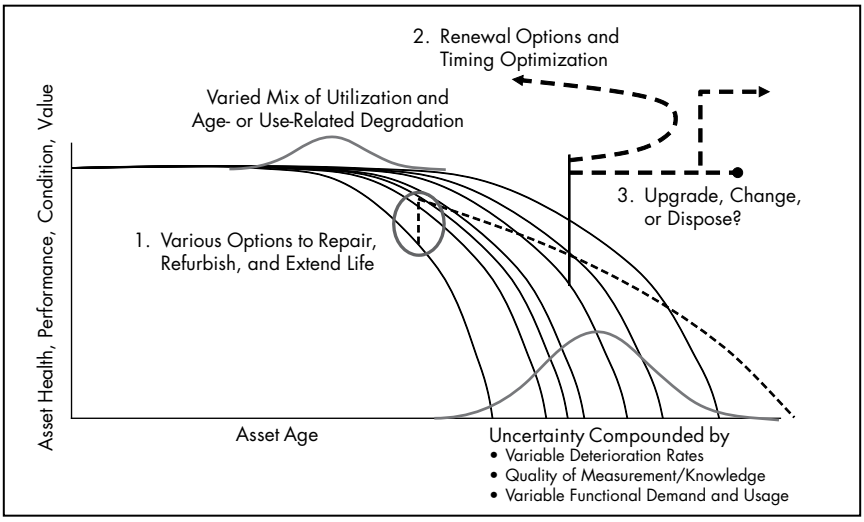
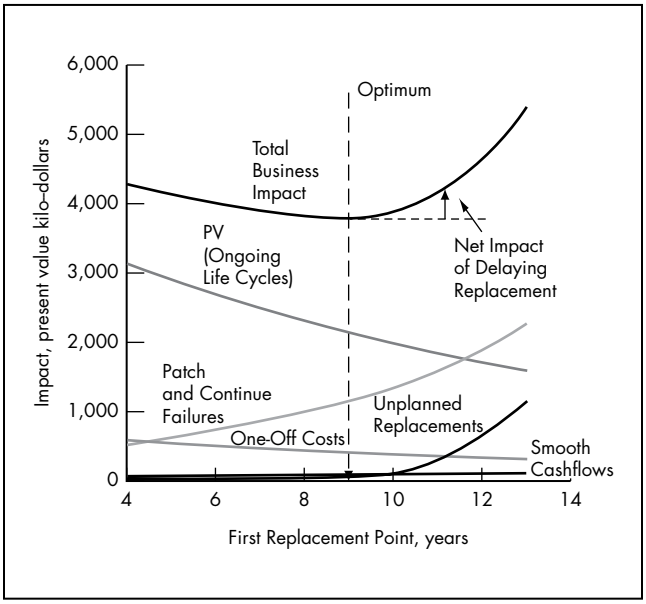


Figure 9.7-23 Options and uncertainties faced in connection with aging equipment



Courtesy of Decision Support Tools Ltd.

Figure 9.7-24 Justifying optimal asset replacement timing

it provides a model for coordinating and optimizing cross-departmental activities toward common value-for-money goals. At the level of the discrete mine, production system, or equipment, it provides language, processes, and tools for determining what is worth doing when and where. The PAS 55 standard (which is now evolving into a full ISO standard, as 21 countries have already endorsed this proposal) provides a generic and well-proven checklist of good practices, offering help in determining existing strengths and weaknesses, planning improvement programs, and demonstrating competent governance to various stakeholders. More importantly, asset management is consistently proving to yield very substantial performance, cost, and risk benefits compared with traditional siloed functional departments with their conflicting objectives.

Clearly, significant challenges are involved in adopting a joined-up, whole life-cycle asset management model, not least in the education, culture change, and leadership styles that are needed. However, the prizes are so great, and the logic (once demonstrated) so evident, that all mining enterprises can be expected to develop and/or adopt such an approach to their activities during the coming years.

ACKNOWLEDGMENTS

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CHAPTER 9.8

Automation and Robotics

Paul Lever

INTRODUCTION

The need to improve the productivity, safety, and profitability of mining is driving the widespread application of automation technologies in mines. Automation in its many forms is something that miners interact with on a daily basis but may be unaware of its existence. The current implementation of mining automation and robotics has not yet significantly changed mining processes. It has, however, begun to demonstrate its potential value to improve the productivity and safety of these mining processes and associated unit operations. This has lead to a growing vision within the industry that fully automated/robotic mining will be a major part of mining in the near future. This vision includes mines that have the following capabilities:

- Automated personnel and equipment tracking
- Automated materials handling—trucks, loaders, conveyors, sizers
- Smart drills—automated drilling of holes and recognition of material characteristics
- Accurate and automated movement and positioning of all mining equipment
- Automated mechanical mining systems
- Remote supervision from distant locations
- Intelligent and integrated control over all mining processes to optimize resource value

Benefits of Mining Automation

Why does this vision of a fully autonomous mine exist today? Traditionally, the main perceived benefits for introducing mining automation have been

- Improved safety—Removing operators from hazardous and stressful mining environments;
- Higher productivity—Through the improved performance of individual machines (more metric tons per hour) and reduced downtime (automated machines see less duty than human-operated machines); and
- Reduced labor costs—Automation removes operators from machines. Penman (2002) suggests labor accounts for approximately 20% of the cost of operating a large haul truck.

Although these benefits are still appropriate today, the current renewed interest in automation and robotics in mining is related to the following perceived additional benefits:

- Limiting operational variance. Automation allows machines to be controlled so that their output is well defined. For example, an automated shovel would load more consistent bucket payloads. This allows a truck size to be selected for the shovel that will be consistently loaded to its target payload.
- Improved precision. Automation means mining tasks are executed at their planned locations and times. For example, an automated blasthole drill will drill blastholes at their precise location and specified depth as designated by the blast pattern (Holmes 2006). This means the outcomes of blasts are more consistent and generate the desired rock fragmentation, which has significant downstream benefits.

The more significant benefit of limiting operational variance and improved precision is that they enable production consistency. This means that mine designs and, more specifically, mine plans, as well as production and schedules can be generated where there is a valid expectation that they can be achieved. Automation allows mining processes to be controlled more effectively (more like manufacturing facilities) where individual processes as well as the complete mining process can be optimized. The introduction of automation is expected to serve as a catalyst for new credos for mining similar to those that have evolved through introduction of total quality management methodologies to manufacturing.

Definitions

To understand what mining automation and robotics are and what potential impact they have on mining operations, it is important to define the basic terms:

- **Mechanized:** Operations performed by machines
- **Automatic:** Does not make decisions but completes task by following well-defined rules
- **Semiautomatic:** Partly automatic and partly manually controlled

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- **Automation:** Mining tasks completed by machines without human workers
- **Autonomous:** Functions independently without human supervision
- **Robotics:** Machines with high-level capabilities to sense and reason about their environment. Such machines are required for successful automation of tasks in high variable and unpredictable mining environments.
- **Intelligent:** Machines with the ability to learn, understand, and deal with new situations

What Makes Mining Automation Challenging?

Automation is used extensively in many other industries and has demonstrated great value in improving productivity (Gaimon 1985; Carlsson 1995). Its successful use in the manufacturing sector is in the automation of well-defined manufacturing cycles of identical components in a well-known and well-structured environment. Mining is typically conducted as a series of discrete steps, or unit operations, by groups of equipment types working in tandem, such as drilling, blasting, loading, hauling, ground control, and materials processing. Here the highly variable and unpredictable mining environment affects the successful execution of each or sequences of unit operations. Thus, automated mining systems must be able to sense, reason, and adapt to this unpredictable environment in order to function effectively. In addition, these systems need to operate for 365 days per year in very harsh mining environments. For these reasons, many existing automation technologies from other industries are not readily transferred into mining.

The growth of robotics, driven primarily by the military, has provided the tools necessary to develop autonomous mining systems. These developments have been driven by low-cost increases in computing power; new algorithms for signal processing, perception, and control; and in particular, new sensing technologies such as Global Positioning Systems (GPSs) and radar and laser systems (Durrant-Whyte 2009). These tools are essential to accurately locate, control, and coordinate the activities of robotic machines. Onboard sensors can generate real-time maps of the mine environment (geometry and geology) around these machines so they can be optimally controlled. This spatial control information is essential to plan and schedule the complete mining process (including machines). This will make mining operations more precise and predictable (like a factory) and ultimately dramatically more productive.

Fully autonomous mining that encompasses the complete spectrum of mining processes will be expensive and require significant technical development. More importantly, it will only be achieved through significant change management of mining culture to accept these innovations. Mine automation technologies will achieve the highest probability of success using a multistep implementation program. The resulting early introduction of technologies can provide mining benefits such as

- Operator aids that improve machine performance and reduce machine damage,
- Improved mine sensing systems that can provide managers with better information about the state of the mine,
- New mine models that allow managers to execute operational decisions,
- Semiautonomous and remote-control capabilities that remove operators to safer environments, and
- Collision-avoidance technologies that prevent accidents.

Mining Robotics—The Science

Field robotics is now well established in the research, commercial, and military arenas. Mining robotics is a stream or application within field robotics. The basic science (with associate sensors, computing hardware, etc.) that drives the technology in this field is not being developed by the mining industry; however, mining robotics is using and reengineering this technology. Several good textbooks and journals explore the extensive science and technology of field robotics. Buehler et al. (2010), Siegwart and Nourbakhsh (2004), Everett (1995), and Ge (2006) are excellent references for more detailed information.

Evidence of the close link between researchers in field robotics and mining equipment manufacturers is Caterpillar's sponsorship of the Carnegie Mellon University (CMU) participation in the Defense Advanced Research Projects Agency (DARPA) Grand Challenge competition. This competition tests the performance of autonomous ground vehicles in a series of different outdoor environments and demonstrates the state of the art in mobile robotics. CMU won the last competition in 2007 and has been in a longstanding partnership with Caterpillar to develop automated mining equipment (Binning 2009).

A very large body of basic research into the development of autonomous vehicles includes major themes such as route planning (Al-Hasan and Vachtsevanos 2002; Frazzoli et al. 2002; Salichs and Moreno 2000); obstacle detection and avoidance (Azouaoui and Chohra 2002*)*; close maneuvering strategies (Gomez-Bravo et al. 2001); and autonomous vehicle control system design (Pereira 2001).

MINING AUTOMATION AND ROBOTICS IN PRACTICE

This section provides an overview of selected automation solutions for the mining industry. Some are commercially available, others are being trialed at mine sites as precommercial technologies, and some are still in the research and development (R&D) stage. Detailed published information about many of these systems is difficult to find as manufacturers and mines are reluctant to release information that might hurt their competitive advantage.

Autonomous Haulage Systems—Technology

Autonomous haulage systems are generally divided into two major divisions: site-level systems, sometimes called the office site manager, and machine-level systems, sometimes called the onboard controls.

Site-Level Automation

The site-level systems provide an interface between the human (and software-based) mine planners, provide data for business tracking, and provide optimization and control over the haulage system machines. *Mine planning* is defined as determining the goals and priorities and communicating them to the mine management tools, and *mine management* is defined as optimization of existing resources to accomplish the goals.

Mine planning. This requires significant human interaction to set goals such as prioritizing production volume versus

cost factors (fuel usage, risk of machine damage) or prioritizing short-term production versus long-term mine efficiency. The challenge in mine planning is to ensure that both explicit and implicit goals are accurately translated to the more automated mine management system.

Mine management. Fortunately, there is a large body of experience with mine management systems. However, a new generation of mine management systems will be required to address the requirements of autonomous systems. The next generation of such systems will require much more detailed mine models and many new features to manage autonomous equipment.

Machine-Level Automation

Machine-level automation requires the ability to

- Understand what tasks need to be accomplished (high-level planning);
- Determine location (positioning);
- Perceive the environment based on the location (perception);
- Based on the environment and position, plan future tasks to achieve the desired goals (task-level planning);
- Execute the planned primitives (primitive execution); and
- Handle exceptions.

High-level planning. High-level planning is the ability to interpret directions from the mine management system and to select the appropriate behaviors. For example, if an autonomous truck is directed to a load area, it needs to understand the path, validate the path, and ensure that it has the required behaviors and capability to execute the plan.

Positioning. Positioning is the ability to determine the machine's pose, meaning its location and orientation. For autonomous machines, an accurate reliable pose is critical. Although absolute accuracy is actually not as critical, the ability to accurately register the machine's pose to the mine model is a requirement. Multiple types of positioning must be considered:

- For aboveground applications, the most common real-time kinetic Global Navigation Satellite Systems (RTK GNSSs) are
  - RTK GNSS + machine sensors (such as odometers),
  - RTK GNSS + machine sensors + inertial,
  - RTK GNSS + machine sensors + inertial + perception-based positioning, and
  - RTK GNSS + machine sensors + inertial + pseudolites (ground-based satellites or reference stations).
- For belowground applications, the most common types of positioning systems are
  - Radio frequency (RF) based distance measurement,
  - Perception-based positioning + machine sensors, and
  - Perception-based positioning + machine sensors + inertial.

These positioning systems are well known in the industry. Perception-based positioning systems have been used in restricted environments such as factory settings for more than two decades (Caterpillar 1991) and in underground mining applications for nearly a decade (Ferret 2003). RTK GNSS-based positioning systems have been used to provide centimeter-level accuracy in construction applications for a decade (Saghravani et al. 2009). RTK GNSSs are susceptible to poor GNSS coverage and must be augmented to provide very high levels of availability. Unfortunately, specific areas (near the poles) and areas with blockage to the east and west can have significant problems with GNSS coverage. However, governments around the world have pledged strong support to the GNSS infrastructure; and as more satellites are put in service, the GNSS coverage should continue to increase substantially. The most well-proven technology to augment GNSS for short durations is using inertial sensors plus odometry. Unfortunately, this technology only has short-duration accuracy. Other possibilities for augmentation include pseudolites, often called ground-based satellites, or RF-ranging beacons. These systems allow individual mines to establish their own ground-based positioning system, which, unfortunately, can be very expensive to install and maintain.

Clearly, the more types of independent positioning sources that are available, the more robust the positioning solutions will be.

Perception. Autonomous machines generally need a good understanding of their surroundings to accomplish object detection and sometimes assist with positioning. The perception system usually requires information from the mine model and the positioning system to accurately determine and report the location of objects. Perception sensors can come in several different forms: radar, laser, vision, and sonar (not discussed here).

Radar has been used since the early 1900s to detect ships and large metal objects. More recently, millimeter-wave radars have been developed to detect objects for adaptive cruise controls and autonomous machines. Radar has the advantages of long range and "seeing" through dust and fog. However, radar suffers from relatively poor resolution when compared to laser-based systems. The size of an object that can be detected depends on the object's radar cross section (RCS), defined as the projected area of a sphere that would return the same signal to the transmitter. The most effective means to increase the effect of RCS is to install corner reflectors. Commercial units are sold for small boats to improve their detection ability by radar. Corner reflectors can be thought of as "tail lights" for radar transmitters.

Laser systems provide the best resolution but suffer from obscurants such as dust and fog. Newer lasers overcome much of this limitation by using multiple reflections from each point or from time gating the return to ensure that small, diffused particles (dust, fog, rain, and snow) do not obscure other objects. As the density of particles increase, the effectiveness of this method diminishes. As with radar, retroreflectors (such as tail lights or reflective tape) can be added to targets to greatly enhance their visibility to laser.

Vision has significant promise for low-cost object detection and object recognition that is most like human perception. There has been a tremendous amount of work in this field and rapid progress is being made, but so far, vision is limited to relatively short-range applications. Vision also is sensitive to shadows, dust, fog, rain, snow, and reflective surfaces (e.g., a reflection in a puddle may be confused as a real object).

Combining multiple spectrums has significant advantages. For example, combining radar and laser gives the longer range and improved penetration of radar with the laser's ability to detect smaller obstacles and improved object classification. As another example, combining laser and vision allowed the TerraMax team (in the Grand Challenge competition mentioned previously) to recognize tumbleweeds in the desert that stumped a vision-only system in earlier tests (Broggi et al. 2010).

Unfortunately, there are currently no practical standardized tests by which to easily determine the capabilities of object-detection systems under all conditions.

Another consideration for the practical use of autonomous systems is that none of the obstacle-detection systems available today have the ability to differentiate a pile of small loose rocks (e.g., a windrow left by a motor grader) from a single large rock that could damage tires.

Task-level planning. Task-level planning is defined as establishing a sequence of task or behaviors to accomplish a goal and is generally rule- or constraint-based. As expected, task planning varies significantly based on the type of goal, the number of constraints, the variability of the environment, and the flexibility of the automated system. One of the common machine-level planning tasks is object avoidance.

Primitive execution. Executing planned primitives is the fundamental machine operation; some examples include drilling, tramming, loading, grading, stability control, dumping, and ripping. These operations generally require continuous closed-loop control and are probably the closest machine operation to a human skill.

Exception handling. Handling exceptions is the ability for an autonomous machine to recognize that it does not have the means or required primitive ability to handle the existing situation, and thus it resorts to fail-safe behavior—usually stopping and asking for help.

The building blocks for autonomous haulage system technology have been developed over the past two decades to the point where commercially viable automation systems for underground mining are now available, and commercially viable haulage systems for surface mines will be available within the next few years. In the near future, autonomous and intelligent remotely operated machines will provide the opportunity for remote operations centers that allow miners to work in the relative comfort and safety of an office environment. These first-generation systems should rapidly evolve to provide continuous improvement in efficiency, productivity, and machine availability.

Autonomous Surface Dump Trucks

One of the principle benefits from using fully automated dump trucks (ADTs) is the direct reduction of labor costs. Penman (2002) found that for two-axle dump trucks in the 220-t (metric tons) class, labor accounts for approximately 25% of operating costs, fuel approximately 45%, and tires approximately 30%. For the larger, 300-plus-t class truck, Penman found the approximate cost split is labor 15%, fuel 30%, and tires 55%. Other benefits include improved safety through the removal of operators from hazardous areas and reductions in machine duty (rate of accumulated damage), and fuel consumption through more consistent operation. For example, the world's first fully automated straddle carrier system at the Port of Brisbane has reduced energy costs per container move by 40% (Kalmar Industries 2007).

The enabling technologies (navigation, truck control, and collision detection) for ADTs exist in a semimature form, and prototype ADTs integrating these technologies have been developed and tested. Commercial ADT systems are not yet available. Of the major haul truck manufacturers, Komatsu and Caterpillar have both developed and demonstrated autonomous dump truck technologies.

Komatsu currently has two sites operating its ADT technology, called FrontRunner. Codelco, at its Gabriela Mistral mine in Chile, has been operating 11 autonomous haul trucks since 1997. More recently, Rio Tinto, at its West Angelas mine in Western Australia, has been operating five trucks since late 2008 (Cribb 2010). Both sites are using 930E-4 electric-drive trucks with a payload of 300 t, and the trials are intended to test autonomous trucks in high-production scenarios. In both cases, the ADTs are not working on the same haul roads with manually driven trucks.

Komatsu's FrontRunner autonomous haulage system operates as a comprehensive fleet management system for mines. The haul trucks are equipped with vehicle controllers, high-precision GPS, obstacle-detection technology, and a wireless network communication system. The system has the capability to navigate a haul route, dump automatically to hoppers or to the ground, and work with some (but not all) loading equipment. The system leverages off several mature technologies, notably the GPS and inertial navigation systems for navigation, and millimeter-wave radar and laser systems for safety and collision detection.

These technologies are integrated using largely fixed automation strategies. Trucks navigate from a so-called pit database, which serves the purpose of a haulage map and contains largely geometrical information, notably the boundaries of the haul road, the truck's travel path, and the boundaries of the loading and dumping areas. The boundaries represent the extent of the ADT's allowed safe operation and are established by driving a light vehicle fitted with high-precision (differential) GPS around the haulage perimeter. The safe-working-area boundaries in load areas are updated as the shovel moves. As an ADT is manually driven along the required haul route at the required speed, the truck's control computer records position, speed, and direction, defining the truck's travel path. The truck's control system (steering, braking, engine, and dumping functions) has the truck "replay" this path to navigate from the load area to the dump zone and back again. The use of the teach–replay approach used in current ADTs simplifies the problem of planning the truck's course or path. The truck doesn't need to plan its path; it knows its path having learned it in the training run. The repeatability of the replayed paths under this approach is high, but it is necessary to add "dither" (a small error in truck path control so that the truck does not always follow the exact same path on the road for every trip, thus causing wear over a broader surface of the road) to the steering action to distribute road wear.

A central control computer manages the ADTs on a particular haul. Each ADT continuously communicates key data (position, speed, and heading) back to the central control computer, which provides general haulage management including tracking each truck's trajectory and anticipating collisions. An onboard safety and collision-avoidance system based on millimeter-wave radar and laser-sensing technologies is used. This system has an overriding authority to bring the truck to a stop if an obstacle or safety hazard is detected.

Automated Underground Loading and Haulage

In underground mines, the restricted environment in which equipment operates has aided in its automation. In addition, some of the greatest potential hazards to miners occur when operating in and around underground mobile equipment. This was an early driver to automate primary underground mobile materials handling equipment. Load-haul-dump units (LHDs)

and trucks operating underground follow well-defined routes in repetitive operating cycles. These, coupled with a well-structured underground operating environment, made deployment of early autonomous vehicle technologies feasible. As early as 1988, King (1988) predicted a positive economic return for the use of semiautonomous LHDs. Caterpillar Global Mining (2008) describes recent trials of automated LHDs at the Malmberget mine in Sweden showing an increase in productivity of between 10% and 20%.

Autonomous LHDs and underground trucks are now available from several manufacturers. These are the Caterpillar MineGem system (Caterpillar Global Mining 2008), Sandvik's AutoMine system, and Atlas Copco's Scooptram Automation system. At this time, more than 10 mine sites worldwide are operating or plan to operate either or both autonomous LHDs and underground trucks. These include De Beers' Finsch diamond mine in South Africa (Faurie 2007), Codelco's El Teniente mine in Chile, and the Stalwell gold mine in Australia (Caterpillar 2007).

Both systems have the underlying premise of removing the operator from the machine to a remote operating station, where the machine can be teleoperated. This station could be located underground, on the surface at the mine site, or at some distant location. Operation requires onboard cameras, computer control of machine functions (steering, braking, acceleration, bucket motions [LHDs], etc.), high-speed network communications, and associated safety systems. However, simple teleoperation is stressful for operators and less productive than manual operation.

The solution is operator-assisted automated steering during teleoperation. Here the operator does not physically steer the system but views the machine's location in real time on a mine plan (typically on a computer screen) and then uses a joystick to give the machine the direction of travel. In Caterpillar's MineGem system, this is called copilot mode.

Finally, in autonomous mode the remote operator provides a goal for the machine (LHD or truck) and the self-guidance system controls the vehicle. In Caterpillar's MineGem system this is called autopilot mode. Typically, the operator fills the LHD's bucket in remote mode and then enters autopilot mode to tram to the dump point, empty the bucket, and return to the loading point. Thus, in autopilot mode, several machines can be operated by a single operator. For safety reasons, remotely controlled and autonomous machines need to operate in exclusion zones that strictly control access.

The key to operator-assisted automated steering and autonomous navigation for LHDs and trucks is the use of a laser–radar range-sensing system. This system maps the location of objects (typically walls) around the machine and determines their location by comparing the measured profiles to an existing database provided from the mine map. This technique is known as simultaneous localization and mapping. Thus, knowing the location on the map relative to the nearest walls, the control system can drive the LHD to the desired location.

Automated Dozers

Dozers are typically used for a wide range of applications in mining. These include profile construction, cleanup around other machines, ripping, and utility work. Thus the level of complexity required to automate a dozer is very much a function of the individual task needs, such as the following examples:

- Adjusting ripper depth and angle, and dozer speed and direction, to ensure good ripping performance is key for an automated ripping dozer. Here, interaction with other machines is often limited.
- In many cases, dozers create structured profiles for other machines to work on or from. Here, material movement accuracy may be important.
- Dozers that undertake utility work such as dragging pipes and cables, moving power boxes, and pushing scrapers may have dedicated sensors and automation capabilities. Here, the dozers require the ability to rapidly move from one application to another.
- Interaction with other machines, and detecting where and what to clean up, is important for dozers required to undertake cleanup tasks. An automated dozer would have to detect when and what to clean up and interact with other machines (shovels, trucks, etc.) in a variety of modes.

The automation of dozers to remove operators from machines appears feasible for dozer operations where interactions with other machines is limited, and complex sensing and recognition of subtle changes in the machine's environment are not required. These cases do not require development and use of complex sensing systems, advanced automation infrastructure, or control of complex machine–material interactions.

Automated dozers are not commercially available today. Several original equipment manufacturers (OEMs) and third-party providers have GPS-based dozer operator aids. These include, from Caterpillar for example, "an in-cab display that gives operators easy to understand color diagrams of where to cut and fill. The system uses onboard computers, software, data radios and centimeter-level GPS receivers to constantly monitor work and update the plan" (Caterpillar 2006b). Remote-control solutions have also been available for dozers for many years. These are typically retrofits by third parties.

Research into better solutions for remote-control dozers continues. A recent project funded by the National Institute for Occupational Safety and Health in the United States developed a remote vision system for dozers on coal stockpiles. The project involved Consol Energy and Caterpillar as participants. This work is described in Schiffbauer et al. (2007).

Many of the enabling technologies (dozer control, positioning, and navigation) for automated dozers have been used in remote-control applications for some time and therefore exist in a semimature form. It is expected that automated dozers with the complete capability to build profiles, clean up, and rip will be achievable in the next 5 to 10 years.

Dessureault et al. (2007) and Holmes (2006) describe an automated dozer project at Freeport-McMoRan's San Juan (Arizona, United States) test site. The target application for this dozer is ripping. The dozer sensors include GPS, Sick lasers, and cameras (for teleremote operation).

Automated Blasthole Drills

Automated blasthole drills (ABDs) have the potential to generate a number of benefits. These include

- Reductions in machine duty (rate of accumulated damage) through smoother computer-controlled operations, and

improved blasting performance through more accurate and consistent drilling;
- Reduced labor costs; and
- Improved safety through removal of the operator from a hazardous and dusty environment.

Automation can also optimize drilling performance through control of bit loading, drill rpm, and torque parameters. Increased penetration rates, lower bit wear, and reduced drill costs will result. It is also desirable that holes are drilled at locations specified by the designed blasting pattern. Inaccurate holes can lead to poor blasting fragmentation, inappropriate muck-pile shape for loading, and so on.

ABDs also have the potential to generate benefits that include implementing drill-based, high-level rock recognition systems to provide detailed rock type and structure information that can be used in blast design. Consistent drill operation through computer control of the drilling process enables these systems to operate effectively.

Thus, the enabling technologies (navigation, drill control, and remote control) for ABDs have existed in a reasonably mature form for some time. Prototype ABDs integrating these technologies are currently under development. Holmes (2006) describes the development in 2005 of an automated blasthole drill called the Advanced Rotary Drill Vector Automated Radio Control (ARDVARC). Holmes states, "Preliminary results of ARDVARC trials as compared to normal manual drilling operations has shown a +15% productivity gain, improved pattern and hole quality, and lower machine duty cycle. Additional gains achieved from hole-to-hole positioning are expected." The technology was tested on a BE49R drill but has been transferred to the Atlas Copco PV271 drill.

Rio Tinto has also been developing an automated blasthole drill that is currently operating at its West Angelas iron ore mine. (The outcomes of this project are described in detail later in this chapter.) This machine is designed to drill automatically in both rotary and rotary-percussion modes. In addition, the drill is capable of performing rock recognition during the drilling process in order to build a rock mass description of the area to be blasted. This description is then used in the blast-design process. This drill also has full video remote-control capabilities.

The typical attributes of autonomous blasthole drills currently under development are

- Automatic drilling of the hole from collaring though pipe removal when either manned or autonomous;
- Navigation from hole to hole within a drill pattern;
- Completion of drill pattern autonomously including accurate setup, relocating holes (due to hole failures or impossible collaring), inclined holes, and drill bit changes;
- High-level task instructions from a drill-and-blast management system;
- Monitoring of self-condition (e.g., airflow and pressure, oil pressures, rpm, component failures) and appropriate reaction (e.g., report bit failure back to drill-and-blast management system);
- Monitoring of drill parameters (thrust, rpm, vibration, and other active and passive geosensors) and using this information for rock-recognition information that can be fed into a drill-and-blast design system; and
- Video remote control of all drilling functions.

Remote Operations Centers

A remote operations center (ROC) enables supervision, control, analysis, and data acquisition from afar. In simplest terms, an ROC can be regarded as a platform for enabling process automation and business integration. An ROC enables

- Enhanced occupational health and safety by removing operators and maintainers from risk exposure;
- Reduced labor costs by relocating high-cost, knowledge-intensive labor away from mine sites to urban centers;
- Increased productivity through
  - Identification of inefficiencies at operating interfaces,
  - Collaborative planning between functions (operations, maintenance, and procurement),
  - Sharing of experience and knowledge across mine sites,
  - Process visibility along the process chain; and
- Potential to lock in benefits through knowledge capture and reuse.

Implementing an ROC provides an important catalyst for change within an organization. Current workflow patterns must be evaluated and modified or adapted. Implementing ROCs therefore provides an important stimulus for changing work practices and driving an organization to achieve higher levels of labor productivity.

Interest in applying ROCs is growing within the mining industry. Examples of early adopters within the mining industry are Rio Tinto Iron Ore (RTIO) in Western Australia and Freeport-McMoRan in their Arizona (United States) operations.

Schweikart (2007) outlines the business drivers for RTIO's implementation of an ROC:

- High cost of supporting remote staff
- Desire for increased staff retention
- Business integration
- Faster, better decisions

The ROC will also supervise the autonomous systems operating at RTIO's West Angelas iron ore mine site. A fleet of five autonomous 320-t Komatsu haul trucks are operating at West Angelas, as well as one automated blasthole drill rig (Trounson 2007).

The removal of the need to maintain a large work force on-site saves ancillary camp expenses; remote working allowances; and fly-in/fly-out expenses, which can account for significant costs. Additionally, integration of operations (site, transport, and infrastructure) in one center removes "silo mentality," where people are focused only on the component of the operation they are involved in and fail to see the effects on the whole mining process, creating bottlenecks at interfaces.

Improved telecommunications infrastructure, in particular telephone networks and fiber-optic links, are cited by Dicker (2007) as key technology enablers for the ROC development. RTIO is evaluating the remote support of maintenance personnel via head-mounted displays and wearable computers. Support would make extensive use of video conferencing, remote desktop access, and voice-over IP technologies. RTIO is also evaluating remote collaborative planning via the use of augmented reality (virtual reality based on real sensor information) and touch tables (Schweikart 2007).

Schweikart outlines a 4-year plan to achieve remote-integrated operations. The first step of this plan is to integrate disciplines (planning, operations, and maintenance functions) at site level, and to begin the transfer of people from site to ROC. The second step is integration per asset (site operations integration) through the transition to remote operation and deployment of operations models. The third step is integration across sites. Because of safety concerns and relatively static operating environments, it is likely that ROC technology will be implemented first on fixed assets, such as beneficiation plants.

RTIO is investing particular attention to aspects of change management and reevaluation of workflow. Scweikart emphasizes that changing to a remote supervision and operations model necessitates a change in the way that work and information flow is managed, and that replicating existing workflows may not improve productivity.

Coyle and Holmes (2007) indicate that the installation of Freeport-McMoRan's ROC on-site was justified on the elimination of inefficiencies across operational boundaries. The ROC is also seen as playing an important role in "institutionalizing the knowledge" of maintenance and operations experts. Here, the vision of the future has tactical operations personnel co-located in a "room of truth." Verified information is made available on a real-time basis so that subject-matter experts can assist less-skilled operatives in the field. Crosspollination of ideas and shared operational awareness are forecast to lead to significant efficiency gains for the organization.

Automated Digging

Large-capacity shovels and hydraulic excavators (called diggers in this section) with bucket capacities to 70 m^3 are critical production units at most open-cut mine sites, and there is an ongoing imperative to improve their productivity. Automation is seen as one of the strategies by which improvements can be realized.

The automation of diggers presently stands at an interesting nexus. Most of the technology needed to realize the automated digger exists (albeit at varying levels of maturity), and developments in other sectors, including industrial robotics and the automotive and aerospace industries, are delivering further advances that enhance technical feasibility.

However, the cost and risk of introducing automated diggers to mining operations currently outweighs the perceived benefits. An automated digger must operate as part of the overall operation of a mine with implications on mine infrastructure, operational practice, work-force skills, and site culture. The gap is significant and no mining company is actively pursuing the introduction of automated diggers to their operations, notwithstanding the several significant mine automation initiatives taking place worldwide (e.g., Rio Tinto's Mine of the Future at the West Angelas mine site).

The transition to automated diggers will likely occur over the next 20 years. It will almost certainly be staged through incremental stepping-stone technologies that, in themselves, bring productivity and reliability benefits and allow the risks to be understood and controlled and the technology to mature and be proven and accepted in mining environments.

This section attempts to identify, at a fairly high level, automation capabilities that might serve as planks in the bridge to realizing the autonomous digger over the next two decades.

Capabilities of the Autonomous Digger

The autonomous digger

- Excavates material and loads trucks with minimal human intervention;
- Plans and executes repositioning moves using high-level mission statements referenced to the mine plan;
- Feeds up-to-date information garnered from onboard sensors back to the mine plan including terrain information, material diggability, and productivity indicators;
- Manages the dig face and floor to optimize productivity and maintain favorable bench structure;
- Automatically executes ancillary functions including management of the trailing cable (on electric shovels), machine park-up for access, and so on;
- Has advanced status monitoring capabilities to identify situations associated with the machine or the environment requiring attention. This includes monitoring structural and electrical heath trends for early prognostication of events and failures;
- Manages overall activity in the load area including the scheduling and dispatch of trucks, cleanup activities, and so forth; and
- Provides an information-rich teleoperation mode that allows recovery from exceptional situations to be completed remotely.

Benefits of the Autonomous Digger

The benefits of the automated digger stem from the increased consistency and reduced variation that automation brings to equipment operation, resulting in productivity and reliability improvements and the ability to have the machinery operate under its most favorable conditions. Ultimately this translates to increased productivity with lower production costs and energy consumption.

The business case for the autonomous digger is likely to be built around the following benefits:

- Consistent and accurate loading of haul trucks. When trucks are loaded to their rated payloads and the load is correctly distributed across the tray, then the chassis, transmission, and tire life are significantly improved and operating costs reduced.
- Increased digger availability. Automation brings consistency that stands to increase equipment availability through lower duty loadings.
- Safer operation. Most safety incidents associated with diggers are linked to operator error (e.g., digger–truck collision). By marginalizing the root cause, automation has the potential to reduce the frequency of accidents and near misses.

Technology Gaps

Four broad technology gaps must be bridged to realize the autonomous digger:

1. Control strategies must be developed to enable automated machines to operate interdependently with other equipment (manned and automated).
2. Situational awareness capabilities must evolve to the point where they can replace the many and varied functions

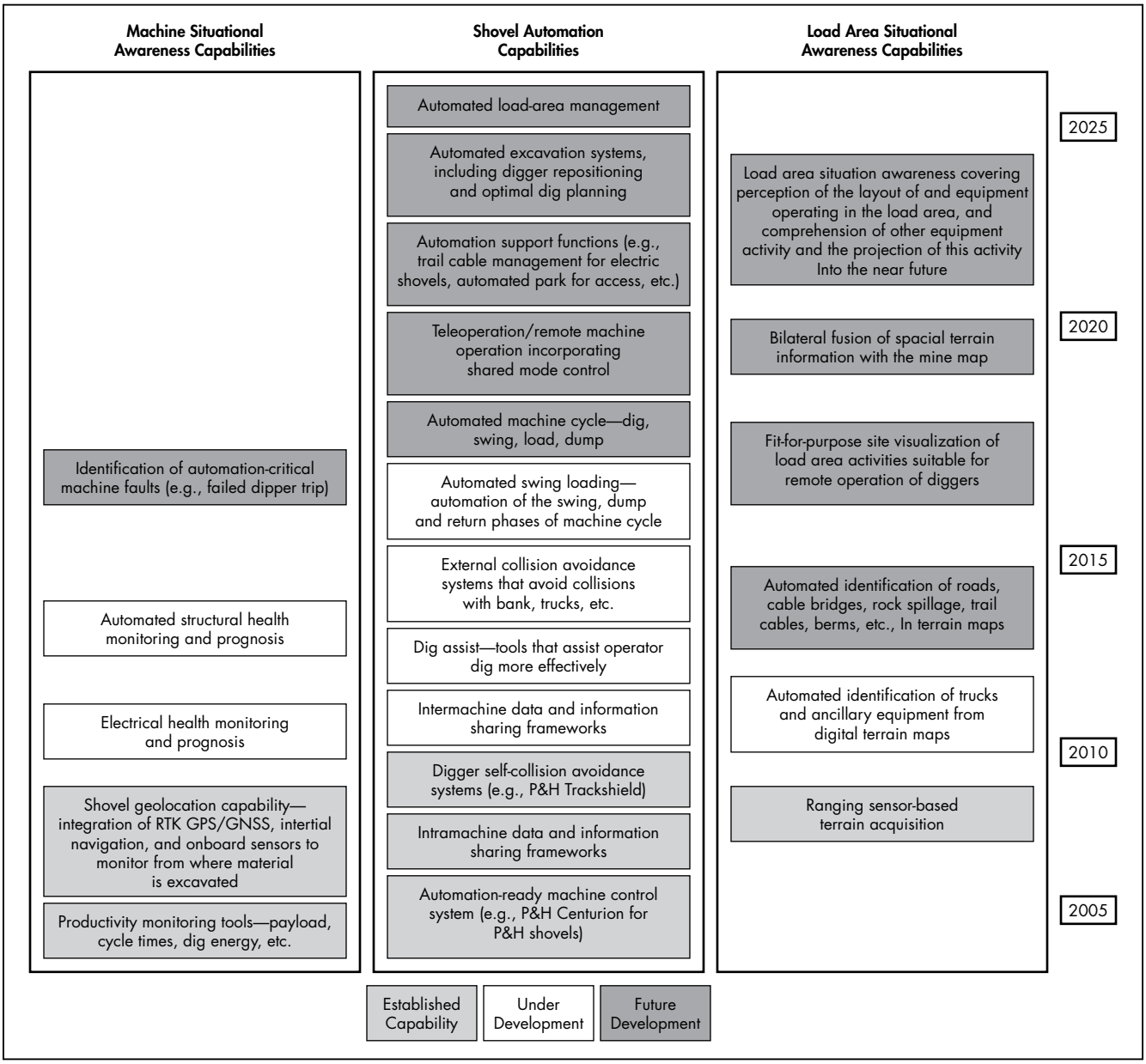


Figure 9.8-1 Autonomous digger capability plan to 2025

currently performed by human operators in planning and actions, and monitoring the status of the machines.
3. Technologies are required that enable effective integration of automated machinery into mine systems.
4. Work-force skills must be enhanced to support deployment of high-end automation technologies.

A significant component of the shovel automation problem is systems integration, including management of interactions with trucks and other equipment and integration of the autonomous shovel into the mine plan. Although in the long term there will, almost certainly, be multiple technology providers working to agreed automation standards, at this time the problem is not sufficiently well defined for effective standardization efforts. The likely scenario for the short-to-medium term is that various equipment and technology providers will integrate their proprietary systems by ad-hoc methods on an as-needed basis under pressure applied by the end user (the mining companies), or one provider will come to dominate the market and set a de facto standard for integration.

The long time horizon for achieving a fully automated digger system mandates a multigenerational technology plan with commercial outcomes that prove technology components and support and maintain the development effort toward its end goal. Figure 9.8-1 gives a digger automation capability plan to 2025 with capabilities divided into three categories:

1. **Machine situational awareness capabilities.** These include such things as machine performance monitoring, functional safety capabilities support automation requirements, electrical and structural health monitoring.

2. **Shovel automation capabilities.** These incrementally build from operator assists such as collision-avoidance tools through automated digging to management of the load area.
3. **Load area situational awareness capabilities.** These build an evolving richer situational awareness model of the load area with knowledge of digger and truck positioning at the lowest level, building toward the full perception of elements in the load area, the comprehension of their meaning, and the projection of their status in the near future.

Implicit in this capability map is that machine and load area situational awareness capabilities feed into shovel-automation capabilities. The timeline is indicative as is the ordering, but it is generally expected that capabilities will flow in this order.

The plan has been organized so that each identified capability could, in principle, serve as a commercialized outcome. The continual delivery of technology products that have productivity or maintenance benefits in their own right and contribute incrementally to the autonomous digger represents the only practical strategy. R&D-based innovation at all levels of the plan is strongly needed, and the most likely scenario is that R&D will be completed by consortiums made up of an equipment manufacturer, the end user, and one or more third-party technology providers including research organizations and universities.

Current State of Automation

Fully automated shovels are not commercially available. However, shovel automation has been the focus of extensive research over the past few years. Dunbabin and Corke (2006) describes automation work completed on a 1/7-scale model cable shovel that demonstrated the ability to automatically perform multiple truck-loading passes that included excavation of the dig face, swinging with obstacle avoidance, identifying an awaiting truck tray, determining an optimal loading strategy, and dumping the material.

McAree et al. (2007) describe collaboration between the Cooperative Research Centre for Mining (CRCMining) and P&H to develop advanced technologies leading toward shovel automation. A P&H/CRCMining laboratory has been set up in a quarry north of Brisbane, Australia, and includes a full-sized P&H 2100BLE electric cable shovel. This laboratory is used to develop and evaluate technologies that increase the productivity of electric mining shovels.

The Australian Coal Association Research Program (ACARP) is currently funding a project to develop, to proof-of-concept, an automated swing-loading technology for electric mining shovels at this facility. A joint CRCMining and Australian Commonwealth Scientific and Industrial Research Organisation (CSIRO) team are developing technologies for a shovel that cycles the swing, dump, and return functions. This includes automatically identifying the location of a truck or in-pit crusher–conveyor to be loaded; planning and executing a minimum path from dig to dump location without the dipper colliding with the truck, crusher, or bank; and dumping without spillage and swinging back to tuck position for the next dig.

Automated digging is probably the most complex component of loader automation. Automation technologies to assist digging are commercially available for several of the autonomous LHD products, but not for large-surface loading machines. A considerable body of research in automated digging exists. Singh (1997), Hemami and Hassani (2009), and Lever (2001) provide reviews of this research area.

Longwall Automation

Two basic control systems exist in a longwall operation. One control system operates in the horizontal plane to control the plan-view geometry of the longwall face, and the second operates in the vertical plane to control the roof- and floor-cutting horizons within the coal seam. Added to these functions are systems for armored face conveyor (AFC) control, including chain tensioning and load sharing, and shearer haulage control.

In simplest terms, in plan view, the longwall face should be straight and perpendicular to the gate roads. If the face is straight, both mechanical stresses on the armored face conveyor and roof support geotechnical issues are minimized. The process to achieve this situation is known as face alignment. As the longwall retreats, the assembly of supports should not creep toward either main or tailgates. To achieve this result in practice, often the face line is angled, introducing so-called tailgate lead or lag with respect to the main gate so that in sloping seams, the same creep-minimizing result can be obtained. Managing the lateral position of the longwall equipment in the panel is called creep control. Automatic face alignment and automatic creep control are two functions that can be applied to effectively provide automation of longwall plan view geometry. Available systems are discussed in this section.

In the vertical plane, the automation situation is more complex. The goal of the longwall, similar to any mining operation, is to maximize extraction of product and minimize extraction of waste. This means the longwall shearer should operate so that roof- and floor-cutting horizons are entirely within the seam, or in some cases within a selected band within the seam. Achievement of this goal is known as horizon control. Automated horizon control needs to at least emulate and at best entirely replace human-operator strategies for steering the shearer within the seam. Again, the available automation solutions for horizon control will be discussed.

Undeniably, longwall automation has the potential to deliver significant advantages in both productivity and safety (Henderson 2007). Interruptions to the mining process are basic causes of decreased productivity. Continuous automatic face alignment, for example, can virtually eliminate stoppages caused by current string line–based manual alignment processes, and optimal alignment of AFC components minimizes wear and reduces consequent equipment breakdown or change-out delays. Close control of face geometry can also improve geotechnical performance of the roof-support system where an automation system can ensure that shields are set consistently.

Automatic horizon control can more effectively steer the longwall in the seam to minimize product dilution and can also contribute to more-effective strata control by accurately leaving coal on the roof and/or floor to protect weaker strata. In the safety context, automation provides the ability for the mine to remove people from hazardous areas, minimizing exposure to dust, heat, noise, and danger from roof and face falls.

Automatic Face Alignment and Creep Control

Automated face alignment is now a mature technology, and all longwall roof support manufacturers offer systems that

deliver effective face alignment automation solutions. These systems allow desired face profiles to be entered into shield-control systems through appropriate graphical user interfaces, and shield-control systems are capable of moving individual shields by calculated distances and are thus also able to coordinate the motion of assemblies of shields to achieve desired profiles along the entire face. Shield hydraulic control systems have been enhanced to allow accurate control of double-acting (D/A) ram motion, and sensors to reliably measure D/A ram travel have been incorporated into shield designs.

Until recently, the major defect in automatic face alignment has been the inability to automatically and reliably measure the actual geometry of the longwall. True closed-loop control of face alignment can only be achieved by comparing the physically measured location of the face with the target location and then minimizing the resultant error. Previous methods were largely based on measuring the accumulated motion of shield D/A rams to indicate face geometry, which gave only an approximate solution with increasing accumulation errors. This situation has been remedied recently through the development of the LASC (Longwall Automation Steering Committee) technology, which measures three-dimensional shearer position directly through an inertial navigation-based SPMS (shearer position measurement system). Because the shearer actually cuts the face, direct measurement of shearer position is the best indicator of face geometry. Open specifications for SPMS data outputs have been devised enabling LASC technology to be applied to any combination of face equipment. The reader is referred to www.lascautomation.com for details of LASC specifications and more-detailed LASC technology descriptions. All face equipment manufacturers now offer LASC technology to the market.

Sensors that measure the position of longwall face equipment relative to the gate-road ribs can be used to provide a measure of automatic creep control. If motion of gate-end equipment relative to the ribs is detected, appropriate lead or lag of the face can be introduced into the basic face geometry input to the automation system. Although LASC laser-based creep sensors are now available to provide this measurement, transformation of this information directly into tailgate lead or lag values is highly site dependent and requires operator input.

Face Alignment Control System Example

Contemporary graphical user interfaces (GUIs) for longwall roof support control systems can be used to display the state of sections of a complete longwall face that can exceed 200 shields in current operations. They can also display the leg hydraulic pressures. This display gives immediate information regarding the hydraulic performance of the supports and can also show through color change when particular shields are not set correctly or are being excessively loaded by the roof strata.

When the face has become misaligned with respect to the target face line, it is seen as an uncut wedge developing between the desired profile of the unmined coal and the AFC. To realign the face, a wedge cut must be executed in the current shearer run to realign the supports perpendicular to the longwall block. This is achieved by making proportional changes in the shield advance distances programmed to introduce a compensating wedge shape in the face, eliminating the misalignment as the shields advance after the shearer passes. The success of this kind of control depends on accurate measurement of face profile and D/A ram extension as outlined in this chapter.

It is difficult to accurately quantify productivity benefits due to contribution from specific automation system elements. However, in the case of LASC-based face alignment, sustained productivity improvement of 130 t/h was reported (Reid 2008). Other productivity improvement information is company-confidential to individual equipment suppliers.

Automated Horizon Control

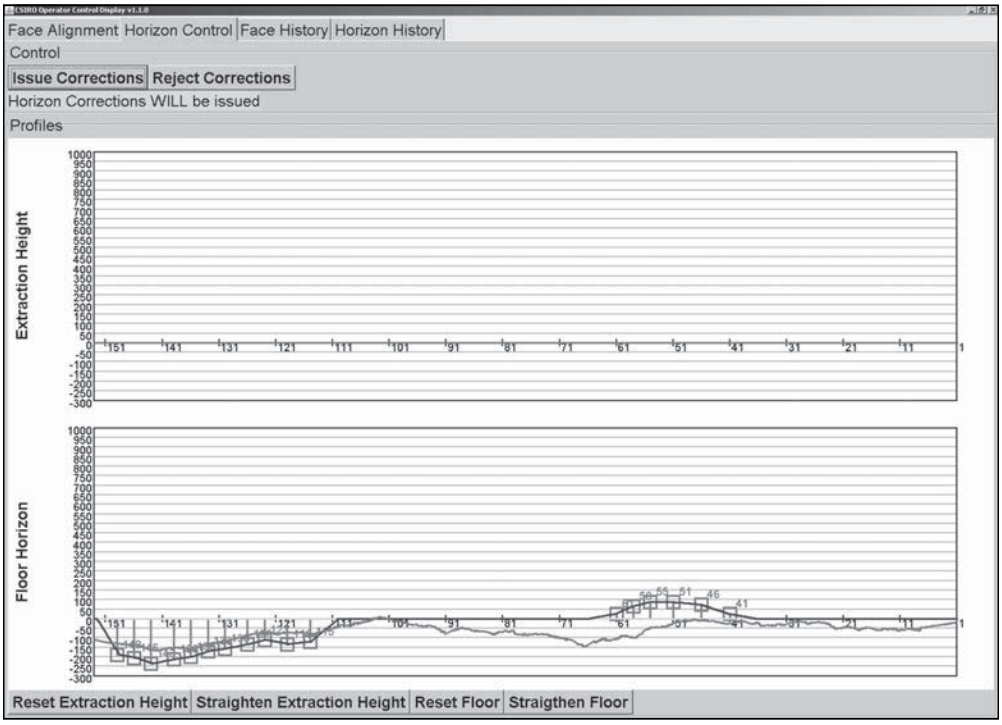
Longwall horizon control—manual or automated—is a challenging task. Whereas the overall goal of horizon control is to ensure that as far as possible only coal is extracted, there are a multitude of site-specific considerations that detract from a mine's ability to achieve that goal. It might not be possible to detect seam boundaries effectively in order to prevent the shearer drums from cutting surrounding rock. In the case of thick seams where visible or other cues to indicate seam trajectory are absent, it is difficult to steer on a consistent path between the gate roads. In other cases, seam undulations along the face or into the panel might mean that equipment cannot articulate sufficiently to track the actual horizons, and roof or floor is cut as a consequence. Problems in detecting fault conditions ahead of mining could mean it is necessary to mine through the faulted zone at short notice.

Automation of horizon control is in the developing stages. The most successful strategy so far has been based on training an automation system using a human operator employing traditional manual cues for horizon detection and control to steer the shearer. The learned extraction process is then repeated automatically until some departure from the horizon-control strategy is observed, such as roof or floor being cut. The training process is then repeated. The advantage of this process is that operator exposure to face conditions is at least reduced. The accuracy of this process has been improved by the availability of the LASC technology, which gives high-accuracy floor-horizon measurements. All shearer manufacturers offer systems that are variations on this basic strategy.

The next stage in the process is to use sensors that can replace the horizon-sensing capability of human operators, or bring to bear new horizon-sensing results. Coal interface detection (CID) sensors based on natural gamma emission by surrounding strata, thermal infrared detection, ground-penetrating radar, optical marker band tracking, and other methods have been developed with varying results. The only commercial CID sensors currently available are based on natural gamma radiation and detection of electromagnetic propagation differences between coal and surrounding strata; several of the other methods are still in the research stage.

Figure 9.8-2 shows a GUI for an interactive automated horizon control system based on the LASC technology. The GUI accepts and displays horizon information over the full face length. The upper screen enables an operator to enter adjustments to a nominal extraction-height setting that has been independently set in the shearer control system. The flat line shows that no operator adjustments have been selected.

The lower screen enables the operator to view recommended floor profiles and to input adjustments manually. The system then automatically generates LASC-recommended floor-height adjustment for the next complete shearer run. In its simplest form, this adjustment is generated by extrapolating the average floor profile from the previous five shears. At this stage of development of automated horizon control, the operator is in the loop. The operator has the ability to input manual floor-height adjustment on a per-shield basis.



Courtesy of CSIRO Division of Earth Science and Resource Engineering.

Figure 9.8-2 A prototype automated horizon control system user interface

The operator's task is to input manual horizon inputs to match the recommended profile as closely as possible. The operator is also able to take into account off-system inputs such as physical observations along the face or known conditions that would preclude the shearer being able to execute the automation system's recommended track. Close matching of manual to automatic settings is shown on the left-hand side of the display in Figure 9.8-2. On the right-hand side, the operator has chosen a floor profile that does not closely follow the predicted floor alignment.

Figure 9.8-2 shows the smooth profile that the automation system generates compared to the manual inputs. This profile then becomes the floor-height adjustment target for the OEM's shearer-control system.

"State-Based" Shearer Automation

In recent years, one of the most successful initiatives has been the development of programmed shearer automation on a logical state basis. The concept is similar to the training method of horizon control described earlier. The operating parameters required for the shearer to completely execute a single pass of a particular cutting sequence are defined as a series of logical states that encompass combinations of ranging arm positions, haulage speeds, and various sensor outputs including motor currents, position measurements, and so on. As the shearer travels along the face, it sequences through the previously defined states. If no anomalous states are encountered, the pass will be executed successfully and the machine will commence the next pass. Error states can be defined to handle operational exceptions, but some errors will cause the system to halt when human intervention is required. When this system was first introduced, significant productivity improvements resulted immediately. This was attributed largely to the consistency that was possible when operation was based on a fixed programmed sequence and not subject to variable operator inputs.

Challenges to Overcome

The automation technologies described in this chapter are able to successfully automate the routine operations of a longwall. The major problems encountered in longwall automation are concerned primarily with the management of exceptions to normal operations caused by the fact that an underground mine is actually not a factory environment and is subject to the vagaries of nature. To achieve full automation and the "workerless face," many more sensors need to be developed to replace the observing roles carried out by face operators at present. Additionally, there are issues with ensuring that equipment operates to the standards required for automation in a harsh, hazardous environment.

Mine Requirements for Longwall Automation

The face alignment and programmed shearer automation systems can be implemented in most mines. In the case of horizon control, it is necessary to establish whether local seam conditions exist that allow horizon sensors to be effective. Equipment manufacturers can offer advice as to which sensors will be appropriate.

For modern automation systems, high-quality data communications links to sensors and face equipment are required. Longwall manufacturers now provide Ethernet connectivity on face equipment as a matter of course. For these systems to be effective at a mine, corresponding communications infrastructure must be available between the face, the surface, and wider into the mining company's and its service providers' data communication networks.

Optimal longwall automation results can be achieved only if all elements in the mining process are operating reliably and at peak performance. Automation systems require particular attention to engineering and maintenance standards at the mine to ensure that all components of the system are effective. An automation culture must be developed and maintained in the work force so that automated operation is fundamental to production.

Dragline Automation

The rate of overburden removal is almost always the bottleneck in open-cut coal mining, and draglines (the most cost-effective means of overburden removal) are the tools of choice. To address this production bottleneck, various strategies have been adopted. First, manufacturers have increased in size and power over the past 40 years, but there appears to be a physical limit in just how large these machines can become before they are impractical. Second, mines have put substantial effort into operator training and production monitoring while also routinely overloading the machines. These strategies are unable to yield further improvement, but continually increasing stripping ratios (which have increased from 2–5:1 to 8–15:1 today) mean that the need is still acute.

Automation is therefore increasingly seen as the next phase in the evolution of the dragline and the industry's quest for increased production. In addition, many in the industry believe that automation can also offer the possibility of lower maintenance costs through the control of the machine within its design limits (a task that human operators often struggle with). Finally, automation systems can deliver consistency and an equivalent skill level that can result in a system that overall will outperform human operators—it may not be as fast as the best operator at the beginning of a shift, but because of its consistent operation, it will on average outperform the pool of human operators. Automation further promises a means of overcoming this skill shortage and would seem inevitable in the mining industry.

Automation Development

The history of dragline automation can be traced back to the late 1980s when BHP Billiton performed a series of trials at a mine in central Queensland, Australia (Allison 1992). These trials attempted to show that a dragline could be partially automated by replaying the control signals of an experienced operator. However, one of the conclusions of these trials was that the instantaneous swing angle of the bucket beneath the boom must be taken into account, making the simple replay strategy infeasible; that is, there must be some kind of bucket swing feedback into the control system. In 1993, CSIRO researchers funded by the mining industry began to investigate bucket-swing sensing and the subsequent automatic control of a dragline's swing motion. Initial experiments determined that vision systems, touted at the time as the solution to the bucket swing-angle sensing issue, could not operate around the clock in the adverse weather and visibility conditions typically experienced at mines. Instead, these camera-based prototypes were replaced by two-dimensional (2-D) scanning laser range finders, which became commercially available in 1995. These sensors proved to be very robust and adaptable to the weather and lighting conditions as well as the mechanically harsh environment experienced at the end of a dragline boom where they were mounted. With such a system it became possible to accurately measure the swing angle of the hoist ropes and hence the angle the bucket-rope plane made with the vertical boom plane (Roberts et al. 1999).

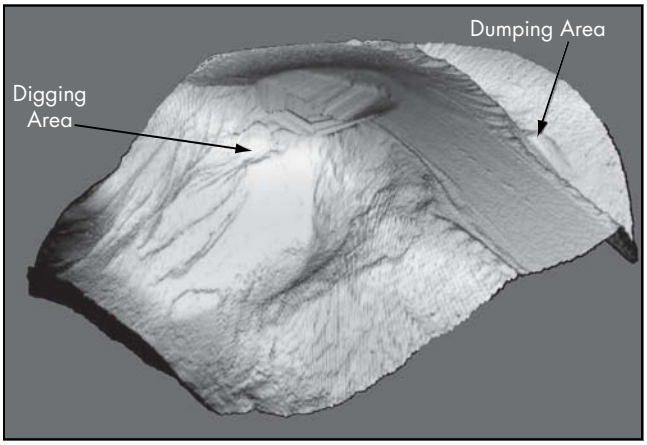
Automation of BE1370 Dragline Demonstration

A full-scale proof-of-concept automated swing-control system was developed on a BE1370 dragline at the Tarong coal mine in South East Queensland, Australia. It performed a fully automatic dig-to-dump-to-dig motion, whereas bucket filling remained the responsibility of the operator. A key component of the system was active controls—the joysticks and pedals were driven by small motors to affect computer control of the dragline, but the operator could override at any time and regain control by moving one of the joysticks. This system was further developed on a BE1350 at the Boundary Hill pit in central Queensland, where in 2002 a 2-week production trial was undertaken (Corke et al. 2003). In total, 12,235 cycles were performed, 3,042 using the automation system and 9,193 fully manually. More than 250,000 t of material were moved autonomously during this trial. In summary, three significant conclusions were reached. First, the system performed more than half of its cycles in better time than the average of the operators working over that period. Experienced operators observing the dragline could not tell a manual cycle from an automatic one. Second, the performance of the automation system was compromised by the system's lack of accurate terrain information. The system relied on the operators marking the position of the spoil piles, highwall, and any other obstacles. This was done by placing the bucket over these locations and pressing a button on the joystick controller. However, a safety margin was introduced to ensure that the bucket never collided with the terrain, and this typically added a 5%–10% time penalty to each automated cycle (due to overhoisting). Third, it was clear that the automation system treated the dragline more gently than the human operators. No specific maintenance benefits could be derived from such a short trial, but the time history of control outputs from the automation system were far smoother than the human operators.

The same CSIRO team between 2002 and 2006 addressed the lack of real-time situational awareness around the dragline where they developed a digital terrain mapping (DTM) system. As well as allowing an automation system with a smaller safety margin, the accurate knowledge of terrain can be used by the automation system to plan optimal bucket trajectories. The successful use of 2-D laser scanning systems for rope location (Roberts et al. 1999) inspired the CSIRO researchers to trial a longer-range scanner on the boom tip, giving it a view of the terrain immediately beneath the boom. With this arrangement, all the terrain around the dragline could be mapped as the machine rotated (Roberts et al. 2003). Initial standalone trials (without the automated swing control) proved the concept, and a 4-week production trial was undertaken in 2006 on the Boundary Hill BE1350. A DTM produced during this trial is shown in Figure 9.8-3. This trial showed that DTMs could be produced routinely in a production environment and that the DTMs themselves were useful to the mine planners and surveyors, even without an automated dragline control system. Further, a level of detail never envisaged was obtained in that the material movement of every cycle could be recorded if desired.

Automated Dragline Digging and Precision Dumping

In 2005, the CSIRO team demonstrated a fully autonomous tenth-scale model dragline that incorporated bucket-swing



Courtesy of CSIRO.

Figure 9.8-3 Digital terrain map produced during a 4-week production trial

feedback, generation of DTMs, optimal bucket-path planning, and autonomous digging (Dunbabin et al. 2006).

The practice of precision dumping has again been raised now that autonomous draglines have been shown to be feasible. The concept of a dragline being capable of accurately dumping its load into a hopper and feeding into a conveyor or into trucks could transform certain mining operations. The CSIRO team has shown how the tenth-scale model dragline can reliably dump material directly into a truck tray (scaled accordingly).

Current Status

The past 15 years have seen a number of proof-of-concepts and demonstrations in full production environments with trials lasting multiple weeks (Winstanley et al. 2007). Dragline automation is now technically possible; economics and the market will determine when it is finally introduced.

FUTURE CHALLENGES

A major challenge over the next decade is to automate the unit operations themselves, understanding that some of the equipment in a group may remain under direct human control or be semiautomated. Thus, each automated machine must be able to operate interdependently with the other machinery associated with the unit operation. The absence of this capability currently is a major barrier to the deployment of automation technologies in mining. Understanding how to achieve safe, interdependent operation in mixed fleets is the most important and significant technical barrier to the further development of mining automation technologies.

McAree and Lever (2003) provide recommendations for where investments in surface mining automation research will provide the short-to-medium-term benefits to the mining industry while making progress toward full automation. They are as follows:

- **Advanced sensing technologies for the mine environment.** This includes the processing of sensor data to extract information and sensor fusion for combining data from several sensor sources. Imaging technologies for terrain and local-area mapping are important, as current technologies are sensitive to dust, vibration, and temperature variations.
- **Use of mine-wide information systems to facilitate unit interactions.** Mine-wide information systems play a critical role in controlling the interactions of automated equipment. This task will be complicated significantly if common communications protocol is not adopted that would allow complete interoperability between sensors and mine information software from different vendors.
- **Extended use of existing and new sensors aboard equipment.** Data from onboard sensors can be used not only to monitor and report equipment status and performance, but to develop detailed understanding of equipment operations and their interaction with mining processes. Understanding these operations is an important step for successful automation.
- **Developing duty meters to manage the trade-offs between equipment productivity and damage, and to predict failures.** Automation systems must perform the many maintenance, fault detection, and isolation functions that operators currently execute. It is important to understand how machine performance and duty trade against each other to optimize the productivity of automated equipment. Poor availability poses the same problem for automated equipment as it does for manually operated equipment.

The widespread acceptance of automation and robotics technologies in mining over the coming decades will depend on a number of factors, including

- Simple and effective integration with mining processes;
- Changes to existing mining processes to simplify the use of automation;
- Management acceptance to ensure automation is seen as a benefit, not a threat, to the mining work force;
- Implementation occurring in stages starting with narrow domain or task requirements and then increasing capability and complexity (machines then systems);
- Meeting mine requirements of productivity, cost, and flexibility;
- Balancing technical complexity with robustness;
- Transparent operation wherein mine personnel must clearly understand the capabilities and limitations of the systems; and
- Turnkey systems, as much as possible.

A major threat to the uptake of mining automation is the desire for a quick success, which leads to shortcuts in technology development and poor implementations.

A VIEW OF AUTOMATION AND THE MINE OF THE FUTURE

For decades, visionaries have predicted that autonomous machines would be critical to the future of mining. The major drivers have been safety, the need to work in remote locations, the lack of operators in general, long-term reliability improvement, and efficiency gains. These visions are rapidly becoming a reality.

Safety Advantages

One of the most often-cited reasons for autonomous mining is safety. Although the fatality rate published by the Mine Safety and Health Administration (MSHA 2009) shows a reduction in fatalities, significant improvement is still needed. Despite tremendous effort on the part of regulatory agencies, mining

companies, and equipment suppliers, the total number of fatalities in the United States has on average not improved substantially since 2003. Similar statistics from Australia, South America, and China show that further safety improvements are very challenging to achieve and potentially even harder to maintain. One clear strategy to reduce injuries and to easily maintain the reductions in injuries is to remove people from harm's way. For example, it is estimated that automation and ROCs offer the opportunity to remove up to 80% of haulage-system operators from surface mines. Fewer people involved in mining operations translates to fewer opportunities for safety-related incidents and fewer opportunities for long-term disabilities caused by repetitive operations.

Integrated Site Awareness

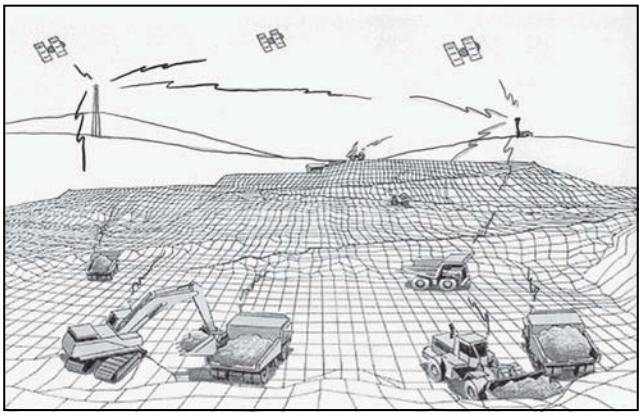
Another significant safety advantage provided by autonomous systems can be achieved through integrated site awareness or situational awareness. Probably the most easily understood example of situational awareness is the accident prevention provided by air traffic controls. By tracking most planes and controlling corridors, the situational awareness provided by air traffic controls helps reduce mid-air collisions. Studies indicate that situational awareness can lead to substantially improved performance in complex military environments (Endsley 1995). Although managing a mine or operating mining equipment may not be as complex as operating a military aircraft or as demanding as managing a battlefield, the continual, around-the-clock nature of the mine operation makes site awareness just as critical (Figure 9.8-4). A well-integrated mine of the near future will allow all machines and potentially mobile objects to be tracked in real time. Integrated site awareness will allow both automated and human machines to easily perceive and comprehend the current situation, which should lead to better decisions and a safer environment for both.

No Fatigue and Minimized Opportunity for Operator Error

Although recent estimates of the effect of fatigue in accidents vary considerably (from 2% to 41% according to De Gennaro et al. 2001), fatigue is clearly recognized as a significant contributor to mine accidents. Furthermore, operator error caused by all sources (including fatigue) may account for as much as 88% of all accidents (Heinrich et al. 1980). It is commonly recognized that the best means to avoid accidents is to design out the opportunity for the accident. Using automated systems significantly reduces the opportunity for errors and provides a means to accurately track, correct, and control any remaining issues. Automated systems never get tired, bored, distracted, or inattentive. There will certainly be challenges related to performance in all environmental conditions, but these should be well understood and will improve over time.

Remote Locations

It is anticipated that many new mines will be located in regions with low population densities. The cost to develop housing, schools, hospitals, and other required infrastructure can significantly impact the ability for a given site to be a cost leader (have a production cost below the average cost in the industry). Many remote sites are supported by a large number of fly-in/fly-out (FIFO) employees, which can lead to "FIFO fatigue" and potential higher turnover rates (due to the variability in mining, it is difficult to make accurate assessments).



Courtesy of Caterpillar, Inc.

Figure 9.8-4 Communication infrastructure for a future mine

Informal feedback from the mining industry suggests that it costs mine sites 100% of an annual salary to replace a worker and a further 50% to train the replacement to a desired productive level (Rowland Communication Group 2004). The cost to attract workers and the worker turnover at remote locations can severely impact the operating cost, production, and safety at these remote mine-site locations. Reducing the total number of workers via automation and allowing more of the remaining workers to work at ROCs could significantly reduce FIFO fatigue and may reduce employee turnover.

Availability of Operators

Scarcity of skilled operators during boom times is often cited as a major problem for mining expansion (Barta 2005). Certainly mines that follow the volatility exploitation model (essentially, mines that are only operated during times of high commodity prices) will make it difficult for the industry to maintain a stable, highly trained work force. Using autonomous systems could minimize the volatility and reduce the start-up and shutdown times of higher-cost operations.

Efficiency

Another key reason for interest in automation is the expected gains in overall relative efficiency, where relative efficiency is defined as tons moved per day relative to a nonautomated mine with the same contingent of machines. Several factors can dramatically influence the efficiency gains at any particular site including load size, cycle time (speed, queuing time, spotting time, loading time, and dumping time), accuracy (percentage of loads delivered to the right location), machine availability, and operator availability (discounted for training time, breaks, absence, etc.). For haulage systems, these can be written as

- Relative productivity = (relative load × relative accuracy × relative machine availability × relative operator availability) / relative cycle time
- Relative load = (autonomous machine load) / (manual machine load). Expect this ratio to equal 100%.
- Relative accuracy. This is the percentage of total loads (waste and ore) that are dumped in the correct location. The relative accuracy should improve slightly for automated machines.

- Relative machine availability (the percentage of time that the machine is down due to a mechanical or electrical failure). It is assumed that machine availability will improve over time with automation, due to the more-consistent machine operation.
- Relative operator availability. Operator availability can vary widely depending largely on the local mining process. This is an area of substantial potential gain for autonomous systems. For example, if an existing mine with no automation achieves 16 h/d of actual working time and the same mine with an automated haulage system can achieve 22 h/d of operation, it is a 38% increase.
- Relative cycle time. This is the average total time per cycle with an autonomous system divided by the average total time per cycle without an autonomous system. Relative cycle time includes the loading time, transporting time, dump time, and queuing time. For underground mining systems that are already using remotely controlled machines, automation should provide a significant improvement in cycle time. But for haulage systems that are currently using manned machines, the relative cycle time may be lower for initial automated systems. However, due to the expected rapid evolution of autonomous systems, it is anticipated that future generations will match the cycle times of well-controlled mines that operate machines within their design limits. In other words, the machines will be operated according to their design limits and the mine rules.

Efficiency Gains Though Process Control

Arguably, the greatest potential improvement may be in process control. The availability of machines that operate consistently and as instructed provides the opportunity for mine operators to fine-tune their operations. Future mine systems will allow every machine, person, and process to be accurately tracked. This will provide unique capabilities to accurately model all mine processes using real-time models to improve mine-planning capabilities. The autonomous machine will be the most salient change, but the real driving force for improvement will be the consistent, predictable process coordinated by an information-rich environment.

An example of a related industry is that of grade-control systems used in construction. These have shown process control–related improvements of 30%–50% and associated fuel and carbon dioxide reductions of 43% (Caterpillar 2006a). Process control is expected to be a strong area of dramatic innovation as autonomous machines begin to be deployed. Although independent data is not available, a recent article regarding the deployment of Caterpillar's MineGem technology at the Jundee gold mine in Western Australia states that "MineGem cut down on wear and tear on the machines." According to Gary Mills, Jundee's mining manager,

> What was happening previously too is that the boggers were bouncing off the walls and causing a lot of damage.…The laser-driven boggers stay off the walls, so you don't get any damage even though they can go faster.…The need for secondary stockpiles in the mine has also been removed by the faster tramming speeds, cutting down that cost as well (Haycock 2009).

Challenges and Changes Required for Successful Automated Systems

Clearly, putting an autonomous machine into an otherwise unchanged mine will not provide the expected gains. For example, in general, autonomous trucks will initially travel at slower speeds than human-operated trucks, causing significant loss of efficiency for the human-operated trucks.

Information System Improvements

One of the biggest changes will be the amount of information needed to run a mine smoothly. An autonomous mine will be very much information driven including everything from the detailed mine model to product levels, payloads, and the location of every asset. This will drive several requirements on communications and information infrastructure:

- High-bandwidth communications with quality of service capability will be a requirement. It is strongly suggested to have dedicated communications channels to avoid issues with lost or late communications.
- Tracking and communicating the location of every asset will be a key layer of the obstacle-avoidance strategy. Neither people nor machines should be allowed to operate within an autonomy zone without proper electronic tagging.
- Every asset will need to provide vital information such as its position, heading, health, status, destination, detected objects, and potential road (map) changes. In addition, it would be useful to communicate any potential changes about the road surface (wet, rough surface, soft, etc.).
- This information will be in addition to the existing productivity and maintenance information that is used for typical mine management.

Accurate Mine and Machine Model

Maintaining an up-to-date mine model will be absolutely critical to autonomous machines. Although machines should have secondary sensors to detect inconsistencies in the mine model, autonomous machines will by necessity assume that the mine model is correct. Most machines should be programmed to halt if they detect inconsistencies in the mine model. Analysis of the inconsistencies and self-correction would be the next phase of development for automated systems. The definition of the mine model will require orders of magnitude more detailed than existing mine models that are used for automated assignments or dispatching.

All mining machines will have to be modeled as well. The fidelity of the machine model will be dependent on the class of machine such as autonomous machines, interactive machines (such as loading machines), support machines, and mobile platforms (light plants, pumps, etc.). Autonomous machines, including both kinematic and dynamic models, will need to be very accurately modeled. Interactive machines (such as loaders) will need to have a very accurate kinematic model and accurate positioning. Support machines may be modeled much more simply, and mobile platforms may be modeled simply as an exclusion zone. Any significant changes to these machines will need to be tracked and modeled (e.g., a pickup pulling a trailer suddenly appears to have grown in length).

Required Operational Changes

Successful installation of an autonomous system will require operational changes in several key areas:

- **Working environment.** Obtaining consistent predictable productivity will require a reasonably predictable working environment. Anything that alters the working environment such as water trucks will need to be accurately controlled and monitored.
- **Improved information system maintenance.** Maintaining the electronic infrastructure will include positioning systems and dedicated high-bandwidth communications systems. In general, automated systems will require either a dedicated network or a redundant network with quality of service assurance. In addition, a backup network should be installed for emergency and autonomy stop functions.
- **Avoiding mixed fleets.** In general, autonomous machines will travel at different speeds than human-operated machines (faster in some cases and slower in others), and mixing manned and unmanned machines may cause significant loss of efficiency for the human-operated trucks. Automated equipment may be programmed to be more "sensitive" to human-operated machines, potentially slowing down to avoid any risk of an accident, and will, therefore, tend to be less efficient.

Machine Health and Maintenance

Site-level machine-health monitoring will become more critical as autonomous machines become more prevalent. In addition, there are still items that are only detectable by human operators (some vibrations, noises, etc.). Development of new prognostics and diagnostics will be required, and maintenance procedures will become much more critical.

CASE STUDY: AUTOMATED SURFACE BLASTHOLE DRILLING AT THE WEST ANGELAS IRON ORE MINE

This section provides an overview of blasthole drilling automation and uses the data associated with the implementation of this application at Rio Tinto's West Angelas iron ore mine.

An automated surface drill is generally considered one that can tram, level, and drill automatically. Automated blasthole drilling is considered easier to adopt into an existing mine fleet than other large, automated mobile equipment due to its relatively slow movement and generally isolated operation.

The automation of surface blasthole drilling has made significant advances into the mining industry since 2005. Some of the cited benefits are

- Repeatability of drilling process
- Lower total cost per meter
- Increased production per drill
- Longer use of assets
- Lower labor needs
- Better hole stability
- Better predictability of blast fragmentation

Issues with Implementation

During the implementation of a new automated drill system, there can be an expectation that the system will perform at a level at least comparable to the existing system. If the system is not performing within expectation, the technology may not be adopted.

Table 9.8-1 Basic sensors for an automated drill

Description	Typical Sensors Used
Drill-rig position	High-precision Global Positioning Systems, ground-based satellites, inertial measurement units, lasers
Drill-hole depth	Lasers, string encoders, wheel encoders, magnetic pulse
Drill-rotation velocity	Magnetic pickup on drill-drive gearbox, flowmeter on rotation hydraulic circuit
Drill-rig roll and pitch	Tiltmeters
Track speed	Track encoders

Source: McHugh 2009.

To plan for implementation success, mine operators should consider the following elements:

- **Drill automation approach.** If the drill automation approach is different from the current manual drilling approach, this must be clearly explained to the stakeholders. Some degree of customization of the automation to suit the ground conditions should occur before the drill is used in full production.
- **Automation sensors.** The automation system critical sensors must be accurate and able to survive in a mining environment. Sensors that detect the surrounding environment (e.g., vision systems) must be of sufficient resolution to determine changing conditions such as collapsed faces or large boulders to prevent catastrophic damage to the machine. Typical sensors required to automate a large blasthole drill are displayed in Table 9.8-1.
- **Communication infrastructure.** If the drill is to be remotely supervised, the communication system selected must be suitable for the application. This will require site surveys and consideration of future requirements.
- **Service support and training.** A sustainable production system requires service support and training. The model of service and support must be clearly defined before implementation.
- **Other dependent mine processes.** The drill automation will affect other processes such as normal drill servicing, sampling, and charging due to more restricted access to the drill area. New procedures will be required before implementation.
- **Safety.** Mine operators may choose soft procedural safety controls rather than more expensive engineering controls that are difficult to maintain in a rapidly changing mine environment. The automation of blasthole drilling may not require interaction with other mine equipment, thus reducing system complexity. The use of remotely operated automated machines provides a much safer environment for drill operators, removing the operator from potentially hazardous areas such as unstable pit walls, unstable pit floors, hazardous airborne dust, and hazardous noise.

  The use of an independent emergency shutdown system that can stop the engine as fast as practicable would provide a level of redundancy in the case of uncontrolled movement. This is very important if a remote operator is supervising the automated drill or drills.

Table 9.8-2 Comparison of drill accuracy for automated and manual operation

Accuracy	Manual	Manual with High-Precision GPS Guidance	Automated
Drill-collar position average error against design, m	0.563	0.259	0.144
Drill-collar depth average error against design, m	0.51	0.33	0.05

Source: McHugh 2009.

Table 9.8-3 Comparison of drill productivity under automated and manual operation

Productivity	Manual	Manual with High-Precision GPS Guidance	Automated
Tramming time median, s	75	43	58
Leveling time mean, s	35	28	40

Source: McHugh 2009.

Table 9.8-4 Comparison of overall drill productivity for automated and manual operation

Productivity	Drill 3 Auto	Drill 3 Manual	Drill 3 Total	Drill 4 Total	Drill 5 Total	Drill 11 Total	Drill 12 Total	Drill 104 Total
Total drill length, m	36,728	3,599	40,326	44,365	25,551	38,398	40,220	38,485
Total drill time, h	576	55	631	675	506	651	815	708
Penetration rate, m/h	64	65	64	66	50	59	49	54
Availability, %			91	89	76	89	93	94
Use, %			51	55	44	55	65	56

Source: McHugh 2009.

Automation Capability

The automation of the large-surface blasthole rigs has advanced at a faster rate than the smaller articulated boom rigs. Larger rigs are considered easier to automate as they have fewer degrees of mast movement. They are generally located in mines with large, simpler patterns with generally vertical rotary holes. The benches are generally well maintained and provide the ideal environment for navigation and leveling. Automation of rotary drilling is considered easier than hammer drilling. Hammer drilling requires a precise rotation speed that must adapt according to the penetration rate. Rotary drilling, on the other hand, will generally have a fixed rotation speed that does not vary as much with the geology.

Mine operators will always encounter different drilling environments, and the more accepted automated drill systems are the ones that allow intervention either manually or remotely to deal with exceptions. This feature is important as it generally allows little loss in production over completely manual systems, aiding in system acceptance.

The automation of some ancillary functions of the drilling process will require significant engineering changes to occur such as bit changing and mode of drilling. Since these processes use consumables of the drill that wear quickly, in the near term they are probably better handled in a semiautomated mode.

To make the drill ready for automation it is generally considered that the drill must be set up for electronic control. Many of the new surface drills use the controller-area network bus that was developed for the automotive industry.

Advantages of Drill Automation

Automation of surface blasthole drill rigs has demonstrated greater accuracy and repeatability with similar rates of productivity compared to manual operation. Remote supervision of automated drills has allowed one operator to control more than one drill.

Field trials of a proprietary automated system on an 80-t Terex SKSS16 rotary and percussion blasthole drill at the West Angelas iron ore mine demonstrated significant improvements in accuracy over manual operation. The summary of the results are displayed in Tables 9.8-2 and 9.8-3.

Table 9.8-4 compares the same automated drill performance against five other same-model drills operated at the mine in manual mode over a period of 9 weeks. The remotely supervised drill was able to drill more than 90% of all holes in automation mode.

The high degree of repeatability of the automated drilling process allows mine operators to choose a drilling approach that best suits their operation with a high degree of confidence that the drill will achieve it. The selected drill approach should ultimately help achieve the overall mine strategy. The repeatability also helps mine operators to better understand the trade-offs in different approaches. The mine geology can have a significant impact on which approach is required at different areas of the mine, and therefore it is unlikely that one single approach will be optimal in achieving the mine strategy. Approaches may include

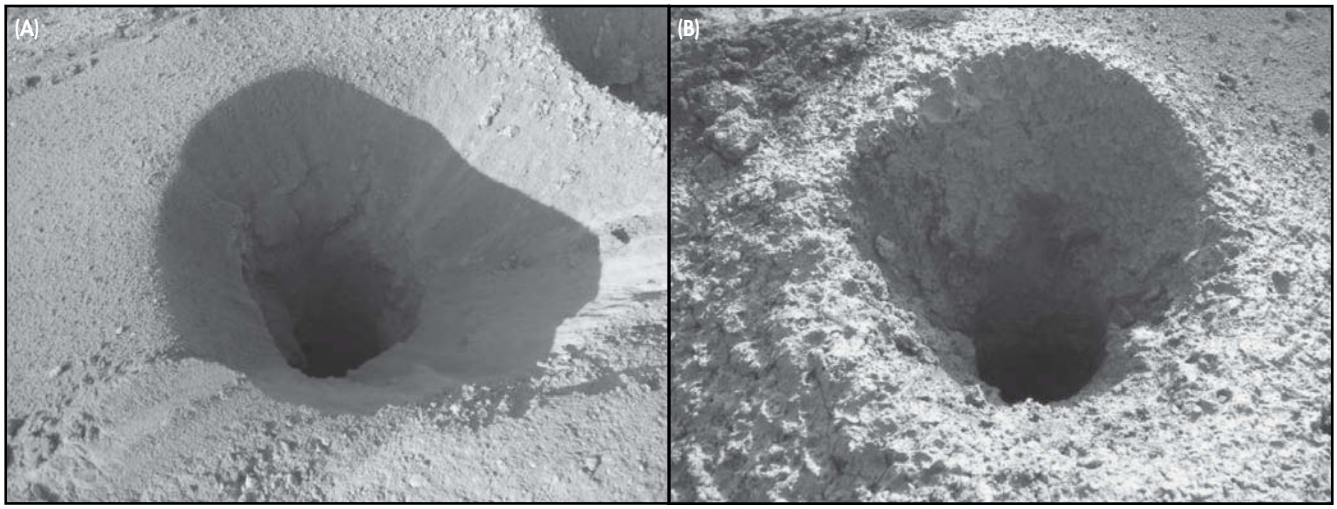
- Trading off production performance for improved drill bit wear rates,
- Drilling the straightest hole,
- Maintaining hole stability to prevent redrilling, and
- Lowering overall stress and vibration on the drill mast and chassis.

In a manual drilling operation, expert drillers are those who can best follow the approach selected by management and adjust the strategy according to the changing geology and condition of the drill. The automation of these approaches will generally reduce the time required to train a new operator to perform like an expert driller. Figure 9.8-5 is a comparison of a manually drilled hole and an automated hole drilled 5 m apart in the same geology. The automated drill shows the correct amount of water injection to maintain a stable collar in that geology.

Disadvantages of Drill Automation

Although the advantages of automation are significant, there remain some disadvantages:

- The automated drill is best-suited to stable geology.



Source: McHugh 2009.
Figure 9.8-5 Comparison of drill collars between (A) manual and (B) automated drill holes

- It can be slower than certain parts of the drill cycle as it maintains the level of accuracy required.
- Sensor failure on the drill is common and could stop the whole system.
- Without significant sensors and processing, the drill may not detect the changing drilling environment such as
  - Collapsing collars,
  - Collapsing holes,
  - Collapsed bench edges,
  - Obstacles such as rocks fallen on the bench, and
  - Potential interaction with other equipment such as explosive-charging carriers.
- Detection of worn bits or drill-string failure can be difficult.
- High-precision GPS may be difficult in deep pits and may require ground-based satellite augmentation.

Challenges to Overcome

To improve the function and reliability of automated systems, several factors need to be addressed:

- Provision for reliable sensors capable of withstanding extreme conditions experienced on the drill
- If operating remotely, reliable wireless communication with significant bandwidth
- The use of prior geological knowledge to adjust the automated settings
- Adequate vision coverage to supervise the rig remotely
- Complex movement path planning between rows
- Integration with other mine equipment such as explosives trucks

Advances and Productivity

Wireless communication for supervision of automated drills enables the use of other real-time sensors for purposes such as rock recognition, automated sampling, and downhole sensors. Prior knowledge of the rock behavior could be incorporated into the automated function design.

The use of remotely supervised automated drills enables the operation of at least two drills at once. As automation becomes more reliable and remotely supervised, it should be possible to run more than three units with one operator.

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CHAPTER 9.9

Mine Infrastructure Maintenance

Uday Kumar, Peter Knights, and Mahinda Kuruppu

INTRODUCTION

This chapter is about maintenance of mine infrastructure so that a mine can safely and economically operate at or above design capacity without unscheduled stoppages or disturbances.

The term *mine infrastructure* refers to major capital installations at surface and underground mine sites. Mine infrastructure is very site specific and depends on the mine location, size, mining method, equipment used, and work force. It includes access roads and declines, hoisting and ventilation shafts and equipment, drainage and pumping facilities, backfill preparation and delivery systems, power generation plant and distribution systems, workshops, service and warehouse facilities, offices, changehouse, chilled-water plant, wastewater management systems, and construction facilities. Other on-site infrastructure includes ore treatment plant and laboratories, communication networks, and equipment for automation of mine operations.

The primary objective of mine infrastructure maintenance is to ensure that production targets are met on time, safely, and economically. Modern mining is mechanized and automated, requiring a high level of availability and reliability of mine infrastructure and maintenance preparedness to ensure that operations are maintained at designed (mine) capacity. The mine production system is critically dependent on the availability, reliability, and capacity of mine infrastructure. The substantial level of system integration in modern mines has elevated the criticality of mine infrastructure. Furthermore, mechanized and automated mining systems are expected to safely perform at designed capacity around the clock, making access difficult for maintenance workers and complicating the task of maintenance scheduling.

Because of the diversity of solutions to specific infrastructure requirements, this chapter will not detail maintenance requirements for all types of infrastructure. Rather, it concentrates on general principles of infrastructure maintenance management and presents some of the critical issues and challenges related to maintenance of the following systems:

- Hoisting systems
- Underground roadways
- Electrical power distribution systems
- Mine dewatering systems
- Maintenance workshop facilities

PERFORMANCE OF INFRASTRUCTURE

Availability and reliability performance is the commonly used measure of infrastructure performance. Maintainability performance is the universal measure of the maintenance friendliness of the infrastructure. Availability and maintainability are not only measures of system performance, but they also have considerable influence on the financial performance and success of mining businesses.

Availability

Designing and using infrastructure systems that are maintenance friendly significantly improves safety and system availability. Availability of infrastructure means that infrastructure is available for delivering the services for the specified period of time. In general, infrastructure availability can be defined as

$$\frac{\text{available time for service}}{\text{total scheduled time for service}}$$

The International Electrotechnical Commission (IEC 2010) defines *availability* as follows: "The ability of an item to be in a state to perform a required function under given conditions at a given instant of time or over a given time interval, assuming that the required external resources are provided."

Availability performance is described by the collective term *dependability* and is a function of reliability performance, maintainability performance, and maintenance support performance as illustrated in Figure 9.9-1.

Reliability and maintainability are design characteristics of the infrastructure that are determined during the design and development phase of the system's life cycle. Maintenance support, on the other hand, is dependent on the users' organizational

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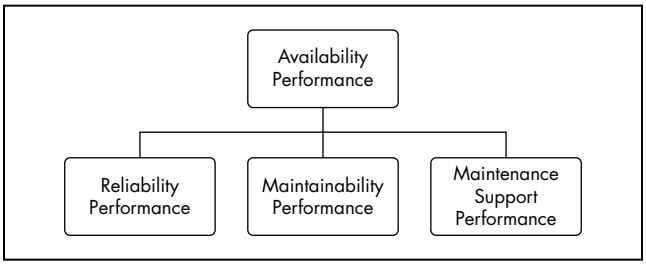


Figure 9.9-1 Dependability and availability concepts

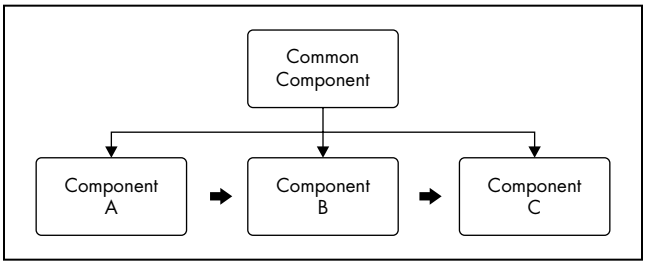


Figure 9.9-2 Reliability block diagram showing common infrastructure component

and operational need, including their operating environments. These factors influence the risks (or uncertainties) related to the availability and capacity performance of the mining infrastructure. A high availability performance level is an important and critical requirement for most of these systems. The factors influencing availability and capacity performance of mining infrastructure include reliability, maintainability, and maintenance support (Figure 9.9-1). Even though high operating reliability characteristics are desirable to reduce the costs of maintenance and maintenance preparedness, the benefits of improved maintainability should not be overlooked in achieving the required availability performance. High maintainability performance makes a system easy to maintain and repair, resulting in high availability performance (see Kumar and Akersten 2008).

Reliability

The reliability of a system is a measure of the probability of its survival as a function of time. In the mining industry, mean time between failures (MTBF) is often used as a proxy for measuring system reliability, where *failure* is defined as unscheduled downtime due to equipment (not process) malfunction. Block models are often used to model the reliability of systems. Systems can be characterized according to whether their principal components are series or parallel connected. In series-connected systems, the failure of one component causes the entire system to fail. This is typical of longwall systems, where, for example, the failure of the face or main gate conveyor disrupts coal production. Parallel systems are designed with some redundancy. For example, a mine dewatering station should include at least one standby pump. Should the main pump fail, the standby can be activated to prevent flooding.

The mine infrastructure elements listed previously are either common components to production systems (see Figure 9.9-2) or are series connected, and so many items of mine infrastructure rate highly on the scale of critical systems for maintenance purposes. For example, every component in a longwall mining system requires power to function. Failure of the power distribution system will cause all components to fail. A mine hoist is a very important series-connected element in the productive system. Loss of hoist capability will result in production losses and could jeopardize the future of a mine depending on the extent of the downtime required to restore functionality.

Maintainability

High maintainability performance and, in turn, high availability performance is obtained when the system is easy to maintain and repair. In general, maintainability is measured by the mean repair time, often called mean time to repair (MTTR), which includes the total time for fault finding and the actual time spent carrying out the repair. The way in which an infrastructure system is designed and built has a direct bearing on the maintainability of the system.

Capacity

Many of the mechanized and automated mining operations are expected to perform safely at designed capacity most of the time or around the clock, making it extremely difficult for infrastructure engineers and managers to get a time slot to maintain them so as to guarantee a high level of reliability, availability, and capacity. The issue of possession of mine infrastructure for maintenance is critical in a mining system operating 24 hours a day and seven days a week.

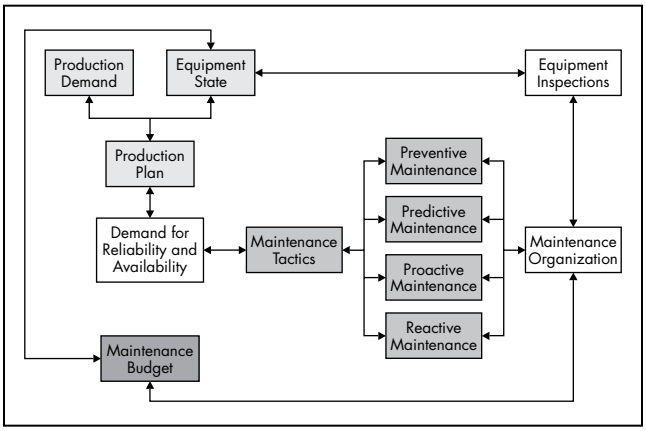
It is also important to define the capacity of infrastructure. It is possible for infrastructure to be available, but the prevalent operating condition or physical condition may restrict capacity. For example, a mine hoist may be forced to operate below rated capacity because worn guide rails restrict hoisting speeds. Similarly, a poorly maintained haul road can restrict speeds and lower transport capacity. Roads may be filled with potholes, or a lack of water for sprinkling may mean a lot of dust is generated, reducing visibility for truck drivers and forcing slower operations. Maintenance of roads, haulages, and other infrastructure systems directly affects their capacity.

Therefore, it is important to measure the performance of mine utilities and infrastructure in terms of both availability and capacity. To ensure adequate availability and capacity, high levels of reliability, maintainability, and maintenance system support.

MAINTENANCE TACTICS

Maintenance tactics, sometimes called maintenance policies or maintenance strategies, are defined on the basis of failure type and consequences of failure. Four basic classes of tactics are available for implementation:

1. Reactive tactics are run-to-failure policies—that is, repair only on breakdown. This approach is also known as corrective maintenance.
2. Preventive tactics are interval or time-based maintenance activities and include inspections, lubrication, adjustment, calibration, cleaning, component replacement, and overhaul.
3. Predictive tactics are condition-based maintenance and repair activities. Predictive maintenance techniques rely on a measure of performance deterioration to intervene prior to failure and include oil analysis, vibration monitoring, and thermography.
4. Proactive tactics are based on a root cause. They include design-out maintenance (design modifications), reliability in procurement and warehousing, operator-driven



Source: Nilo 2004.

Figure 9.9-3 Integrated maintenance activities

reliability, and operating restrictions designed to eliminate the root cause of problems.

A tactic is defined by the type of maintenance and repair action to be realized and the criteria for determining when to apply that action. Unscheduled maintenance work arises as a result of reactive tactics. Scheduled maintenance comprises preventive, predictive, and proactive maintenance work (see Figure 9.9-3).

The production plan is formulated based on estimated sales demand and the current state of the productive equipment and infrastructure. The production plan drives the asset management plan, which specifies availability, reliability, and capability targets for productive equipment and infrastructure assets. These targets drive the selection of maintenance tactics within the infrastructure maintenance plan. These choices in turn determine the maintenance workload, which dictates the level of resources (labor, equipment, and tools) required. In the real world, the maintenance budget restricts the extent of these resources. The maintenance plan must be adjusted by reviewing the type and frequency of tactics to accommodate the budget.

Selection of Maintenance Tactics

Infrastructure assets are constructed to fulfill a useful role or purpose, known as the *primary function* of the asset. The purpose can be defined by answering the question "What do we want the asset to do?" For example, the primary function of a hoist system might be stated as "to lift 5,000 t/d of ore from the 300-m level to the surface stockpile."

However, *secondary functions* often must also be fulfilled. These include environmental integrity, safety, structural integrity, control, containment, comfort, and appearance. A secondary function of the hoist system is that the skip must decelerate to a creep speed before each station. *Protective functions* are a subset of secondary functions and provide protection against environmental damage or risks to health or safety.

In his book *Reliability-Centered Maintenance*, Moubray (1997) defines a functional failure as "the inability of an asset to fulfil a function to a standard of performance which is acceptable to the user." Reliability centered maintenance (RCM) attempts to guide the selection of maintenance tactics by

- Preventing loss of functions that might lead to health, safety, and environmental risk exposure; and
- Preserving functions necessary for operating capability.

In the RCM methodology, the following questions are asked in order:

1. What are the functions and associated desired standard of performance of the asset in its present operating context?
2. In what way can it fail to fulfill its functions? (Functional failure)
3. What causes each functional failure? (Failure modes)
4. What happens when each failure occurs? (Failure effects)
5. In what way does each failure matter? (Failure consequence)
6. What should be done to predict or prevent each failure? (Proactive tasks)
7. What should be done if a suitable proactive task cannot be found?

Figure 9.9-4 shows a simplified RCM decision tree for selecting maintenance tactics. For each failure mode being evaluated, maintenance tactics are defined by beginning the logic at the top left corner of the decision tree. It can be seen that maintenance tactics are defined first on the basis of the consequence of the failure and second on the failure type (wear out, random, burn-in, or early life failures).

RCM decision trees such as that shown in Figure 9.9-4 are extremely useful for defining the type of maintenance activity to apply. However, they do not help to define the frequency with which these activities should be applied.

A trade-off exists between the frequency with which preventive maintenance is scheduled and the cost of lost production incurred. If maintenance is insufficient, the result is a high likelihood of breakdown and significant production losses. At the other extreme, overmaintaining an asset also results in production losses. As Figure 9.9-5 shows, there exists an optimal frequency with which to schedule preventive maintenance activities. Determining this optimum, however, is not a trivial task and will not be dealt with in this chapter. Readers are referred to Jardine and Tsang (2006).

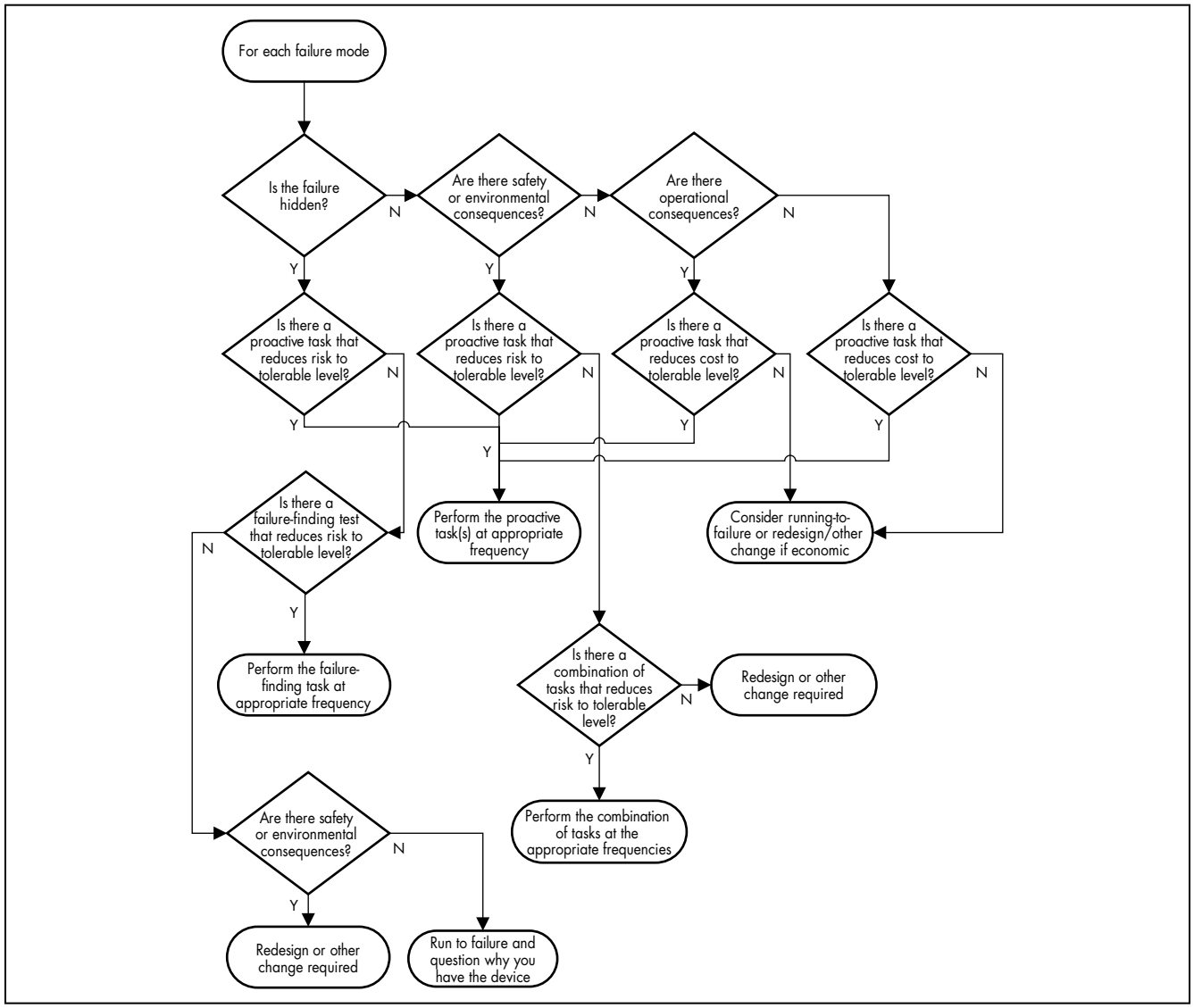
LIFE-CYCLE COSTS AND MAINTENANCE STRATEGIES

The root causes of reliability problems (Figure 9.9-6) are not just due to the maintenance process. The way infrastructure assets are specified and designed, the price and quality criteria adopted during the purchase process, the way certain assets are stored (e.g., shafts of electric motors must be regularly rotated to prevent flattening of ball and roller bearings), the way in which installation and commissioning are controlled, and the way equipment is operated all contribute to potential failures during operation. World-class maintenance organizations use root-cause analysis techniques to pinpoint the potential causes of failures. They then design specification, purchasing, warehouse, and operating procedures aimed at reducing the incidence of failure during operation. Maintenance activities that involve design modifications are known as design-out maintenance.

SYNCHRONIZING THE MAINTENANCE PLAN WITH THE PRODUCTION PLAN

It is important to synchronize the mine infrastructure maintenance schedule to that of the mine production plan and the production equipment maintenance schedules. Key to this is the implementation of a process to determine maintenance and repair priorities. Such processes are usually dependent on the evaluation of the effects and consequences of failure modes.

One way to determine the criticality of each failure mode is to quantify the expected frequency and impact of the failure



Source: Reyes-Picknell 2005.

Figure 9.9-4 Simplified logic for reliability centered maintenance

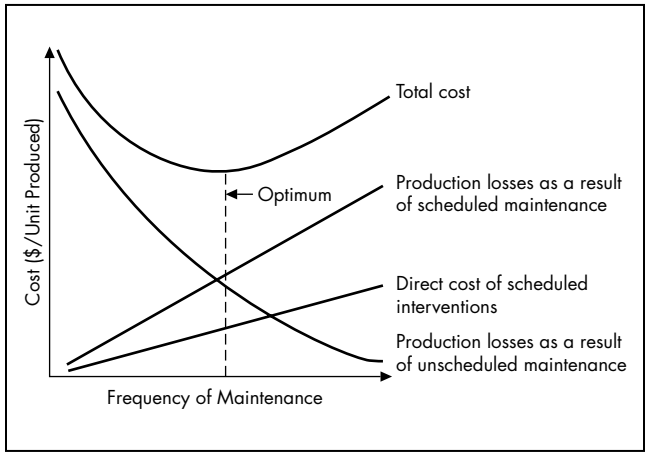
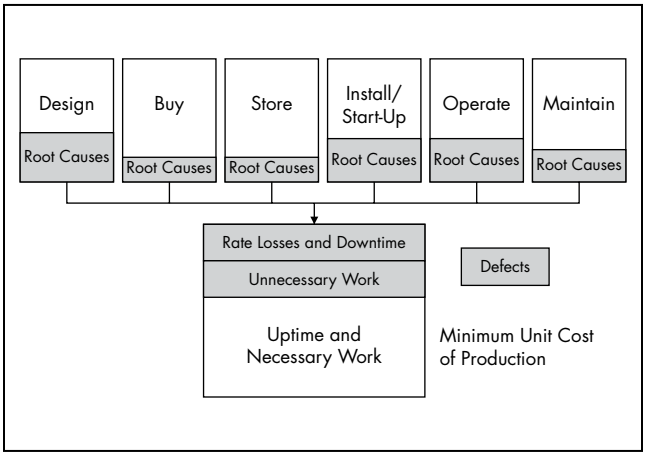
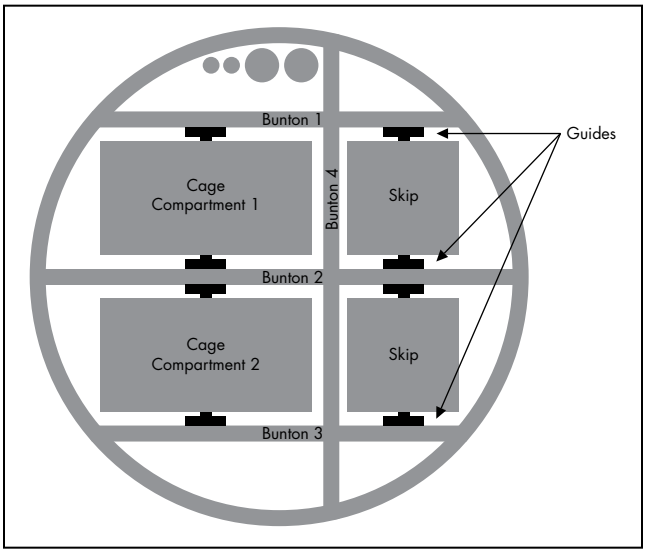


Figure 9.9-5 Optimal frequency of preventive maintenance



Source: Moore 2008.

Figure 9.9-6 Life-cycle reliability approach



Source: Northcote 2004.
Figure 9.9-7 Typical shaft cross section

and determine whether, during normal operating conditions, the failure will be evident to an operator. Quality Standard QS-9000 guidelines assign a rating using an integer scale of 1 to 10 to the frequency, severity, and detectability of each failure mode (Stamatis 1996). The severity rating needs to consider safety, environmental, and economic impacts. A risk priority number (RPN) can then be calculated for each failure mode by multiplying the three assigned ratings.

Grouping of the resulting RPN values allows low-, medium-, and high-priority maintenance and repair tasks to be determined. This in turn helps in scheduling. High-priority tasks have greater reason to interrupt production for immediate attention. Medium-priority tasks can be grouped and completed during the next scheduled preventive maintenance, whereas noncritical tasks can often be deferred until the next overhaul window.

MAINTENANCE PERFORMANCE MEASUREMENT

The cost of mine maintenance and its influence on the effectiveness of the total system can be significant. Mine managers need to know how outputs from investments in the maintenance of infrastructure will contribute to business goals. Without any formal measures of performance, it is difficult to plan, control, and improve. The main purpose of performance measurement is to determine whether systems and organizations are performing as they should. This helps determine resource allocation and subsequent control of outcomes.

One way to compare outcome measures with performance targets is to benchmark performance with competitors. *Maintenance performance indicator* (MPI) is the term used to describe an indicator that is a product of several metrics and is used to measure maintenance (Wireman 1998; Parida and Kumar 2006). Mining companies use many different forms and types of MPIs to measure the performance of maintenance processes. MPIs show the value of the contribution of maintenance process toward the ultimate objective of the company as a whole.

MINE INFRASTRUCTURE EXAMPLES

The following discussion deals with specific applications of mine infrastructure maintenance related to the provision of safe access to the ore body via shafts and roadways in underground mines. It also addresses the maintenance of electrical distribution and dewatering systems in underground and surface mines.

Mine Hoisting Systems

Access to ore reserves in underground mines is achieved either by declines or by shafts. The economic cutoff depth for decline haulage depends on a multitude of factors including production rate, ore grades, metallurgical recovery factors, truck capacity, and haulage speeds. Deeper underground mines are serviced via shaft hoisting systems. Apart from providing access to the mine's ore body, the shaft also serves as an opening for mine ventilation and a conduit for mine services such as water and compressed air systems. Shaft availability is invariably at a premium, as downtime required to inspect, maintain, and repair shafts directly translates to an opportunity cost of lost production. Engineers charged with shaft maintenance are tasked with achieving an appropriate balance between downtime costs and shaft integrity.

This section describes the fundamental maintenance management requirements of hoisting systems for underground mines.

Major Hoist Components

A hoist system consists of a winder, headframe, shaft steelwork and guides, ropes, conveyances, and counterweights. Shafts are typically divided into several compartments. Figure 9.9-7 shows a typical shaft cross section with cages for personnel and equipment access and skips for the conveyance of ore and waste. The winder plant consists of winder drums, a braking system, drive motors, and control system.

Shaft Structures

This section draws heavily on the work of G.J. Krige (2005).

The maintenance of shaft structures is becoming an increasingly important factor in the operation of underground mines. Because profit margins in deep underground mines are tight, the life of old shaft infrastructure is increasingly being extended.

Two key parameters are strength and functionality. Shaft guides must have sufficient strength to resist the impact forces resulting from the passage of conveyance and they must be functional in guiding a conveyance without derailment.

Shaft Structure Deterioration Mechanisms

Engineers charged with the maintenance of hoisting infrastructure are frequently called on to assess the suitability of shaft equipment with various levels of deterioration for safe hoisting. Thus it is necessary to be able to identify, measure, and manage the principal deterioration mechanisms. These mechanisms are caused by operation of the shaft or by rock movements due to mining or seismic activities.

Deterioration from Shaft Operation

The following deterioration mechanisms are related to the operation of shafts.

Steel corrosion. The most common deterioration mechanism in deep shafts is corrosion of steel comprising shaft buntons and guides, pipes and pipe support brackets, and station or platform structures. Generalized corrosion affects the entire exposed area of all steel structures, and localized corrosion affects specific areas of the structure.

Generalized corrosion results from ambient atmospheric conditions within the shaft, including the gaseous content,

humidity, and temperature of air. Typically, shafts with upcast air are more prone to generalized corrosion than shafts with fresh downcast air. Generalized corrosion uniformly reduces steel thickness and leads to a uniform reduction in the residual strength of steel components.

Localized corrosion results when corrosive chemicals and moisture are trapped in pockets where spillage can accumulate—cable trays, flat horizontal surfaces, or at the ends of buntons—as a result of water running down the shaft lining. Localized corrosion results in the reduction of strength of the steel member at a particular point.

Mechanical damage. Mechanical damage may be caused by falling objects in the shaft or by collision of shaft conveyances and shaft steelwork. Minor mechanical damage may also be caused by spillage, but it seldom causes significant loss of strength or functionality of steelwork. Major mechanical damage often causes complete loss of functionality without significantly impairing strength.

Fatigue damage. Fatigue damage of steel shaft structures is caused by repeated loading cycles during the operation of the shaft. The loading cycle results from the passage of conveyances within the shaft. Load magnitude is determined by guide misalignment, bunton stiffness, hoisting speed, and conveyance effective mass. Fatigue cracking at welded joints is a real possibility, usually occurring at or close to connections. Loss in strength and functionality is similar to that caused by localized corrosion.

Wear. Wear is a deterioration mechanism usually confined to conveyance guides. Wear does not compromise strength, but it may affect functionality. Localized wear may be indicative of misalignment of buntons due to rock movement.

Deterioration from Movement

Deterioration mechanisms related to rock movements or seismicity within a mine include those described in the following paragraphs.

Vertical strain. Ongoing mining can progressively reduce the area that has been supporting the weight of overlying rock, resulting in vertical compressive strain. This vertical strain in the rock can be induced into the shaft guides. A compressive strain causes bowing of the guides between bunton levels. Strain limits should not exceed the closure of permissible gaps left between shaft guides.

Horizontal ride. Some shafts are constructed to pass through the ore body. This is typical of reef mining as practiced in South Africa. Redistribution of stresses following orebody extraction can result in horizontal dislocation of the shaft above and below the reef. It may lead to severe distortion of the buntons installed between the hanging wall and footwall of the ore body.

Vertical closure and squeezing. Changes in rock stress caused by mining can produce vertical and horizontal displacement of the shaft. Unless adequate provision has been made for vertical closure during the design and construction of the shaft, shaft steelwork can become severely distorted. Redistribution of horizontal stresses as a result of mining activities can cause inward pressure on shaft walls, squeezing or distorting the shaft section. This leads to cracking or crushing of concrete in the shaft lining, buckling of shaft steelwork and piping, and misalignment of guides. Changes in horizontal stress state have also reduced clearance between shaft conveyances and shaft steelwork or walls, increasing the likelihood of impact of the conveyance with steelwork.

Shaft Structure Maintenance

A three-step strategy is proposed for defining the maintenance strategy for deep shaft structures:

1. Define limits and requirements using analytical techniques. This means that the misalignment and deflection of deep shaft structures must not exceed the specified maximum. Analytical techniques based on well-established shaft design procedures (e.g., SANS 10208-3 [2001]) can be used to establish acceptable limits to deterioration.
2. Implement measurement and inspection procedures. Information must be captured, recorded, and reported on a regular basis by mine personnel working in the shaft who know the shaft well. The deterioration condition should be described within a limited number of categories. There should be enough categories to enable meaningful trending while still allowing memorization and simple categorization. Table 9.9-1 lists six shaft categories developed by Anglo American Corporation (Krige 2005). Color coding can be used as an effective reporting means, with green indicating structural members classified as category 0 or 1 condition, yellow categories 2 and 3, and red categories 4 and 5.
3. Monitor trends and schedule maintenance. Most shaft deterioration takes place slowly, so changes may not be noticed. It is important to track deterioration over time (Table 9.9-2). Gradual movements of bolts in slotted holes and gaps between structural members may be observed, and, quite suddenly, further ground movement can result in buckling of buntons. Corrosion can progressively reduce the stiffness of conveyance guiding systems over many years. Again, quite suddenly, with no specifically observed, changes, conveyances may start to derail because reduced stiffness has led to dynamic displacement of the conveyance exceeding the upper limit.

Condition Monitoring of Guides

Hecker (2005) provides a useful summary of predictive maintenance techniques used to assess guide alignment. These methods include the following.

Plumb lines. The most common method of aligning shaft steelwork is based on vertically suspended plumb lines. Often the distances to be measured are large, causing plumb lines to sway because of air movement in the shaft. To overcome this, steady-state brackets can be installed to hold the lines still at the points in the shaft, and the plumb bobs can be suspended inside oil pots. Because lines have to be suspended outside of compartments, distances often have to be measured at awkward angles. Both factors can lead to measurement errors.

Laser beams. Calver (1982) described a method of mounting a laser onto a guide. A target is placed on the same guide at a distance away from the laser. This distance is chosen such that refraction caused by water in the air and temperature gradients is minimized. The target is then moved progressively closer to the laser and fixed to the guide at intermediate positions. At each position, the target is adjusted so that the laser beam coincides with the center of the target, allowing a deflection profile of the guide to be established.

Table 9.9-1 Shaft condition categories

Condition Category	Description of Steelwork Condition	Use of Shaft
0	New condition; no deterioration	No precautions necessary
1	Good condition; some deterioration but no loss of strength	No precautions necessary
2	Reasonable condition; less than 5% loss of strength	No precautions necessary; minor maintenance advisable to retard rate of deterioration
3	Poor condition; loss of strength of between 5% and 20%	No precautions necessary; planning of major maintenance advisable to protect asset
4	Severe deterioration; loss of strength of between 20% and 50%	Reduced speed hoisting advisable; planning and execution of major maintenance necessary
5	Critical deterioration; loss of strength exceeding 50%	Expert assessment of shaft use necessary; execution of major maintenance critical

Source: Krige 2005.

Table 9.9-2 Recommended maximum period between inspections

Condition Category	Period Between Inspections by Shaft Personnel	Period Between Inspections by Structural Engineer
0	2 years	3 years
1	2 years	3 years
2	1 year	2 years
3	1 year	2 years
4	6 months	1 year
5	2 months	6 months
Special condition	As required by nature of the problem	As required by nature of the problem

Source: Krige 2005.

Inertial platform. Inertial platforms were first developed in Belgium and South Africa. Such a system consists of a platform suspended from a conveyance. Proximity transducers on the platform measure the distance between the platform and the guides. Accelerometers, and sometimes gyroscopes, measure platform motion. Double integration of the accelerometer signals can be used to calculate the path that the platform follows down the shaft. The deflection profile of each guide can be calculated by adding the distance between the platform and the guide to the amount by which the platform deviates from the vertical in each direction.

Moving-beam systems. Moving-beam systems consist of beams running along the guide rails. Guide curvature can be measured by recording the distance between the center of the beam and the guide face, or by measuring the inclination of the beam. The guide shape can then be calculated from the successive curvature readings obtained (Hecker 2005).

Hoist Ropes

Wire hoisting ropes are constructed of an assembly of steel wires that operate at high stress levels and are almost invariably subjected to fluctuating rope loads. A significant source of stress fluctuation is the repeated bending and straightening of ropes as they run over sheaves.

Greenway (2005) provides the following list of factors affecting the life of winding ropes:

- Manufacturing standards and quality
- Rope design and selection for envisaged duty
- The provision and selection of a satisfactory operating environment
- The control of loading of in-service ropes

Rope safety is affected by the following factors (Greenway 2005):

- Control and limitation of manufacturing defects
- Effective assessment of rope condition to identify the optimal moment to discard the rope
- Detection of abnormal rope damage
- Control and limitation of severe peak rope loads

Chaplin (2005) lists the following factors that contribute to degradation and fatigue of hoisting ropes.

Tensile fatigue. Tensile fatigue results from the fluctuation of axial tensile loading. Changes in the attached mass and acceleration are the primary sources of axial load fluctuation. For this type of fatigue, dominant parameters are the tensile load range, mean tensile load, rope construction and wire grade, environment (including lack of effective lubricant or exposure to corrosion), and manufacturing quality.

Bending-of-sheaves fatigue. This form of fatigue involves repeated bending while under constant tensile load. The primary sources of stress fluctuation in this mechanism are the local changes in wire curvature as the rope adjusts to the radius of the sheave or the drum. Restriction of the source of fatigue stresses to change in wire curvature requires that individual wires can slide with respect to one another. Any constraint to this freedom, or ineffective lubrication or internal corrosion, will impair fatigue endurance.

Free bending fatigue. Free bending fatigue involves fluctuating bending deformation that does not involve contact with another body and that is typically excited by system dynamics. In fixed-rope applications, this type of bending often takes place adjacent to a termination.

Torsion fatigue. A consequence of the geometry of many ropes is that they generate a torque in response to a tensile load when the ends are constrained. Some rope constructions are designed to minimize this tendency to rotate, but these can have other disadvantages such as not being as robust. Under certain applications of wire ropes, such as moorings of offshore systems, torsion fatigue can be a dominant mode of failure.

Combined modes. In practice, life of a hoisting rope is dictated by a combination of fatigue modes, usually tensile and bending-of-sheaves fatigue. Briem (2005) provides a

useful method for estimating the fatigue life of wire hoisting ropes on drum winders as defined by a limiting number of single bending cycles.

Hoist Rope Inspection Procedures

To ensure safe operation, mining companies are required to inspect hoist ropes. Visual inspections are performed daily and electromagnetic induction tests are performed every three to four months. During visual inspection, rope speed is reduced so that an inspector can look for visible signs of rope defects. This typically results in the loss of about half an hour per day of shaft time.

Electromagnetic (EM) analysis of hoist ropes uses a data capture system that wraps around the head of a rope, allowing it unhindered passage. The EM head induces eddy currents within the rope, and a signal or trace is returned that is proportional to rope section integrity and thickness. Local faults are generally detected as local spikes on the EM trace, and loss of metallic area shows up as a sustained dip in the trace.

Young and Delaney (2005) describe the development of an automated hoist-rope inspection system in which an EM detection head was integrated with a vision system and hoist drum encoders. Pattern recognition software linked to the camera data picks up visible signs of deterioration. Digital signal processing of the EM traces is as outlined previously.

Underground Roadway Maintenance

Haulage roads in any mine are an integral part of the production system and make a significant contribution to the mine productivity, safety, and economics. A haulage road can be broadly classified in one of three categories:

1. Permanent haul road
2. Service haul road
3. Temporary haul road

All haulage roads need a carefully planned maintenance program. Otherwise, they can turn into a major production bottleneck.

Underground roads provide access from the shaft or decline and the ore body. Good roads are essential to moving personnel and materials around an underground mine. Poor roads result in maintenance downtime, which restricts access to parts of the ore body. The costs include lost production; damage to mobile equipment, resulting in increased tire and vehicle repair costs and lack of vehicle availability; and road-related accidents, resulting in injury and compensation claims.

Degradation Mechanisms

The following text draws heavily on a paper by Logan and Seedsman (1995).

Road defects are the result of interactions of the excavation process, pavement materials, traffic, environmental conditions, and road construction. Causes can be linked to poor floor horizon control, lack of drainage, poor compaction, or the use of inappropriate material to carry high-axle loads.

Water plays a critical role in the degradation of underground roads. Only small quantities of water are required to deteriorate pavements. Key degradation mechanisms include the following:

- *Potholes*, which form when induced water pressures cause layer separation. Areas where poor drainage has led to water retention are particularly susceptible to potholes.
- *Ruts*, which are deformations in wheel paths in parallel directions. This is a characteristic of soft clay floors.
- *Shoving*, whereby road material is moved sideways by cornering, braking, or sliding wheel forces.
- *Slippery surfaces* caused by presence of clay materials or sediments on the surface of the roadways.

Maintenance of Underground Roadways

Maintaining underground roadway surfaces has three main objectives:

1. To provide a good riding surface
2. To minimize safety hazards for vehicular traffic
3. To provide a free-draining surface

As a road deteriorates, several maintenance procedures of moderate cost can be used to prolong its life. These include grading, compaction, drainage maintenance, and resurfacing.

Grading profiles. Logan and Seedsman (1995) regard grading as the single most important function in maintaining an underground road network. Regular grading to maintain crossfalls, eliminate surface defects, and restore drainage profiles reduces the effects of surface water. Lack of adequate crossfalls is one of the principal problems associated with roads. Crossfalls should normally be in the range of 1:25 to 1:17.

Compaction. After grading, compaction of pavement materials can assist pavement performance and minimize maintenance intervals by increasing the strength of the pavement and its resistance to water.

Drainage. These maintenance tasks for drainage systems are essential:

- Checking that flow from drippers is collected and directed away from the pavement
- Cleaning roadway drains of debris to ensure free flow of water away from the pavement
- Regularly inspecting and maintaining roadway pumps to ensure effective operation and clear the inlet of debris
- Regularly cleaning settling pumps to ensure effective pump operation and prevent overflow
- Checking and flushing drain lines to prevent sediment buildup and reduce duty on pumps
- Locating and addressing new sources of water

Resurfacing. Pavement resurfacing may be required as a result of degradation of the road aggregate, incorrect selection of pavement material, traffic abrasion, shoving and mixing of the road base with weaker material, or scouring and erosion due to water flow. If resurfacing is required, the weaker material should be removed and new material should be added and mixed with existing strong materials. The pavement should then be shaped with a grader and compacted, and drainage should be reestablished.

Electrical Power Distribution Systems

This section describes the essential features of electrical power systems used in surface and underground mines and their maintenance management. Because mining operations are highly mechanized, mines and processing plants consume large amounts of electricity. The mining engineer is expected to be aware of the utility, applications, limitations, and risks involved in the use of electrical power, as well as the maintenance of electrical infrastructure.

Alternating current (AC) is the standard power supply, although direct current (DC) is used in certain specific applications such as motors requiring speed control. The transmission and distribution of AC power requires voltage variation using a network of transformers. The main transformer on-site receives grid power to the mine site. Alternatively, diesel electric AC power generators can be used in areas where the electrical grid power is inaccessible.

Mining operations require mechanical power for most work, and mechanical power is often facilitated by the use of electrical motors to drive various accessories—for example, material-handling equipment, ventilation fans, and hydraulic equipment such as rock drills. Therefore, mining engineers need to have an understanding of the maintenance aspects of motors and motor starters, transformers, and distribution cables that power mining equipment. To a very large extent, predictive maintenance methods are replacing the traditional usage-based maintenance in the management of electrical infrastructure in mines.

Stages of Power Distribution and Power Factor

Most mines are large consumers of electrical power. If the power is supplied by the electrical mains grid where the transmission voltage is 33 kV or more, the voltage is usually stepped down to 11 kV using a main transformer located on-site. Alternatively, if the power is supplied by a diesel generator, the voltage may need to be stepped up to 11 kV. Power is then distributed via a network of feeder cables to substations located on the surface facilities and in underground locations. The substation transformers modify the voltage as required by the applications. In surface mines, the main consumer is electrically driven mining equipment such as draglines and shovels. In underground mines, the main consumers are ventilation fans, large drilling machines used for mine production and rock reinforcement, and dewatering pumps. Generally a substation transformer supplies power to all the applications in its locality.

The power factor is one of the significant parameters that should be monitored in the use of electrical power. It is defined as the ratio by which the product of voltage and current is multiplied to obtain the true electrical power (Rizzoni 2007). Hence

$$\text{power factor} = \cos \Phi = \frac{\text{true power}}{\text{apparent power}}$$

where $\cos \Phi$ is the cosine of the phase angle between the current and voltage vectors. The product of V and I is called apparent power, and

$$P_T = V \times I \times \cos \Phi$$

where P_T is true power, V is voltage, and I is current.

The power factor is affected by the inductive and capacitive components of the electrical infrastructure, which includes both the distribution system and the consuming appliances. If the power factor is low, the current increases to supply true power given in kilowatts. Increasing the power factor close to unity is significant in any power distribution system to limit the current flow through the circuit. This is known as power factor correction. In practice, most circuits are inductive and as a result cause currents to lag behind the line voltage. A circuit with a leading current (capacitive circuit) is seldom found. However, these effects do occur in long-distance power lines. These are the two major causes of low power factor:

1. Lightly loaded electric motors and transformers (this equipment should be designed to run at or near full load)
2. Fluorescent lighting circuits (these lights have to be considered carefully and steps taken to improve their power factor)

Correcting the power factor is part of daily maintenance in a large power consumer. The optimum power factor achievable is about 0.85.

Safety Aspects of Electrical Power

A range of protection devices and methodologies may be applied for the safe use of electrical power. To maintain safety, it is vital that these devices be properly installed, inspected, and maintained and that correct practices be followed in their use.

Earth Leakage Protection

Earth fault protection is a crucial aspect in the use of electrical power in underground and open-pit mines where high-voltage electrical cables are frequently used to power mining equipment. The main purpose of earth fault protection is to safeguard personnel and electrical apparatuses. Unless properly controlled, the occurrence of an earth fault can be hazardous at a mine because it may cause

- Voltages from machine frame to earth, which are dangerous to personnel; or
- Electric arcing underground, which may initiate an explosion or fire.

The most common apparatus faults in mines occur in power cables, as they are susceptible to damage caused by heavy equipment. This is particularly true of flexible trailing cables that supply power to mobile mining machines. They are laid over ground or close to ground level, and it is almost impossible to prevent their damage by road traffic and during relocation of the cables. Cable construction is such as to provide every phase conductor with an individual conductor screen so that crushing would expose a single phase, creating an earth fault current. A protective device such as an earth leakage relay would then isolate the cable and contain the sparking within the cable before the occurrence of a heavy short-circuit current due to a phase-to-phase fault. Coordination among protective units is introduced to allow the unit closest to the fault to isolate the faulty circuit without causing the healthy part of the system to be deenergized.

Effect of Current Exposure to Human Body

Though desirable, it is almost impractical to provide automatic protection against electrocution as a result of direct contact with a live conductor, particularly where the electrical reticulation is exposed to a humid or damp atmosphere. To appreciate this, it will be necessary to study the effects of current values on human beings (Table 9.9-3).

As the table indicates, passage of current as low as 20 mA through the human body can cause loss of muscular control to the extent that an individual cannot disengage from the live conductor, and a current in excess of 50 mA is sufficient to produce a critical heart condition from which there is little or

Table 9.9-3 Effect of current exposure on the human body

Current Level, mA	Effect
<1	No sensation
1–8	Sensation of shock, not painful; individual can let go at will, as muscular control is not lost
9–15	Painful shock; individual can let go at will, as muscular control is not lost
16–20	Painful shock; muscular control of adjacent muscles lost; individual cannot let go
21–50	Painful shock, severe muscular contractions, breathing difficulty
51–200	Ventricular fibrillation (a heart condition that may result in death)
201 and over	Severe burns; muscular contractions so severe that chest muscles clamp heart and stop it for the duration of shock (this prevents ventricular fibrillation)

no chance of recovery. For effective safeguard against electrocution, it would be necessary to introduce earth leakage protection designed to operate with a fault current below 20 mA, which in most cases would be impractical.

Methods of Earth Fault Protection

Two main methods are used for earth fault protection. They are earth leakage protection and earth continuity protection. The earth leakage protection devices are activated by any unbalanced current of any phase of a three-phase circuit caused by an earth leakage. The leakage current should not exceed 1 or 2 A for most applications, and it is limited to 30 mA for portable apparatuses operated by personnel (Mines Safety and Inspection Regulations 1995). The principal purpose of earth continuity protection is to ensure a continuity of earth conductors throughout the length of the supply cable and associated connectors. All trailing cables have either a separate central conductor called a pilot wire or they use a pilot-wireless monitor through the cable conductors. The pilot-wire/pilot-wireless systems are connected to a solid-state frequency-controlled system to ensure the continuity of the ground conductor. In case of damage, if the signal level becomes too low (indicating excessive resistance in the pilot-wire/pilot-wireless system) or the frequency is outside tolerance, the monitor indicates a fault condition and opens (i.e., trips) the control relay. Most mine regulations specify that automatic protective devices must be tested periodically for proper operation (Mulvaney 2008).

Other protection devices that ensure safe power usage are surge diverters, which protect the circuit from overload surges such as those due to lightning; circuit breakers, which respond to overloads and quench the resulting arc in a way that will not affect other parts of the system; fuses, which open the circuit when excessive current flows through it; and manually operated isolators and switches.

Maintenance of Major Components Used in Electrical Power Reticulation

The major components that require maintenance are the transformers, switchgear, and protective equipment. The protective equipment is also used with ventilation fans, pump motors, electrically powered mining machines such as rock drills, and trailing cables. Usage-based maintenance is commonly practiced in many mines. However, predictive maintenance based on condition monitoring improves reliability and should be implemented wherever possible. The following points relevant to the maintenance management of major components are suggested:

- Transformers use air or oil as a medium for cooling and insulation. Regular monitoring of the condition of oil is useful as a predictive tool to determine the performance of the transformer. Oil analysis gives information about the integrity of insulation of the transformer windings, dissolved gases resulting from any arcing within the windings, water contamination, and the degree to which the oil itself has deteriorated. Arcing may occur as a result of breaking down of insulation properties in oil. A thorough visual examination for oil leaks, including temperature measurement and a test of relief valve operation, should be carried out periodically.
- As trailing cables are subjected to frequent movement in a harsh mining environment, they accumulate damage over time. The insulation resistance is one indicator of the accumulated damage. Hipot or megger testing are popular insulation test methods practiced by many companies. However, both methods damage the cable and reduce its operational life. It is not recommended to conduct destructive testing of cable insulation. IEEE 400-2001 (2002) states that partial discharge testing is nondestructive and more accurately gauges insulation integrity. This information is useful for timely replacement of the cable. Trailing cables should be equipped with both earth leakage relays and earth continuity relays.
- Mines have typical apparatuses such as dewatering pumps and ventilation fans. Suppliers of this equipment usually provide maintenance information. However, it is the responsibility of the mine's personnel to perform the routine maintenance.
- Dewatering pumps must be equipped with devices such as flow-sensing relays, reservoir water level sensors, and position indicators of inlet and outlet valves to make sure that the pump does not engage unless there is adequate water. The proper functioning of these devices should be checked periodically. Portable dewatering pumps are not generally equipped with all the flow-sensing devices. However, they are usually fitted with current-sensing devices that stop the pump if there is a large variation of the load such as that caused by loss of water. Also required are regular lubrication of motor bearings and switching contact maintenance of motor starters.
- Primary ventilation fans are heavy electromechanical machines. The bearings on which the motor shaft is mounted are subjected to wear resulting in increased vibration and heating. A periodic measurement of the bearing temperature and vibration monitoring of the fan reveals the condition of the bearings and the dynamics of the fan rotation, allowing prior action to be taken before a failure occurs.
- Rock drills are subjected to heavy vibration. Therefore in addition to periodic checking of the trailing cable, the electric panel should be checked for loose contacts.
- Motor starters should be selected considering the applications. For example, variable-frequency-drive starters help to regulate the motor speed, and they can be very cost-effective for use with primary and secondary ventilation fans where constant full-speed operation is not required.

Also, these starters work well with automatic and remote control systems. Other types of starters commonly used are soft starters having electronic control of starting voltage, star-delta starters, and direct on-line (DOL) starters. The DOL starters are suitable only for low power applications, as the starting current of the motor can be as much as seven times the full load current. Because of the presence of moving parts and the heavy current experienced by starters, wear is usually high, and the frequency of inspection and maintenance should be adjusted as required.

- Mine regulations usually specify that earth leakage relays need to be checked for proper operation at frequent intervals (e.g., monthly). Also, power cables and earth grounding must be checked periodically.
- The resources needed for maintaining the electrical infrastructure depends on the extent and complexity of the electrical equipment. Most mines require a lead hand who can also act as a supervisor, and other electricians. At least one electrician shoud be required to work during each shift. Specialized contractors may be needed for specialized or statutory work such as that dealing with high voltage. Also required are an electrical maintenance shop and an inventory of consumables and test equipment.

A reliability-focused maintenance plan should be developed to keep the mine electrical system in good working order. Necessary precautions must be taken to minimize the risk of earth fault, as mines use flexible power cables and associated gear for power distribution. Mine regulations stipulate the periodic testing of earthing systems, protective devices, cables, and other electrical equipment, and use of a logbook is required to maintain all testing and examination records. During maintenance of mining equipment, personal danger tags are used to communicate with equipment users that their equipment is being serviced.

Dewatering Systems

Open-pit mines vary in size and depth. Large pits can be several kilometers in length and width and up to a kilometer deep. In most instances, the depth is greater than that of the natural water table. A variety of methods are used to remove water ingress. Some mines remove water continuously via bore pumps located around the pit. Others pump away water collected in a sump at the bottom of the pit. Regardless of the method employed, it is necessary to use one or more pumps that can remove the peak water ingress that may occur during the rainy season. This will keep the drawdown level (i.e., depressed water table level) below the pit depth. Dewatering of pits situated close to lakes or nearby abandoned mines can be challenging if seepage occurs.

Selection of Pumps Used for Dewatering

Many types of pumps are used to remove water from mines. The factors that determine the type of pump are flow rate, pressure or elevation, density of water, and suction head that should not exceed a given limit. Centrifugal pumps are adequate for most dewatering applications and are commonly used for lifting water up to about 300 m (1,000 ft). Multistage pumps can be used for removing water at greater depths (Munson et al. 2006). In underground mines, positive displacement rotary pumps are popular because of their self-priming capability. Plunger and diaphragm pumps are also used.

The design of centrifugal pumps is a highly developed field. Actual pump characteristics are experimentally determined through tests and supplied by the pump manufacturers. Typically they include a graphical relationship between the total head and the discharge as well as identification of the zone where the pump efficiency is high. For a given head and a discharge, a suitable pump can be selected considering its efficiency as well. Furthermore, pump characteristics also include what is known as net positive suction head. This is useful to determine the maximum suction height of the pump while allowing for pressure losses in the pipe.

Head Loss in Pipes

Head loss in piping should be taken into account in determining the overall pressure head required for pumping. This consists of the pipe friction loss and the shock loss when the fluid passes through sharp bends and turns (which should be eliminated or minimized during construction and maintenance). Pipe manufacturers usually provide the friction losses. Piping handbooks give the shock loss factors applicable to sharp changes in the piping geometry (Crowe et al. 2008). Alternatively, losses due to bends and other geometric changes of a pipe can be expressed as an equivalent increase in the length of the pipe subjected to frictional resistance. Often it is more convenient to include the effect of geometric changes as an added length, which is provided by the pipe manufacturers.

Planning and Managing Surface Mine Dewatering

For mining to be carried out without the hindrance of groundwater ingress, many managers seek to reduce the level of the water table below the depth of the mining pit. The usual strategy for doing so is to pump from several bore water pumps before constructing the surface mine and during mine operation. Alternatively, the water collected in sumps at the pit bottom is removed in some mining operations. The following considerations can be useful during planning and operation of the dewatering system.

- The estimate of water ingress should be based on a hydrological study. The existence of nearby abandoned mines or large reservoirs such as lakes must be considered. Previous experience with groundwater in bores or drill holes in and around the location can also be useful, as can construction of test bores.
- Place of discharge should be decided considering the geology. If the ground is permeable, water must be sent to a reservoir far distant from the mine.
- Electric submersible bore pumps, which are available in a variety of sizes and capacities, can be used. Bore pumps are usually multistage centrifugal pumps. Although they are quite reliable, they can be easily damaged by letting them run without water and/or under cavitating conditions. Hence flow-sensing switches should be incorporated to ensure that pumps stop automatically when adequate water is unavailable. They may be reenergized after a preset time lapse. Pump manufacturers provide automatic pump controllers and alarm transmitters.
- In many sites, the yield of water increases substantially for weeks or months and then gradually lessens. Pumping should continue until the drawdown is below the pit bottom. The drawdown can be measured by inspecting the level of water in drill holes or well bores.

- In some cases, dewatering is done using a sump located inside the mining pit. Centrifugal pumps with horizontal- or vertical-axial configuration can be used as appropriate. However, they should be located near the sump so suction head restrictions are satisfied. Pumps driven by diesel engines are commonly used at some locations inside a surface mine if electrical power supply is unavailable. If the depth of the mine is great and/or the delivery distance is large, a transfer tank may be required on the ground level adjacent to the pit for temporary storage of water. Silt may be removed by sending it through a precipitation tank. In this case, secondary pumps are needed for delivery of water to the final destination.
- Pumps configured to operate in parallel increase the flow rate without increasing the total head. Those configured to operate in series increase the total head without increasing the flow rate.
- The main considerations in selecting a pump for dewatering are capability, reliability, portability, and wear resistance. Both reliability and wear resistance are affected by operational conditions.
- The correct grade of polyethylene pipe, one that can withstand the applied pressure, should be used. The size of the pipe should be determined considering the flow rate. A velocity of 3 m/s (9.8 ft/s) is an upper limit considering the frictional losses in a pipeline. A minimum velocity of about 1 m/s (3.2 ft/s) is required to prevent settling of solid matter inside piping. The settling velocity depends on the maximum size of the solid particles carried by water. To minimize damage to the piping and to reduce maintenance, all piping must be securely fastened. This is necessary because pipes tend to move about when subjected to internal pressure and in particular when dynamic transient pressures occur.
- If the delivery pipe passes through steep elevations, one should be aware of the possible occurrence of large negative pressure in the falling side of the pipeline. Vacuum relief valves may be required to prevent pipe collapse. Similar conditions may occur when the pump is suddenly stopped where the water is delivered through a long pipeline.
- The life of a pump, particularly if it is used for pumping saline or corrosive waters, is affected by the material it is made of. The expected mine life should be considered when pump material is being selected. Even a long-lasting and expensive pump, if it is operated under abrasive conditions, may require regular replacement of the impeller or volute chamber.
- Some important considerations in the design and maintenance of sumps and transfer tanks are to provide adequate excess capacity to cater for pump downtime, install trash screens and grit traps ahead of the pump intake, and facilitate the regular cleaning of deposited material in screens and grit traps.
- A vacuum gauge installed in the suction side of a pump gives useful information about its performance. An unusually high suction pressure may indicate a blocked suction pipe and/or cavitation occurring, whereas a low value may indicate that air has leaked into the suction pipe or that the pump has not been primed. Similarly, an unusually low reading in a pressure gauge installed in the delivery side may indicate no flow, excessive air in the water, and so on. An increase in the delivery pressure indicates a blockage or a closed valve in the delivery pipe. Excessive vibration in a pump may be an indication of an unbalanced or worn-out impeller, worn bearings, or a damaged shaft (Kuruppu 2008).

Table 9.9-4 Maintenance strategies

Type/Tactic	Details
Preventive maintenance	Interval-based maintenance including cleaning, inspection, and replacement of parts
Predictive maintenance	Measurement of performance factors such as flow rate, suction, and delivery pressures; maintenance and repair according to needs and prior to failure
Proactive maintenance	If failure is repetitive, reengineering of system to eliminate root cause of failure

Table 9.9-5 Factors to consider when selecting a maintenance strategy

Factor	Description
Reliability	At a minimum, the predictive maintenance strategy is necessary to ensure high reliability. In some cases, proactive maintenance may be necessary to eliminate the root cause of failure in the existing systems.
Availability and capability	Most equipment requires at least preventive maintenance. The service interval should depend on the demand for equipment (e.g., operational hours per week) and the recommended interval of parts replacement.

Maintenance Management of Dewatering Infrastructure

The maintenance management strategy should be designed to minimize the total dewatering cost over the planned life of the pit. This may include adding more resources such as equipment for performance monitoring, replacement of parts, routine cleaning and inspections, and reengineering of the existing infrastructure if the current system is not reliable. An optimum level of maintenance is that which corresponds to the lowest total dewatering cost. Table 9.9-4 gives a list of maintenance strategies commonly adopted by maintenance personnel. Table 9.9-5 shows the factors that should be considered when formulating a maintenance strategy. Reactive maintenance, which would be required in case of equipment failure, can be minimized by adopting these strategies.

Adequate resources must be available to meet the requirements of maintenance. The resources usually required are skilled personnel such as team leaders, plumbers and fitters, parts required for pumps and prime movers (e.g., diesel engines), accessories such as piping and pipe fixtures, and material-handling equipment.

In conclusion, dewatering of pits requires proper consideration of the rate of water ingress and the total pressure head needed for suction and delivery of water. The total head depends on suction height, delivery height, and friction loss in the piping. Once the pumping system is operational, the reliability of the pump can be adversely affected by air entrapment due to low water level and by abrasive material such as sand mixed with water. The latter causes rapid wear in the impeller and the housing. Other problems are clogging in the suction and delivery pipes and the occurrence of transient pressures in the delivery

pipe resulting from sudden interruption to pumping. To minimize damage to pumps and piping and the disruption of dewatering, these issues require proper planning, monitoring, and control.

Maintenance Workshop Facilities

For a mine system to operate successfully according to specifications, there must be a variety of supporting maintenance workshops and facilities near mining activities. These can be managed by the mining companies or, if the maintenance of mining equipment and systems is completely outsourced, by a private contractor.

These facilities consist of central workshops shared by repair and service personnel. Such workshops are equipped with all the necessary facilities—welding, painting, drilling lathing, metal drilling, machine lubricants, and so on. Minor repairs for mobile equipment are often performed at the site of operation, but when a piece of equipment needs to be overhauled or needs major maintenance and repair, it can be moved to central workshops. As mining systems become more extensive, the operation and management of the maintenance facilities must be integrated into mine operation functions. Infrastructure facility management is increasingly important for reducing mining systems' operating costs and improving their reliability. Judicious location and operation of such facilities will reduce the transport- and logistics-related costs involved in operating and maintaining mining systems.

CONCLUSION

A characteristic of many of the mine infrastructure elements is that they are either common components to production systems or are series connected. Because of this characteristic, the business risks associated with stoppages or service reduction due to poor maintenance can be significant. Due to design deficiencies or to operational and environmental stresses, infrastructure systems are not completely failure free, and they are sometimes unable to meet capacity requirements. Therefore, a high degree of maintenance preparedness for the mine infrastructure is necessary to meet performance targets. In some mines, this preparedness is best achieved by integrating maintenance organization into operations departments. In others, especially mines having a high degree of process complexity, the preferred solution is to maintain a separate engineering department.

With the potential reemergence of the minerals boom, infrastructure utilization will become of critical importance. This in turn will limit the availability and extent of opportunities to perform routine preventive maintenance of mine infrastructure, requiring continuous revision of tactics employed to maintain infrastructure assets.

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CHAPTER 9.10

Systems Engineering

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INTRODUCTION

Systems engineering is a field consisting of a broad range of techniques that can be used to quantitatively model, analyze, and optimize a system. Thus, it includes a whole range of methods, including modeling techniques such as discrete event simulation and artificial intelligence, analysis techniques such as statistical tests and Six Sigma, and optimization techniques such as linear programming and gradient methods. To a practitioner, however, the subject could be limited to the techniques that are directly applicable to their field. Because mining spans a variety of activities, the mining industry uses every systems-engineering technique available.

Because of space limitations, this chapter cannot cover the entire range of techniques relevant to the industry. For this reason the topics included were selected based on certain criteria. Topics are covered if they are relevant to large portions of the mining industry, especially if the topic can be usefully covered in brief. Most topics, however, do not fall under this category, and most are very complicated techniques that require entire books for proper presentation and are important only for a narrow application in the industry. Additionally, they may have been widely discussed in the literature (mining and otherwise). Therefore, these techniques are only introduced here, with the focus being on "good practices."

This chapter discusses data collection, modeling techniques, analysis techniques, and optimization.

CORE FUNDAMENTALS: DATA COLLECTION

Optimization is essentially an act of balancing competing constraints with the constraints being defined by data from the process. Because erroneous constraint definitions could result in gravely suboptimal optimization, it is imperative that all data be collected carefully and described accurately.

Measuring something fixed and definite, such as the length of a machine, is easy. However, measuring activities that are variable is not easy, because many measurements have to be made, with "many" not necessarily being a very clear number. Additionally, when measurements are expensive, there is a constant struggle between "how many" and "representative."

Determining Sample Size

This section discusses methods of determining the required sample size.

Minimum Number of Samples to Estimate Within a Certain Error Range

If the standard deviation, σ, of the process being measured is known, then the sample size, n, required to obtain an error range ($\pm$) of δ is calculated as follows (NIST/SEMATECH 2006):

$$n = z_{\alpha}^{2}\left(\frac{\sigma}{\delta}\right)^{2}$$

where z_{α} is the value on the normal distribution curve for a probability of α.

This equation assumes normality, and it also requires *a priori* knowledge of the process's standard deviation. If the distribution is not normal, but the number of samples exceeds 30, then the normality assumption will probably be fine. It is when the number of samples is low that a wrongful normality assumption is harmful.

Example 1. The time to cut a face was previously determined to be normally distributed in a mine, with an average cutting time and standard deviation of 43 and 15 minutes, respectively. Management would like to sample the cutting times before and after the modifications to know if the modifications did make an impact. How many samples (of cutting time) are needed if the average cutting time is to be estimated (at a 95% confidence interval) within 2 minutes of true mean?

Solution. A 95% confidence interval implies $\alpha = 5\%$, which yields $z_{\alpha} = 1.96$ (round off to 2). Given that $\sigma = 15$ and $\delta = 2$, to estimate the mean cutting time within 2 minutes of the true mean, management needs to take $(2)^2 \times (15^2/2^2) = 225$ samples.

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Sampling Proportions

Frequently, the focus of a small sampling exercise may be to see if a given machine or process is running or not. Sample values in such cases are binary, that is, a machine is running or not running, and the ultimate measurement is a proportion (such as the machine is running X% of the time). For such cases, the minimum sample size required is given by the following equation (NISH/SEMATECH 2006):

$$n = z_{\alpha}^2\left(\frac{pq}{\delta^2}\right)$$

where

p and q = two proportions (running/not running)
δ = margin of error in estimating the proportion

A time study for sampling proportions should include at least 10 successes and 10 failures. Studies focused on equipment reliability and statistics such as mean time between failures are directed to the literature on reliability engineering (O'Connor 2002; NISH/SEMATECH 2006).

Example 2. The main belt conveyor at a mine typically runs 90% of the time. It is down the rest of the time. Management would like to buy a new motor to improve reliability. How many samples (running/not running) should management take if they wish to estimate (with 95% confidence) the proportion of the time (±2%) that the belt is running?

Solution. A 95% confidence interval implies α = 5%, which yields z_{α} = 1.96 (round off to 2). Given that p = 0.90, q = 0.10, and δ = 0.02, then $n = 2^2(0.9 \times 0.1/0.02^2) = 900$ samples.

In this case there is an additional question (not addressed here): When should one collect the 900 samples? The answer to that question lies in the 90% estimate given in the problem. If management sampled the belt every hour to obtain the 90% estimate, then the 900 samples should be obtained on an hourly basis. If the belt fails very infrequently, the Poisson distribution discussion that follows should be considered.

Poisson Distribution

Poisson distribution governs sparse data (i.e., rare events). For a Poisson distribution, the minimum sample size is given by the following (van Belle 2008):

$$n = \frac{4}{\left(\sqrt{\mu_0} - \sqrt{\mu_1}\right)^2}$$

where μ_0 and μ_1 are the estimated means of two populations that are to be compared.

Example 3. A machine breaks down approximately every 10.5 hours. A new maintenance program is expected to increase the time between breakdowns by 2 hours. How many samples should be taken to conduct the two-mean comparison tests?

Solution. The number of samples needed, n, is as follows:

$$n = \frac{4}{\left(\sqrt{10.5} - \sqrt{12.5}\right)^2} = 45$$

Working with Any Sample Size

Sampling/measurement is useful even if a large number of samples cannot be taken. Comparison of means and confidence interval for means are very valuable tools whether or not a large number of samples are taken.

Example 4. In the previous example, management could not take 225 time-study samples on the continuous mining machine (CMM). Before the changes were made, 38 time-study samples (n_1) revealed the cutting time to be normally distributed with an average of 41 minutes (μ_1) with a standard deviation of 14 minutes (σ_1). After the changes were made, 32 samples (n_2) revealed a cutting time average of 38 minutes (μ_2) with a standard deviation of 11 minutes (σ_s). Did the changes improve the CMM (with 95% confidence)?

Solution. This problem is solved by performing the following four steps:

1. Compute the pooled degrees of freedom, ν, as follows:

$$\nu = n_1 + n_2 - 2 = 68$$

2. Compute the pooled variance, s^2:

$$s^2 = \frac{(n_1 - 1)\sigma_1^2 + (n_2 - 1)\sigma_2^2}{\nu} = \frac{37 \times 14^2 + 31 \times 11^2}{68} = 161.8$$

3. Compute the t-statistic:

$$\text{t-statistic} = \frac{\mu_1 - \mu_2}{\sqrt{s^2\left(\frac{1}{n_1} + \frac{1}{n_2}\right)}} = \frac{41 - 38}{\sqrt{161.8\left(\frac{1}{38} + \frac{1}{32}\right)}} = 0.98$$

4. Look up t-table for t_{critical} for α = 0.05 (two tail) and ν = 58. This gives t_{critical} = 2.0.

Because the computed t-statistic is lower than t_{critical}, one cannot claim that the changes made any difference.

Example 5. In Example 1, what is the confidence interval (95% confidence) for the obtained average cutting time in the first time study (n = 38, μ = 41, and σ = 14).

Solution. The problem is solved by performing the following steps:

1. Confidence interval for mean = $\sigma/\sqrt{n}$ = $14/\sqrt{38}$ = 2.71
2. t-statistic for 95% confidence interval (α = 0.05, two tail) = 2.0
3. Confidence interval = $\pm 2.0 \times 2.71 = \pm 5.4$

The time study truly implied that the average cutting mean is between 35.6 (i.e., 41 – 5.4) and 46.4 (i.e., 41 + 5.4).

Time Studies

When the minimum sample sizes are known, the next step is to actually collect the samples. Time studies are a common way that processes are sampled. Simply speaking, time study is the process of measuring the frequency and duration of activities. Time studies can be very insightful, as they quantify the interaction between activities. Even though most interactions are known, the magnitude of the interaction can sometimes be a surprise. When properly designed, time studies can be used to identify bottlenecks, set performance standards, and guide system redesign. A typical sequence for a time study is as follows:

1. **Identify the goals.** Cleary spell out the goals of the study.
2. **Gather intimate knowledge of the system.** Visit the location, meet people, and identify potential hazards. Learn about factors (such as shift changes) that may compromise the integrity of the study.
3. **Plan for the act of time study.** Identify observation locations, prepare for the environment (moisture, dust, noise,

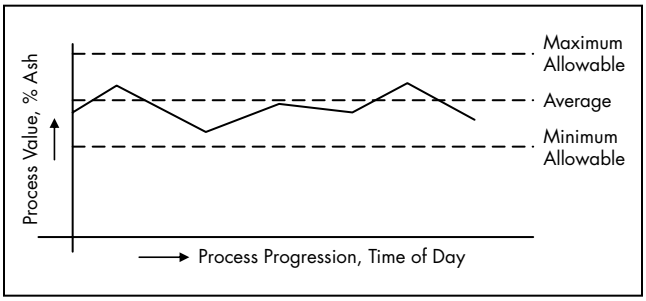


Figure 9.10-1 Example control chart for a coal washing plant

etc.), create forms or obtain batteries and software for personal computing devices. If there is more than one person involved in the time study, ensure that every member understands the definition of an "activity," especially when to start and stop.

4. **Collect long-term data.** Not all critical activities may occur during the time study. This is especially true of major breakdowns. Thus, it is important to review long-term records.
5. **Analyze or utilize time-study information.** Look beyond "averages." Do not assume normality. Explore the role of variability through simulation.

Advances in sensor and mine communications technology is eroding the role of performing time studies in areas that are popular time-study targets, such as truck-shovel utilization. Most mining equipment has sensors that measure and record basic operational data (such as cycle time, truck load, and speed) and health data (such as engine temperatures or breakdown) that can feed directly into a production-simulation exercise. While the wealth of data can vastly improve production simulation, one must be careful not to implicitly trust automatically gathered data. Errors have been known to occur, not just because of undetected malfunctioning sensors, but also due to errors in data warehousing, including conceptual errors such as different periods of aggregation for different data streams and errors in upstream databases that contribute to the corporate warehouse (such as a misspelled operator name).

MODELING TECHNIQUES

This section introduces three common systems-modeling techniques.

Statistical Process Control

Statistical process control (SPC) procedures are implemented to track a normally distributed process in real time with an eye on quality limits. The term *process* is meant as a key performance indicator of the system. Thus, while coal washing may be the process that is being tracked, the actual measurement being utilized in the tracking could be the average ash content (Figure 9.10-1) of washed coal. The intent of SPC is to identify when a process is out of control, so that remedial actions can be taken immediately. In Figure 9.10-1, the upper and lower limits for the process are shown. The process is deemed out of control whenever a process measurement strays past them. In many cases, a process may not be deemed out of control unless several process values are out of bounds.

In the simplest of forms, SPC requires a regular measurement of the process. These measurements are then plotted on a chart that has quality limits. The limits are usually either two standard deviations (95% confidence limit) or three standard deviations (>99.5% confidence limit) from the mean.

Because the mean and standard deviation estimates of the process directly impact when the process is deemed "in control" or "out of control," it is critical that these be estimated after a rigorous study of the process. An appropriate number of independent samples should be taken. However, this may be easier said than done because many mining processes are inherently correlated in time. For example, the ash content is directly related to the seam quality. When a particular area is being mined, all quality data will be similar. Thus, a 100-short-ton batch of coal may all have similar properties. The inherent correlation in processes is what makes SPC challenging.

Inherent correlation between samples close in time can be detected by plotting $P(t)$ versus $P(t + \lambda)$, where $P(t)$ is the profits received at the end of period t and λ is some lag time. For example, assume that measurements are made every minute. If 500 data points were collected, then $P(t)$ is the entire data set. If $\lambda = 25$, then $P(t + 25)$ or lag 25 data, would be the series starting at Sample 26 and ending at Sample 500; $P(t + 25)$ would have 475 points. Next, $P(t)$ is plotted against $P(t + 25)$. Of course, the last 25 points in $P(t)$ have to be discarded so that it also has 475 points, such as $P(t + 25)$. If this plot reveals no correlation (i.e., $R^2 < |0.05|$), samples 25 minutes apart should be used to draw the SPC chart.

Discrete Event Simulation

To understand discrete event simulation (DES), one must look at processes as a group of non-deterministic activities. Non-deterministic activities are those whose duration cannot be predicted with certainty. Some activities occur in a sequence, while some occur in parallel. At any given time, many activities are in progression, while some activities may start or end. For example, as one truck heads toward the mill, another is headed to the dump, while yet another may be headed to an excavator. During this time, a drill may be drilling blastholes or an excavator may be loading. The start or end of an activity is termed an "event." DES is a process simulation technique that models a process as a series of events. Simulation is controlled by a calendar of events called the "future events list" (FEL), with time jumping from one event to the next. Events are generated according to the definition of individual activities, with most activities being stochastically defined (i.e., as a statistical distribution). As events are generated, their start and end times are written to the FEL.

DES is very useful because it is able to represent a process or system with all its inherent variability. The complex interactions between events can be understood and their effects accurately quantified. The effect of probabilistic interactions is the most difficult to quantify in the methods that do not take into account the stochastic nature of most activities. Additionally, even if their stochastic nature is accounted for, it is almost impossible to quantify the combined effect of events, each of which follows a different distribution and is intertwined with other events.

The key to a successful DES model is good data. Time-study or other data that are used in the model should be representative of the activity. Maintenance and operational data beyond the time-study period should be reviewed to identify and quantify long-term trends. Distribution fitting should be accurate and should be correct for the activity. For example, for manual tasks, a right-tailed distribution is preferred (Yingling et al. 1999).

There are a variety of DES languages, including Arena simulation software (Rockwell Automation) and GPSS World (Minuteman Software). An excellent resource for mine simulation is Sturgul (2000).

Artificial Intelligence: Neural Networks

Artificial intelligence is a field encompassing a variety of tools—a popular one of which is neural networks. Neural networks are used widely in the mining industry, including in ore reserve estimation, process control, and machine health monitoring. This topic has been presented widely in the literature and, therefore, the reader is directed to standard references (Hagan et al. 1996; Sarle 1997; Haykin 2008) for a fundamental presentation. However, the many subtleties of neural networks that impact performance, but are not discussed widely in common literature, are presented here.

Neural networks are simply numerical models that describe through equations (i.e., $y = f(\boldsymbol{x})$) the relationship between a set of inputs, $\boldsymbol{x}$, and a given output, y. What makes them tricky to use is that there is little theoretical guidance on the modeling process. This is compounded by the multitude of software products, such as business intelligence tools, that make it "easy" to apply neural networks, resulting in many wrong or suboptimal applications. This is especially true in mining because mining-related data often have some amount on built-in unreliability.

Data Subdivision

As with any modeling process, neural models are tested prior to their use. Modeling and testing are done by splitting the data set into modeling and testing subsets. Typically, data are randomly subdivided (such as 75% modeling, 25% testing). However, as shown by Ganguli and Bandopadhyay (2003), random subdivision can result (with substantial probability) in two subsets that are not statistically similar. It is obvious that the modeling subset should be similar to the testing subset; otherwise it is akin to studying English but being tested on French. This problem is pronounced when the data set is sparse. A neural network requires the modeling subset to be further split into training and calibration subsets. Thus, this modeling approach requires the data to be split into three similar subsets: training, calibration, and testing (TCT). Usually, the split is 60%, 20%, and 20% between the three subsets, though there is no hard and fast rule.

Techniques such as genetic algorithms have been used to split data sets into TCT subsets (Ganguli et al. 2003; Samanta et al. 2004a). Some also first presplit the data set into multiple groups/categories, each of which contributes to TCT subsets (Samanta et al. 2004b; Yu et al. 2004). In any case, the intent is to arrive at three subsets that are "similar." It is up to the modeler to decide on what constitutes similarity. No matter what strategy is used in constructing TCT, the following two rules should be adhered to (especially in sparse data sets):

1. The training subset should contain the highest and lowest values. This way, the neural network is exposed to a broad range of data during training.
2. When data grouping/segmentation is done prior to constructing the TCT subset, it is possible that some groups may not have sufficient data for the three TCT subsets. In such cases, samples are assigned to training first, followed by calibration. Testing has the last priority.

The point of this discussion is that data should be subdivided with care rather than randomly. If data are divided randomly, one should verify that the subsets are similar.

Neural Network Design

The choice of architecture is an important neural network design parameter. The architecture of a neural network implies its size (in terms of number of neurons), number of layers, and choice of activation functions. Some of these are discussed in this section.

Size of the hidden layer, number of neurons. As a general rule, the total number of weights and biases should be less than half the number of training samples. The total number of weights and biases in a single-layer neural network, for N_{ip} inputs, N_{hl} hidden layer neurons (with associated N_{hl} bias weights), and one output (with an associated bias weight), is as follows:

$$N_{\text{weights + biases}} = (N_{ip} \times N_{hl} + N_{hl}) + (N_{hl} + 1)$$

Too many neurons result in too many neural network parameters being estimated from too few samples. Thus, the estimates may not be reliable. The most desirable neural network is the one that meets performance criteria with the least number of weights and biases.

Number of inputs. It should be remembered that even if one has sufficient data to define more inputs, more inputs might not be necessary. As Ganguli et al. (2006) discovered while modeling a semiautogenous grinding mill, not every input helped improve model performance (despite common belief on their usefulness). In other words, the modeler should experiment by eliminating some inputs so that the model has the least number of inputs, weights, and biases for the same neural network performance. When eliminating inputs, ideal candidates are those that are highly correlated to other inputs or those on which the modeler has the least confidence (in terms of quality of data). If the inputs are correlated, one could apply techniques such as Gram–Schmidt orthogonalization to remove the correlation between inputs prior to using them.

Training algorithm. The next important design factor is the selection of a training algorithm, including learning rates and momentum. Simply speaking, training involves optimizing the following nonlinear objective function:

$$\varepsilon = f(\boldsymbol{w})$$

where

ε = error
$\boldsymbol{w}$ = set of weights and biases

Optimization of nonlinear functions is a very mature field, with no clear guidance on the "best" algorithm. This includes the work of Samanta et al. (2006), who applied (and discussed) various training algorithms for ore reserve estimation and did not find any clear leaders.

However, "Neural Net FAQ" (Sarle 1997), which is an excellent Internet resource on neural networks, recommends the following:

- Use Gauss–Newton algorithms (such as Levenberg–Marquardt) for small number of weights and biases. These are resource-intensive algorithms that require a

Table 9.10-1 Suggested data descriptors for common distributions*

Distribution	Required Descriptors	Intuitive Descriptors
Normal	Mean, μ Variance, σ	expected value = μ; variance = σ
Poisson	γ	expected value = γ; variance = γ
Lognormal	Mean of lognormal data, μ_{LN} Variance, σ_{LN}	expected value = $e^{(2\times\mu_{LN}+\sigma_{LN}^2)/2}$; variance = $(e^{\sigma_{LN}^2}-1)\,e^{(2\times\mu_{LN}+\sigma_{LN}^2)}$
Beta	The shape parameters, α and β	expected value = $\frac{\alpha}{\alpha+\beta}$; variance = $\frac{\alpha\beta}{(\alpha+\beta)^2(\alpha+\beta+1)}$

*Deciles and quartiles are always helpful additional descriptors.

storage capacity proportional to the square of the number of weights and biases.
- Conjugate-gradient-type algorithms are less resource intensive and require a storage capacity proportional to the number of weights and biases. Thus, they are best for large problems.

No algorithm guarantees finding the global optima. Therefore, it is customary to train the neural network several times, starting with a different set of random weights each time.

Advanced concepts. Many different neural network models can be fitted to a given data set. Often it is very difficult to decide which model is "best"—is it the one with the best prediction correlation (r^2), the one with the best root-mean-square error, or some other measure? How does one combine competing performance measures? Additionally, what if the testing subset performance is very different from training and calibration subsets? While not all answers are easy, a common-sense solution would be to pick a network that performs equally well across all subsets.

There is no reason, however, to restrict oneself to one model to represent the data set. "Ensemble modeling" (Dutta et al. 2006) can be done where predictions from multiple neural network models are combined to obtain the final prediction. Because ensemble models reduce variance and not bias, ensemble members should ideally be low bias predictors. Other ensemble modeling concepts include boosting (Samanta et al. 2005) and modular networks (Sharkey 1999).

ANALYSIS TECHNIQUES

This section presents two common analysis techniques after a brief discussion on how to describe data.

Basic Description

The most common (and insufficient because it does not describe the variability) descriptor of data is the average or mean. However, average assumes that the data are normally distributed. Some common data types encountered in mining time studies are rarely normal. Cycle times are often Poisson (distribution), while others, such as yes/no-type (or on/off) data, are typically binomial. Thus, before data are described, their distribution should be confirmed.

After the distribution is confirmed, data should be described in a way that is consistent with its distribution. It should also describe or indicate the spread or variability. Ideally, the description would be intuitive. Thus, while the phrase "time to load is lognormally distributed with a mean and standard deviation of 0.25 and 0.6" is an accurate and probably sufficient description of the time to load, a more intuitive description would be that the "expected value" of time to load is 1.82 minutes, though the expected value alone is insufficient, as it does not indicate the variability. Thus, the expected value, which is always part of a good description, should be accompanied by the variance and, preferably, the deciles or quartiles for an indication of variability. See Table 9.10-1 for more information.

An important reason for describing the data is to obtain a feel for the process. An additional tool for obtaining a good feel for data is in the form of basic plots (NIST/SEMATECH 2006). Indeed, plotting is considered the first step in exploratory data analysis.

Analyzing Non-Normal Data

Some analysis tools, such as Six Sigma (described later), require normality assumptions. Often, non-normal data can be adapted for such tools, as follows (Padnis n.d.):

1. **Converting data to normal distribution:** There are many available transformations for converting non-normal data to normal data. Examples include Box–Cox and Logit (for on/off data) transforms.

   The Box–Cox transformation is as follows (NIST/SEMATECH 2006):

$$x_i(\lambda) = \frac{x_i^{\lambda}-1}{\lambda}$$

   where

   x_i = i^{th} sample in the data set containing the "n" samples; $\mathbf{X} = (x_1, x_2, \ldots x_n)$
   λ = transformation parameter (usually between –2 and 2)
   $x_i(\lambda)$ = transformed i^{th} value

   λ is selected by trial and error and is the one that maximizes the following function:

$$f(x,\lambda) = \frac{-n}{2}\ln\left[\sum_{i=1}^{n}\frac{(x_i(\lambda)-\bar{x}(\lambda))^2}{n}\right] + (\lambda-1)\sum_{i=1}^{n}\ln(x_i)$$

   where $\bar{x}(\lambda) = \frac{1}{n}\sum_{i=1}^{n} x_i(\lambda)$

   After the data are transformed, one could determine confidence intervals as required by Six Sigma or other processes. These intervals or limits can then be back-transformed to raw data, though back-transformation can be complicated, depending on the transformation, because confidence intervals that are directly back-

Table 9.10-2 Box–Cox transformation

Raw Data, x_i	Transform, $\lambda = -0.1$ $x_i(\lambda)$	$\ln(x_i(\lambda))$	$(x_i(\lambda) - \bar{x}(\lambda))^2$	
1	0.0000	0.0000	2.6466	
13	2.2624	2.5649	0.4040	
6	1.6404	1.7918	0.0002	
3	1.0404	1.0986	0.3439	
93	3.6445	4.5326	4.0709	$f(x,\lambda) = -22.5$
2	0.6697	0.6931	0.9162	
1	0.0000	0.0000	2.6466	
19	2.5505	2.9444	0.8532	
45	3.1659	3.8067	2.3688	
4	1.2945	1.3863	0.1105	
	$\bar{x}(\lambda) = 1.6268$			

transformed need not be reflective of the original data. This is certainly true of lognormal transformation, where the confidence interval is computed differently to account for bias arising from back-transformation. Olsson (2005) suggested that for log-transformed data, the confidence intervals should be computed as follows:

$$\text{confidence interval} = \mu + \frac{s^2}{2} \pm t\sqrt{\frac{s^2}{n} + \frac{s^4}{2\times(n-1)}}$$

where

μ = mean of the transformed data
s^2 = variance
n = number of data points
t = appropriate t-statistic (such as 2.23 for 10 samples and 95% confidence)

The obtained interval can then be directly back-transformed to the original form. Because the Box–Cox transform has no bias issues with back-transformation, confidence intervals can be computed the traditional way ($\mu + t \times s$).

2. **Using subgroup averages:** Subgroups or clusters of non-normal data can be normal. Thus, analysis can be done on these groups, though the results of the analysis must be understood in the context of groups. For example, though truck cycle time may not be normally distributed, the average cycle time per hour may be normally distributed.
3. **Subdividing the data set:** In this strategy, data are subdivided into large groups based on some logical reasoning. Often, such subdivision yields normal data. For example, again assume that truck cycle time is not normal. Also assume that for some internal reason in that mine, truck dispatch system works differently in the day shift than in the evening shift. Treating the day shift data and evening shift data as two different data sets may result in two almost-normal subsets.

Example 6. The following data were collected: 1, 13, 6, 3, 93, 2, 1, 19, 45, and 4. The process requires that 97.5% of the samples be less than 80. Based on the samples, does the process meet the specifications?

Solution. The 97.5% (single tail) requirement defines a distance of two standard deviations from the mean.

Wrong analysis. The following is not correct:

1. Assume normality automatically.
2. Therefore, the sample mean = 18.7, and the standard deviation = 29.4.
3. A statistical analysis says that 97.5% of the data are less than 77.5 ($18.7 + 2 \times 29.4 = 77.5$).
4. It is concluded that the process meets specifications.

Correct analysis. Whenever the standard deviation is close to or greater than the mean, one should be concerned with normality assumptions. The correct analysis is as follows:

1. First, determine the distribution or test for normality. If the distribution is not normal or the data fails a normality test, then transform the data. In this case, the samples (that are truly lognormal) fail the Anderson–Darling test (NIST/SEMATECH 2006) for normality. The Anderson–Darling test is not presented here.
2. Next, convert the data to a normal distribution using Box–Cox transformation. Similarly, compute $f(x,\lambda)$ for λ between –2 and 2, in increments of 0.1. However, in this example, only $f(x,\lambda)$ is computed for $\lambda = -0.2, -0.5, 0.1, 0.2$, and 0.5. The highest $f(x,\lambda)$ occurs at –0.1. Thus, the λ chosen to transform the data is –0.1. The Anderson–Darling test for normality confirms that the transformed data (second column in Table 9.10-2) is indeed normal.
3. Next, conduct confidence interval analysis on transformed data. The transformed data have a mean and standard deviation of 1.62 and 1.26. Therefore, 97.5% of the data falls below 4.14 ($1.62 + 2 \times 1.26 = 4.14$).
4. Finally, back-transform the upper limit (4.14); that is, solve the following equation:

$$4.14 = (x^{-0.1} - 1)/-0.1$$

Therefore, $x = 209.4$.

The Box–Cox transformation can be back-transformed directly, unlike the lognormal transformation.

The process fails. The limit (80), when transformed is 3.54, which is 1.52 standard deviations away from the mean (i.e., $(3.54 - 1.62)/1.26 = 1.52$), which implies 6.6% of the samples are greater than 80.

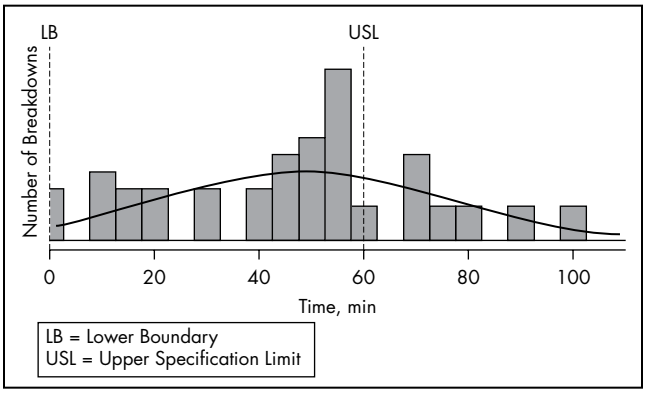


Figure 9.10-2 Histogram of CM breakdowns before implementation of a maintenance program

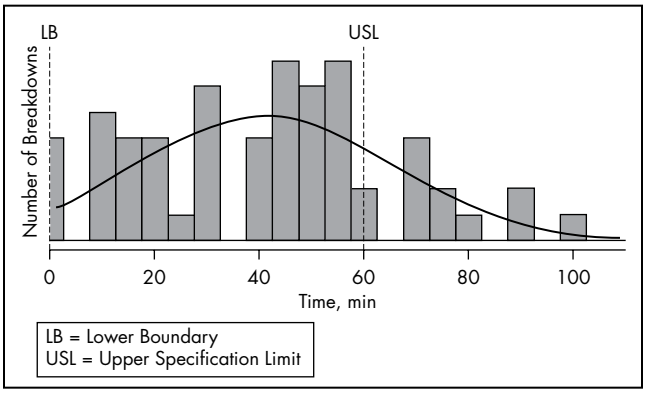


Figure 9.10-3 Histogram of CM breakdowns after implementation of a maintenance program

Six Sigma

Six Sigma is a discipline utilizing strict measuring and data analysis procedures to quantify the quality performance of a process and to then measure and verify changes made in that process. The goal is to achieve a nearly defect-free process in which the rate of defects is at or below 3.4 per million. Using a manufacturing analogy, if the process results follow a normal bell curve (and many do), then 3.4 defects per million or 99.9997% efficiency falls within plus or minus 6 standard deviations (sigmas) of the bell-curve distribution mean. The Six Sigma methodology strives to make the process so well controlled, and hence the distribution curve so tight, that plus or minus 6 standard deviations of the curve fit within the specifications for the product.

Except for mineral processing, applying Six Sigma to mining may not seem straightforward, but with a little adjustment, such as redefining "defects" and "specification limits," it can be quite applicable. Suppose a major change in the preventive maintenance program at an underground coal mine is planned. To see if it has a positive effect on major downtime elements associated with a continuous miner (CM), one can simply define a defect as any downtime in excess of, for example, 1 hour. Assume that the CM downtime data over a time period of 56 shifts is represented by Figure 9.10-2.

In a case such as this, there is no lower specification limit per se. The lower limit is, of course, zero minutes. But it is not a specification; it is a natural boundary to the data. Hence, it is defined as the lower boundary. One should not automatically assume that the collected samples are normally distributed (i.e., Six Sigma only applies when the data are normally distributed). (The previous section gives instructions for handling non-normal data.)

The analysis of the data shown in Figure 9.10-2 shows that the observed data exceeded the upper boundary 250,000 times out of 1 million (or 25% of the time), while the expected performance of the system (the curve) is about 335,000 times out of 1 million exceeding 60 minutes.

Now assume that the data collected in 60 shifts after the implementation of the new maintenance program are shown in Figure 9.10-3. Did the maintenance program make a difference? Figure 9.10-3 "looks" better with a lot more data appearing that is less than the upper specification limit. The observed and expected performance indicators are both improved, and the average downtime of the data shifted from 48.4 to 41.1 minutes. But did the new preventive maintenance program make a verifiable difference in fact?

This is where the second stage of the Six Sigma discipline becomes useful. One must now verify statistically that the shift in the sample mean represents a real change in the population. This is done through the use of various tools, including the hypothesis tests presented earlier. By using these statistical tools, one can determine (within given confidence intervals) whether or not there has been a real shift in the population that the sample represents (which is the goal) or whether the shift in the sample statistics could be due to the natural variation in the data.

After an engineer becomes reasonably familiar with the Six Sigma methodology and the statistical tools (some of which are discussed here), a whole host of applications in any mining scenario become apparent. Six Sigma provides the mining engineer with the knowledge and tools to help him or her virtually eliminate the old style "seat of the pants" decision making. Given some data, or simply the time to gather the data, the mining engineer can give upper management unbiased, objective, and verifiable input into most any decision process.

OPTIMIZATION

The techniques that can provide an optimum solution to the system include linear programming, mixed integer programming, nonlinear programming, and network flow modeling. These are widely published techniques and are covered in detail in standard textbooks (e.g., Fletcher 2000). Additionally, space restrictions do not allow them to be covered here in any detail. Therefore, they will only be introduced here, except for cutoff grade optimization that has a broader impact. Those desiring advanced developments in the optimization field are referred to Optimization Online (2010), an excellent on-line source.

Standard Optimization Techniques: A Brief Introduction

Many constraints in a mine system can be expressed through basic linear equations, with the overall problem being that of balancing these competing constraints to maximize (or minimize) a linear objective function. Mathematically, this can be stated as follows:

$$f(v) = c^T \boldsymbol{v}$$

Table 9.10-3 Grade–tonnage distribution of a hypothetical gold deposit

Grade Category, oz/st	Short tons (in thousands)
0–0.02	70,000
0.02–0.025	7,257
0.025–0.030	6,319
0.030–0.035	5,591
0.035–0.040	4,598
0.040–0.045	4,277
0.045–0.050	3,465
0.050–0.055	2,428
0.055–0.060	2,307
0.060–0.065	1,747
0.065–0.070	1,640
0.070–0.075	1,485
0.075–0.080	1,227
0.080–0.100	3,598
0.100–0.358	9,576

where

$\boldsymbol{v}$ = vector of variables, whose value needs to be determined
c = set of known constants

The constraints on the variables are described as follows:

$$\boldsymbol{Pv} \leq \boldsymbol{q}$$

where

$\boldsymbol{P}$ = matrix of known constants
$\boldsymbol{q}$ = vector of known constants

Linear programming is a set of techniques that provide methods of solving for $\boldsymbol{v}$ that maximize $f(v)$ without violating the constraints. Different linear programming techniques impose different restrictions on $\boldsymbol{v}$. The simplex method requires that the elements of $\boldsymbol{v}$ be all nonnegative, while mixed integer programming requires that they also be whole numbers. Nonlinear programming, on the other hand, allows both $f(v)$ and the constraints to be nonlinear, while quadratic programming allows quadratic terms in $f(v)$. Network flow programming/graph theory involves describing and solving network problems as linear programming problems. Network problems are those that intuitively relate to a network structure, such as a system of pipes (constrained by flow direction and pipe capacity), highway traffic, or conveyor systems. However, they have also been related in abstract to problems with no physical flow at all, such as the famous Lerchs–Grossman algorithm for pit-limit determination. In this problem, the restrictions in how mining can advance can be related to flow. For example, one cannot mine a block of ore unless the blocks above it are mined. The interested reader is directed to Hustrulid and Kuchta (2006) for an elaborate presentation of different pit-optimization techniques.

Cutoff Grade and Scheduling Optimization

Heuristic Cutoff Grade Optimization Algorithms

Cutoff grade is traditionally defined as the grade that is normally used to discriminate between ore and waste within a given ore body. This definition can be extended to mean the grade that is used to differentiate various ore types for different metallurgical processing options. Although the definition of cutoff grade is very precise, the choice of a cutoff grade policy to be used during an exploitation of a deposit is not. Use of simply calculated breakeven cutoff grades during the production would, in most instances, lead to suboptimum exploitation of the resource.

Table 9.10-4 Mine design parameters for the case study

Parameter	Symbol	Value
Price	P	600 $/oz
Sales cost	s	5.00 $/oz
Processing cost	c	19.0 $/st ore
Recovery	y	90%
Mining cost	m	1.20 $/st
Fixed costs	f_s	$10.95 million/yr
Mining capacity	M	Unlimited
Milling capacity	c	1.05 million st
Capital costs	CC	$105
Discount rate	d	15%

Exploitation of a deposit in such a way that the maximum net present value (NPV) is realized at the end of the mine life has been an accepted objective of mining companies. The NPV to be realized from an operation is dependent on interrelated variables such as mining and milling capacities, sequence of extraction, and cutoff grades. These interdependent variables interact in a complex manner in defining the NPV of a project. The sequence of extraction is dependent on the rates of production, the grade distribution of the deposit, and the cutoff grades. Cutoff grades cannot be determined without knowing the extraction sequences and capacities of the mining system. The determination of capacities is directly related to the cutoff grades and extraction sequences.

Mine planning is a process that defines sets of values for each of these variables during life of the project. The biggest challenge during the mine planning is to define the capacities of the mining system that are in perfect harmony with the grade distributions of the deposit through the planned extraction sequence and cutoff grade policy.

For a given set of capacities (the economic costs associated with the capacities within the mining system, the pit extraction sequence, and the prices), there is a unique cutoff grade policy that maximizes the NPV of the project. The determination of these cutoff grades during the life of the mine is the subject of this section.

Traditional Cutoff Grades in Open-Pit Mining

Consider a hypothetical case study where an epithermal gold deposit would be mined by an open pit. Table 9.10-3 gives the grade distribution of the material within the ultimate pit limits of this deposit. Table 9.10-4 gives the assumed capacities and accepted costs to mine this deposit at a 2,720-t/d (3,000-st/d) milling rate.

Traditionally in open-pit mining, a cutoff grade is used to determine if a block of material (free standing, i.e., without any overlying waste) should be mined or not, and another cutoff grade is used to determine whether or not it should be milled or taken to the waste dump.

The first cutoff grade is generally referred to as the ultimate pit cutoff grade, and it is defined as the breakeven grade that equates cost of mining, milling, and refining to the value of the block in terms of recovered metal and the selling price.

ultimate pit cutoff grade

$$= \frac{\text{milling cost} + \text{mining cost}}{(\text{price} - (\text{refining cost} + \text{marketing cost})) \times \text{recovery}}$$

$$= \frac{\$19 + 1.20}{(\$600 - \$5) \times 0.90}$$

The second cutoff grade is referred to as milling cutoff grade and is defined as the breakeven grade that equates cost of milling, refining, and marketing to the value of the block in terms of recovered metal and the selling price.

milling cutoff grade

$$= \frac{\text{milling cost}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$= \frac{\$19}{(\$600 - \$5) \times 0.90} = 0.035 \text{ oz/st}$$

In the calculation of the milling cutoff grade, no mining cost is included because this cutoff is basically applied to those blocks that are already "selected for mining" (by the first cutoff) to get to the higher-grade ore blocks and those blocks that the cost of mining will be incurred regardless of the action to be taken with respect to milling it. Notice that the depreciation costs, the general and administrative (G&A) costs, and the opportunity costs are not included in the cutoff grades given. In the traditional cutoff grades, the basic assumption is that all of these costs, including fixed costs defined as G&A, will be paid by the material whose grade is much higher than the established cutoff grades. The first cutoff grade is used to ensure that no material (unless they are in the way of other high-grade blocks) is taken out of the ground unless all of the direct costs associated with "gaining" the metal can be recovered. This assurance is automatically built into the ultimate pit limit determination algorithms such as Lerchs–Grossmann and the moving cone. The second cutoff grade is used to ensure that any material that provides positive contribution beyond the direct milling, refining, and marketing costs will be milled.

The general characteristics of the traditional cutoff grades

- Are established to satisfy the objective of maximizing the undiscounted profits from a given mining operation,
- Are constant unless the commodity price and the costs change during the life of mine, and
- Do not consider the grade distribution of the deposit.

Heuristic Cutoff Grade Optimization Techniques

Mining the deposit under consideration with a traditional milling cutoff grade of 1.2 g/t (0.035 oz/st) at 95 Mt (1.05 million st) milling capacity results in the exploitation schedule given in Table 9.10-5.

The annual cash flows are given as profits in millions of dollars, and they are determined as follows:

Table 9.10-5 Exploitation schedule for constant cutoff grades

Year	Cutoff Grade, oz/st	Average Grade, oz/st	Q_m*	Q_c†	Q_r‡	Profits, million $/yr
1	0.035	0.102	3.6	1.05	96.3	33.0
2	0.035	0.102	3.6	1.05	96.3	33.0
3	0.035	0.102	3.6	1.05	96.3	33.0
4	0.035	0.102	3.6	1.05	96.3	33.0
5	0.035	0.102	3.6	1.05	96.3	33.0
6	0.035	0.102	3.6	1.05	96.3	33.0
7	0.035	0.102	3.6	1.05	96.3	33.0
8	0.035	0.102	3.6	1.05	96.3	33.0
9	0.035	0.102	3.6	1.05	96.3	33.0
10	0.035	0.102	3.6	1.05	96.3	33.0
11 to 34	0.035	0.102	3.6	1.05	96.3	33.0
35	0.035	0.102	3.4	1.00	91.7	31.4
Total	0.035	0.102	125.8	36.70	3,365.9	1,154.2
NPV = $218.5 million						

*Q_m = amount of total material mined (in millions of short tons) in a given year.
†Q_c = ore tonnage (in millions of short tons) processed by the mill.
‡Q_r = recovered ounces (in thousands) produced in a given year.

$$P_i = (P - s) \times Q_r - Q_c \times c - Q_m \times m$$

where

P_i = annual profits, million $
P = price, $/oz
s = sales cost, $/oz
Q_r = recovered ounces, oz/yr
Q_c = tons processed by mill, million st/yr
c = milling capacity, million st/yr
Q_m = total material mined, million st/yr
m = mining cost, $/st

In the example (Table 9.10-5), a total of 33.3 Mt (36.7 million st) at an average grade of 3.5 g/t (0.102 oz/st) is mined with a stripping ratio of 2.42, and the ore is processed by the mill during the 35 years of mine life. This schedule results in total undiscounted profits of $1,154.2 million and NPV of $218.5 million.

As long as the operator mines and processes the blocks of material with grades greater than or equal to the static cutoff grades without considering deposit characteristics, only the undiscounted profits will be maximized. The maximization of discounted profits versus NPVs are two different things; when the objective is to maximize the NPVs, the maximization of profits without the time value of money amounts to optimization without the capacity constraints of the mining system and thus, when viewed within the framework of constrained NPV optimization, always yields suboptimal NPVs. Realizing the fact that mines that are planned with traditional cutoff grades would not result in maximum NPVs, many approaches have been suggested to modify the traditional cutoff grade policies such that the NPVs from a given operation are improved.

The concept of using cutoff grades higher than breakeven grades during the early years of an operation for a faster recovery of capital investments and using breakeven grades during the later stages of the mine has been practiced in the industry for heuristic NPV optimizations. In this approach,

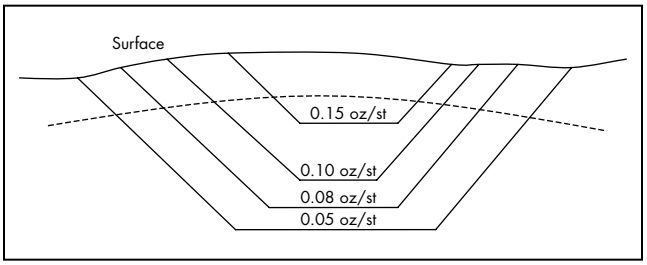


Figure 9.10-4 Idealized cross section of a series of pits for various cutoff grades

the traditional cutoff grade calculation is modified so that they include depreciation, fixed costs, and minimum profit per ton required for a period of time to obtain a much higher cutoff grade during the early years. After the end of the initial period, the minimum profit requirement is removed from the equation to lower the cutoff grades further until the plant is paid off. At that point, the depreciation charges are also dropped where the mine operates with the traditional milling cutoff grades. This concept is demonstrated pictorially in Figure 9.10-4. Analytically, the concept is explained as follows:

- Assume that the $105-million plant capital cost in the case study would be depreciated during the first 10 years by the straight-line method:
  - Depreciation cost per year = $105 million/10 years = $10.5 million/yr
  - Depreciation cost per short ton = $105 million/1.05 st = $10/st of ore
- In addition, assume that a minimum profit of $3.0/st would be imposed to increase the cash flows further during the first 5 years.

Then, the milling cutoff grades, g_{milling}, during the life of mine would be as follows:

Years 1 through 5

$$g_{\text{milling}} = \frac{\text{milling cost + depreciation + minimum prof.}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19 + \$10 + \$3}{(\$600 - \$5) \times 0.9} = 0.060 \text{ oz/st}$$

Years 6 through 10

$$g_{\text{milling}} = \frac{\text{milling cost + depreciation}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19 + \$10}{(\$600 - \$5) \times 0.9} = 0.054 \text{ oz/st}$$

Year 11 through depletion

$$g_{\text{milling}} = \frac{\text{milling cost}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19}{(\$600 - \$5) \times 0.9} = 0.035 \text{ oz/st}$$

The year-by-year tonnage and grade schedule obtained from the modified cutoff grade policy is given in Table 9.10-6. Again, a total 28.44 million st at an average grade of 0.106 oz/st is mined, with an overall stripping ratio of 3.88 and milled during 25 years of mine life. This modified schedule results in total undiscounted profits of $1,112.7 million and an NPV of $355.7 million. The comparison of total undiscounted profits and NPVs in Tables 9.10-3 and 9.10-4 indicate that there was a 3.6% reduction in total undiscounted profits and a 63% increase in the total NPV of the project when the cutoff grade policy was modified from the traditional to the declining approach where cutoff grades were elevated during the initial years.

Table 9.10-6 Yearly tonnage and grade with a declining cutoff grade strategy

Year	Cutoff Grade, oz/st	Average Grade, oz/st	Q_m*	Q_c†	Q_r‡	Profits, million $/yr
1	0.060	0.153	6.9	1.05	144.6	57.8
2	0.060	0.153	6.9	1.05	144.6	57.8
3	0.060	0.153	6.9	1.05	144.6	57.8
4	0.060	0.153	6.9	1.05	144.6	57.8
5	0.060	0.153	6.9	1.05	144.6	57.8
6	0.054	0.141	6.0	1.05	132.8	51.9
7	0.054	0.141	6.0	1.05	132.8	51.9
8	0.054	0.141	6.0	1.05	132.8	51.9
9	0.054	0.141	6.0	1.05	132.8	51.9
10	0.054	0.141	6.0	1.05	132.8	51.9
11 to 27	0.035	0.102	3.6	1.05	96.3	33.0
28	0.035	0.102	0.3	0.09	8.1	2.8
Total	–	–	125.8	28.44	3,032.1	1,112.7
NPV = $355.7 million						

*Q_m = amount of total material mined (in millions of short tons) in a given year.
†Q_c = ore tonnage (in millions of short tons) processed by the mill.
‡Q_r = recovered ounces (in thousands) produced in a given year.

In the previous calculations, the G&A costs were not included in the cutoff grade and profit calculations. The effects of the $10.95 million per year fixed costs on the cutoff grade policy and the resulting total profits and NPVs are computed as follows:

Years 1 through 5

$$g_{\text{milling}} = \frac{\text{milling cost + depreciation + minimum profit + fixed cost}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19 + \$10 + \$3 + \$7.95}{(\$600 - \$5) \times 0.9} = 0.075 \text{ oz/st}$$

Years 6 through 10

$$g_{\text{milling}} = \frac{\text{milling cost + depreciation + fixed cost}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19 + \$10 + \$7.95}{(\$600 - \$5) \times 0.9} = 0.069 \text{ oz/st}$$

Year 11 through to depletion

$$g_{\text{milling}} = \frac{\text{milling cost + fixed cost}}{(\text{price} - \text{refining} + \text{marketing cost}) \times \text{recovery}}$$

$$g_{\text{milling}} = \frac{\$19 + \$7.95}{(\$600 - \$5) \times .9} = 0.050 \text{ oz/st}$$

Table 9.10–7 gives yearly tonnage and grade schedules resulting from the cutoff grade policy that includes fixed costs as part of the cutoff grade and the profit calculation. The policy of declining cutoff grades calculated with depreciation, minimum profit, and the G&A costs further increased the NPV ($357.1 million versus $355.7 million), while overall undiscounted profits were adversely reduced by 20% ($1,112.7 million versus $885.6 million).

Optimizing Cutoff Grades by Lane's Approach

The preceding discussion demonstrated the significant impact of the cutoff grade policy on the NPV of a project. It is generally accepted that the cutoff grade policy that gives higher NPVs is a policy that uses declining cutoff grades throughout the life of the project. The most obvious resulting question is: "How should the cutoff grades for a given mine be determined so that one obtains the highest NPV possible?"

Table 9.10-7 Yearly tonnage and grade schedules with a modified declining cutoff grade strategy (fixed costs included)

Year	Cutoff Grade, oz/st	Average Grade, oz/st	Q_m*	Q_c†	Q_r‡	Profits, million $/yr
1	0.075	0.182	9.2	1.05	171.6	62.8
2	0.075	0.182	9.2	1.05	171.6	62.8
3	0.075	0.182	9.2	1.05	171.6	62.8
4	0.075	0.182	9.2	1.05	171.6	62.8
5	0.075	0.182	9.2	1.05	171.6	62.8
6	0.069	0.169	8.2	1.05	160.0	57.1
7	0.069	0.169	8.2	1.05	160.0	57.1
8	0.069	0.169	8.2	1.05	160.0	57.1
9	0.069	0.169	8.2	1.05	160.0	57.1
10	0.069	0.169	8.2	1.05	160.0	57.1
11 to 17	0.050	0.132	5.4	1.05	124.8	39.5
18	0.050	0.132	1.3	0.26	30.5	9.6
Total	–	–	125.8	18.11	2,562.5	885.6

NPV = $357.1 million

*Q_m = amount of total material mined (in millions of short tons) in a given year.
†Q_c = ore tonnage (in millions of short tons) processed by the mill.
‡Q_r = recovered ounces (in thousands) produced in a given year.

In a paper published in 1964 and in a book published in 1988, K.F. Lane discussed in detail the theoretical background, a general formulation, and a solution algorithm to this problem. In Lane's formulation and theoretical analysis, it was shown that cutoff grade calculations that maximize NPV must include the fixed costs associated with not receiving the future cash flows quicker due to the cutoff grade decision taken now. The cutoff grade equation that maximizes the NPV of the deposit when the system is constrained by the mill capacity is given as follows:

$$g_{\text{milling}}(i) = \frac{c + f + F_i}{(P - s) \times y}$$

where

$g_{\text{milling}}(i)$ = cutoff grade to be used in Year i
i = 1, … N (mine life), years
F_i = opportunity cost per short ton of material milled in Year i (where $F_i = d \times NPV_i/C$)

where

d = discount rate
NPV_i = NPV of the future cash flows of the years (i) to the end of mine life N
C = total processing capacity in Year i

The underlying philosophy in inclusion of the opportunity cost per ton, F_i, in the equation is that every deposit has a given NPV associated with it at a given point in time and that every ton of material processed by the mill during a given year should pay for the cost of not receiving the future cash flows by one year sooner. (In other words, the opportunity cost, F_i, should be viewed as taking the low grade now when higher grades are still available.) The details of this approach are given in the next section.

Table 9.10-8 gives yearly tonnage and grade schedules resulting from Lane's approach. The cutoff grade policy that is determined by this optimizing strategy gives 90% higher NPV and 35% lower profits than the original constant cutoffs determine by the traditional breakeven approach. Even though the total tons mined are the same between the original cutoff policy given in Table 9.10-5 and the optimum policy given in Table 9.10-6, the amount of material milled is lower both in

Table 9.10-8 Annual tonnage and grade schedule using Lane's method

Year	Cutoff Grade, oz/st	Average Grade, oz/st	Q_m*	Q_c†	Q_r‡	Profits, million $/yr	NPV, million $
1	0.161	0.259	18.0	1.05	245.2	95.9	413.8
2	0.152	0.255	17.2	1.05	241.0	94.4	380.0
3	0.142	0.250	16.5	1.05	236.4	92.6	342.6
4	0.131	0.245	15.7	1.05	231.3	90.5	301.4
5	0.120	0.239	14.9	1.05	225.7	88.1	256.1
6	0.107	0.232	14.1	1.05	219.6	85.4	206.4
7	0.092	0.213	12.1	1.05	200.9	76.7	152.0
8	0.079	0.188	9.8	1.05	177.9	65.9	98.1
9	0.065	0.163	7.6	1.05	153.6	53.9	46.9
Total	—	—	125.8	9.45	1931.4	743.4	—

NPV = $413.8 million

*Q_m = amount of total material mined (in millions of short tons) in a given year.
†Q_c = ore tonnage (in millions of short tons) processed by the mill.
‡Q_r = recovered ounces (in thousands) produced in a given year.

tons (i.e., 36.7 million versus 9.45 million st) and in ounces recovered (i.e., 3.37 million versus 1.93 million oz).

The effect of the optimization on the mine life is also significant; shortening of the mine life from 36 years in Table 9.10-5 to 10 years in Table 9.10-8 is the trade-off between the optimum NPV approach versus the traditional breakeven approach. It should be pointed out again that the mining system did not allow for the stockpiling of lower-grade ore for later processing. If stockpiling were allowed, then the stockpiled material has to be worked into the mining schedule given in Table 9.10-8 when it is most appropriate.

Algorithm for Determining Optimum Cutoff Grades for a Single Constraint Problem

The cutoff grades, $g_{\text{milling}}(i)$, given in Lane's equation depend on estimates of NPV_i of the future profits in Year i through the end of mine life from the operation. The NPV_i of the future profits cannot be calculated until the optimum cutoff grades have been decided. The solution to this type of interdependency problem is obtained by an iterative approach, where initial NPV_i values are guessed at first, and at each iteration, they are improved on until the solution converges to a stable optimum answer. This algorithm is as follows:

1. Start with the grade–tonnage curve similar to the one given in Table 9.10-1 for the whole deposit.
2. Assume the most appropriate milling capacity (C) to be used, the selling price (P), refining and marketing cost (s), recovery (y), milling cost (c), mining cost (m), annual fixed costs (f_a), and discount rate (d).
3. Determine the cutoff grade, $g_{\text{milling}}(i)$, to be used in Year i by the following equation:

$$g_{\text{milling}}(i) = \frac{c + f + F_i}{(P - s) \times y}$$

where

$$F_i = (d \times \text{NPV}_i)/C$$
$$f = f_a/C$$

If the initial NPV_i is not known, set NPV_i to zero.

4. From the most current grade–tonnage curve of the deposit, determine the following:
   - ore tonnage, T_c, and grade, g_c, above the cutoff grade, $g_{\text{milling}}(i)$
   - waste tonnage, T_w, that is below the cutoff grade, $g_{\text{milling}}(i)$
   - stripping ratio, $sr = T_w/T_c$
5. If the total reserves calculated in step 4 is greater than the annual milling capacity, set Q_c (the quantity milled in Year i) to $Q_c = C$. Otherwise set it to $Q_c = T_c$. Set Q_m (the quantity mined in Year i) to $Q_m = Q_c\,(1 + sr)$.
6. Determine the annual profit, pi, by using the following equation:

$$P_i = (P - s) \times Q_c \times g_c \times y - Q_c \times (c - f) - Q_m \times m$$

7. Adjust the grade–tonnage curve of the deposit by subtracting ore tons, Q_c, from the grade distribution $(Q_m - Q_c)$ from the cells below the cutoff in proportionate amount such that the shape of the distribution is not changed.
8. If Q_c is less than the milling capacity, C, then set mine life $N = i$ and go to step 9; otherwise set the year indicator to $i = i + 1$ and go to step 3.
9. By using the profits (P_i) estimated in step 6, calculate the incremental NPV_i values for the cash flows to be generated from i to N by using the following equation:

$$NPV_i = \sum_{j=i}^{N} P_j/(1+d)^{j-i+1}$$

For each year $i = 1, N$ (where N is total mine life in years).

10. If the total NPV of future profits for the whole deposit, NPV_i, is not within some tolerance (e.g., ±$500,000) of the total NPV from the previous iteration, go to step 1; otherwise stop; the cutoff grade, g_i, values for Years $i = 1, N$ is the optimum policy that gives maximum NPV of future profits for the operation.

Mine Scheduling and Cutoff Grade Optimization Using Mixed-Integer Linear Programming

A vast amount of work has been done on the development of large-scale production scheduling by using the linear programming and the integer programming models and related solution algorithms since the late 1960s. Johnson (1968), at the University of California at Berkeley, developed one of the most comprehensive mixed-integer linear programming (MILP) models, which is valid even today. This work was followed by Dagdelen's (1985) work in implementing Lagrangian-based solution methodology to MILP models. Application of Lane's cutoff grade optimization algorithm (Lane 1964, 1988) to a gold deposit was described in Dagdelen (1992), and the actual algorithm was given in Dagdelen (1993). The MILP approach as a schedule and cutoff grade optimizer was initially proposed by Dagdelen (1996).

The concept of the MILP approach is demonstrated and discussed in Urbaez and Dagdelen (1999) and was successfully applied to a large-scale gold mine operation with complex process flows at Newmont Mining Corporation's Nevada operations (Hoerger et al. 1999). The use of MILP models for scheduling and cutoff grade optimization on large-scale mining operations involving complex metallurgical process options was further discussed in Dagdelen and Kawahata (2007). Kawahata (2006) worked on improving the solution time for MILP problems by way of the Lagrangian relaxation technique.

The MILP model may be set up do determine optimum schedules and cutoff grades for open-pit and underground mines. For the MILP approach, the mines and stockpiles are treated as the sources to be divided into several "sequences" where sequencing arrangements have to be followed (Figure 9.10-5). To start mining in a given sequence in a period, previous sequences may have to be completely mined out within, or prior to, that period. A given sequence may represent a volume of material as small as a single block, a combination of blocks on a given bench, or a volume as large as a complete phase. Each sequence may consist of a number of "increments." Increments can be based on grade intervals or value intervals if the deposit is a multi-metal deposit (see Figure 9.10-6).

The potential process options and waste dumps are defined as the "destinations" in the MILP model. For a given multiple-mine and multiple-process project, the material flows from each "source" to each "destination."

As the material moves from a given source to a final destination, it is subjected to many operational constraints and associated costs. As the material moves from a given source

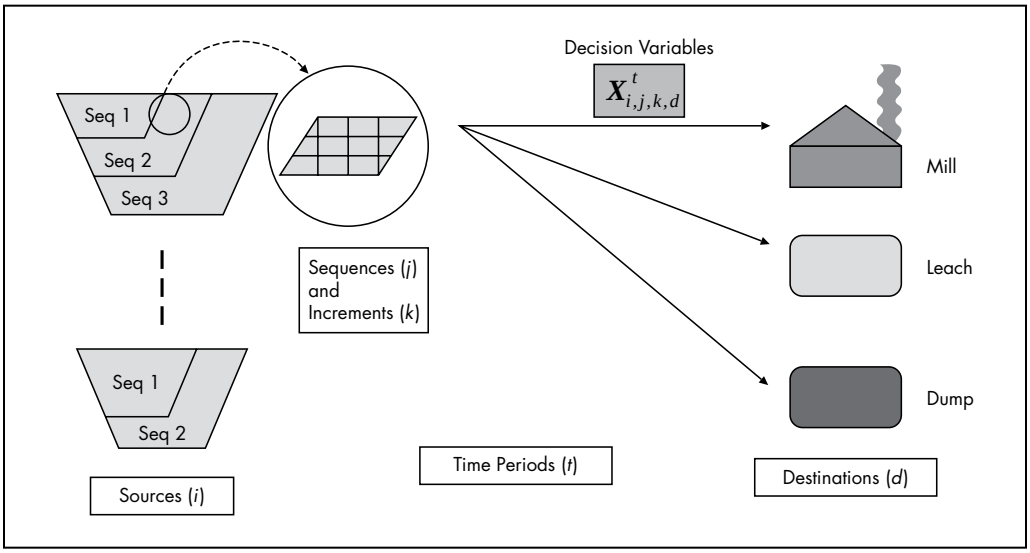


Figure 9.10-5 Production-scheduling decision variables

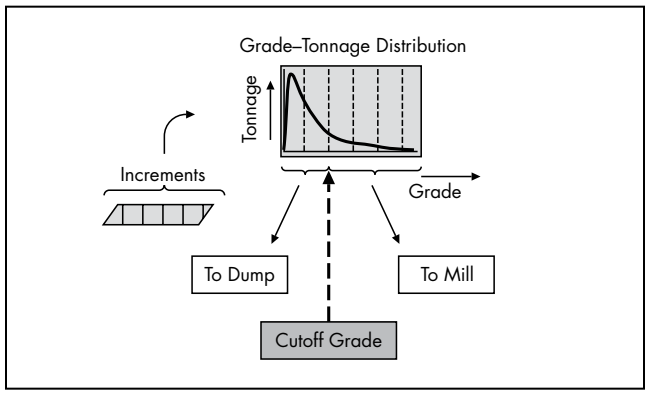


Figure 9.10-6 Cutoff grade determinations in the MILP model

to a given destination, there is an economic value associated with each destination. The objective of the MILP model is to maximize the NPV of the cash flows associated with potentially moving all the material from the sources to destinations such that all of the long-term, operational period requirements and constraints are satisfied.

Objective function. The objective function is defined as follows:

$$MaxZ = \sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}\sum_{d=1}^{D}\sum_{t=1}^{T}\left(SR^t_{i,j,k,d} - OC^t_{i,j,k,d}\right)\times X^t_{i,j,k,d}$$

where

i = source ($i = 1,\ldots,I$)
j = sequence ($j = 1,\ldots,J$)
k = increment ($k = 1,\ldots,K$)
d = destination ($d = 1,\ldots,D$)
t = time period an activity is taken ($t = 1,\ldots,T$)
$SR^t_{i,j,k,d}$ = discounted sales revenue per ton of material mined from source i, sequence j, increment k, and sent to destination d in time period t
$OC^t_{i,j,k,d}$ = discounted operating cost per ton of material mined from source i, sequence j, increment k, and sent to destination d in time period t
$X^t_{i,j,k,d}$ = decision variable, tons of material mined from source i, sequence j, increment k and sent to destination d in time period t

Constraints. The objective function is subject to the following constraints:

1. **Reserve constraints:** The material mined is up to what is available in the geologic reserves, $Res_{i,j,k}$.

$$\sum_{d=1}^{D}\sum_{t=1}^{T} X^t_{i,j,k,d} \le Res_{i,j,k} \qquad \text{for } \forall\, i, j, k.$$

2. **Mining capacity constraints:** Mining capacity at each source can be limited in each time period. $Mcap^t_i$ is the upper limit capacity at source i in time period t.

$$\sum_{j=1}^{J}\sum_{k=1}^{K}\sum_{d=1}^{D} X^t_{i,j,k,d} \le Mcap^t_i \qquad \text{for } \forall\, i, t.$$

3. **Global mining capacity constraints:** Total mining tons from all the sources can be limited in each time period. $GMcap^t$ is the upper limit capacity in time period t.

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}\sum_{d=1}^{D} X^t_{i,j,k,d} \le GMcap^t \qquad \text{for } \forall\, t.$$

4. **Process capacity constraints:** Process capacity at each process destination can be limited in each time period. $Pcap^t_d$ is the upper limit capacity at process destination d in time period t.

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K} X^t_{i,j,k,d} \le Pcap^t_d \qquad \text{for } \forall\, d \{\text{process}\}, t.$$

5. **Attribute blending constraints:** Attribute blending constraints at each process destination can be limited as lower and upper bound in each time period. $Att^n g_{i,j,k}$ is the grade of n^{th} attribute located in source i, sequence j, increment k; $Att^n L_d$ is the lower bound; and $Att^n U_d$ is the upper bound of blending constraints for n^{th} attribute at process destination d.

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}\left(Att^{n}g_{i,j,k}-Att^{n}L_{d}\right)\times X_{i,j,k,d}^{t}\geq 0$$

for $\forall\ n, d \in \{\text{process}\}, t.$

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}\left(Att^{n}g_{i,j,k}-Att^{n}U_{d}\right)\times X_{i,j,k,d}^{t}\leq 0$$

for $\forall\ n, d \in \{\text{process}\}, t.$

6. **Attribute cumulative amount constraints:** Attribute cumulative amount constraints at each process destination can be limited as lower and upper bound in each time period. $Att^{n}g_{i,j,k}$ is the grade of n^{th} attribute located in source i, sequence j, increment k; $Att^{n}CL_{d}$ is the lower bound; and $Att^{n}CU_{d}$ is the upper bound of cumulative amount constraints for n^{th} attribute at process destination d.

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}Att^{n}g_{i,j,k}\times X_{i,j,k,d}^{t}\geq Att^{n}CL_{d}$$

for $\forall n, d \in \{\text{process}\}, t.$

$$\sum_{i=1}^{I}\sum_{j=1}^{J}\sum_{k=1}^{K}Att^{n}g_{i,j,k}\times X_{i,j,k,d}^{t}\leq Att^{n}CU_{d}$$

for $\forall n, d \in \{\text{process}\}, t.$

7. **Sequencing constraints:** To control sequencing constraints, binary variables $Y_{i,\text{previous}}^{t}$ are introduced. A sequence j can only be mined after the set of sequences $\forall$ previous $\in \Gamma j$ are mined out. The following set of constraints control these sequencing arrangements.

$$t\sum_{k=1}^{K}\sum_{d=1}^{D}\sum_{\theta=1}^{t}X_{i,\text{previous},k,d}^{t}-Y_{i,\text{previous}}^{t}\sum_{k=1}^{K}Res_{i,\text{previous},k}\geq 0$$

for $\forall i$, previous $\in \Gamma j, t.$

$$\sum_{k=1}^{K}\sum_{d=1}^{D}\sum_{\theta=1}^{t}X_{i,j,k,d}^{t}-Y_{i,\text{previous}}^{t}\sum_{k=1}^{K}Res_{i,j,k}\leq 0$$

for $\forall i, j$, previous $\in \Gamma j, t.$

8. **Non-negativities:** All the decision variables have to be non-negative numbers.

$X_{i,j,k,d}^{t}\geq 0$ for $\forall i, j, k, d, t.$

$Y_{i,j}^{t}\geq 0$ and binary for $\forall i, j, t.$

These are typical constraints that exist in most of the open-pit mine operations. Other sets of constraints and the stockpile option can also be incorporated, as discussed by Hoerger et al. (1999).

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PART 10

Surface Extraction

CHAPTER 10.1

Introduction to Open-Pit Mining

Andrew Wetherelt and Klaas Peter van der Wielen

Increasing mining costs, declining average ore grades, environmental considerations, and improved health and safety awareness are some of the main challenges facing the mining industry today (Caccetta and Hill 1999). These challenges emphasize the need for optimization of mining, especially when it concerns large-scale mining associated with open-pit operations.

The underlying message expressed in this chapter is that an open-pit mine is an increasingly complex and interdependent system that can only be optimized by careful coordination, management, and harmonization of its individual elements.

DEFINITION OF OPEN-PIT MINING

Open-pit mining can be defined as the process of excavating any near-surface ore deposit by means of an excavation or cut made at the surface, using one or more horizontal benches to extract the ore while dumping overburden and tailings at a dedicated disposal site outside the final pit boundary. Open-pit mining is used for the extraction of both metallic and nonmetallic ores; application of this mining method in coal is less common. Open-pit mining is considered different from quarrying in that it selectively extracts ore rather than an aggregate or a dimensional stone product. The main difference between strip mining—commonly used in the mining of shallow, bedded deposits—and open-pit mining lies in the overburden disposal. In strip mining, overburden is dumped directly onto mined-out panels rather than outside the final pit boundary, as is typical of open-pit mining.

Production tonnages for open pits range from fewer than 15,000 t (metric tons)/yr in small iron ore operations to more than 360 Mt/yr in large porphyry copper operations such as Escondida in Chile. As of 2008, there are approximately 2,500 industrial-scale open-pit metal mines in the world, which is approximately 52% of all industrial-scale mining operations. Iron ore (44%), copper (38%), and gold (15%) together account for 97% of the total open-pit excavation volume (Raw Materials Group 2008).

TYPICAL DEPOSITS

By definition, ore bodies mined through open-pit mining are located at or near the surface. Although the geometry of ore bodies varies from pit to pit, as a general rule it can be said that open-pit mining favors ore bodies that can be mined on a large scale (e.g., extensive ore bodies with low stripping ratios). Porphyry copper deposits such as Chuquicamata and Escondida in Chile, and Bingham Canyon in the United States, are prime examples of such large low-grade ore bodies. Other common ore-body shapes include stratabound and stratiform deposits such as Western Australia's iron ore deposits and the Zambian copper belt mineralization, diatremes typical of kimberlites (Jwaneng in Botswana) and carbonatites (Palabora in South Africa), and stockworks such as the Kalgoorlie (Western Australia) gold deposits.

OPEN-PIT GEOMETRY

The geometry, or layout, of an open-pit operation is discussed in this section. The main considerations are on those parts of the excavation that have to accommodate the main equipment and their operations, namely the benches, haul roads, and overburden disposal site. Two other subjects related to open-pit geometry—pit expansion and transition to underground mining—are also included in the discussion.

Benches

Benches are possibly the most distinguishing feature of an open pit. They are crucial in an operation as they accommodate the active blasting and excavation areas. Benches can be divided into working benches and inactive benches (Hustrulid and Kuchta 2006). Working benches are in the process of being excavated, whereas inactive benches are the remnants of working benches left in place to maintain pit-slope stability. Between main benches, catch benches are left in place to prevent cascading material from compromising safety in active areas of an operation. Figure 10.1-1 shows a simplified geometry of a typical open pit as well as the layout of some of the crucial elements in more detail.

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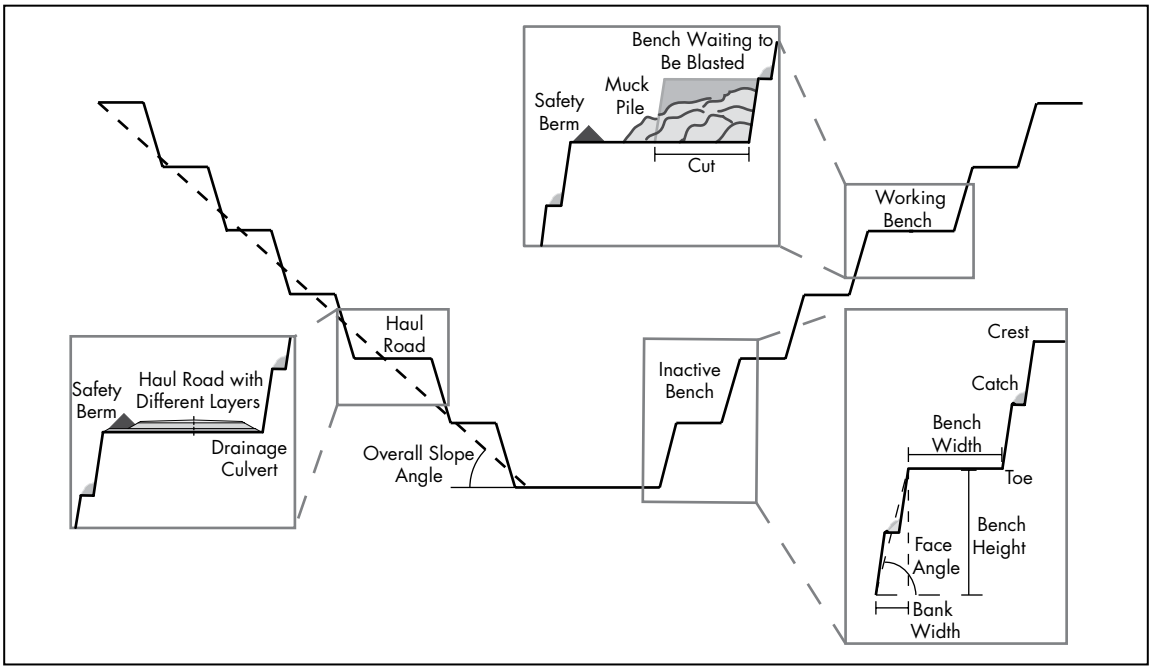


Figure 10.1-1 Typical open-pit geometry

Bench heights typically lie around 15 m. The bench width varies according to equipment size and the type of bench. Working benches should at least be wide enough to accommodate the turning radius of the largest haul truck plus the width of the safety berm (Caterpillar 2006). However, ideally, the bench should at least be wide enough to allow the largest haul trucks to clear the excavator under full acceleration. Depending on the chosen pushback geometry and size of the equipment, the width of a working bench can be anywhere from 30 m to several hundred meters. The width of catch benches is typically between 3 and 5 m but can vary with overall bench height. A small catch berm (~1 to 1.5 m) is usually included at the edge of the catch bench to improve its effectiveness at containing bench-scale rockfalls.

Constituting one of the busiest areas of an open pit, working benches have to accommodate large excavators and dump trucks as well as the muck pile formed after a blast. Therefore, maintaining the quality of a suitable working surface is vital to ensure acceptable safety and productivity levels at an active excavation. As discussed under "Haul Roads," it is important to balance maintenance requirements against the duration and intensity of traffic. This is no less true for benches than it is for haul roads. Floor maintenance resources such as wearing course (also known as road base or road capping), labor, and road maintenance equipment should be allocated according to the added value they have in the operation as a whole. The importance of good floor maintenance on benches is emphasized in research by Ingle (1991), showing that up to 70% of tire damage may occur in active loading and dumping areas. For this purpose, it is important that a well-drained and smooth surface free of rocks is maintained. Dozers or front-end loaders can be used to aid the main excavator in maintaining good floor conditions. Furthermore, they can increase main excavator efficiency by reshaping the muck pile to increase bucket fill factors and possibly aid in the selective mining of ore and barren rock. For health and safety reasons, safety berms (also known as safety benches or windrows) are constructed along crests of benches in a similar manner to those found next to haul roads. The main goal of berms is to stop equipment from backing over the edge of a crest. Generally, a berm with a height equal to the axle height (e.g., at least half of the wheel height) of the largest truck entering an area is not only a safe design but commonly required by mine safety regulations.

Haul Roads

Haul roads constitute a key element of an open-pit mine, providing the main haulage route for ore and overburden from active excavation areas to the pit rim and beyond. Figure 10.1-1 shows the layout of a typical haul road. In light of a trend toward increased gross vehicle mass and haulage distance, detrimental effects of inadequate haul road design, management, and maintenance are becoming increasingly costly (Thompson and Visser 2006). Possible effects are

- Decreased truck and tire life,
- Loss of productivity,
- Poor ride quality, and
- Excessive fugitive dust generation.

All these factors can result in exacerbated vehicle and road maintenance and operating costs. Furthermore, statistics provided by the National Institute for Occupational Safety and Health (NIOSH) show that, in the United States, haul roads are responsible for 20% of lost-time injuries and 42% of fatalities in surface mines (Turin et al. 2001). Lastly, haul roads can significantly impact pit angles and stripping ratios depending on the adopted design and geometry. As such, sound haul road design and management can have a significant positive influence on the safety record, profitability, and environmental impact of a mine.

Thompson and Visser (2006) argue that optimal performance of a haul road network can only be achieved through an integrated approach incorporating (1) geometric, (2) structural,

and (3) functional design as well as (4) the adoption of an optimal management and maintenance strategy. Geometric design covers the basic layout of haul roads based on input criteria such as truck types, traffic intensity, design life of the road, available construction materials, and costs. The next step, structural design, goes into more detail, determining factors such as required materials for road construction material based on the projected design life and traffic intensity. The goal here is to ensure the haul road can accommodate the imposed loads without excessive construction or maintenance costs. Functional design is mainly concerned with providing a safe, vehicle-friendly ride at the best economic performance. Wearing course material selection and performance is crucial during this stage as it is the controlling factor for rolling resistance, fugitive dust generation, ride quality, and surface deterioration rates. The last step, adoption of an optimal management and maintenance strategy, involves developing the most cost-effective approach to maintaining the functionality of the haul road. This subject is covered in more detail later in the "Maintenance and Management" subsection. For a more detailed discussion of design and layout of haul roads, readers are referred to Chapter 10.6 of this handbook.

General Design and Operational Aspects of Haul Roads

Depending on the location and use, haul roads are generally around 3–3.5 and 3.5–4 times wider than the largest truck size on two-way straights and in two-way bends, respectively. Effectively, this places the width of most two-way haul roads between 20 and 35 m, and up to 40 m on bends. For one-way haul roads, a width of 2–2.5 times that of the largest truck size is generally enough. Recommended grades lie between 1 and 8 and 1 and 10 (10%–12.5%), but higher grades are possible when trolley-assist haulage is used. It is important to keep the grade as constant as possible to make truck operation easier and more efficient. Where speeds exceed 15 km/h, corners can be superelevated, although superelevation should not exceed 1 and 10 or 10% (Caterpillar 2006). For better drainage on flat sections, a cross slope of 2% with loaded trucks on the upper part should be considered. On grades, minimum cross slope is required. For safety reasons, angles between roads on intersections should be 90° where possible. Lastly, on two-way sections of haul roads a center berm can be constructed. There is some debate about the effectiveness of such berms—some mines use them; others consider them to be center obstacles without any added value that can cause tire degradation.

Before a shift starts, operators have to perform a basic vehicle check, testing vital systems such as the brakes. Traffic rules on haul roads vary depending on the operation; for example, speed limits for haul roads range from 5 to 40 km/h. Large, loaded trucks in an uphill haul may not be able to achieve the maximum speed. For the rest of a mine site (i.e., workshops, stockyards, crushers, etc.), speed limits generally range from 10 to 20 km/h. In light of safety considerations, trucks traveling in the same direction are normally not allowed to pass, and in most cases passing is also prohibited for light vehicles. In most other aspects, traffic rules on-site closely resemble those of public roads.

Dust

Haul road dust can have a considerable environmental impact, increase maintenance and operations costs, and be a serious safety hazard both in the short term by reducing operator visibility and through long-term exposure, which may cause damage to the respiratory system. The main controlling factors in haul road dust are

- Wind speed at the road surface,
- Traffic volumes and speed on the haul road,
- Particle size distribution of the wearing course material,
- Construction characteristics of the wearing course material, and
- Meteorological conditions at the mine site (Thompson and Visser 2000).

Operator exposure can be decreased significantly by fitting cabs with filtration equipment, airtight seals, and air conditioning as well as increasing the following distance between trucks. Thompson and Visser (2000) state that the most common dust suppression measures include

- The application of a suitable wearing course material,
- Reduction of haulage speeds,
- Regular application of water or chemical suppressants, and
- Sound haul road maintenance.

The selection and application of a suitable wearing course, combined with regular application of water and chemical suppressants, are the most feasible and effective options. Applying a suitable wearing course should be the preferred control measure as it is a preventive rather than mitigating action and it has many beneficial effects in other areas of haul road management and maintenance. The feasibility of watering and chemical suppressants should be evaluated through a cost–benefit analysis. Watering often seems the cheapest alternative, although in light of its short effectiveness and, possibly, limited water supplies, other alternatives may be more feasible (Thompson and Visser 2000). Chemical suppressants are more effective at long-term dust suppression than water. However, as production haul roads are highly dynamic in nature, chemical suppressants generally do not last very long in these environments as continual watering and grading decreases their effectiveness. Furthermore, they do not mitigate effects of material spillages during haulage and are normally more costly. Therefore, continual watering and grading is in many cases the main type of dust suppression for production haul roads, although in some cases a combination of both systems might be the most viable alternative. For auxiliary roads that are more permanent in nature, dust suppressants are a more viable alternative to continual watering and grading.

In addition to the measures discussed previously, avoiding spillages also plays an important role in fugitive dust suppression and, possibly more importantly, prolonging tire life. Chemical suppressants are not effective at suppressing fugitive dust generated from spillages as they are not applied regularly (Thompson and Visser 2000). Watering haul roads will mitigate this problem but suffers from the problems discussed in the previous paragraph. Considering the drawbacks of watering and the effect spillages can have on tire life (as discussed in the "Tire Management" section later in this chapter), avoiding spillages through adequate load placement by the excavator should be preferred over more regular watering.

Maintenance and Management

Rolling resistance is the resistance to motion that a haul truck experiences because of friction. The main contributors are wheel load and road conditions and, to a lesser extent, tire flexing and internal friction. Minimum rolling resistances of

1.5% (radial and dual assemblies) to 2% (cross-ply or single-wheel assemblies) are quoted for rear-dump trucks. Estimates of rolling resistance related to tire penetration indicate an increase in rolling resistance of 0.6%/cm of tire penetration into the haul road (Thompson and Visser 2003). Thompson and Visser also report that similar resistances can arise from road surface deflection or flexing. As haulage is one of the main cost generators in an open-pit mine; rolling resistance reduction can lower capital and operating costs considerably.

Some of the main contributors to a high rolling resistance and bad haul road performance are inadequate wearing course construction and application as well as haul road defects such as potholing, rutting, loose material (dust and stones), corrugations (commonly referred to as washboards), surface cracking, and insufficient drainage. If these defects are left unattended, they may reduce haul road productivity, impede safe vehicle operations, and damage equipment. Five of the main routine maintenance activities for a haul road are

1. Dust suppression measures,
2. Routine surface maintenance,
3. Clearing material spillages,
4. Replacing the wearing course material, and
5. Drainage culvert and shoulder maintenance (Paige-Green and Heath 1999; Thompson and Visser 2003).

All these measures are aimed at maintaining or improving road quality and will mitigate most haul road defects. However, because of limited resources and complex maintenance requirements, a practical maintenance strategy is more effective at mitigation and prevention of these defects.

Semipermanent and temporary haul roads are commissioned and decommissioned during a mine's life. It is vital that the costs of constructing these haul roads are balanced against their design life (Thompson and Visser 2003). Under-expenditure of resources on permanent, high-volume haul roads or over-expenditure on short-term, low-volume haul roads can have serious detrimental effects related to premature failure, compromised health and safety, high maintenance costs for permanent haul roads, or an excessive drain on resources for haul roads with a short, active life span.

Wearing or surface course material plays a major role in the productivity and maintenance requirements of a haul road and the equipment using it. Therefore, the selection, application, and maintenance of wearing course materials are paramount to the good functional performance of a haul road during its operational life. To determine its influence on overall haul road functionality and road user costs, the performance of wearing course materials should be analyzed and benchmarked (Thompson and Visser 2006). Visual inspection of haul roads has traditionally been the main method of determining the maintenance needs for haul roads. However, recent advances in the use of high-precision Global Positioning Systems (GPSs), communications, and equipment monitoring have led to the development of tools for real-time qualitative assessment of haul roads by haul trucks. The integration of accelerometers in haul truck–mounted road monitoring systems may provide quantitative indicators of haul road quality, and as such add a whole new dimension to haul road management (Thompson and Visser 2006).

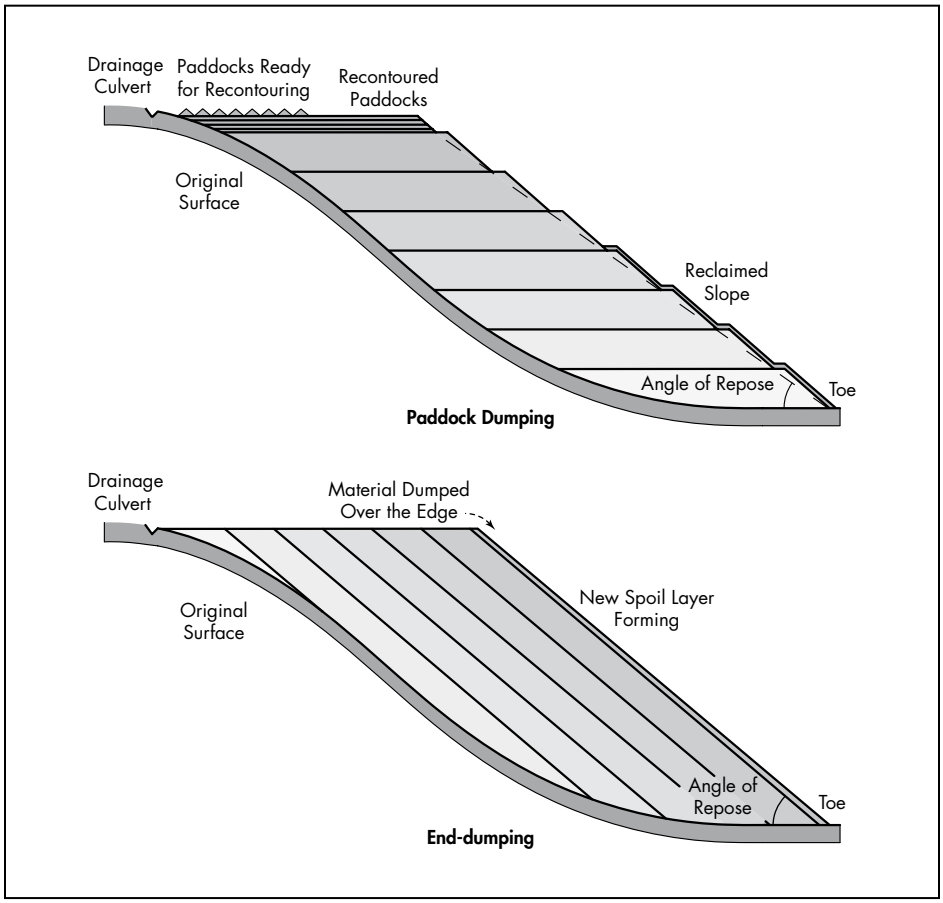
During the scheduling of haul road maintenance, it is important to bear in mind traffic intensity (preferably expressed as gross vehicle mass), function, and projected life span of the road, as well as its maintenance requirements. Routine maintenance and grading may seem like best practice, but when traffic intensities and maintenance requirements are not taken into account, it becomes apparent that this practice may incur excessive maintenance costs on sections of the haul road system with low-traffic intensities. Therefore, optimum maintenance intervals should be determined for haul road maintenance activities, based on a cost–benefit analysis of the relationship between maintenance costs and vehicular cost-incurring factors related to road maintenance, according to Thompson and Visser (2003). They conclude that balancing the optimum maintenance intervals for haul road maintenance and dust suppression with available resources, in conjunction with visual inspection or, ideally, real-time monitoring of haul road performance, should form a solid basis for a practical maintenance strategy for all haul roads in a mining complex. When this proactive, holistic approach to haul road management is implemented, significant gains in operating and maintenance costs, as well as road construction costs, can be achieved.

Two other important elements of a haul road that need maintenance are drainage culverts and safety berms. Similar in size and purpose to safety berms on working benches, safety berms are constructed at the side of a haul road. Drainage culverts are constructed where climatic and hydrogeological conditions dictate their usefulness. They can be located at the pit wall side of the haul road (e.g., where there is a high influx of water) or at both sides of the haul road (e.g., for drainage in areas of high precipitation). Sufficient height of the safety berm and maintaining unobstructed flow in the drainage culverts are essential in ensuring safety on haul roads.

Overburden Disposal

Overburden forms, by far, the largest volume of material produced by most open-pit mines. As overburden generally does not generate any revenue, handling is kept to a minimum. Furthermore, it can contain sulfides or other substances that are potentially damaging to the environment. Consequently, selection of the most suitable site for the overburden embankment involves a trade-off between handling costs related to overburden disposal and the environmental impact of the overburden at a particular site. Minimizing costs involves selecting an overburden embankment site in close proximity to the mine where the environmental impact is as small as possible. Preferably, this site is as close as possible to the projected final pit limit, at the same or a lower elevation as the excavation to minimize upslope haulage costs while maintaining the lowest possible cycle time. With these considerations in mind, optimization of overburden management at a mine site can have a considerable positive influence on the environmental impact and economic viability of a mine.

Overburden is deposited either top-down or bottom-up. Both methods are illustrated in Figure 10.1-2. End-dumping (or top-down dumping) of overburden involves dumping the material over an advancing face. During operation of the dump, only limited reworking of material by dozers is required. Recontouring starts after the end of the dump life. In paddock dumping (also known as bottom-up dumping), the layers of overburden are stacked by dumping on top of the dump, followed with spreading by bulldozers to form relatively thin layers. Paddock dumping is favored from a geotechnical point of view because it allows for more control over the angle of repose of the dumped material and provides a better homogeneity of the material and ultimately better stability (Spitz and Trudinger 2008). Furthermore, paddock dumping provides the possibility



Source: Spitz and Trudinger 2008.

Figure 10.1-2 Comparison between paddock dumping and end-dumping

of concurrent rehabilitation of the overburden embankment and more control over the encapsulation of potentially acid-generating overburden. The lack of homogeneity in top-down dumping provides more potential for settlement and creation of zones of different permeability, both of which can cause more pronounced erosion and eventually instability. Moreover, the lack of homogeneity in rock size can also encourage oxidation of sulfides and consequently acid-mine drainage (Department of Resources, Energy and Tourism, Australia 2006). Paddock dumping does not markedly decrease truck cycle times (Turin et al. 2001). The main advantage of end-dumping is that it is significantly cheaper than paddock dumping due to considerably lower rehandling of overburden both during operation and rehabilitation of the dump site. However, most importantly, paddock dumping is far superior in terms of safety because there is less risk of edge failure and a dump truck falling over an edge when backing up. This point is illustrated by Turin et al. (2001) in an analysis of 10 years of lost-time injuries and fatalities at overburden embankments in the United States. They found that "backing up and falling over an edge" accounts for 73% of fatalities at dumps. Three of the main contributing factors to this are edge failure (23%), the lack of a berm or barrier (35%), and driving through a berm or barrier (31%). This not only shows that proper barrier construction is vital, but also that dump site stability is an important factor. Furthermore, these statistics highlight that, from a safety perspective, paddock dumping, with its better geotechnical control, is the preferred option. In order to further increase safety on an overburden embankment, it is good practice for haul trucks to approach the dumping face from left to right so the operator can inspect the dump berm and dump surface for any tension cracks. The next step is stopping and reversing the haul truck to the dump edge, using the berm as a marker rather than a stopping block prior to tipping.

Pit Expansion

Expansion of an open-pit mine is done in a series of phases, often referred to as pushbacks or cutbacks. From a planning standpoint, a pushback should be aimed at maximizing the financial return from a mine. When planning a pushback, this means taking into account not only the grade of a material but also the costs of development, mining, processing, and marketing (Hall 2009).

The exact geometry of a pushback is very site-specific and depends on a range of factors including ore-body geometry, financial goals, geotechnical consideration, mining equipment, production goals, and long-term planning. Pushbacks can be either conventional or sequential (McCarter 1992). Essentially, both methods push back a pit shell the same distance horizontally; however, a sequential pushback does this through a number of smaller, active benches pushed simultaneously at several levels, whereas a conventional pushback mines the whole horizontal extent of a pushback level before progressing to the next level. Different zones of sequential

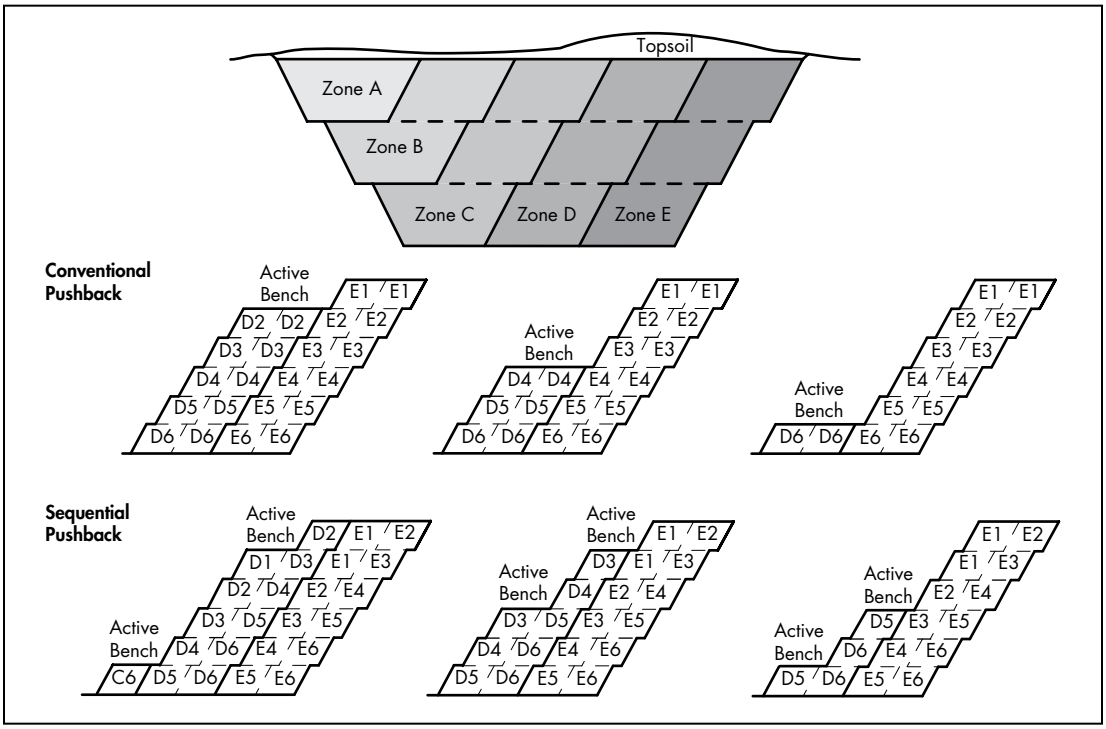


Figure 10.1-3 Conventional and sequential pushbacks

pushbacks are divided by haul roads. Figure 10.1-3 illustrates the differences between sequential and conventional pushbacks. The general pit layout is shown in the top of the figure, divided into different pushback zones (Zone A, Zone B, etc.).

The bottom of the figure illustrates how both methods would mine the same pushback differently. The letter in the blocks denotes the zone, and the number indicates the mining sequence (i.e., mining commences with Block D1, then D2, etc., until the last block is mined).

In a conventional pushback, all activity is concentrated on one level as compared to multiple active levels in a sequential pushback. This means that sequential pushbacks require more complex planning than conventional pushbacks. However, developing several areas simultaneously means there is more flexibility during excavation, allowing superior control over production planning and blending of ore. Furthermore, as sequential pushbacks contain a number of active benches on the pit slope, the working slope angle is lower than the final pit slope angle, which does not have these wider, active benches, resulting in improved slope stability. Conventional pushbacks, on the other hand, are less complex to schedule and the larger active area at a given time allows more working faces within a pushback. Disadvantages include less flexibility in scheduling and blending as well as a greater vulnerability to operational problems.

Presplit blasting is often done at the final reach of a pushback or pit shell. By allowing more control over face angle and back break, this improves long-term stability of the pit face. Sequential pushbacks are more commonly seen in large-scale open pits, whereas conventional pushbacks are more common in shallow and small-scale operations.

Transition to Underground Mining

In some cases, notably with vertically extensive ore bodies, it can be profitable to continue mining by underground methods after the final pit limit has been reached. In recent years several big open pits, such as Palabora, have commenced underground production with others, such as Chuquicamata and Debswana's Jwaneng mine, announcing plans to go underground. The transition from open-pit to underground mining presents a set of unique geotechnical, planning, and management challenges.

With regard to management, the first challenge is deciding on the feasibility of underground mining. Most comparison methods between open-pit and underground mining rely on establishment of a break-even stripping ratio and a comparison of the net present value for the next feasible open-pit pushback to that for an underground mine. After feasibility of underground mining has been proven, timing of underground mining is the next issue to be decided. There are two major considerations in the timing of the transition. First, to maintain continuity of the operation it is important that the underground mine can supplement and eventually take over production from the open pit without major permanent changes in tonnages of ore shipped to the mill. Differences between the open pit and underground ore grade and composition may complicate this issue. A production overlap between open-pit and underground mining is therefore common to allow for a smooth transition from open-pit to underground mining. Second, while a smooth transition requires a production overlap, neither of the two operations should compromise the other's safety. Geotechnical analysis should provide insight into the interaction between the two adjacent operations. Safety in the surface operations could be jeopardized as a result of crown pillar failure or due to mining subsidence-induced slope failure (e.g., when a caving method is used). On the other hand, surface blasting-induced vibrations could compromise safety underground.

Siting of the portal is another major consideration in the transition from open-pit to underground mining. Most

importantly, it is crucial that the stability of the underground mine entrance should not be compromised in any way by surface operations. This is as true for a decline portal within an operating pit as it is for a shaft located near a pit wall. Additionally, when a decline portal is sited within the open-pit walls, surface traffic interactions relating to both excavations should be kept to a minimum. This includes not only keeping haulage routes for both operations separated to the highest extent possible, but also considering the effects of blasting.

UNIT OPERATIONS

In mining, unit operations can be defined as those basic steps necessary for the production of payable material from a deposit, and the auxiliary operations used to support the production (Hartman and Mutmansky 2002). Unit operations in an open-pit production cycle include access, drilling and blasting, excavation practices, and haulage methods. Auxiliary operations and other production-related activities include overburden and topsoil removal, ancillary operations and mine services, and equipment monitoring and maintenance.

Access

The first step in the development of a new mining operation or a pushback is gaining access to the area to be mined. Especially when working in remote areas, an appropriately planned and designed access road with sufficient capacity can make a substantial difference in the initial success of a new operation.

After permanent access to the new mining area has been secured, removal of topsoil and overburden can begin. In mining, overburden refers to all unprofitable material that needs to be excavated to access an ore deposit, including topsoil and overburden. Topsoil refers to the layer of unconsolidated material at the surface that is suitable for sustaining plant growth. Because of the unconsolidated nature of topsoil, it often requires different excavation techniques. Depending on climate, topography, and bedrock geology, topsoil can vary from anywhere between centimeters and tens of meters thick. Overburden refers to the consolidated material underlying the topsoil and generally overlying the ore body. If overburden is encapsulated between two layers of ore, it can be referred to as interburden.

Topsoil Removal

After the initial pit outline has been staked out, vegetation should be removed and any surface water courses should be diverted away from the site. As topsoil is generally free digging, scrapers, bulldozers, front-end loaders, and small hydraulic excavators are the most common equipment used in topsoil stripping. Bulldozers can be used for pushing material onto piles for further excavation by front-end loaders or hydraulic excavators. Alternatively, they can support scraper operation by ripping soil or by pushing scrapers along where they do not have enough traction. Graders are mostly used for precision applications such as haul road construction. Haulage distance is an important consideration in choice of equipment. At short haul distances, scrapers and bulldozers are more economic, whereas a more conventional excavator/truck haulage operation tends to be more economical at longer haul distances.

Before the bedrock is reached, the exposed soil may be a large potential source of dust, and a water bowser or tank truck (a mining truck that has been adapted for the distribution of water) may be required for dust suppression. In many operations, topsoil storage is required for reclamation purposes at the end of mine life. In some cases separate storage of different topsoil and subsoil layers may be necessary to ensure quality of the material. Depending on the duration of topsoil storage, revegetation and erosion control may be required.

In some operations, the excavation may never progress into consolidated material and the practices discussed here may also apply to the excavation techniques used for ore.

Overburden Removal

Removal of overburden is often required before extraction of ore can begin. Regarding materials handling, there are three important differences between ore and overburden:

1. Overburden is not beneficiated and will generally not generate any revenue.
2. Overburden tonnages almost invariably exceed ore tonnages in an open-pit mine.
3. The rock mass characteristics are often different from that of the ore.

The first two points imply that handling of overburden and related costs should be kept to a minimum. To minimize costs related to the handling of overburden, it is often blasted to a coarser fragmentation than ore, and in many cases excavated and hauled by larger-capacity equipment. As a result, in many operations the stripping of overburden is contracted out. Drilling, blasting, excavation, and haulage practices for consolidated material will be discussed in more detail in the following sections.

Drilling and Blasting

Drilling and blasting comprise the first two of the four main stages in the production cycle of an open-pit mine and the most common method of rock breaking. Other rock-breaking methods such as mechanical breaking and surface miners can generally not compete in terms of either production rate or economy and will not be discussed in this chapter.

Blast Design

Blast design is the first and most crucial step in drilling and blasting. First and foremost, blast design is an iterative process, where important factors such as the required fragmentation, production, and muck pile shape are used as a starting point for determining optimal drill-hole diameter, depth and inclination, subdrilling, explosives type, and detonation timing (Hopler 1998). Importantly, operating costs of both the mine and the processing plant are directly related to the fragmentation achieved during blasting (Bhandari 1997). Bench height and subdrilling requirements dictate drill-hole depths. *Subdrilling* is the term used for the length a blasthole extends beyond the excavation level. This is done to reduce the risk of equipment damage as a result of poor floor conditions, which in turn is governed by geological features and the strength of the rock mass. Figure 10.1-4 shows a basic blast and blasthole geometry.

Bench height is generally fixed by ore-body characteristics and geometric and geotechnical considerations and is taken as the starting point for blast design. Selection of a suitable drill-hole diameter is a complex process taking into account a host of factors related to production requirements, rock mass characteristics, environmental considerations, and equipment selection (Hopler 1998). In general, it can be said that larger drill-hole diameters can be operated at larger burdens and spacings. However, this does result in larger

fragments compared to smaller-diameter holes at the same powder factor. Assuming a fixed bench height, a good indication of hole diameter, D, is given by Bhandari (1997) as

$$D = H/120$$

where H is the bench height in meters.

When determining the blasthole diameter, it should be taken into account that there is an inverse relationship between hole diameter and operational costs (Bhandari 1997). Typical values range from 83 to 350 mm depending on the scale of the operation.

Drilling patterns (e.g., burden/spacing ratio) vary substantially in size, and patterns are tailored to specific operations. There are many ways of calculating spacing (S) and burden (B), most of which are based on hole diameter. The following rules are considered a good starting point for the estimation of the spacing and burden (Hustrulid 1999):

$$B = K \times D$$

where K is a constant and D denotes hole diameter in meters, and

$$S = 1.15 \times B$$

where B denotes burden in meters.

The constant K is dependent on the strength and density of the explosive, as well as rock "blastability." It ranges from 20 in dense rock with light explosives to 40 in light rock with heavy explosives, but typically it lies between 25 and 35 (Hustrulid 1999). In strong or blocky rock and to achieve optimal fragmentation, conservative values should be used when calculating the burden. Typical values for the burden range from 3 m in small-scale operations to 10 m in very large-scale operations.

The relationship between burden and spacing—the burden/spacing ratio—can be varied according to the chosen blasting pattern. Square patterns ($S \sim B$) are easy to lay out but result in poor charge distribution (Bhandari 1997). Elongated patterns ($S > B$) are preferred in hard-breaking rock and when there are problems with back break (Hustrulid 1999). Staggering a pattern further complicates layout but, due to superior blasting energy distribution, results in better fragmentation. If there are problems with back break or when superior face angle control is necessary, presplitting can be done with a smaller drill rig.

Stemming is placed on top of the explosive column to ensure efficient use of the explosive energy and to reduce air overpressure. Drill cuttings are often used for stemming. This is a cheap alternative to inserting specialized stemming material, but it is less efficient at containing blast energy, possibly resulting in unsatisfactory blasting results (e.g., vertical flyrock and oversize blocks). Where possible, angular material is preferred as by nature it tends to lock in place better during the detonation process, further improving confinement of the explosive pressure. Appropriate stemming-chip size lies in the range of 10% of the blasthole diameter. A good approximation of stemming depth is $0.7–1 \times B$, and is commonly between 2 and 7 m.

After a drilling pattern is established, a delay sequence should be fitted to this pattern. The first consideration in determining delay intervals is the availability of free faces. A blast should be initiated at the free face and aim at maximizing the use of the free face throughout the blast. When there are no free faces available (e.g., a box cut or sump blast), a diamond cut is the best option, displacing rock upward. In the case of one free face, a chevron pattern (V-pattern) is recommended, although a row-by-row pattern can also be used (Bhandari 1997). The angle of the V can be varied according to the local geological conditions and the desired blasting result. When there are two free faces, an echelon pattern usually produces the best results. Figures 10.1-5 through 10.1-8 show different standardized blast designs with relative detonation sequencing of the rows. Other cuts exist but are generally used in more specialized applications. After a delay sequence has been established, appropriate delay intervals can be assigned to rows or individual holes. Regarding delay intervals, Bhandari (1997) recommends 3–6 ms/m of effective burden (i.e., at the time it is blasted and not when it is drilled). Apart from the drilling pattern, other important considerations during the selection of suitable delay timings are safety of the blast, geology, prevention of surface cutoffs, vibration reduction, and fragmentation requirements. Delay timings should be customized to prevailing geological conditions, even within the same operation.

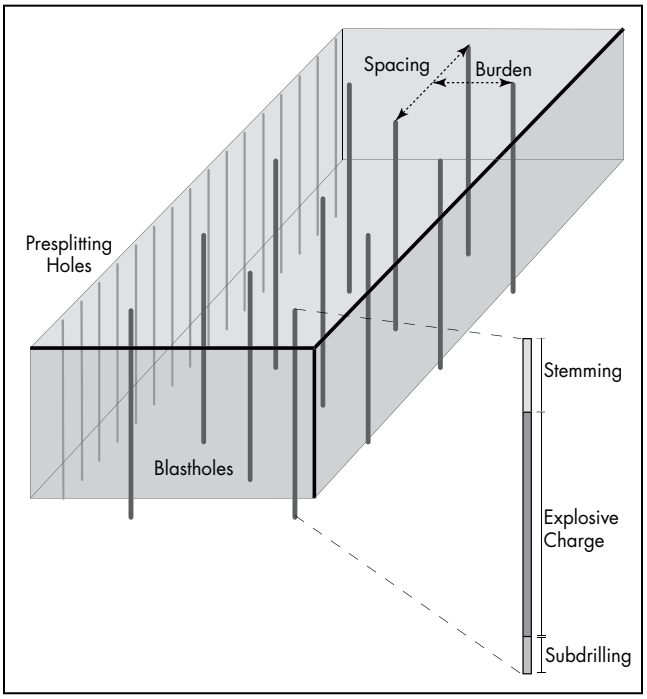
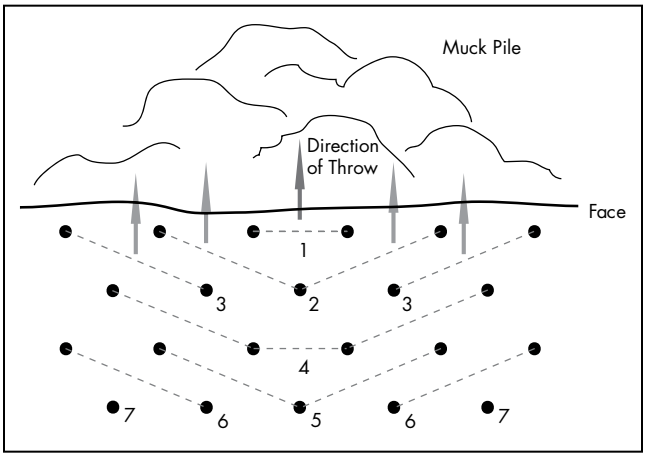


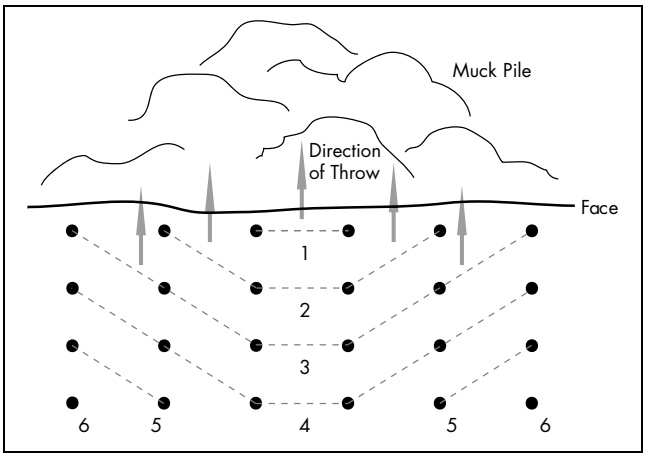
Figure 10.1-4 Blast and blasthole geometry

The shape of the muck pile required by the excavator is another consideration during blast design. High, compact muck piles are generally preferable for rope shovels and hydraulic excavators, whereas a low, flat muck pile is better for front-end loaders (Hustrulid 1999). The delay pattern, point of initiation, and number of rows are the main influences on muck pile shape. The higher the number of rows in a blast, the larger the vertical component of rock movement and the higher the resulting muck pile (Bhandari 1997).

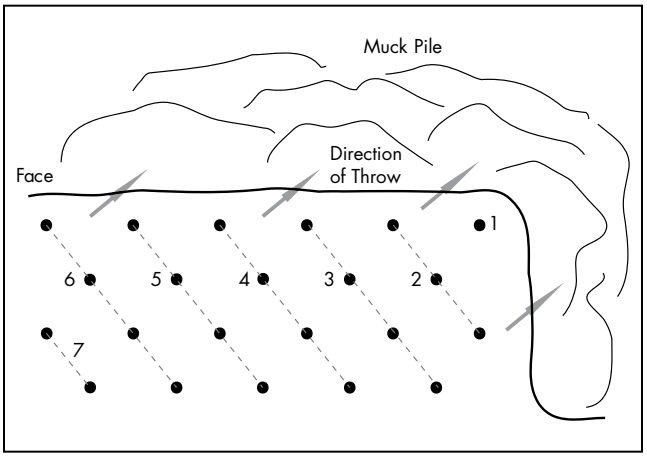
The mining industry is becoming increasingly aware of the benefits of achieving a good fragmentation during blasting. Most importantly, drilling and blasting is, comparatively, the cheapest method of comminution. Therefore, achieving good fragmentation at this early stage can have a significant positive



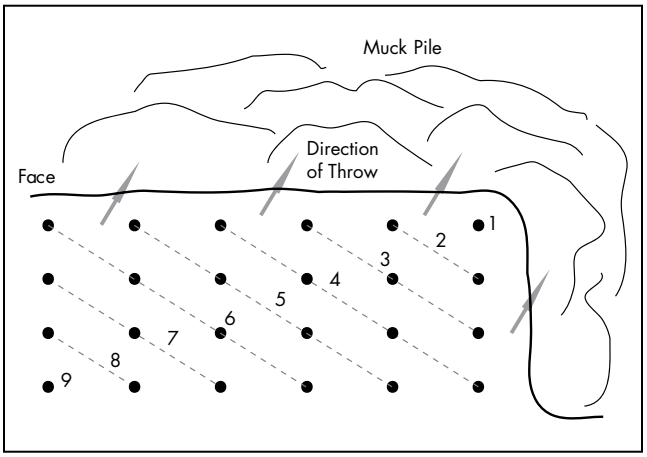
Source: Hustrulid 1999.
Figure 10.1-5 Face blast chevron staggered



Source: Hustrulid 1999.
Figure 10.1-6 Face blast chevron rectangular



Source: Hustrulid 1999.
Figure 10.1-7 Corner blast echelon staggered



Source: Hustrulid 1999.
Figure 10.1-8 Corner blast echelon rectangular

impact on the efficiency and costs of downstream comminution processes (Borquez 2006). Furthermore, better fragmentation allows for better use of the capacity of the excavation and haulage equipment in a mine. On top of that, optimization of blasting at a specific site may suggest that the same results can be obtained by using less explosives, and it may reduce the amount of oversize boulders produced. With these considerations in mind, it becomes clear that determining the optimal fragmentation is a function of not only the effectiveness of the drilling and blasting process but also of excavation, haulage, and the downstream comminution processes.

The first step toward obtaining optimal fragmentation results is adopting a blast design aimed at producing the best fragmentation. Generally, this means implementing a more closely spaced drilling pattern (especially a lower burden) with smaller-diameter blastholes and more accurate timings, but the exact blast design depends on a host of site-specific parameters. Ongoing research is aimed at establishing the most influential factors in fragmentation and how blast design can be geared toward optimized fragmentation using these factors.

Application of high-precision GPSs, accurate drill guidance, and drill monitoring have made it possible to drill blastholes with very little deviation. Together with the use of electronic detonators, these two fairly recent advancements in drilling and blasting have made it possible to consistently produce optimal fragmentation during a blast. These developments are further aided by the introduction of imaging software that can analyze the fragmentation of blasted material, allowing for an iterative approach toward optimal fragmentation.

Drilling

Production drill rigs are usually truck- or crawler-mounted and are powered either by a diesel engine or an electric drive. Pull-down and hoist forces are applied by either hydraulic or chain-hoist systems. A range of systems is available for monitoring machine health and the drilling process.

Production drill rigs are divided into rotary, top hammer, and down-the-hole (DTH) hammer drill rigs. Rotary drill rigs rely on a pull-down force transmitted through a rotating drill string usually with a tricone bit for the cutting action (Australian Drilling Industry Training Committee 1997). Rotary drill rigs are generally most efficient in medium to hard rock and in holes with a diameter larger than approximately 170 mm. Hole depths can extend to more than 80 m in extreme cases. Top hammer drill rigs transmit the hammering force from the drill rig through the drill string down the

hole. They are preferred for hole diameters up to 140 mm, depths down to 20 m, and mostly used in small-scale operations and precision applications such as secondary breaking. Hole straightness and energy loss at drill string joints are the main factors limiting the use of this type of drill rig for deeper and larger-diameter holes. DTH hammer drill rigs rely on compressed air for operating a piston at the end of the drill string to provide the hammering action. Common applications include presplitting, dewatering holes, and other applications where high accuracy is required, although they are also used for blasthole drilling. DTH hammer drill rigs are the most efficient drill rig type in hard to very hard rock. Drill-hole diameters for DTH hammer drill rigs commonly lie between 140 and 170 mm, and depths of up to 40 m are feasible. Selection of the most suitable hole size and drill type for a particular mining operation is a function of balancing projected operating and capital costs with the required rock fragmentation, wall stability, grade control, and production requirements.

Generally, drilling and blasting is the bottleneck in terms of time consumption in the drilling and blasting cycle. It is absolutely essential for the overall productivity of a mine that muck pile volumes are sufficient to keep excavators and the haulage system used to the highest degree at all times (Hopler 1998). Drilling productivity is dependent on rock hardness, drill rig and bit selection, bailing air volume, and engine capacity. Rock hardness is the main determinant in the drillability of a material, and the drill rig and bit should be selected for the prevailing conditions. Engine capacity is the most important factor in fitting a drill rig to the drillability of the material as it determines torque, rotation speed, and pull-down force of the rig. Bailing air, provided by an onboard compressor, is used to clear broken rock from the bottom of the hole. The airflow should be sufficient to clear rock chippings out of the hole but not so high as to cause excessive fugitive dust generation, wear on the drill string, and excessive fuel/electricity consumption.

Blasting

Ammonium nitrate-fuel oil (ANFO) is the most common and cheapest surface blasting agent, followed by emulsions and slurries. The ingredients making up the explosive substance are carried to the blasting location in separate compartments of a specialized truck. For safety reasons the ingredients making up the explosive substance are generally not mixed together by the explosives truck until it is on-site, loading the blastholes. Optimal fragmentation is usually achieved when explosives are distributed so that the lower third of the hole depth contains half of the explosive charge (Hopler 1998). If high drilling costs or problems with fragmentation are experienced, it can be advantageous to use stronger explosives or deck charges, or add additives to increase explosive energy (Bhandari 1997). Shock tube, detonating cord, and trunkline/downhole combinations of these two systems are the most common surface detonation systems. Pyrotechnic millisecond connectors are the most common delay mechanisms, although in recent years electronic detonators have rapidly been gaining ground because of unrivalled delay timing accuracy. In most applications, a detonator is used to set off a primer or booster that will in turn initiate the explosive charge.

Before charging a blasthole, it is good practice to check for water (stagnant or influx) and unexpected voids and to verify drilling was done to the planned pattern, hole depth, and inclination. If water is found, the hole should be dewatered; a polyethylene liner can be inserted and/or a water-resistant explosive can be used.

Secondary blasting may be required to break oversize boulders that are too large for the primary crusher. Mudcapping and blockholing are the two most common secondary blasting methods. Mudcapping involves molding an explosive to a rock surface and covering it with mud. Blockholing requires drilling a hole in the rock and charging it with explosive cartridges. Both processes are expensive and may well produce excessive air overpressure and flyrock. It is essential to be able to see all sides of the boulder in the case of blockholing due to the possible presence of a misfire from the initial blast. Alternatively, mechanical breakers can be used to break oversize boulders, but this technique is often inefficient when dealing with competent rock types.

Excavation

Excavation is the third main stage in the production cycle in a mine. Depending on the size of the operation and the type of haulage system, electric rope shovels, hydraulic excavators, or in some cases large front-end loaders are used in open-pit mining operations.

Equipment

Rope shovel bucket capacities have risen to just over 100 t. The largest bucket capacities for hydraulic excavators currently available are slightly lower than that of rope shovels, topping at 90 t. Front-end loaders normally have capacities of around 36 t, although there are larger models with capacities of up to about 90 t.

Until recently, rope shovels were the sole players in the 60 to 100-t range. However, there are now several hydraulic excavators competing directly with rope shovels in this size range. Rope shovels are still the standard in high-production, low-cost mines because of their reliability and long life. In the intermediate size range (30 to 60 t), the choice of equipment is to a large degree site-dependent, opting either for the flexibility of a hydraulic excavator or the reliability of a rope shovel. In the smaller size range (<30 t bucket capacity), hydraulic excavators account for almost all orders and are slowly replacing rope shovels.

The net digging force in hydraulic excavators is a combination of break-out or curling force (bucket-tilt cylinders), boom force (boom cylinders), and the crowd force (stick cylinders). In rope shovels, the net digging force consists of the crowd force (crowd machinery) and rope pull (hoisting machinery).

For hydraulic excavators, face shovel and backhoe configurations are available. Face shovel configurations are preferred in harder rock and with higher rock faces. Backhoe configurations allow for more selective digging and faster cycle times as swing angles can be reduced when loading a truck on a lower level. Front-end loaders are the preferred choice for specialist jobs where their mobility and flexibility can be used to a maximum extent. Examples include blending operations, road and infrastructure construction, working in the confined space of a drop cut, and as support or backup for larger excavators.

Excavation equipment can be evaluated in terms of productivity (metric tons per hour) and efficiency (cost per metric ton). The bucket fill factor is an important consideration in the overall productivity of an excavator and is dependent on the

truck coverage, "diggability" of the material, operator skills, and net digging forces from the excavator. Other important factors in achieving acceptable productivity and efficiencies from excavators are matching the trucks to excavator sizes (ideally three or four loading passes), selecting the right excavator for the bench height, and providing enough working space for the excavator and trucks to operate (Hustrulid and Kuchta 2006).

Excavation Practices

Excavation of a new level often begins with a drop cut. Alternatively, when excavations are still above topography (i.e., on a hillside), a new excavation is begun from an access road without the need for a drop cut. The confined working space in a drop cut often requires the use of a backhoe excavator from the top level or a front-end loader driving in and out of the drop cut. Alternatively, a truck can turn and reverse into position next to a small excavator in the drop cut. Lateral extension of the bench is done through frontal or parallel cuts. Frontal cuts involve cutting adjacent niches into the muck pile rather than having an excavator moving with the muck pile parallel to the bench wall as done in parallel cuts (Hustrulid and Kuchta 2006).

As shown in Figures 10.1-9 through 10.1-11, trucks can drive by or stop and reverse into a position next to the excavator. Hustrulid and Kuchta (2006) have written a comprehensive description of different possible modes of truck–excavator operation. According to them, available working space, the necessary swing angle of the excavator, and truck positioning time are the major considerations in the selection of the type of operation. Drive-by operations are most suitable for parallel cuts, optimizing efficiency by reducing positioning time of the truck. Frontal cuts can require excessive swing angles by the loader, making it inefficient. The disadvantage of drive-by operations is that they require larger working areas and ideally a separate ingress and egress route from the loading area. Stop-and-reverse operations can be employed in combination with both parallel and frontal cuts. They require less operating space and are more efficient from the excavator point of view. When there is sufficient space on the bench, a truck can turn without the need for reversing.

With regard to operating techniques, it is generally considered good practice to excavate the farthest or hardest-to-dig material in the first pass while waiting for the truck to position. The intermediate passes can be positioned where the operator deems fit and the last pass can be used for floor cleanup. This is only necessary when there is no dedicated support equipment for floor cleanup. Correct load placement is crucial in avoiding spillages and excessive truck wear. Ideally, loads are centered on the center line of the body (longitudinally) above the hoist cylinders of the truck (laterally) with no material on the headboard and enough freeboard on the sides and rear (Caterpillar 2006). Considering optimal loading of dump trucks, an original equipment manufacturer (OEM) recommends what they call the 10/10/20 payload policy. This policy states that "no more than 10% of loads may exceed 10% over the target payload and no loads may exceed 20% of the target payload" (Caterpillar 2006). Average cycle times range from 25 to 27 seconds for hydraulic shovels in backhoe and face shovel configuration, respectively, to an average of 35 seconds for rope shovels and 38 seconds for large front-end loaders (Caterpillar 2006). In well-fragmented rock, rope shovels commonly have bucket fill factors around 100% to 105%, for backhoe excavators this lies between 80% and 110%, and large front-end loaders generally can attain bucket fill factors between 90% and 110%. Bucket fill factors of more than 100% are achieved by heaping material in the bucket. Backhoe excavators are most efficient on benches no higher than the length of the stick with short swing angles, loading a truck on a lower level.

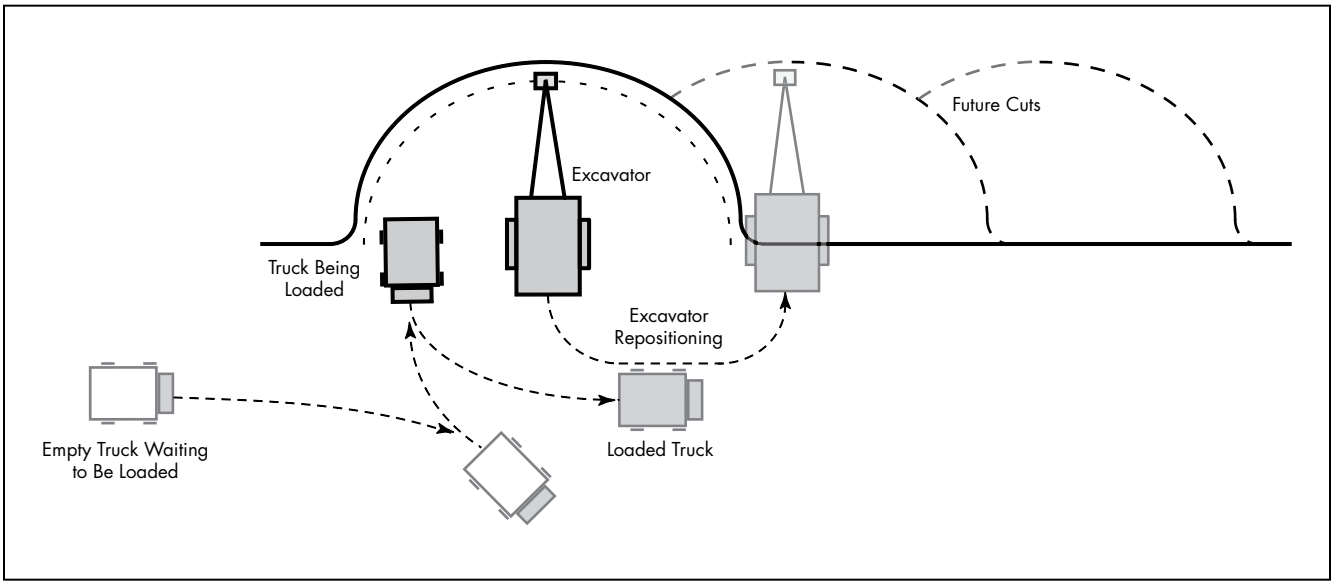
Some of the main productivity losses for excavators are idle time while waiting for trucks, excavator relocation (especially with rope shovels), poor operator skills, bad digging conditions (i.e., poorly fragmented rock, wet material, poor underfoot conditions, incorrect muck pile shape and dimensions), and unplanned downtime. For front-end loaders, there are the additional requirements of good floor condition maintenance and good drainage to prevent excessive tire damage. Careful scheduling can increase productivity by reducing idle time and excavator relocation time. Reduction of excavator cycle times can also improve the productivity of a mining operation markedly. To illustrate this point, in a typical operation, with all other factors unchanged, a reduction of 5% in the excavator cycle time can equate to up to 40 extra truckloads per day. Implementing cycle time improvements suggested by personnel, OEMs, or consultants is crucial in capitalizing on possible cycle time reductions. Among the possible improvements are better floor condition maintenance, adequate operator training, and excavator-optimized blast design to ensure the correct fragmentation and muck pile shape. Furthermore, excavator manufacturers play a pivotal role in providing technological advances such as increased engine performance, monitoring, and automation to reduce possible cycle time.

Grade Control, Reconciliation, and Selective Mining

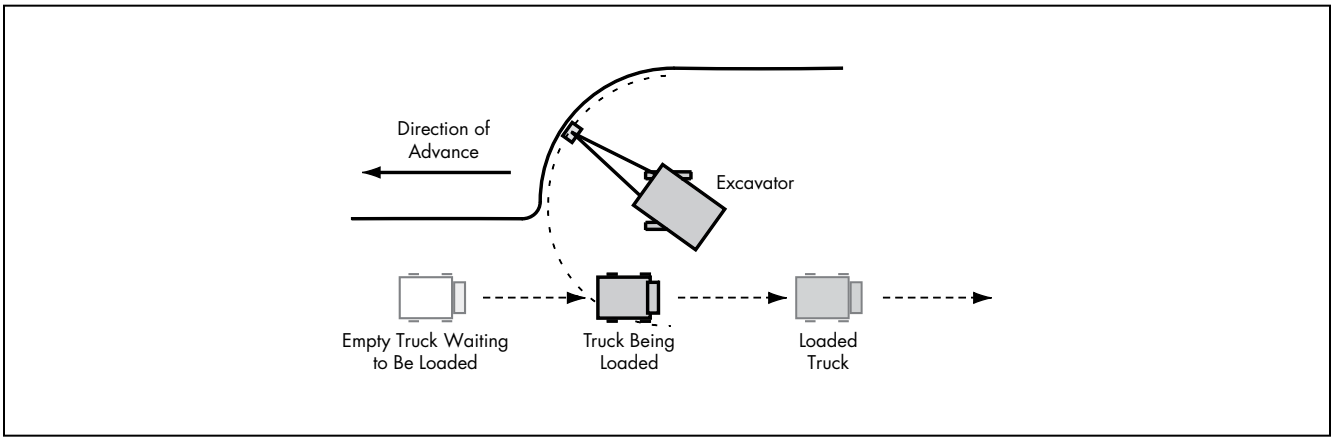
Open-pit mining has historically been considered a bulk mining method with low selectivity. However, a global trend toward increasingly challenging open-pit mining conditions combined with the need to control costs and optimize mill performance have accentuated the need for optimum fragmentation, selective mining, and improvement in grade control. In line with this need, technological advances make selective mining increasingly feasible by decreasing the selective mining units: the minimum volume of material that can be extracted by an excavator without significant dilution. The ultimate goal of grade control and selective mining is to ensure a constant mill feed, as well as minimizing ore loss and dilution. Depending on the operation, profits from improved grade control, grade reconciliation, and selective mining can be larger than from any other operational improvement and as such they deserve thorough attention (Sinclair and Blackwell 2002).

Grade control requirements and practices are largely dependent on the commodity. First, the commodity price controls the implications of ore loss, and management has to justify the extra expenditures relating to selective extraction. Second, the increased mill performance due to lower dilution must justify any additional actions required during the mining processes. Last, the style of mineralization, often commodity-specific, dictates whether grade control is geared more toward ore/barren rock discrimination or whether it is focused on grade and stockpile control (Davis 1992).

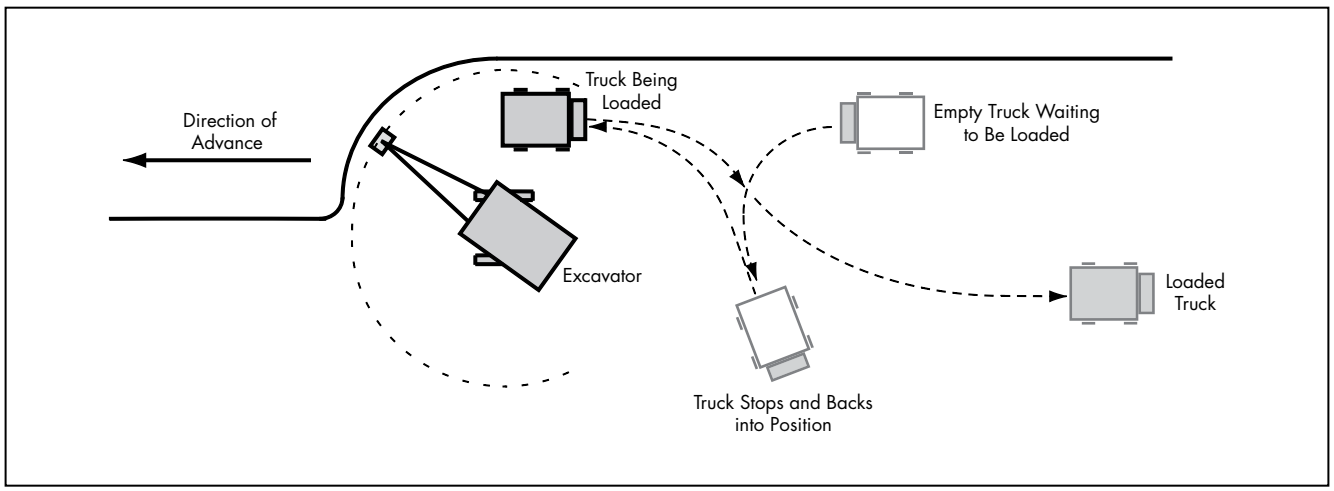
Unbiased sample acquisition, meticulous sample processing, and accurate sample analysis on a short time scale are keys to effective grade control. In hard-rock operations, sampling blasthole cuttings is the most effective option for grade control, although sometimes a drill rig is dedicated solely to grade-control drilling (Annels 1991). In this case, timing of



Source: Hustrulid and Kuchta 2006.
Figure 10.1-9 Frontal cut



Source: Hustrulid and Kuchta 2006.
Figure 10.1-10 Drive-by operation



Source: Hustrulid and Kuchta 2006.
Figure 10.1-11 Stop-and-reverse parallel operation

sample acquisition is crucial because of the risk of contamination during drilling. Operations mining unconsolidated material often employ a continuous trencher (commonly referred to as a ditch witch) to produce samples. Best results are obtained by trenching at right angles to the predicted ore-body orientation. Sample results should be used to model grade distributions in planning or geostatistical software, while taking into account the minimum selective mining unit of an excavator. On the basis of the produced models, a clear demarcation of ore and barren rock in active operating areas can be provided, often done using colored flags or pegs. Hanging colored ribbons down the face can further aid excavator operators in discerning ore and barren rock.

The next step in grade control is the selective mining of material. A distinction should be made between free-digging material (e.g., laterites) and consolidated material, because drilling and blasting aggravates potential ore losses and dilution (Davis 1992). In free-digging material, excavators often work across the strike of the ore body. *Flitches* refer to the steps or thin "lifts" in which a bench is mined. Depending on ore-body geometry and the minimum selective mining unit of an excavator, they can be as small as 1 m, although 2.5 m is a more common size. Ideally, mineralized portions of the flitch are removed prior to the barren rock. To minimize risk of dilution and ore loss in a drilling and blasting operation, it is important that excessive intermixing and movement of blasted material is inhibited. When a whole blast is located in ore, spacing and burden can be reduced to increase fragmentation and subsequently reduce comminution costs. Alternatively, when a blast only breaks up barren rock, increased spacings and larger burdens reduce fragmentation and decrease production costs. If ore and barren rock are intermixed on a subblast level, they are blasted together before demarcation of both zones. Bulldozers can be used to separate ore and barren rock before excavating, although this can result in "smearing" of the ore or barren rock margin. In any case, supervising excavations closely is important to ensure correct dispatching of trucks and, possibly, to visually discern ore/barren rock contacts during excavation. After the ore is loaded into a truck, it should be dispatched to the dump or the correct stockpile. The use of modern equipment dispatching systems greatly aids in dispatching trucks to the correct destination.

Apart from ore/barren rock discrimination and assigning metallurgical grades to material, grade control also provides a basis for reconciliation of mill production figures, geostatistical models, and pit production tonnages and grades (Davis 1992). Discrepancies between the mine production and mill production can serve as an indication of poor mine or mill performance, a lack of communication between different departments in a mine, sampling error, lab error, or errors in the geostatistical model.

Haulage Systems

The fourth and last main stage of the production cycle in a mine is haulage. Rigid-frame haul trucks have dominated haulage in open-pit mining operations for decades, although in some cases articulated dump trucks (ADTs) have proven a viable alternative, and sometimes rail haulage is still being used. Furthermore, longer haulage distances in many large pits, availability problems with haul trucks, and improvements in technology have revived interest in in-pit crushing and conveying (IPCC).

Rigid-frame trucks, possibly in combination with trolley-assist hauling, are the preferred choice for haulage in most open-pit mines. Compared to ADTs, they are far superior in payload capacity, achieve better speeds in most road conditions, and have lower maintenance requirements. However, in certain cases such as small-scale operations and operations that struggle to maintain adequate surface conditions, the flexibility and versatility of ADTs can pay dividends. IPCC systems are another alternative to conventional truck haulage. Whether IPCC is economically viable is a function of production, duration of the operation, and the distance and vertical lift of the haulage route.

Trucks

Payload capacities for rigid-frame haul trucks currently range from around 25 t to just over 360 t. Depending on the truck size and manufacturer, there is a choice between mechanical-drive and electrical-drive systems. Mechanical-drive systems use a diesel engine in combination with a mechanical power train, whereas diesel–electric systems rely on a diesel-powered alternator to generate electricity for electric motors (mostly AC). Diesel–electric AC systems dominate the larger truck sizes (>150-t payload), while payload capacities for ADTs generally do not exceed 50 t.

Truck cycle times depend on the type of excavator, capacity of the truck, and haulage distance. Assuming good truck–excavator capacity matching and good digging conditions, trucks can be loaded in approximately 100 to 180 seconds, although this can be longer for front-end loaders. Spotting at the excavator typically takes between 40 and 60 seconds.

In the highly interactive system of today's open-pit mine, productivity of the haulage system is largely dependent on the performance of other activities in the mine, notably the haul road and dump maintenance and the excavator efficiency. For that reason, problems with haul truck productivity and reliability can often be traced back to poor performance of other parts of the operation. The operational interdependence of haul trucks and excavators comes to light during synchronization of their use and during loading. Production scheduling can reduce idle times for trucks and excavators. Payload monitoring and good communication between truck and excavator operators is important when trying to achieve optimal loading. Similarly, good operator training as well as a high degree of coordination, communication, and visual confirmation on the part of both the excavator and the truck operator is required for adequate truck spotting. Ideally, excavator operators should communicate the correct position for a truck to the truck operator, rather than relying on the judgment of the latter to get the truck in the right place. Good communication is especially crucial when double-spotting trucks. This method has the potential to reduce excavator and truck idle time but it is more complex than single-spotting or drive-by operations. Lastly, effective dispatching can maximize the use of trucks and excavators by providing a better synchronization between the two.

Other considerations in dump truck productivity are haul road performance, floor conditions in active loading and dumping areas, and retention of material in the truck's bed. Poor haul road performance (e.g., haul road defects and high rolling resistance) can reduce productivity and reliability of dump trucks. Likewise, poor floor conditions in active loading and dumping areas (e.g., benches and dumps) can also affect productivity and reliability of dump trucks and especially their tires. In both cases, management of the floor conditions in a

manner similar to that of haul roads can have a positive impact on haul truck performance. Additionally, retention of material in both the excavator bucket and the truck bed can pose a challenge to operations, especially in arctic conditions or where material has a high clay or moisture content. Adaptation of the truck bed design (e.g., rubber floor mats or circulation of exhaust fumes through the bed to prevent freezing) can alleviate this problem.

Tire Management

Tires are a basic constituent of a dump truck and in recent years tire shortage has been a major challenge for the mining industry. Prolonging tire life can result in big savings, considering that off-the-road tires for large dump trucks can cost more than $50,000 (2009 estimate), while excessive downtime of trucks can result in even higher costs.

More than 80% of tires fail before they wear out. Of all failures, approximately 45% are caused by cuts from spilled material and 30% by impacts with large rock fragments (Caterpillar 2007). Maintaining the correct inflation pressure, maintaining good floor conditions, and ensuring good truck handling and tire awareness by operators throughout the mine are some of the main aspects that can increase tire life. For truck operators, especially when it comes to removing spillages, good communication between truck operators and the road maintenance department is crucial. Analysis of scrap tires and monitoring of tire performance can provide valuable insight into causes of tire failure and possible prevention of premature tire failure. Furthermore, as a response to tire shortages, several major tire manufacturers now offer tire management systems. These systems use software, sensors integrated into tires, and dedicated, handheld tire-monitoring devices to measure and benchmark tire performance.

Trolley-Assist Haulage

Trolley-assist haul trucks are haul trucks that have been adapted from the standard diesel–electric system to a system that relies on pantographs to connect to an electrified overhead line for power supply. Historically, use of this system has been concentrated in southern Africa, but with rising diesel prices, interest in trolley-assist hauling from other parts of the world is increasing. Major infrastructures that need to be installed for the use of trolley-assist haulage include the overhead lines, truck conversion packages, and trolley substations. These conversion packages can be fitted on the majority of diesel–electric trucks.

Increased power supply from the overhead line, compared to a truck-based diesel generator, allows maximum use of the capacity of the electric motor in a truck. As a result, reduced cycle times are possible, and consequently, truck fleets can be reduced. This allows for productivity increases and possibly reduced capital investment costs related to purchasing fewer trucks. Furthermore, as the diesel engine is idling while on trolley assist, fuel consumption and ultimately time between engine overhauls can be reduced significantly. Lastly, energy can be recovered into the trolley supply grid when trucks are moving downgrade.

There is no generic way to determine whether an operation would benefit economically from a trolley-assist system. Savings related to trolley-assist usage are directly proportional to the number of kilometers traveled on the trolley-assist line by the entire truck fleet (Hutnyak Consulting, personal communication). Other important factors to consider in determining the possible savings related to trolley-assist haulage are the truck fleet size, designed life of a ramp, vertical lift of the haulage route, and traffic densities. As such, trolley systems are not suitable for every operation. First, the number of trolley-assist kilometers must be large enough to offset additional investments required for the trolley-assist infrastructure. This is most likely in large mines with a long life and an extensive diesel–electric haul truck fleet. Second, the difference between fuel costs and electricity costs must justify the transition to trolley-assist haulage. Lastly, trolley-assist haulage inherently reduces the flexibility of a haul road system by fixing haul routes. Therefore, trolley-assist hauling is economically most attractive to extensive mines with long, permanent, uphill haul roads and a large truck fleet in regions with high diesel costs relative to electricity costs.

Higher haul truck speeds, more traction, and highly fixed routes increase the chance of rutting and other haul road defects. The main adverse effects of this are reduced productivity and possibly pantograph damage when trucks are rejected from the trolley line. Therefore, haul road maintenance is even more important on trolley-assisted haul roads. The recent adoption of AC drives now allows trucks to connect to and run on trolley assist at variable speeds, negating one of the main disadvantages in the past when they relied on DC drives. Furthermore, sensors can now aid operators in staying underneath the overhead line.

In conclusion, it can be said that trolley-assist hauling can be economically viable, especially now that technological advances have solved some former DC-related disadvantages of trolley systems. However, because of mine-specific circumstances, economic viability of trolley systems should be evaluated on a mine-by-mine basis.

In-Pit Crushing and Conveying Systems

IPCC systems typically rely on gyratory, impact, cone, or jaw crushers to feed an overland conveyor belt that transports material to the mill or overburden embankment. They can be classified into mobile and semimobile systems. Mobile systems are crawler mounted and are often fed directly by an excavator. Having capacities of less than 1,500 t/h, these systems are usually found in small open-pit mines or quarries. Semimobile systems are mostly based on gyratory crushers fed either directly from trucks or from truck-fed apron feeders. They can only be moved with specialized equipment, hence the name semimobile. Having far higher capacities (up to 14,000 t/h) than fully mobile systems, these systems are suitable for mines with very large production tonnages.

The most common conveyor belt configuration is a standard trough-type conveyor. However, this type of conveyor belt suffers from limited curve radii (minimum of ~400 m for large overland conveyor systems) and slope angles (maximum of 16°–18°) it can scale. Pipe conveyors are a relatively recent development used to negotiate tighter curves. They are essentially rubber conveyor belts folded into a pipe shape with idler rollers. As idlers constrain the belt from all sides, far tighter curves can be negotiated. A further advantage is the reduction of spillages and fugitive dust generation. The disadvantage of a pipe conveyor is its limited capacity. To overcome the slope angle limitation, conveyors can be led up switchbacks or a dedicated trench can be excavated for the conveyor belt at the desired angle. Alternatively, one of several high-angle conveyor systems can be used such as the sandwich design or the pocket-lift design. The sandwich design, as the name

suggests, sandwiches materials between two conveyor belts kept in place by idlers. The pocket-lift design relies on material being carried in pockets created by wrinkling the belt. A system similar to the pipe conveyor can also be used for high-angle conveying, provided that material is sufficiently confined.

As mentioned earlier, the economic viability of IPCC depends on production tonnages, duration of the operation, haulage distance, and vertical lift. As a general rule, it can be said that if production exceeds approximately 100,000 t/d, when haulage distances surpass 5 km or when the vertical lift exceeds 250 m and if the installation can be in operation for at least 7–8 years, the benefits of IPCC can offset the higher capital costs of this installation. The economic benefits of IPCC rely on the potential to significantly reduce truck haulage distances and consequently, fuel consumption, haul road and truck maintenance costs, and labor requirements.

Furthermore, operations using IPCC are less prone to tire, equipment, or labor shortages. Other advantages over conventional truck haulage systems include lower carbon emissions and improved safety. The resulting reduction in overall mining costs has led to a revived interest in IPCC. IPCC, especially semimobile systems, reduces the flexibility of a mining operation with respect to pit expansion and pushbacks. Additionally, crusher moves and unplanned downtime of the conveyor belt can have serious impacts on the overall productivity of the system. Careful selection of crusher locations should minimize downtime due to crusher relocation.

Advantages of IPCC systems are best realized in large, high-volume open pits with a long mine life. However, regardless of mine life and size, materializing the potential cost savings still requires a detailed economic feasibility study incorporating site-specific operational, geological, and economical aspects.

Ancillary Equipment and Mine Services

Ancillary equipment and mine services, also referred to as auxiliary operations, are "all activities supporting but not directly contributing to the production of ore" (Hartman and Mutmansky 2002). Among the more prominent and important auxiliary operations in an open-pit mine are power/fuel supply and distribution, haul road construction and maintenance, in-pit water management, and the communications infrastructure. These activities do not generate revenue directly. Nonetheless, it is critical for the overall efficiency of a mining operation that these activities are given adequate attention. Bulldozers, front-end loaders, graders, water bowsers, and fuel/lube trucks are some of the most important pieces of ancillary equipment. Graders and water trucks are absolutely essential to haul road maintenance, which in turn is one of the most important elements of an efficiently operating surface mine. The main role of bulldozers and front-end loaders is maintaining active loading and dumping areas, preventing tire damage, and ensuring effective loading and dumping.

The increasing awareness of health and safety within the mining industry has raised attention for mine rescue service. Many large open-pit mines have one or more ambulances, a fire truck or firefighting equipment, and some have advanced rescue trucks with highly trained crews available 24 hours per day to respond to emergencies. Many operations also have an airstrip or heliport locations for emergency life-flight services if needed.

With many large excavators running on electricity, the power system is undoubtedly the most important mine service. Electricity can be supplied either by a utility company (often more economical) or by an on-site generator (in remote areas). Typically, an open-pit primary distribution system consists of a ring bus or main that partially or completely encloses the pit. The distribution voltage is normally 4.16 kV, but 7.2, 6.9, or 13.8 kV are sometimes used (Morley 1990). Radial ties complete the circuit from the bus to the switchhouses located in the pit. Where necessary, substations are employed where equipment voltages are lower than the main power loop. The power distribution evolves throughout a mine life; substations and other components of the power distribution system are mobile so they can follow mining equipment into a new working area. More details on mine power supplies are found elsewhere in this handbook.

In-pit water management, lighting, and communication infrastructure are three other important mine services. The benefits of efficient in-pit water management are discussed in the "In-Pit Water Management" section in this chapter. Flood lights are important as they enable around-the-clock production, which markedly increases productivity at a modern open-pit mine, especially at high latitudes.

The communication infrastructure in mining operations is rapidly evolving into an essential part of the operation as a response to the ongoing increase in complexity of open-pit mining operations. While initially used mainly for radio communications, the communication infrastructure now also carries information essential for equipment dispatching and monitoring (discussed in more detail in the next section). A well-implemented communications structure in a surface excavation can improve safety, efficiency, and productivity by enabling real-time dispatching and monitoring of mining equipment as well as voice and video communications. Additionally, mine services such as the pumping system can now be monitored and controlled remotely; slope-stability monitoring can be centralized to a large degree; and cameras can be located in critical areas such as the digging face, primary crusher, and other key areas. Several different types of networks are available for surface excavations and the choice of system is site-specific. Open standard wireless local area networks such as IEEE 802.11b/g/n are becoming commonplace as they are cheaper and more versatile than proprietary radio systems, satellite phones, or terrestrial phones. Depending on the application, wireless networks can comprise two or more discrete points (i.e., point-to-multipoint networks) or as a more flexible, all-encompassing mesh network (e.g., Bluetooth, Zigbee, IEEE 802.15.4, or similar standards).

Equipment Dispatching and Monitoring

Advances in GPS positioning, mechanical health monitoring, and production monitoring are contributing to an ever-increasing trend toward automation of open-pit mines. Truck dispatching is now commonplace in most operations, and the benefits and potentials of various other monitoring systems are increasingly being recognized.

All major OEMs include a basic machine management package with their equipment. Their most important function is to monitor the health of vital machine functions such as the power train, suspension, and brakes to detect abnormal conditions or impending failure. By enabling proactive rather

than responsive maintenance, operator safety can be improved while significantly reducing downtime and maintenance costs. The option exists to have OEM engineers analyze the data and make maintenance recommendations remotely. These packages can be expanded with additional features to increase productivity, such as payload monitoring and road analysis in trucks; bucket load monitoring and motion tracking for excavators; and vibration reduction feedback systems, GPS-based drill positioning systems, and drill progress monitoring for drill rigs.

Equipment dispatching systems are offered by most OEMs and several third parties. Originally, these GPS-based systems were geared toward dispatching of dump trucks to the correct areas. However, by applying the same principles to drill rigs, and to ore blending, auxiliary, and other mobile support equipment, modern systems are better described as equipment management systems. Integration of payload monitoring, cycle times, and other production-related variables have further revolutionized equipment management in mines. Tracking these parameters over time allows benchmarking and bottleneck identification in the production cycle. By providing benchmarks and highlighting previously unnoticed bottlenecks in the production cycle, modern equipment management systems can prove a valuable tool in the optimization of an open-pit operation.

Extensive research is being done into the use of autonomous surface mining systems, and the first pilot-scale tests on the use of haul trucks and drill rigs are now under way. However, the use of autonomous surface mining systems is still in its infancy, and the realization of the huge potential of these systems will be an ongoing process in the next few decades. An in-depth discussion of current developments in autonomous mining systems is found in Chapter 9.8 of this handbook.

Production Planning

Adequate production planning is absolutely essential in achieving the highest possible return from a mining operation (Caccetta and Hill 1999). It is a complex process for evaluating a host of variables including ore-body geometry, mill-feed requirements, and equipment fleet-related factors to achieve the required production. Accordingly, in recent years a large amount of research has been dedicated to improving mathematical and financial principles behind production planning. On the basis of these principles, pit optimization and planning software have been developed and included in major mining software packages to aid in production planning. In many cases, production planning is not simply a case of planning the mining sequence for an ore body. Rather, it should be a cost–benefit analysis not only on the extraction of ore but also on the required development and installation of infrastructure required for ore extraction and the interaction between the two in terms of economic performance, safety, availability of labor, and availability of equipment (Hall 2009).

Depending on the type of production planning, a host of factors such as staff and equipment fleet dispatching and availability, pit expansion, processing alternatives, and commodity prices come together in production planning to meet targets set by the management of a mining corporation. It may be contracted out to a consultancy or it can be done in-house by engineers at the operation or sometimes at a dedicated office.

Production planning can be divided into long-term and short-term planning. Long-term (or strategic) planning concentrates on defining a goal and as such is undertaken on a multiple-years to life-of-mine time scale (Kear 2006). Short-term (or tactical) planning, on the other hand, is more geared toward achieving a goal and is done on a time scale ranging from day-to-day planning to multiple-year production plans. Inherently, long-term planning is more concerned with providing frameworks for pit design and expansions (e.g., pushbacks), production schedules, rehabilitation plans, and equipment selection, whereas short-term planning focuses on equipment use, maximization of productivity, and ultimately meeting productions targets. In modern operations, equipment-dispatching systems play an important role in short-term production planning. Short-term production schedules should have a degree of flexibility in them to respond to unanticipated changes in the production environment. These changes range from simple matters such as a haul road defect to complicated issues such as a slope failure. Assessment and, where necessary, adjustment of short-term production schedules as a result of these changes should be implemented in a timely fashion.

OPEN-PIT MINING AND THE ENVIRONMENT

As mentioned in the introduction of this chapter, the influence of mining on the environment is becoming an increasingly important consideration before, during, and after the mining process. As such, discussions of the environmental impacts, as well as mine closure and rehabilitation of open-pit mines, are warranted. Furthermore, to connect several other deliberations within this chapter, a discussion of geotechnical considerations and slope monitoring is included together with a section on in-pit water management.

Geotechnical Considerations and Slope Monitoring

Throughout this chapter, there have been several references to the importance of adequate geotechnical design and slope-stability monitoring of an open-pit mine. Subsequently, a generic discussion of these topics is appropriate. Readers are referred to Chapters 8.3 and 8.5 of this handbook for a thorough discussion of the subjects of geotechnical instrumentation and slope stability.

Rock mass parameters are an important consideration from the feasibility study onward; they not only affect pit layout and geometry, but also blasting practices and, to a lesser extent, mining equipment selection and the layout of the comminution circuit (Wyllie and Mah 2004). Most importantly, rock mass parameters directly determine the steepest possible slope and face angle while maintaining an acceptable factor of safety, thereby having a major influence on the profitability of an operation. Furthermore, they can rule out the use of certain areas of an excavation for important infrastructure such as the main haul road.

In an active excavation, continual slope monitoring is crucial in predicting and preventing slope failures and, when failure is imminent, mitigating the effects of a slope failure. A comprehensive slope-stability monitoring program reduces the risk of major production delays or even sterilization of part of a reserve permanently as a consequence of a slope failure. Moreover, it ensures overall safety of personnel and equipment in an operation. A survey showed oversteepened slopes, failure to appreciate the effects of water, and rockfalls to be the main causes of injuries, fatalities, and damage to equipment (Sullivan 2006). Adequate design and monitoring can largely prevent these situations from occurring. Another situation where pit-slope monitoring is important is when there are

active underground workings in close proximity to an open-pit mine. Crown pillar failure or caving-related subsidence can permanently cease surface excavation activities.

Slope-stability monitoring techniques can be divided into surface and subsurface monitoring techniques (Wyllie and Mah 2004). Surface monitoring techniques include visual survey, direct measurement techniques, prism monitoring, laser systems, and radar systems. Visual survey includes visual inspection of slopes as well as mapping of structural discontinuities (and more recently, photogrammetry). Although the discontinuity mapping and photogrammetry can provide valuable insights into failure mechanisms that cannot be gained using other techniques, they do not provide quantitative data on slope stability and therefore should be supplemented by at least one of the methods discussed in the following paragraphs.

Direct measurement techniques include crack width meters, tilt meters, and other similar devices. These are low-tech methods to provide accurate indications of minor displacements in a possible failure zone. Prism monitoring relies on the use of total stations at permanent, stable reference positions to determine the distance to prisms mounted in areas of instability. From the change in spatial coordinates of targets over time, the displacement velocity and direction can be calculated. It is a very cost-effective method but it is vulnerable to atmospheric conditions such as excessive dust or mist. Laser systems rely on a laser scanner to produce a three-dimensional point cloud model of a slope. The higher density of points compared to prism monitoring makes laser scanning more comprehensive than conventional surveying techniques. A further advantage of laser scanning is that it can aid in photogrammetry and the mapping of discontinuities. Radar systems are similar to laser systems but provide higher accuracy. The drawbacks of radar systems are that they can only monitor one single area at a time compared to a broader picture as gained through laser scanning, and they are generally more costly. However, because of the unrivaled accuracy (submillimeter), they are often used to monitor the highest risk areas, such as working faces or areas of known instability.

Subsurface monitoring techniques include time–domain reflectometry, borehole probes, extensometers, and inclinometers. These techniques rely on measurement of changes of the inclination or other characteristics of a borehole that could indicate deterioration of stability. Additionally, seismic monitoring techniques are used. These rely on geophones registering acoustic emissions associated with failure events.

The most cost-effective approach to slope-stability monitoring is generally a combination of several of these techniques where they are used to complement one another. For instance, laser systems or prism monitoring can be used to determine overall stability of pit slopes and identify possible failure zones. If instability of a slope is detected, extensometers or radar systems can be used for more precise determination of movements in this area.

A last important consideration in slope stability is the presence of groundwater. Phreatic levels in and around a mine are crucial for maintaining pit-wall stability, especially in areas with clayey material or where the rock is heavily affected by structural discontinuities (Wyllie and Mah 2004). If the climate has periods of prolonged frost, the freezing/thawing cycles can further aggravate the negative effects of groundwater on slope stability. Piezometers are the main tool for determining groundwater levels. These, together with rain gauges, can act as an early warning system and serve as a basis for adjustment of the rate of water extraction from dewatering wells to prevent groundwater-induced failures. Apart from the geotechnical implications of groundwater in and around a pit, there are also major production-related considerations associated with in-pit water. These are discussed in the following section.

With mining being a business enterprise, geotechnical design, monitoring, and stabilization of an open-pit mine is ultimately a matter of economics: balancing the benefits and costs of stabilization against the costs and implications of a slope failure (Pine 1992; Wyllie and Mah 2004). This is in sharp contrast to civil engineering where the social and financial consequences of failure can be far more extensive (Pine 1992). A combination of this different approach to risk management together with the bigger scale of potential slope failures means that civil engineering solutions to slope stabilization are generally not feasible in surface excavations. Furthermore, surface excavations can possibly tolerate a degree of slope failure that would be unacceptable in civil engineering applications (Wyllie and Mah 2004). As a result of this, lowering of the water table and decreasing the slope angle are often the only practical options in surface mining, although in some cases applying civil engineering solutions has been economically viable (Wyllie and Mah 2004).

In-Pit Water Management

Responsible water management is essential in minimizing the environmental impact of most mining operations. Furthermore, there are considerable geotechnical, operational, and economical advantages to in-pit water management (i.e., more stable pit walls and a lower stripping ratio). A sound approach to in-pit water management requires the development of drainage strategies for both surface and groundwater and continuous monitoring of the performance of the water management plan (Department of Resources, Energy and Tourism, Australia 2008). For this, a thorough assessment of local geology, rock mass characteristics, hydrogeology, surface hydrology, and local climate are required. When deciding on dewatering methods, it is important to consider not only the above-mentioned factors but also the logistics related to open-pit mine dewatering. This includes the interaction between the chosen dewatering method, dewatering-related infrastructure (i.e., power supply and water transport from the well) and unit operations in a mine (Atkinson 2000).

To maximize the advantages of dewatering, sections of an open-pit mine must be dewatered before mining begins. Ideally, wells, drain holes, pump lines, and other dewatering infrastructure are situated such that they do not require rerouting as mining progresses. Additionally, the destination of water removed from the pit is an important consideration. It can be reinjected elsewhere, used in mineral-processing operations, or it can be treated and discharged into surface water courses.

There are several advantages to a correctly implemented dewatering program. First and foremost, dewatering pit slopes improves and maintains slope stability. This results in safer working conditions and allows for steeper slopes, lowering the stripping ratio. Second, a lower moisture content of blasted material increases diggability and reduces haulage costs as dry material has a lower mass than wet material, and there is less retention of material in the excavator bucket and the truck bed. Detrimental effects of wet haul roads include unsafe traffic conditions, more tire cuts, and increased rolling resistance. Lastly, water influx into blastholes is decreased,

reducing the need for blasthole dewatering or the use of more expensive water-resistant explosives.

Rain can constitute one of the main influxes of water into an open pit. As such, a thorough understanding of the typical local climate and its extremities (e.g., monsoons) is invaluable when formulating the water management plan for an operation. Rain gauges can be a further aid, monitoring rainfall and providing an indication of increased influx and possibly the demand to adjust the pump rate.

Stream diversion and dewatering wells are the main water management activities outside a pit perimeter. Inside the pit perimeter, dewatering wells, sumps, horizontal drain holes, and in some cases drainage adits or grouting are used for water management. Stream diversion, both of permanent and ephemeral streams, is done during the development stage of a mine before or concurrent with topsoil removal. Depending on the hydrogeology, dewatering wells can be situated within pit boundaries or outside of them (Atkinson 2000). Where the direction of flow is mainly lateral, dewatering wells outside the pit perimeter are generally more effective. A further advantage of dewatering wells outside the pit perimeter is that they can be installed before mining commences. However, they are less effective at preventing vertical inflow through the pit floor. To mitigate this, in-pit (vertical) dewatering wells are more effective (Atkinson 2000). This type of well creates more drawdown in the pit than dewatering wells outside the pit perimeter, but they cannot be installed prior to mining, and it generally also requires more complicated logistics. In-pit horizontal drain holes are used for locally depressurizing targeted areas (Atkinson 2000). They are an inexpensive method that can significantly increase slope stability because the small scale of these holes allows them to be installed quickly, targeting specific problem areas. However, they can only be installed after mining begins, they suffer from freezing effects in arctic areas, and water removal usually requires a sump downstream of the drain holes. Sumps are catchment basins at the base of a pit that serve the purpose of collecting in-pit water so it can be pumped back out. Grouting refers to the injection of chemicals that block pores in the rock to provide a barrier that prevents groundwater influx. This is a costly option that is only effective if there is a very well-defined geological feature producing the majority of water influx in a pit. Drainage adits around a pit shell serve the same purpose as drain holes. Because of the labor involved, it is a very costly and inflexible option that is becoming increasingly scarce.

Environmental Issues

In recent years, the pressure on mining companies to minimize the environmental impact of their operations has increased considerably. As a result, prevention and mitigation of detrimental effects to the environment are now high on the agenda of modern operations. Some discussion of environmental considerations related to mining in general is available elsewhere in this handbook, so this chapter focuses on environmental issues specific to open-pit mining.

One of the largest environmental impacts associated with open-pit mining stems from the physical change of the landform as a result of the vast quantities of material moved (Spitz and Trudinger 2008). The physical change of the landscape encompasses both the excavation itself and the disposal sites for overburden and tailings. The excavation will have a large impact on the visual amenity of the landscape, drainage patterns, and groundwater levels. Furthermore, the higher stripping ratios and generally lower ore grades mean that surface mines vastly exceed underground mines in the overburden volume that is generated, both from ore processing and overburden. As a result, a larger surface footprint is taken up by the overburden embankments and tailings impoundment generated by open-pit mining. Associated with the larger volume is not only a larger aesthetic impact on the landscape, but also a higher potential risk of spillage of toxins into the environment. Additionally, the changes in topography make the site more susceptible to erosion. Concurrent or postmining revegetation and placement of geotextiles on erosion-prone surfaces can provide sufficient protection against both wind and water erosion.

Chemical contaminants, increased turbidity, changes in flow patterns, and higher susceptibility to flash floods are the main effects of open-pit mining on the surface water regime (Spitz and Trudinger 2008). Chemical contamination and suspended solids can generally be removed in treatment plants. Channeling or rerouting streams away from vulnerable areas can be done to prevent erosion and mobilization of contaminants. Lowering of the groundwater level in an area due to mining may result in vegetation losses (and consequent changes in fauna), ground settlement, and lower flow rates from spring-fed surface water. Most detrimental effects of groundwater level control can only be effectively mitigated after mine closure.

Effects of blasting include excessive vibration and air overpressure, as well as dust, fumes, and possibly flyrock. The maximum allowed peak particle velocity (PPV) depends on the vicinity of populated areas and national or regional regulations. The U.S. Bureau of Mines found that cosmetic damage to houses can start at a PPV of 12 mm/s at a frequency of 10 Hz. The onset of damage is dependent on both vibration frequency and construction quality of a building, but in general it can be said that a higher frequency needs a higher PPV to be damaging (Siskind et al. 1989). Flyrock should be avoided altogether. Correct blast design should minimize environmental effects from blasting, and good communication with local residents can reduce perception of blasting by the public.

In populated areas, dust, noise, and road traffic effects are more pronounced than at remote mine sites. Noise causes disturbance of wildlife and annoyances both with residents and operators. The emphasis on noise management should be on reducing noise and limiting exposure time (Department of Consumer and Employment Protection of the Government of Western Australia 2005). In this context, hearing protection is regarded as an interim noise-protection measure unless other measures are demonstrably impractical. Cabs on modern equipment are designed to reduce noise exposure and as such play an important role in limiting exposure time. Lower perception of noise by the public can only be achieved by taking noise reduction into account during production planning, control of noise at the source, and noise barriers. Typical exposure limits imposed by legislative bodies range from 80 to 90 dB(A) for average exposure levels and peak exposure levels from 135 to 140 dB(A) (NIOSH 1998).

Depending on the particle size, shape, and chemical composition, dust can cause physical or chemical contamination of equipment and soils as well as respiratory and dermatological problems, reduced visibility, and coating of vegetation

(Department of Resources, Energy and Tourism, Australia 2008). The extent of the impacts of dust is highly dependent on climatic conditions and dust composition. Dust is generated by drilling and blasting, excavation, haulage, dumping, and processing of material, or it can emanate from poorly vegetated or bare areas like tailings impoundments in combination with wind. Dust generated from drilling can be suppressed by adding water to the bailing air from the drill hole, and by employing drill deck shrouds that envelop the drill stem. Haul roads are the most significant contributor of dust in surface mine operations, emitting between 78% and 97% of all dust.

As discussed earlier in the section on haul roads, dust suppression measures include spraying with water or chemical dust suppressants, and compacting or changing wearing course material. If loading, hauling, and dumping causes significant dust release, wetting the material before excavation can be considered, although this has detrimental effects on the efficiency of excavation and haulage. Water sprays around other sources such as stockpiles and the mill can be used to prevent fugitive dust from these areas. In Australia, maximum exposure levels for PM10 (dust with a median diameter <10 μm) lies at 50 $\mu g/m^3$ on a 24-hour average (Department of Resources, Energy and Tourism, Australia 2008). Toxicity of the particulate matter is dependent largely on silica content and exposure levels should be adjusted accordingly.

An increase in mining-related surface traffic on public roads may cause congestion, damaged or polluted roads, dust, noise, and unsafe situations. Where unacceptable situations occur, other forms of haulage, careful route planning, and mitigation of effects can relieve pressure on the road network, but often the problems cannot be solved completely.

Mine Rehabilitation and Closure

The main objective of mine closure is to ensure physical and chemical stability, as well as restoration of the ecosystem in all areas disturbed by the mining operation (Spitz and Trudinger 2008).

Before closure of a mine, ore robbing or scavenging can be done. This practice involves extraction of residual ore tied up in haul roads and other areas that did not allow extraction of the ore for operational reasons. One example of ore scavenging was done at Palabora, South Africa, to supplement underground ore production. It is important to realize that ore scavenging may reduce factors of safety for slopes to dangerous levels and should only be done after careful analysis of the probability, extent, and consequences of slope failure. Furthermore, it may inhibit access into the pit, should this prove necessary in the future.

Mine closure aspects specific to open-pit mining mostly arise from the large surface impact and large volumes moved (Spitz and Trudinger 2008). Careful planning during the development and operational stage and taking into account postmining use of an operation can significantly reduce postmining site disturbances and costs. Generally, it is attempted to put the postmining landscape to an equal or better use compared to the premining landscape. Stabilization and erosion protection of steep slopes and unstable faces are crucial for ensuring physical and chemical stability. This usually involves recontouring to stabilize pit faces and embankments to approximate original contours of the landscape, followed by establishment of vegetation or liner placement for erosion protection. Flooding of the excavation allows it to be more successful at blending in with the landscape and natural ecosystem, it prevents oxidation of sulfides and subsequent potential acid mine drainage, and it aids in the prevention of reestablishment of drainage patterns. Backfilling is another option for rehabilitation of an excavation but may be a very costly option that is generally only considered when there is a depleted open-pit mine nearby, when there is an active excavation or other supply of large volume of material in close proximity to the pit to be backfilled, or when backfilling is explicitly demanded by environmental regulations.

Maintaining chemical stability of overburden and tailings typically involves the prevention of acid mine drainage and subsequent metal leaching, as well as immobilization of any other toxic chemicals left over from the mining process (Spitz and Trudinger 2008). Flooding or sealing any potentially acid-generating material prevents oxidation of sulfides and the resulting acidification of water. If mine drainage is expected to contain any contaminants, a treatment plant should be erected with sufficient capacity to treat all effluent water and possible storm surges.

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CHAPTER 10.2

Open-Pit Planning and Design

David Whittle

INTRODUCTION

Open-pit planning and design is a decision-making process that leads to a realistic and actionable plan to profitably harvest mineral resources. Planning can be carried out for a wide range of time frames from the very short (e.g., the next shift of mining activities) to the very long (e.g., the profitable extraction over the complete life of the mine). This chapter is focused on the long term and on the creation of long-term value. It is about maximizing the value of the mine to the company.

Excellence in long-term mine planning involves making good decisions in pit design, mine sequencing, production rate, process method, ore selection, and mining method. All these decisions need to be made simultaneously and in conditions of great uncertainty, such as the following:

- Are the geotechnical conditions well understood? Will the pit walls survive future seismic and weather events?
- What is the resource under the ground really like? How much ore is available and how will it respond to the planned processing methods?
- What will commodity price and costs be next year? In 30 years?

Welcome to the very exciting (and the very uncertain) world of open-pit mine planning and design. To excel in this field, the planning team will need to know something about mathematics and engineering, business and economics, and risk and uncertainty. It is impossible to cover all of these subjects completely in one chapter. Therefore, this chapter has been written to equip the reader with a basic framework and a good set of analytical tools to allow one to have a fair degree of competence in open-pit mine planning. The chapter is organized as follows:

- **Planning process:** This section describes an overall planning process for open-pit mine planning and design.
- **Principles:** This section outlines guiding principles for the open-pit mine planning process. Adoption of these principles is a major assumption for the rest of the chapter. For example, net present value (NPV) maximization is established as the primary objective, and this is treated as an assumption in the rest of the chapter.
- **Open-pit planning techniques:** This section provides information on basic techniques used in open-pit mine planning. Techniques are presented in their approximate order of application.
- **Models and tools:** This section provides information about model structures, optimization, and workshop techniques.

PLANNING PROCESS

The process described is generic and can be adapted for many types of projects, including open-pit mine planning. The process is summarized in Table 10.2-1, and the details are provided in the following discussion. The process is described as a sequence of steps for clarity. In practice, however, it is not uncommon to cycle back to earlier steps. For example, when new information becomes available in Step 2 (preparation of inputs), it may give rise to a change in the purpose and scope of the study (Step 1).

The whole process may be applied once, or iteratively to cover the range from a broad study of resource potential, to a narrow optimization study intended to generate a detailed actionable plan. The difference between the broad and narrow application is reflected in the study's purpose and scope.

General Preparation

Any open-pit mine planning process requires leadership, management, and resources, with responsibilities shared by a number of people. Example responsibilities and role assignments are shown in Table 10.2-2.

Stakeholder analysis is conducted to determine the purpose and scope of the process, to generate inputs to the study, and to define the structure of the outputs. Engagement should be considered with all stakeholders, including internal stakeholders (e.g., senior management, planning team members, users of the plans, and resource providers) and external stakeholders (e.g., investors, customers, government agencies, and local communities).

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The Steering Committee defines the purpose and scope of the study, subject to the outcomes of the stakeholder analysis. Examples of purposes and scopes are provided for different study types in Tables 10.2-3 and 10.2-4.

Preparation of Inputs

The next five subsections provide guidance on the main categories of information that should be prepared for the open-pit planning process. Much of the information will be uncertain, and one of the principles, described later, is to treat uncertainty explicitly. One of the ways to deal with uncertain data is to employ the method described in the "Workshop for Building a Model of Highly Uncertain Data" section near the end of this chapter.

Resource Information

Information concerning exploration and mining rights in and near the area of interest must be prepared. If the scope of the mine planning project extends to possible acquisitions, then information should be gathered about the mineral rights held by others at nearby properties. The rights held by others for alternate resources on the property should also be considered. For example, it is not uncommon for coal rights and coal-bed methane rights to be issued separately. Although these rights apply to different resources, it is very difficult to harvest one resource without impacting operations designed to harvest the other. Separately held rights can lead to very significant operational constraints and should be considered early in the planning process.

Block models should be prepared for the areas of interest. The reader is referred to the "Resource Models" subsection later in this chapter for information about block models.

Current Operations

The following information about the current state of the operation needs to be obtained or prepared:

- A process flow sheet encompassing mining, operations, and shipping operations (if applicable)
- Bottlenecks in the business:
  - The current bottleneck (the component in the flow sheet that determines overall system output) with information about reliability and throughput variance
  - Growth bottlenecks (what is standing in the way of mine expansion?)
- Significant drivers of recovery, dilution, loss, time in process, and cost
- A cost model detailing fixed and variable costs for each part of the process flow sheet

Project Alternatives

Information about potential technology alternatives for the expansion of an existing mine or the development of a new mine must be prepared. It may helpful to conduct a brainstorming session to generate a broad range of alternatives for consideration. The reader is referred to the "Workshop for Framing" section near the end of this chapter for further information.

For each alternative, information must be prepared that includes performance expectations (expected throughput rates and variances, etc.) and cost data (capital, fixed, variable, and terminal costs).

Table 10.2-1 Steps in the process

Step	Action	Details
1	General preparation	Assign roles (Table 10.2-2). Perform stakeholder analysis. Determine purpose and scope.
2	Preparation of inputs for the study	Prepare the following: • Information about exploration and mining rights in the area of interest • Resource information, including reports, maps, drill-hole data, and block models for the area of interest • For existing mines, information about the mining operations and equipment, processing facilities, and infrastructure (Include data for operational performance, variability, capacity, and costs.) • Information about alternate mining and processing methods that may be considered in the study • Information about applicable mining and environmental laws and details of exploration and mining rights • Forecasts of future commodity demand and price • Forecasts of future costs
3	Framing	Determine the set of decisions and alternatives to be tested as part of the study.
4	Assessment	Conduct the modeling, optimization, and economic evaluation of alternatives. Conduct risk assessment on main candidates for selection.
5	Selection	Select the best alternative plan(s). Select the actions to preserve future options.

Table 10.2-2 Planning project roles

Typical Role	Responsibilities	Typical Composition
Steering Committee	• Conducts the stakeholder analysis • Sets purpose and scope for the study • Provides resources (including personnel) • Monitors progress	Heads of departments who provide input or people to the planning process
Planning project manager	• Manages the process. • Ensures study objectives are met within budget and on time	A person with project management and mine planning skills
Planning team	• Conducts the work	Full- and part-time members with appropriate skills for the work to be completed

Whenever the term *expected* is used in this chapter, it should be understood to mean "qualitatively," that is, "the most likely." If a mathematical definition is preferred, then it should be defined as "the probability-weighted mean of all possible values."

For legal, environment, and community issues, information should be prepared about mining and environmental laws, applicable to the area of interest, that will potentially limit the approach to mining or processing, or affect costs.

In relation to local communities, information should be prepared to address the following:

- The potential for the community to provide people, services, or infrastructure
- The expectations of the community to share in the benefits of the mine proceeding

Table 10.2-3 Example purposes and scopes for different types of studies for an existing mine

Study Type and Frequency	Purpose	Scope
Resource potential study Frequency: Every 3 to 4 years or whenever key information (price forecasts, resource estimates) changes significantly	Explore a wide range of expansion and development options in pursuit of maximum value	• No constraints on capital or on major changes to the operation • Short- and medium-term (<3 years) optimization decisions are not considered • Long-term price forecasts are considered
Life-of-mine plan Frequency: Updated annually	Determine the actionable life-of-mine plan that will maximize value, provide the basis for reserve reporting, and set the scene for medium-term production planning	• Major operational changes are subject to permitting and capital approvals • Focus on value-maximizing optimization decisions • Long-term and short-term price forecasts are considered

Table 10.2-4 Example purposes and scopes for different types of studies for a potential mine or a mine expansion

Type	Purpose	Scope
Concept study	Seek to identify a concept for development or expansion that has a reasonable prospect of creating value	• No constraints on capital or on major changes to the operation • Short- and medium-term (less than 3 years) optimization decisions are not considered • Long-term price forecasts are considered
Prefeasibility study	Examine a wide range of strategic alternatives for development or expansion, and select the single plan that would create the greatest value at an acceptable risk	• Major operational changes are subject to permitting and capital approvals • Focus on value-maximizing optimization decisions • Long- and short-term price forecasts are considered
Feasibility study	Optimize the single plan (output from prefeasibility study) to create an actionable plan that meets company requirements for investment approval	

- The community's concerns regarding safety, noise, dust, and impacts on the environment
- Any other factors that may lead to the community being negatively or positively impacted by the mine

In evaluating matters relating to legal, environment, and community, it may be relatively easy to determine the minimum requirement for compliance, and a desire to minimize costs may be the motivation to pursue that outcome. However, planning only for minimum compliance may not be the most effective way to progress a project. The government and community are significant stakeholders, and their support for the project will be strongly influenced by the mine's ability to meet their needs and expectations. Accordingly, it is important to evaluate other alternatives that seek to actively garner the support of government and communities. Not only the costs associated with different alternatives should be considered but also their potential to avoid or cause delays and impacts on social license.

Prices

Commodity price is the single most important variable impacting critical mine-planning decisions. It has a direct bearing on the optimal rate of production, the final pit size, and the desired degree of operational flexibility that is built into the operation. If it were possible to know future prices with certainty and precision, that knowledge could be applied as follows:

- Near-term prices up to about 10 years into the future have the greatest impact on the determination of the optimal production rate for the mine.
- The price at the end of the mine's life has the greatest impact on the size of the final pit.
- The variable nature of the price (measured over 3 to 36 months) has the greatest impact on the degree of flexibility designed into the operation.

Commodity price is, however, the most difficult variable to predict. Many different forecasts can rationally be made for future prices, and they all have one thing in common—they are all almost certainly wrong. Because of this, a mine should not rely on only one price forecast. It should use at least the following three:

1. Mid-price forecast: The expected price
2. High-price forecast: A somewhat optimistic forecast
3. Low-price forecast: A somewhat pessimistic forecast

Price forecasts are generally sourced from either the mining company's economist or an external consultant or research company. Whatever the source, it is important to recognize that forecasts do not normally model all aspects of future price behavior. This fact becomes apparent when a comparison is made between a chart showing historic prices and a corresponding price forecast chart (e.g., see Figure 10.2-1). They are characteristically different. A historic price chart will show the price as it happened—subject to the effects of market discontinuities, changes in economic conditions, investment cycles, balances and imbalances between supply and demand, and the influence of market speculators. None of these price drivers can be predicted with certainty, and some cannot be predicted at all. Forecasts can only be made based on scenarios. The following are some example coal price scenarios:

- Declining coal resources will constrain supply, leading to a price increase.
- Ongoing mining technology improvements will reduce coal industry costs, thereby reducing the coal price.
- Coal supply currently exceeds demand, and when marginal producers close down, prices will rise.
- Emerging clean energy technology will reduce the demand for coal, leading to price decline.
- Economic growth in developing nations will increase the demand for coal, thereby increasing price.

Consequently, most price forecasts are, in effect, straight-line assumptions modified by a scenario. Examples of simple forecasts are provided in Figure 10.2-1.

In addition to the simple models for price forecasts, planners are increasingly using stochastic models for price in an effort to better plan for the random nature of prices. Methods include *random walks* and *mean reverting random walks*. The

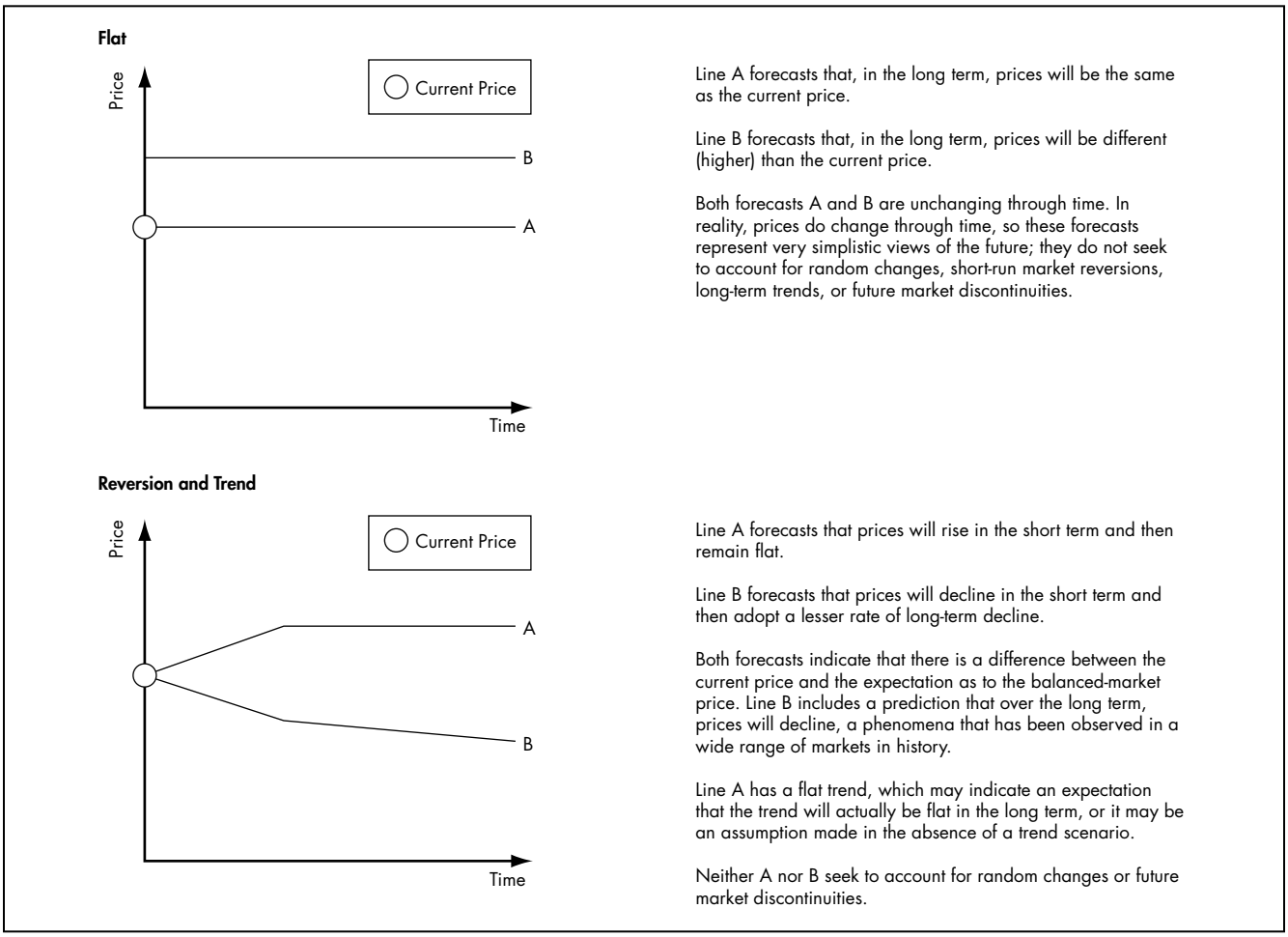


Figure 10.2-1 Simple price forecasts

use of these approaches can lead to better decisions as to the degree of flexibility in an operation; however, the detailed discussion of these methods is beyond the scope of this chapter.

When obtaining price forecasts, one should inquire as to the following:

- What aspects of future price behavior (long-run stable price, long-run trend, near-term reversion, short-run random) are modeled in the forecast?
- Is the forecast in real or nominal currency?
- Are there intentional biases (i.e., intentionally conservative or optimistic) in the forecast?

Most planning should proceed with the mid-price forecast, but the high- and low-price forecasts can be used to test sensitivities and to develop options.

Most planners are familiar with sensitivity analysis. The idea is to determine how much a project value changes with a change in a major input variable. Sensitivity analysis often proceeds by varying the input variable by an arbitrary amount (e.g., ±10%). However, the results of the sensitivity analysis would be a lot more useful if the cases tested were more representative of the uncertainty in the input variable. The high- and low-price forecasts should serve this purpose. Apart from their use in sensitivity analysis, high-price and low-price forecasts can also be used to develop options, discussed later in this chapter.

Costs

All costs need to be rendered in a structure suitable for cash flow modeling in one or more optimization systems. Planners should not rely on only one set of cost forecasts but instead should use at least the following three:

1. Mid-cost forecast: The expected costs
2. Low-cost forecast: A somewhat optimistic forecast
3. High-cost forecast: A somewhat pessimistic forecast

Some costs are market based, and the concepts and models described for forecasting prices may equally be applied to the forecasting of market-based costs. Examples of market-based costs include oil, shipping, equipment, and construction services.

Assumption Sets

To facilitate the assessment of alternatives, it is very useful to define and document a number of assumption sets. Each assumption set consists of price, cost, and geological and operational settings. The first assumption set should be the reference assumptions set and should include price forecasts at expected values (the mid-price forecast) and cost, recovery, throughput rates, geological and geotechnical models (including pit slopes) at expected values.

Secondary assumption sets may be built to represent different scenarios for future conditions or resource quality/volumes. The following are examples:

- **High-price assumption set:** This is the same as the reference assumptions set except that the high-price forecast is used (along with any associated impacts on costs).
- **Low-price assumption set:** This is the same as the reference assumptions set except that the low-price forecast is used (along with any associated impacts on costs).
- **High-resource assumption set:** This is the same as the reference assumptions set except that the high-price case from the resource range is used.
- **Low-resource assumption set:** This is the same as the reference assumptions set except that the low-price case from the resource range is used.

Each of the above-mentioned assumption sets should have a reasonable (>10%) probability of being the most representative of current or future facts. As analysis continues, more assumption sets may be created. For example, in testing a broad range of production rates, the impact of rate on mining width, dilution, and other factors should be carefully considered, and appropriate assumptions should be determined and documented.

In some cases, market-based cost movements are correlated to commodity price movements. If that is the case, then the correlation should be reflected in the high- and low-price assumption sets by including corresponding changes to the correlated costs.

Framing

The framing process leads to a set of high-order decisions and alternatives to be tested as part of the study. The result of the framing exercise is a strategy table, an example of which is shown in Table 10.2-5.

Table 10.2-5 Example of a strategy table showing key decisions (column headings) and alternatives*

Mining	Processing	Water	Power	Transportation
Current plan (sublevel stope mining)	4.5 Mt (current)	3,000 ML artesian (current)	Gas-fired (current)	0.75 Mt truck and rail (current)
Open pit in northern zone; continue sublevel stope mining in southern zone	6.0 Mt (brownfield)	6,400 ML artesian	Expand gas-fired to pipe capacity	**Concentrate slurry pipeline**
Open pit in northern zone with new decline from base of pit	10 Mt (brownfield)	**3,000 ML artesian plus desalination to meet demand**	**Grid connection**	1.5 Mt truck and rail
Open-pit everything	**11.5 Mt (7-Mt greenfield)**			

*A hypothesis is represented by the choice of one alternative from each column. A "big capital" hypothesis is indicated in bold type.

Assessment

The framing exercise gives rise to a set of hypotheses (possible best planning decisions) to be tested. Each hypothesis should be subject to simulation and optimization to determine its economic value. The initial assessment should be conducted using the reference assumptions set. This assessment will reveal which set of decisions yields the highest value. The structure of the conclusion is as follows: *If* the reference assumptions set is true, *then* the [defined set of] decisions are optimal.

Example 1. Using the reference assumptions, a pit of 60 Mt ore feeding a 5-Mt processing plant is optimal.

Assessment should continue with the application of other assumption sets to determine the optimal set of decisions for each. The generalized structure of the conclusions is as follows: *If* the [named] assumptions set is true, *then* the [defined set of] decisions are optimal.

Example 2. If the high-price assumptions are true, then a pit of 75 Mt ore feeding an 8-Mt processing plant is optimal.

The final stage of assessment involves identifying a list of candidate decisions to carry forward and subjecting these decisions to risk assessment.

Selection

The plan is selected based on the assessment results. The following is a recommended structure.

- **Best-case plan:** A theoretical best-case plan with few constraints and no secondary objectives applied. This is generated using the reference assumptions set. For further information, the reader is referred to the "Establishing a Theoretical Best-Case Plan" section later in this chapter.
- **Optimized plan:** The plan based on decisions that are optimal for the reference assumption set, with all constraints and secondary objectives applied. The NPV change associated with the application of constraints and secondary objectives should be documented.
- **Risk management actions:** These are modifications to the optimized plan designed to manage risks identified in the risk management process. The cost to NPV resulting from the application of risk management actions should be documented. For further information, the reader is referred to the "Identifying and Managing Risk" section later in this chapter.
- **Option preservation actions:** These are modifications to the plan designed to respond to alternate assumption sets. A simple model for the preservation of options can be applied.
- **Final plan:** This is the optimized plan as modified by the risk management and option preservation actions.

PRINCIPLES

The following principles are posited as a foundation for open-pit mine planning:

- Employ evidence-based decision-making methods.
- The primary objective is the maximization of NPV.
- Treat secondary objectives as constraints.
- Do not apply public reporting rules to the input of the planning process.
- Deal with uncertainty and risk explicitly rather than by practicing conservatism.

Details of each principle are discussed in the following four sections.

Evidence-Based Decision Making

Evidence-based decision-making methods should be employed as follows:

- Seek to avoid bias in data gathering and in decision making.
- Assess information for its probability of being correct or predictive.
- Account for gaps in information and understand how those gaps contribute to uncertainty.
- Avoid using rules of thumb. Rules of thumb are simple approximations or rough methods for estimating or measuring something. They are usually handed around from person to person without sufficient explanation of the limitations and hidden assumptions underpinning them. Such approximations are only useful if all their limitations and hidden assumptions are understood.
- Apply defined decision-making processes and analysis methods. For example, applying the processes and methods described in this chapter will help to avoid making poor decisions.
- Learn to recognize and avoid logical fallacies. An example of a logical fallacy is: "The mine must be either open pit or underground." This example is called a false dichotomy because in reality there are other possible choices (e.g., a combination of open-pit and underground mining and in-situ leaching). Interested readers can find information about logical fallacies on the Internet.

Primary and Secondary Objectives

The primary objective is the maximization of NPV. This is the one goal that must be achieved, subject only to the constraints of the secondary objectives. The pursuit of the primary objective is one of optimization, and only the maximum possible value will be satisfactory.

The primary objective is a matter of choice for a company engaged in the planning activity, and a company is free to choose any primary objective it likes. However, in mining, NPV maximization is a very common primary objective, and with a desire to elevate its readability, this chapter has been framed with the assumption that NPV maximization is the primary objective.

Secondary objectives should be treated as constraints. In the mathematical framework for optimization, it is only possible to have one objective (to be minimized or maximized). However, in business, it is not uncommon to nominate multiple objectives. To profitably employ the mathematical frameworks for optimization in open-pit mine planning, one of the objectives must be nominated as the primary (e.g., maximize NPV) and all other objectives treated as constraints. Constraints are specified by minima or maxima (or both) and act to limit or modify the pursuit of the primary objective. Examples of secondary objectives are shown in Table 10.2-6.

Secondary objectives applied as constraints cannot enhance the pursuit of the primary objective; they can only inhibit it. A sacrifice is made on the primary objective (i.e., NPV is reduced) when a secondary objective is applied. Secondary objectives should only be applied sparingly, and the resultant sacrifice of NPV should always be reported.

Separation of Public Reporting from Planning

Public reporting rules should not be applied to the inputs of the planning process. This principle is particularly important when considering publicly reported mineral resources and ore reserves. Rules for public reporting are framed to provide a basis for consistency in reporting across all reporting entities and act as a filter on available information. For example, the Joint Ore Reserves Committee (JORC) Code requires the competent person to make a preliminary judgment as to the technical and economic factors likely to influence the prospect of economic extraction and to apply those factors to ensure that the reported mineral resources have a "reasonable prospect of economic extraction" (JORC 2004). (The JORC Code applies in Australia, but provisions are similar in many other jurisdictions.)

The competent person's preliminary judgment as to economic factors acts as a filter to the full range of data regarding in-situ mineralization. It is much better for mine planners to be exposed to the full range of data regarding in-situ mineralization, including inferred resources and potential mineralization beyond inferred resources. The planner can then apply more comprehensive economic criteria and analysis to determine the subset of in-situ mineralization that is economically extractable.

When the planning process is complete, the rules and conventions of public reporting should be applied to generate accurate public reports.

Explicit Treatment of Uncertainty and Risk

Uncertainty and risk should be dealt with explicitly rather than through the practice of conservatism. It is common for mine planners to practice conservatism as a means of dealing with uncertainty or reducing risk. Two examples are (1) deliberately overestimating costs to compensate for the potential for costs to be accidentally underestimated, and (2) deliberately underestimating the grades to compensate the potential for accidentally overestimating them.

- Acknowledge that there is uncertainty associated with most inputs to the mine planning process.
- Understand the inadequacy of single estimates of uncertain inputs. Even when a single estimate is an expected value, its exclusive use can lead to poor planning decisions because it does not represent the probabilistic range. An uncertain input is best represented as a continuous or discrete probability distribution, and the full range of potential consequences of that distribution must be understood.
- Risk management should be applied when the full range of consequences of input uncertainty is understood.

OPEN-PIT PLANNING TECHNIQUES

Basic Technique for Generating Life-of-Mine Plans

Table 10.2-7 shows a recommended series of steps to achieve the technical design of an open-pit mine and its long-term schedule. The steps are iterative, and at various points it is necessary to cycle through the steps in the pursuit of a good overall plan.

Pit Parameterization Methods

Applying pit optimization to a single resource model and a set of assumptions will produce one pit outline—the outline that maximizes the net cash flows (undiscounted) for a mine and often called the "cash pit." However, it is useful to have a number of pit outlines available for analysis—to be used as candidates for final pit designs and pushback designs. A technique pioneered by Matheron (1975), referred to as pit

Table 10.2-6 Examples of secondary objectives

Secondary Objective (Constraint)	Rationale
Maintain a favorable position on the industry cost curve. *Example:* Do not exceed a cash cost of $0.80/kg.	Reducing costs is always a sub-objective to maximize profit. However, for strategic reasons, low costs may override the objective to maximize profit. This is the case if a company has a relative cost objective—for example, to only produce copper if it can be produced at a marginal cost in the bottom half or quartile of the marginal costs for the industry.
Declare a large reserve. *Example*: Plan a mine that will allow the reporting of a minimum reserve of 5 Mt.	A major nonfinancial measure used in the valuation of shares is the reserve. Appleyard (1997) writes: "The reserve is probably the major input in a company's ability to raise debt or equity finance and, if listed, in its rating on the share market." In seeking to maximize their share price, some companies may be prepared to sacrifice NPV to quote a higher reserve.
Conserve capital. *Example:* Do not exceed $100 million in capital spending.	It is sometimes not possible to access the capital required to achieve the optimal rate of production.
Stretch the mine life so that it spans at least two price cycles. *Example:* Reduce the production rate to ensure a mine life of at least 10 years.	Many commodity prices are thought to have cyclical behavior. For example, every 3 to 7 years, the price will move from high to low and return to high again. The risk of having too much of the mine life occurring in low-price times is reduced if the mine life is extended.
Avoid risk. *Example:* Invest in the $50 million processing plant that can be funded from free cash flow, even though the $100-million processing plant promises a higher NPV.	Walls and Eggert (1996) performed some research on the decision-making behavior of mining CEOs and found that they do not generally make highest NPV decisions—their decisions being tempered by risk-averse behavior. Walls found that CEOs of smaller mining companies tend to be more risk-averse and suggested that this is because for small companies, the higher risk projects carried with them a greater risk of company failure. Smith (1997) suggested that, rather than choose the maximum NPV production rate, this rate should be seen as the maximum rate and that the actual rate should be less. Part of the reasoning behind the attenuation is that a maximum NPV production rate tends to lead to an unreasonably short mine life, leaving minimum time to recover from early difficulties.

parameterization, harnesses the pit-optimization process to deliver a set of nesting pit shells that serve the purpose well. Several pit parameterization techniques are now in use, each with varying advantages and disadvantages. The following techniques are described in this section: revenue factor, cost factor, vertical bounding, and mining direction.

Revenue Factor

In the revenue-factor technique, a range of factors is applied to the block revenue to alter its value in consecutive optimization runs. The application of a lower revenue factor leads to ore becoming less valuable or being reclassified as waste, and this in turn leads to a smaller pit. A typical range of revenue factors is 0.10 through 1.00 in steps of 0.01. By running pit optimization repeatedly with incrementally different revenue factors, a range of pits from large to small is obtained. The pits so produced are often referred to as shells and are numbered 1-*n* from smallest to largest. The shells do not overlap and are good candidates from which to choose pushback designs.

Cost Factor

Similar effects can be obtained by applying factors to either mining or processing costs or by direct manipulation of cutoffs. A typical range of cost factors is 1.00 through 20.00 in 100 equal steps, with higher cost factors yielding smaller pits.

The main advantage of this class of pit parameterization is that it usually gives rise to a large range of shells that, if used as the basis of pushback designs, can lead to high NPV. The main disadvantage is that the shells for vertically oriented ore bodies are often impractical, with shells forming as concentric rings, sharing no common wall.

Vertical Bounding

The vertical-bounding technique starts with all benches in the model being sterilized, with the exception of the highest bench.

Table 10.2-7 Typical steps in creating an open-pit plan

Step	Details	Reference Sections in This Chapter
1	Choose a hypothesis from the strategy table (Table 10.2-5) to test. Prepare an assumption set and a resource model for analysis.	Workshop for Framing Assumption Sets Resource Information
2	Apply a pit optimization tool to generate a set of pit shells.	Pit Parameterization Methods Mine Planning Optimization
3	Determine a theoretical best-case plan and choose the final pit.	Establishing a Theoretical Best-Case Plan
4	Choose the production rate.	Determining the Optimal Production Rate
5	Choose a plan for a final pit.	A Plan for a Final Pit Allowing for Topographical Features Allowing for Blending
6	Choose the number of pushbacks and their basic design.	Pushbacks Allowing for Mining Width
7	Optimize the schedule and processing cutoffs.	Determining Processing Cutoffs Mine Planning Optimization

The pit optimizer is run only on the top bench, and this may or may not lead to the creation of a pit (depending on whether there is any ore in the top bench). The process proceeds by progressively unsterilizing benches in the model and rerunning the pit optimizer. The result is a set of nested pits, and the advantage of this approach is that the shells that it produces can often have a better mining width at the base of pushbacks compared to the revenue-factor or cost-factor approaches. The main disadvantage is that this approach may provide too little or too much working space, depending on the nature of the ore body, and there is not much that can be done to make the optimizer produce a better outcome.

Mining Direction

The mining-direction technique is similar to the vertical-bounding technique, but the progressive unsterilization is done in a direction chosen by the planner. In many circumstances, the nested pit shells produced by the technique are more practical than those produced by other techniques, as there is a propensity for shells to have a common wall. The technique is implemented though user-defined formulas in the pit optimization software, acting to progressively unsterilize blocks according to their distance from a defined point or plane in the model. The process is normally run several times using different origin points or planes. The results can then be compared and evaluated based on their NPV and practicality.

The main advantage of this technique is that it can quickly generate a range of alternative sets of pushbacks that are more practical to mine.

Establishing a Theoretical Best-Case Plan

When applying optimization tools to open-pit mine planning, it is useful to start by generating plans that have as few constraints as possible. This theoretical best-case plan can be prepared as follows:

1. Generate multiple pit shells using the revenue-factor method and use all of them as pushbacks. This may lead to an unusually high number of pushbacks, but that is of no concern when creating this theoretical best-case plan.
2. Apply a simple schedule that limits prestripping (e.g., one that mines out each pushback in turn) or apply schedule optimization.
3. Only apply one production constraint at a time (i.e., only one of mining, process, or selling limits) and experiment with that limit over a wide range until NPV is maximized.
4. Do not apply sinking rate limits or mining-width constraints.
5. Ignore all secondary objectives.

The plan will almost certainly be impractical, but it provides a very useful internal benchmark for the project. The benchmark can be used to evaluate the NPV impact of imposing secondary objectives and of the constraints applied to make the plan more practical. When a practical plan can be created with an NPV close to the theoretical best-case plan, then it is known to be a good plan (practical and with a high value), and there is little to be gained by further refining the pit designs and schedules.

Determining the Optimal Production Rate

The optimal production rate is the rate at which NPV for the project is maximized. The simplest method to determine the optimal production rate is to calculate the NPV for a range of different alternatives. The testing is done by scheduling the mine for a range of different assumption sets, with each assumption set representing a different production rate alternative with its associated capital; variable costs; fixed costs; and selectivity, pit geometry, and infrastructure requirements. As the rate increases, there are several impacts on operations and efficiencies in the pit, and these should be reflected appropriately in the assumption sets. The following are examples:

- Congestion will increase, leading to longer cycle times unless additional mining width is provided.
- Larger equipment may be required, leading to an increase in the size of the selective mining unit and an increase in dilution and/or mining loss.
- Demands on ramps will increase, leading to a need for wider or additional ramps.

The expenditure of capital unlocks productive capacity. If the level of capital spending is increased, it causes some upward and some downward impacts on value, as follows:

- The capital spending reduces the NPV directly.
- The capital spending increases production throughput, which in turn increases net positive cash flows in each period and increases the NPV.
- The mine life is reduced, thereby reducing the number of periods in which positive NPV can occur and bringing forward terminal costs.
- The fixed costs per period associated with maintaining the level of production may go up.
- The variable costs associated with processing each unit of production may go down.

The results, once graphed, should be a convex curve, such as the one shown in Figure 10.2-2.

The overall production rates for metals mines are ordinarily set by the comminution (crushing and grinding) function, whereas for other types of mines, the pace may be set by a different part of the value chain. For example, in an iron ore operation, the pace is likely to be set by the capacity of the rail network or the port.

Why are they different? In each case, the pace is set at the economic design bottleneck, that is, that part of the process chain that *should* be designed as the bottleneck of the operation. The choice is based on examination of the capital costs and fixed costs of different parts of the process chain. Considering only capital costs for a moment, the economic design bottleneck is that part of the process chain with the highest ratio of capital to product-unit throughput. For example, if it costs $50 million to establish mining equipment and $100 million to establish process equipment to achieve a given level of product output, then processing, rather than mining, is the design bottleneck. In a metals mine, the capital intensity of the comminution circuit normally outweighs the capital intensity of the mining operation or finished goods shipping. Therefore, it is common for the pace in metals mines to be set in comminution. In contrast, the cost to install matching mining or finished goods shipping is relatively low, so it is uncommon for these functions to constitute the economic design bottleneck for a metal mine. In an iron ore operation, the design bottleneck is likely to be in the rail or the port. This is because the capital intensity for these parts of the value chain is relatively high. Fixed costs associated with maintaining a certain level of production also contribute to the determination of the design bottleneck, but it is beyond the scope of this chapter to discuss this in detail.

When the economic design bottleneck has been determined, it should be used as the primary lever for testing increased or decreased production rates for the whole mine. The design bottleneck constraint should only be applied in early stages of the analysis; a wide range of production rates must be tested. Secondary constraints should only be introduced in later stages of the analysis, and their impact on value and overall throughput should be carefully examined.

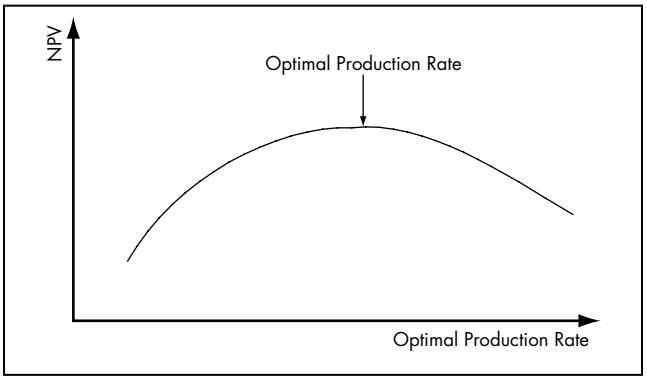


Figure 10.2-2 Relationship between production rate and NPV

Throughout the planning process, focus should be maintained on the economic design bottleneck. If a schedule emerges for consideration in which this constraint is underutilized in some periods, then this is a strong indication that the schedule or plans are not as good as they should be.

Determining Processing Cutoffs

In practice, mine planners rely on software to calculate or optimize the setting of processing cutoffs. However, it is important to understand the basic concepts and not merely place blind faith in the software. To that end, a simple explanation of some of the concepts is provided, including the calculation of marginal cutoffs for simple situations and descriptions of some of the issues encountered when seeking to effect ore/waste discrimination in more complex situations.

As part of the open-pit planning process it is necessary to determine the treatment of run-of-mine material. The alternatives typically include the following:

- Treat as ore and allocate to one of the available processing pathways.
- Treat as ore but defer processing by allocating the material to a grade stockpile (the material would later be reclaimed for processing).
- Treat as waste and allocate to a waste dump.

The traditional method is to differentiate material by its product grade and the application of cutoffs. In a simple case, material with a grade above a defined product grade cutoff will be processed and material below the cutoff grade will be treated as waste.

The cutoff grade is calculated such that material will be processed only if processing will lead to a net increase in value. If the objective is to maximize net cash flows, the cutoff is calculated as the grade at which the net cash flow from processing is the same as the net cash flow from treating the material as waste. The mathematics for this is shown in Equations 10.2-1 through 10.2-6. The conventional term for this type of cutoff is *marginal cutoff*.

The simplest way to think about this is to consider material as it emerges from the pit, at the pit rim, with all mining costs paid. The marginal cutoff is set at the grade at which the cost of processing exactly equals the revenue generated by the recovered product.

The cash flows associated with treating the material as waste, C_{mw}, can be calculated with the following equation:

$$C_{mw} = -M \times C_m \qquad (10.2\text{-}1)$$

where

M = mass of the material
C_m = cost to mine a ton of material

The cash flows associated with treating the material as ore, C_{mo}, can be calculated with the following equation:

$$C_{mo} = P \times R \times M \times G - M \times C_p - M \times C_m \qquad (10.2\text{-}2)$$

where

P = price obtained for a unit of product
R = metallurgical recovery in the process
G = grade of the material
C_p = cost to sell a ton of material

The cutoff grade is calculated as the grade (G) at which the net cash flow from processing (C_{mo}) is the same as the net cash flow from treating the material as waste (C_{mw}):

$$C_{mo} = C_{mw} \qquad (10.2\text{-}3)$$

or

$$P \times R \times M \times G - M \times C_p - M \times C_m = -M \times C_m \qquad (10.2\text{-}4)$$

$$P \times R \times M \times G = Cp \times M \qquad (10.2\text{-}5)$$

$$G = Cp/(P \times R) \qquad (10.2\text{-}6)$$

The following are some important observations as to the marginal cutoff shown in Equation 10.2-6:

- The cutoff is independent of the mining cost.
- The cutoff is independent of the grade distribution in the ore body.

The derivation of Equation 10.2-6 is easy to understand, and this method can be applied in very simple cases if the simple objective (maximization of net cash flows) is acceptable. However, the mathematics becomes more complicated when multiple products or multiple processes are available. This is because of the following:

- Multiple-product mines require the application of multiple-product grade cutoffs—one cutoff per product. Figure 10.2-3 shows graphically how two product grade cutoffs are applied.
- Multiple-process mines require the application of cutoffs for each process—one cutoff per product per period per process. Figure 10.2-4 shows how multiple process cutoffs are applied.

When the objective is changed from maximization of net cash flows to the maximization of NPV, the calculations become intractable without the use of optimization software. This is because decisions made in one period affect the discounted value of material processed in subsequent periods. Instead of calculating just one grade cutoff per product and per process, a separate set is required for each time period. If, for example, a mine has two products, three processing methods, and operates for 20 years, then 120 cutoffs must be calculated, and the setting of cutoffs in each period affects the optimal setting of all others.

When one further complexity is added, the whole concept of discriminating based on product grades alone becomes problematic. When the processing recovery for one product

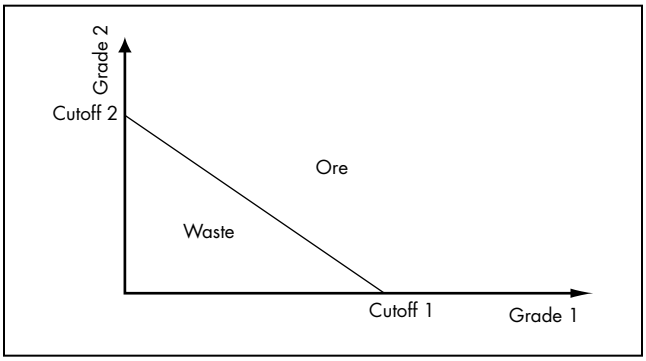


Figure 10.2-3 Application of multiple-product grade cutoffs

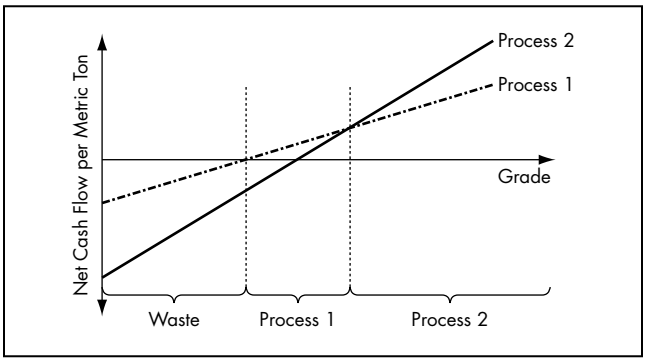


Figure 10.2-4 Application of cutoffs for multiple processes

(e.g., gold) is affected by the grade of another product (e.g., copper), the application of product grade cutoffs, as shown in Figure 10.2-3, becomes inaccurate—it can lead to the misclassification of some ore as waste and some waste as ore. The solution is to use a cutoff method based on cash flows rather than product grades. This allows for the avoidance of the misclassification described above, and it also significantly reduces the number of decision variables. In optimization software, this is usually referred to as the cash-flow method, and it is very easy to apply when the objective is to maximize undiscounted net cash flows. Each parcel of material is assigned to the processing path that will maximize the profit (or minimize the loss) for the parcel.

If the objective is to maximize NPV, then a related approach called the DC method can be used. The DC method extends the cash-flow method used for marginal cutoff calculation and allows for its application to NPV maximization problems.

Figures 10.2-5 and 10.2-6 illustrate the application of the DC method to two processes: "mill" and "heap," respectively. The illustrations are for two processes only, but the approach is applicable to any number of processes.

Allowing for Topographical Features

Mine planners often need to deal with topographical features such as existing infrastructure, roads, and rivers that are difficult or expensive to move. Two methods are described here to deal with such features. The first method is forcing the pit optimizer to avoid undermining them, and the second method is allowing the pit optimizevr to undermine them only if there is economic justification to do so.

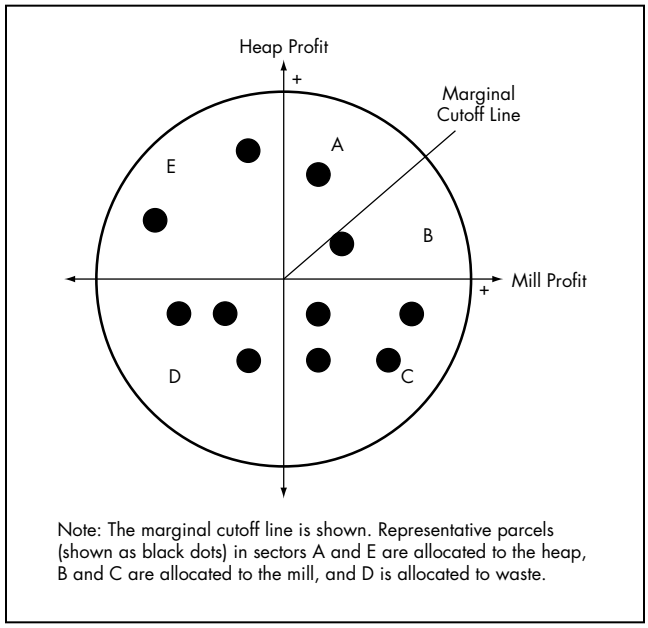


Figure 10.2-5 DC diagram indicating the domain for parcels in a given increment

The first technique involves the application of extremely high mining costs to certain blocks in the model and is effective at stopping a pit optimizer undermining defined topographical features. Figure 10.2-7 shows the surface-level blocks in a model. The shaded blocks represent the area that cannot be mined because of the presence of a river (black line). Assigning extremely high mining costs (e.g., $10 million per metric ton) to these blocks will make them uneconomic to mine, no matter how much ore is under them.

The second technique involves the definition of a set of arcs to model a topographical feature and to associate with this set the cost of relocating the feature. Arcs are normally used to model the pit slope constraints in pit optimization, but their application can be extended to modeling other things, including topographical features that are either expensive or impossible to mine through. Figure 10.2-8 shows how arcs can be used to represent such a constraint. The dotted line shows the approximate southern boundary of the river. The river may be relocated further to the north at a cost, but without relocation, the dotted-line boundary is the furthest north that a pit can extend. The arcs, shown as arrows in the diagram, trace a path along the boundary, and then one final arc completes the loop. The effect of this loop of arcs is that if a pit outline were to include any one of the linked blocks, then all blocks must be mined. The cost of relocating the river is built into the cost of mining one of the blocks; in this example, the shaded block. It does not matter which block carries the cost, provided that the cost is only carried once. The effect of this arrangement is that the pit optimizer will mine through the boundary only if doing so will maximize the value of the pit (i.e., only if relocating the river is economically justified).

Allowing for Safety Berms and Ramps

Pit optimizers cannot model safety berms and ramps directly. Instead an average pit slope angle must be calculated for the purpose of pit optimization, which takes into account the

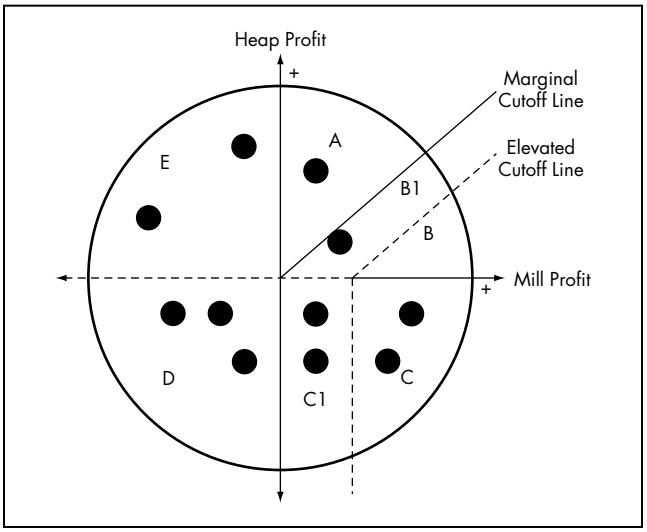


Figure 10.2-6 DC diagram indicating the application of an elevated cutoff on the material leading to the reallocation of parcel in sector B1 to heap and C1 to waste

safety berms and the width and number of times a ramp is expected to intersect a pit wall. When the pit optimization work is complete, the safety berms and ramps should be inserted into the design.

A geotechnical engineer prescribes the inter-ramp angle. It is the maximum slope that can be safely sustained between ramps and the base or the rim of the pit. The geotechnical engineer will prescribe the vertical frequency and width of safety berms, which must be taken into account in the calculation of the inter-ramp angle.

The average pit slope for a given sector of pit wall is based on the inter-ramp angle but adds an allowance for ramps. To calculate the average pit slope for a given section of pit wall, the following information must be known:

- The design width of the ramps—the width must take into account the types of equipment that must use it, the corresponding height and position of the safety windrow, the need for drainage and utilities, and the requirements for geotechnical stability
- The number of times the ramp is expected to cross this section of wall
- The height of the wall from the base of the pit to the pit rim

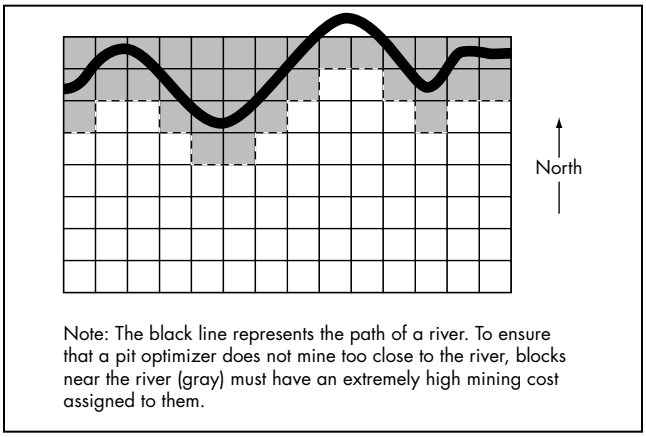
In the early stages of study, these things may be unknown or poorly understood. The solution is to work iteratively to a solution throughout the planning process, adjusting the average pit slopes (Equation 10.2-8) for use in pit optimization as better information becomes known.

Referring to Figure 10.2-9, the horizontal distance from toe to crest without ramps (B) is calculated using Equation 10.2-7:

$$B = (C/\tan(x°)) \tag{10.2-7}$$

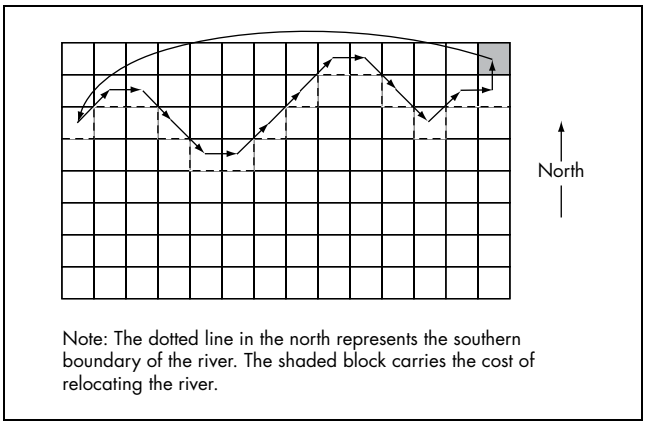
where

C = depth of the pit
$x°$ = inter-ramp slope angle



Note: The black line represents the path of a river. To ensure that a pit optimizer does not mine too close to the river, blocks near the river (gray) must have an extremely high mining cost assigned to them.

Figure 10.2-7 Plan showing the surface level of blocks



Note: The dotted line in the north represents the southern boundary of the river. The shaded block carries the cost of relocating the river.

Figure 10.2-8 Plan showing how the path of a river is defined using arcs

The average pit slope $y°$ is calculated using the following equation:

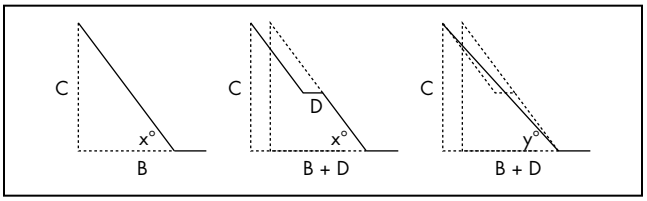
$$y° = \text{atan}(C/((C/\tan(x°)) + D)) \tag{10.2-8}$$

where D is the width of the ramp.

A Plan for a Final Pit

In the life of a mine, the plan for the final pit may change many times. The changes occur as new information becomes available, for example, new drill results that extend the resource, revised price and cost forecasts, or improved technology. Information constantly changes, and to the greatest extent possible, plans should be refined and improved over time to take advantage of the new information. The final pit only comes into existence when mining stops, and up until that time, its final form is uncertain. Consequently, the use of the term *final pit plan* is discouraged. It implies that both the pit and the plan are final. The plan is rarely final, so the term *plan for a final pit* is preferred.

Having established the plan for the final pit as a changeable thing, for practical purposes it must manifest at any given time as something concrete—something that can be tested with various economic assumptions, schedules, and



Courtesy of Gemcom Software International.

Figure 10.2-9 Allowing for ramps and safety berms in the average slope *y°*

mining rates and adjusted for different mining widths and ramp designs. At the beginning of a planning project, a reasonable proxy for the final pit can be found in the optimum pit outlines determined in the pit optimization process. The pits corresponding to revenue factors 0.75 through 1.5 provide a good spectrum of pit sizes for consideration in the early part of the planning process. In examining this range of pits, the existence of common walls or floors must be determined. Common walls and floors indicate aspects of the pit shape that are stable for a range of economic assumptions. These stable areas can be useful in mine planning, but before relying on them as design features, one must try to understand why they are stable. It could be that the geological model includes a boundary that limits the ore zone and leads to a common wall in a range of pits, or that the geological model shows ore extending only to a certain depth, thereby limiting the depth of the pit. In these situations, the reliability of the geological information should be tested as follows:

- Could it be that the boundary indicated in the geological model is not exactly where it is shown? Could there be unexpected folds or faults?
- Is the vertical limit of the modeled ore based on positive information (e.g., drilling penetrated barren rock below the ore) or the absence of it (e.g., drilling has not been extended below this level)?

This sort of examination may prompt further data gathering or reinterpretation of the geology. In the meantime, planning should proceed as follows:

1. Use the largest pit in the range to establish a reasonable maximum footprint. This footprint will help in the positioning of infrastructure, waste dumps, and leach pads. Positioning these things outside the maximum footprint will reduce the risk of incurring the capital outlay to relocate them in the future. However, this risk reduction comes at a cost: the cost of hauling waste and ore over greater distances. The final decision as to distancing should be made on the balance of costs of longer haulage and the benefit of reducing the risk of future relocation.
2. Use the revenue factor 1.0 pit as a proxy for the final pit, for the purposes of determining the mining schedule and production rate.
3. As testing of the mining schedule and production rate proceeds, retest pits in the range of revenue factors 0.9 through 1.0, and adopt the one with the highest NPV as the new proxy final pit.
4. Refine the pit shapes based on ramp designs and mining widths. Produce a range of pit sizes and test them to determine the pit that yields the highest NPV.

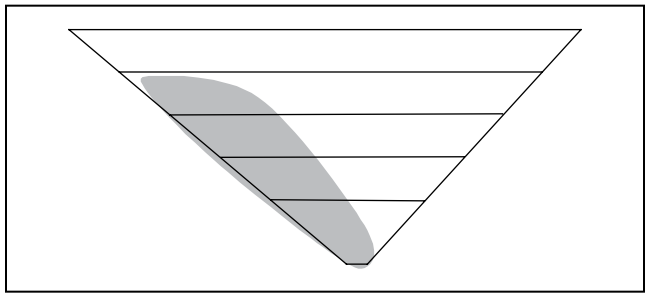


Figure 10.2-10 A worst-case mining sequence. The stripping ratio in the early part of the mine life is extremely high, requiring a significant amount of waste mining to access the ore.

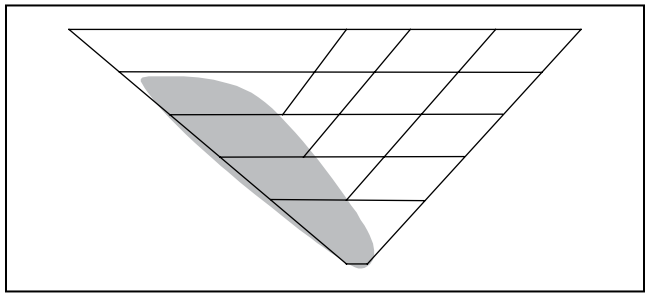


Figure 10.2-11 Mining sequence with four pushbacks. By mining each pushback in turn, the amount of prestripping required to access early ore is significantly reduced.

Pushbacks

Two mine sequences are illustrated in Figures 10.2-10 and 10.2-11. The first is simple to achieve, with excellent mining access and low complexity. However, the stripping ratio in the early part of the mine life is very high and then decreases over time. In this sequence, much of the cost of stripping is incurred early in the mine life, and access to ore is delayed. This sequence leads to poor NPV and for that reason is often referred to as a worst-case sequence (worst for NPV). In Figure 10.2-11, a mining sequence with four pushbacks is shown. By mining each pushback in turn, the prestripping is deferred and ore access is achieved earlier. The stripping ratio starts out low and ends up high. This sequence is more complicated than the first, but it will yield a higher NPV.

The sequence shown in Figure 10.2-11 is similar to the best-case schedule described previously. It has a many pushbacks and a high NPV. The trade-off is practicality. Specifically, too many pushbacks can lead to excessive expense to maintain multiple working slopes, vertical advance rates that are difficult or impossible to achieve, operational problems associated with moving equipment around multiple working faces, lack of adequate mining width, and extra cost associated with reworking ramps.

If these problems are experienced with a design that has many pushbacks, it is likely that the NPV calculated for that particular case is unachievable. On the other hand, if all pushbacks were eliminated, practicality could be assured, but for many ore bodies there will be a significant loss in NPV. How should the need for pushbacks be determined? If pushbacks are needed, how many should be employed? The simplest way to determine the need for pushbacks is to experiment with different numbers of them. The two extremes have already been

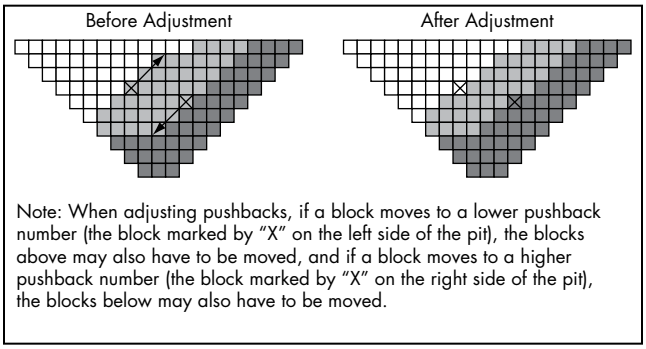


Figure 10.2-12 Pushback adjustments

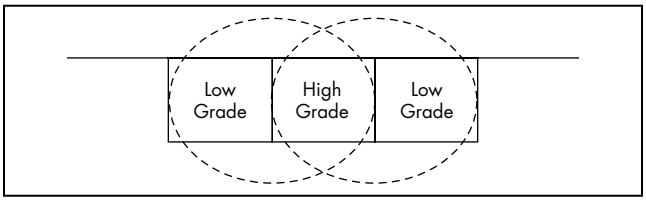


Figure 10.2-13 In this example, low-grade ore can be blended with high-grade ore in a 1:1 ratio to make salable product

discussed—no pushbacks and many pushbacks. The value difference between these two schedules will provide an indication as to the value-adding potential of pushbacks. If the value difference between a best-case and worst-case schedule is small, then there is little or no advantage in employing pushbacks. However, if the value difference between a best-case and worst-case schedule is great, employing some number of pushbacks will add significant value.

In a best-case schedule, many pushbacks are used, often in the range of 50 to 100. This is not a practical number, but it is easy to generate this many, and it provides a useful benchmark. In practice it is possible to achieve very nearly the same value as a best-case schedule merely by sizing each pushback to contain about a year's production. However, the success of this approach is to some extent illusory. If NPV is being calculated on annual rests, which is usually the case for long-term planning, then the calculation of NPV will be no different, for example, for 12 annual pushbacks compared to 48 quarterly pushbacks. The NPV calculation is insensitive to anything that happens on a timescale shorter than 1 year. Annual pushbacks are a good starting point but may still be more than is required to achieve a good NPV. Therefore, the recommended approach is as follows:

1. Compare the value of a best-case schedule and a worst-case schedule. If the values are similar, then there is little advantage in employing pushbacks. Focus instead on practicality and operational efficiency.
2. If the value difference between best case (many pushbacks) and worst case (no pushbacks) is great, then an examination of pushback alternatives is warranted. Start to simplify the best-case schedule by choosing pushbacks that each represent about a year of production. Also estimate the maximum number of pushbacks that can fit given the mining-width constraint. This is where practical requirements and theoretical best-case NPV collide. If pushbacks are too small to be mined, then the reported NPV is theoretical. Proceed with no more pushbacks than can be practically achieved.
3. Experiment with reduced numbers of pushbacks, adjusting their size to achieve the most favorable schedule. If a stripping hurdle occurs in a schedule, it will often be associated with the conclusion of one pushback and the commencement of another. Try to increase slightly the size of the concluding pushback, or try decreasing the size of the commencing pushback to overcome the stripping hurdle. Test the NPV of each alternative and settle on the smallest number of pushbacks that will achieve NPV that is close to the best-case benchmark.

Allowing for Mining Width

When a pit optimizer generates a set of pit shells, it takes no account of mining width. As a result, there is no guarantee that any pair of shells will obey mining-width constraints to the extent that good mine planning requires. A mine planner can take care to choose pit shells that exhibit most of the attributes required, but inevitably some intervention will be required to adjust the shape of the pits. Some software packages have options to apply mining-width constraints subject to a user-defined set of parameters. Without access to such systems, manual design techniques should be applied to adjust for mining width.

Whenever an adjustment is made to a pushback in one bench, it can affect the shape of the pushback in benches below or above it (refer to Figure 10.2-12). Any such changes to pushbacks will almost certainly lead to a loss of NPV.

Allowing for Blending

In the field of open-pit mine planning, few would doubt the enormous impact that the Lerchs–Grossman pit optimization method has had. The method is highly efficient, and it can be guaranteed to produce an optimal solution. However, it can only produce an optimal solution if each block in the model is assigned an independent value, and it is not always possible to do that. This can certainly be the case when run-of-mine material is blended to make a salable product. A simple three-block model is shown in Figure 10.2-13. In this example, low-grade ore can be blended with high-grade ore in a 1:1 ratio to make salable product. Assuming all blocks contain identical tonnages of material, the high-grade block should be mined, but which of the two low-grade blocks should be mined? Either one or the other of the low-grade blocks should be mined, but not both. There are two optimal solutions to this problem, and no matter how the economic model is built, it is unlikely that the Lerchs–Grossman method will be able to find either solution.

So, how does one define the pit shape for blended products? Lerchs–Grossman pit optimization can still be used to contribute to the solution, but it cannot be relied on to the same extent as for nonblended products. The Lerchs–Grossman method must be supplemented by a technique to prepare models for optimization and test for the optimality of solutions. These will be discussed in reverse order.

To describe the test for optimality, it is necessary to first establish a typical pricing model for blended products. Figure 10.2-14 shows a typical pricing arrangement based on the product grade. There is a target grade and a price applicable for blended product with that grade. Above the target, a premium is paid, and below the target, a penalty is applied.

If the grade gets too low, the product is rejected altogether. In addition to being affected by product grade, blended product price may also be affected by the grade of deleterious elements. Figure 10.2-15 shows a typical structure for the premium or penalty for a deleterious element. If the grade is too high, the product is rejected altogether.

In most cases, the best value for a mine will be achieved when it can consistently produce blended product that just meets or slightly exceeds the target grade requirements. This is because the gradient on the price curve changes at the target grade—the rate at which a premium is paid is less than the rate at which penalties are applied if the product is worse than the target grade. The implication for pit design is that the average grades (product grade and deleterious element grades) of all ore mined must equal the grades of the on-specification product. That sets an important condition for optimality of the pit—that the average grades of all material flagged as ore and included in the pit must be equal to or slightly better than the target grades.

This condition can be considered as it might be applied to the simple example in Figure 10.2-13, but to ensure that there is just one optimal solution, the data can be perturbed by assigning slightly different tonnages to the two low-grade blocks. The left-hand low-grade block contains 10,000 t—the same tonnage as the high-grade block. The right-hand low-grade block contains 11,000 t. There is now just one optimal solution: Mine the left low-grade block and the high-grade block. This is because if the right-hand block was substituted for the left, then an extra 1,000 t of low-grade material would be mined adding to cost, but not increasing the amount of on-target blended product that can be produced.

With this perturbation, one is a little closer to applying the Lerchs–Grossman optimization method. But a method is still needed to prepare the model for optimization, that is, one that will bias the optimizer to choose a pit containing the desired average grade characteristics without biasing the outcome in any other way. The most flexible method involves the use of parametric pricing with price attenuators to increase or decrease the pit optimizer's propensity to include different material in the pit outline. The calculation of the attenuated price, P_a, is shown in Equation 10.2-9 for one product grade and one deleterious element grade, but it can be extended to include two or more deleterious element grades.

$$P_a = P_b\left(\frac{A_p(G_p - T_p)}{T_p} + \frac{A_d(G_d - T_d)}{T_d} + 1\right) \qquad (10.2\text{-}9)$$

where

P_b = price paid for 1 t of on-specification blend
A_p = attenuator for product
G_p = product grade for a block
T_p = target product grade
A_d = attenuator for the deleterious element
G_d = deleterious element grade for a block
T_d = target deleterious element grade

The attenuated price, P_a, is a function of price attenuators A_p and A_d as well as the target grades and prices and the block grades. With the attenuators A_p and A_d set to zero, the attenuated price is equal to the target price, no matter what the block grades are. If an attenuator is set to a positive value, the attenuated price increases for higher grades and decreases for lower grades. If an attenuator is set to a negative value, then the effect is reversed—the attenuated price decreases for higher grades and increases for lower grades. By manipulating the attenuators, relative block values are changed to increase or decrease their propensity to be included in a pit outline by the pit optimizer.

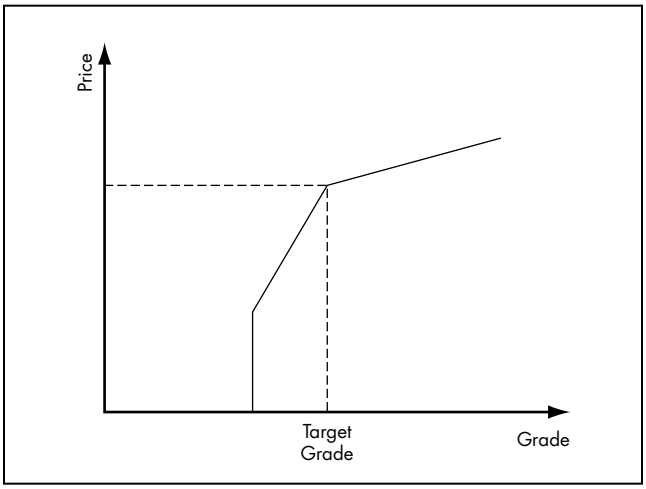


Figure 10.2-14 Typical structure for a sales contract for a blended product. There is a target grade and a price applicable for blended product with that grade. Above the target a premium is paid and below the target a penalty is applied. If the grade gets too low, the product is rejected altogether.

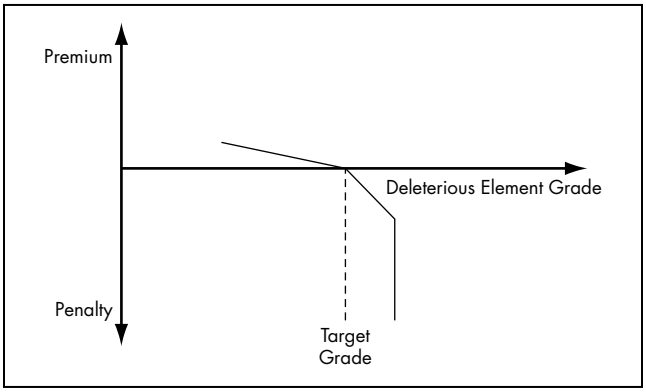


Figure 10.2-15 Typical structure for the premium or penalty for a deleterious element. If the grade of the deleterious element gets too high, the product is rejected altogether.

Two additional tools are now required to conduct pit optimization: a technique to prepare models for optimization and a test for the optimality of solutions. The application of pit optimization with these tools is as follows:

1. Prepare the model for optimization, calculating block values using Equation 10.2-9 with attenuators set to zero.
2. Run the pit optimizer.
3. Evaluate the mined ore in the pit by calculating the average grades for product and deleterious element. Compare these average grades with the target grades. If the average grades are equal to or slightly better than the target grades, then the result is acceptable and you can stop. If not, go to Step 4.

4. Adjust the price attenuators. If, for example, a decrease in the average grade is desired, reduce the value of the corresponding attenuator and go to Step 2.

This is an iterative process with an indeterminate number of steps. It is possible that the approach will not be able to achieve a solution, and here are the following two reasons that it might fail:

1. There may be more than one optimal solution, as was the case shown in Figure 10.2-13. One may experience this problem by being unable to find a satisfactory setting for an attenuator, with a tiny change in the attenuator leading to an extreme change in the average grade mined. In the Figure 10.2-13 case, tiny changes in the attenuator would lead to either both or none of the low-grade blocks being mined. A possible workaround to this problem is to perturb the data, for example, by changing the cost of mining by depth or distance from facilities.
2. There may be an abundance of certain material types that must be mined, with only some of it being useable as product. This is illustrated in Figure 10.2-16. One may experience this problem by being unable to find a satisfactory setting for an attenuator, with a tiny change in the attenuator leading to an extreme change in the average grade mined. This is similar to the case illustrated in Figure 10.2-13, except that, provided that the high-grade block is worth mining, there is no choice but to mine both low-grade blocks overlaying it. In this case, one should settle on the most conservative set of price attenuators that yields a pit with better than target grades. After the pit optimization process, additional material can be added to the blend to bring the actual grades down to the target grades.

Analyzing the Case for Flexibility

The beginning of this chapter has focused on the long-term trends for prices, not the short-term fluctuations. However, short-term fluctuations will surely occur, and mines need to test the impact and assess the case for building flexibility into an operation to allow it to respond to fluctuations. An example of flexibility in a metals mine is the ability to change the processing cutoff grade. When prices are high, it may be advantageous to increase the processing cutoff. This increases the output of product, allowing the mine to take full advantage of the high price. To be able to make this change, the mine must have the ability to move quickly to a higher mining rate because raising the processing cutoff will lead to a higher strip ratio, and to keep the rate of ore constant, the mining rate must be increased. The benefit of flexibility in this case is the ability to take advantage of the higher prices; the cost is the additional cost of having mining flexibility.

Price is the most important economic driver for all mines, and it is also quite variable. Other important economic drivers may exist that are just as uncertain as price, and it is important to analyze the case for flexibility. The following are other examples:

- Providing excess capacity on ramps allows mining rates to be increased quickly.
- Allowing some flexibility in the comminution functions allows short-term recovery and throughput changes to be made in response to unexpected ore characteristics.

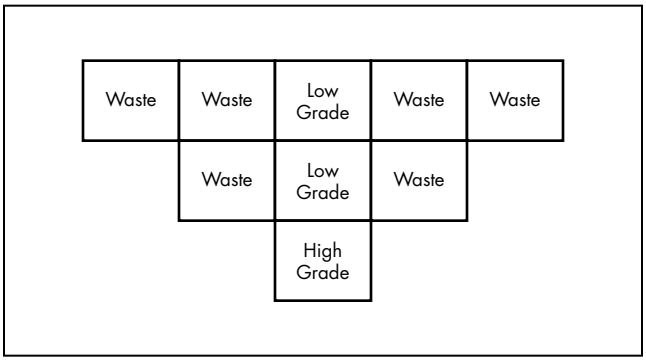


Figure 10.2-16 In this example, abundant low-grade material overlays the high-grade material, and there is no way to mine the material in the desired 1:1 ratio. Provided that the stripping can be supported, the best solution would be to mine both low-grade blocks and the high-grade block and treat half of the low-grade material as waste.

- Maintaining spare mining capacity and an inventory of exposed ore provides more short-term options to manage head grade and throughput.

Identifying and Managing Risks

The inputs to mine planning have varying levels of uncertainty, and uncertainty can translate to risk. Steps must be taken to reduce risks to an acceptable level. To achieve a successful outcome it is necessary to manage all risk aspects closely, including the potential for risk/reward—that is, the acceptance of additional risk (within an approved tolerability limit) resulting in increased rewards.

Each mining company needs to develop an organizational risk management process and culture that provides its management and shareholders with certainty and confidence in its planning process. Organizational management processes are dealt with in detail in other literature. Whichever approach is adopted, it is important that this process accomplish the following:

- Identifies, assesses, and manages both strategic risks associated with the mine plan as well as tactical and other risks that the plan must manage
- Supports the planning process through the objective analysis of the contributing/influencing uncertainties and their consequences
- Designs, implements, and monitors effective risk controls that either reduce the probability of the risk occurring or the consequence/impact of an event when it occurs
- Applies risk management equally to risk reduction/avoidance and to opportunity/upside potential

The result of the analysis will be changes to the mine plan that impact risk and NPV.

Determining Options for Preservation

An option in finance is the right, but not the obligation, to execute a transaction at a later time. An option can be bought and has a value because it exposes the owner to the possibility of the later transaction. In mine planning there are many decisions that are conceptually comparable to an option in finance.

Table 10.2-8 Example of a preserved option

Description of Option	Low-Grade Ore To Be Processed Rather Than Wasted
Trigger for option	Success hypothesis (low-grade ore processing method) is true
Probability, P_o	The study team estimates 70% probability of success, P_e, in achieving a solution with \$2/t lower processing cost and 84% recovery
Success value, V_{oe}	\$150 million
Actions and costs to preserve	Reschedule to avoid moving the low-grade ore until the success of the low-grade processing method is known. Cost of reschedule ~ \$20 million (NPV)
Option value, V_o	\$85 million

Table 10.2-9 Minimum requirements for an underground mine plan—for the purposes of calculating the underground value *Y* of blocks in a model

Information Requirement	Description
Stope envelopes	The shapes that contain mined ore, defined as a set of blocks (stope blocks)
Aboveground value	The value of a stope block after it reaches the surface. The value can be calculated as grade × recovery × price – processing cost × tons
Mining and haulage costs	The costs to mine a stope block and bring it to the surface for processing

For example, maintaining flexibility in a mining fleet has a cost, but it provides the ability in the future to quickly increase mining rate, allowing the processing cutoff to be increased, thereby increasing monthly product output. If this option is available, it is one that could be exercised during periods of very high commodity prices. Without having an option such as this, the product output could not be changed in response to changes in the market. An option must be exercisable in the following situations:

- A credible future price regime
- Response to positive exploration results
- Response to a positive research-and-development outcome
- Other circumstances that are credible and that have non-zero probabilities of occurring

If it is possible to estimate the probability that the future option will become available, the value of the exercisable option, V_o, is calculated using Equation 10.2-10:

$$V_o = P_e \times V_{oe} - C_{po} \quad (10.2\text{-}10)$$

where

P_e = probability that a trigger event will occur (triggers the option)
V_{oe} = value (discounted cash flows) of the option if it were exercised
C_{po} = cost (discounted cash flows) incurred in preserving the option

If it is not possible to estimate the probability of a trigger occurring, then an alternative is to calculate the minimum probability, $P_e\ min$, that is required for the option value to be positive ($V_o > 0$). If $P_e\ min$ is sufficiently low, then it is reasonable to conclude that the option value will be positive. Table 10.2-8 shows an example of a preserved option.

Determining the Open-Pit/Underground Interface

For some ore bodies, mining is possible by either open-pit or underground methods. If the choice is between applying one method and the other, then a decision as to which method to choose can be made by comparing the best open-pit plan with the best underground plan.

In the case that a combination of the two methods may be employed, then a decision must be made as to where one method ends and the other begins. When both open-pit and underground mining methods are used, a great many problems can arise. These include the complexity of sequencing the commissioning and operation of the mines and the costs and risks of managing the geotechnical and safety consequences of digging a pit near an underground mine. The detailed analysis of these issues is beyond the scope of this chapter. What follows is a description of a simple method to determine an interface between the two methods, with some parts of the ore body being taken by the pit while others are left for underground mining.

The interface between the two methods will occur at a depth or location at which the open-pit method becomes less economic that the underground method. The determination of this interface can be achieved by applying a pit optimizer to a specially modified block model. The modifications enable the calculation of the value of each block as the difference between its open-pit value and its underground value. This difference represents the net benefit of mining a block by open-pit method instead of by underground method. The following is an important logical assumption that underpins this analysis: If a block could be mined by underground method and it is not mined by open-pit method, then it will be mined by underground method.

A block can be considered that has value X if mined by open-pit method and a value Y if mined by underground method. The value Y will be achieved if the block is not included in the pit, because it will instead be mined by underground method. The value X will be achieved if the block is included in the pit outline, but the improvement in value is not X, it is only $X – Y$. This is the value that should be used for the purposes of pit optimization—the difference in value between open pit and underground.

The calculation of the underground value Y relies on the application of an underground planning process (beyond the scope of this chapter to describe). Instead, here are some guidelines as to how the underground value of blocks should be considered in the open-pit mine planning process. The calculation is described in principle only as the application details of this method depend on the modeling tools included with the pit optimization software being used. The minimum requirements are described in Table 10.2-9. The underground value Y is equal to the aboveground value, less the mining and haulage costs.

Now that the underground value of each block is known, it should be deducted from the value of each block in the model prior to pit optimization. Pit optimization proceeds with block values equal to the open-pit value, less the underground value ($X – Y$). The result will, in most cases, be a pit that is smaller than it would have been if the block values had not been adjusted. It is possible for the pit to stay the same, but it is not common. That part of the underground mine plan

Table 10.2-10 Treatment of rehabilitation and closure costs

Type of Cash Flow	Description	Treatment
Rehabilitation bond	A bond paid to the government against mine rehabilitation. In the event that rehabilitation is carried out effectively by the mining company, the bond is returned.	The bond can be treated like working capital. There is an opportunity cost (interest foregone) associated with the bond, and it is incurred for as long as mining continues and until rehabilitation is complete. This interest cost should be treated as a fixed cost during the life of the mine.
Waste-related continuous rehabilitation cost	Waste dumps may be continuously rehabilitated to their final form and cover.	These costs are incurred roughly in the same period as mining occurs and are proportional to the tons of waste (assuming a certain waste dump format and rehabilitation plan). The costs should be modeled as a variable cost per ton of waste.
Ore-related continuous rehabilitation cost	Tailings may be treated and stabilized on a continuous basis.	This cost is incurred roughly in the same period as processing occurs and is proportional to the tons of ore (for a given treatment and stabilization process). The sum of the costs should be modeled, a variable cost per ton of ore.
Terminal ore- and waste-related rehabilitation cost	Recontouring a pit, replacing the topsoil, and revegetating at the end of a mine life is required.	Treated as a terminal cost. This terminal cost will vary with the size of the final pit.
Terminal ore-related rehabilitation cost	This includes stabilizing and covering tailings dams.	This cost is treated as a terminal cost. This terminal cost will be different for different totals of ore tons.
Terminal facilities cost	The removal of buildings and equipment and preparing the land for its future use is required.	This should be treated as a terminal cost.

that was not mined out by the open-pit mine remains as an underground mine.

Development costs are not included in the calculation of the underground block value. These costs are important, but there is no satisfactory method for apportioning the development costs to stope blocks in the model. The alternative is to deal with the development costs outside the pit-optimization process. If some blocks are mined in the open pit that could have been mined by underground method, then the initial underground mine plan assumptions are no longer valid. This is particularly true for development that is justified by its ability to gain access to several stope blocks. If some of those stope blocks are removed from the underground mine plan, then the justification for the underground development must be reassessed. This reassessment may lead to further changes to the underground mine plan, specifically the removal of any development that can no longer be justified and, consequently, the removal of any remaining stope blocks that rely on that development. If this is the case, then the whole process of determining the open-pit/underground interface should be repeated, but now based on the new (smaller) underground mine plan.

Another consideration is that the pit optimizer will take no account of the separation between open-pit and underground mining. Without proper separation, either the open pit or the underground or both mines will become unstable and unsafe. A geotechnical engineer should be consulted and adjustments made to the plans to ensure that both plans can be safely executed.

Anticipating and Allowing for Mine Closure

Mine rehabilitation and closure are described in detail in other chapters of this handbook. In this chapter, the treatment of closure is limited to the cash-flow effects and their impact on mine planning decisions. Table 10.2-10 shows different types of rehabilitation costs and their recommended treatment.

The treatment of costs shown in Table 10.2-10 must be simplified for the purposes of pit optimization because in that process, only variable costs can be modeled.

- Fixed costs (e.g., interest associated with a rehabilitation bond) should be treated as implicit fixed costs.
- Terminal costs should be treated in the same way as capital.

MODELS AND TOOLS

Introduction to Optimization

Optimization is a term that can be used in a general sense to mean a process through which an outcome is made as good as it possibly can be through the adjustment of inputs, structures, or methods. The term also has a more precise mathematical definition—which means to "find the optimal value of a function, often subject to constraints."

Optimal (value) in mathematical terms means one of the following:

- **Minimal value:** The lowest possible value (to optimize total cost is to minimize it)
- **Maximal value:** The highest possible value (to optimize total profit is to maximize it)

A model for optimization includes the following:

- **Objective:** This is the item to be optimized.
- **Objective function:** The objective function is a mathematical expression, or some other kind of mathematical model that calculates the thing that is to be optimized subject to the variables described next.
- **Input data:** Costs, prices, and efficiencies are all examples of input data on which the calculation of the objective value depends.
- **Decision variables:** To optimize the objective, an optimization process must find the appropriate settings for one or more decision variables. An optimization problem may have one or many decision variables.
- **Constraints:** The settings of the decision variables are subject to constraints, both on the decision variables themselves and on functions of the decision variables.
- **Optimization method:** This is a method for determining the appropriate settings for the decision variable such that the objective is optimized.

Mine Planning Optimization

Tables 10.2-11 through 10.2-13 show basic descriptions of the most commonly applied optimization processes in open-pit mine planning. These optimization methods are available in a variety of commercial software packages.

Table 10.2-11 Typical formulation for pit optimization

Objective	Maximize the value of the pit (undiscounted).
Input data	A dollar value must be provided for each block in the block model framework. The block values are calculated based on the expected cash flows if the block is mined and (optionally) processed. Blocks can have negative values (e.g., waste) or positive values (e.g., ore). Terminal costs and capital costs are not included in the model except in special cases.
Objective function	The total value of the pit is the sum of the values of all blocks selected for inclusion in the pit.
Decision variables	There is one decision variable for each block in the model with two possible settings: *include* or *do not include* in the pit. The solution is presented as the set of blocks that constitutes the pit.
Constraints	Blocks may only be included in the pit if the blocks above it are also included in the pit. "Blocks above" includes any blocks on higher benches that need to be mined to satisfy pit slope requirements. These block dependencies are defined by arcs. An arc points from one block to another, indicating that if the first block is included in the pit outline, then the other must also be included. Figure 10.2-17 shows a two-dimensional illustration of the use of arcs to define block dependencies. If the shaded block is to be included in the pit outline, then the three blocks above it must be mined to maintain 45° pit slopes. Every other block in the model has a similar set of arcs, so mining dependencies are maintained all the way up to the surface. If the shaded block is mined, then all blocks in a cone (defined by the pit slopes) above the shaded block must also be mined. Figure 10.2-18 shows an example for 31° slopes. The accuracy with which slopes can be modeled is determined by the number of arcs. Most pit-optimization software allows the user to control how arcs are generated. If more arcs are generated, pit slope accuracy is improved, at the expense of processing speed. Arcs can also be used to model constraints other than pit slopes.
Optimization method	The most common optimization method is the one described by Lerchs and Grossmann (1965). This method is framed in graph theory and follows a repeating set of steps until a specific test for optimality is met. The result is a subset of blocks that obeys all block dependencies defined by the arcs and has the highest positive value.

Table 10.2-12 Typical formulation for schedule optimization

Objective	Maximize NPV.
Input data	A dollar value must be provided for each block in the model framework (similar to the requirements for pit optimization). In addition, the ore, process stream allocation, and waste tonnage information for each block is required. Blocks are often aggregated into logical mining units with a defined set of precedences. The aggregation is done to reduce the complexity of the optimization problem.
Objective function	Determine the schedule of blocks (or logical mining units) that will maximize the NPV of the mine.
Decision variables	The period in which each block or logical mining unit is mined
Constraints	Constraints always include maximum rates for mining and processing through each processing stream and product sales. Many software systems also allow additional constraints such as a maximum vertical rate of advance.
Optimization method	Schedule optimization is often formulated as a linear or mixed-integer programming problem in mine schedule optimization software.

Table 10.2-13 Typical formulation for cutoff and stockpile optimization

Objective	Maximize NPV.
Input data	A sequence of material made up of discrete increments. Each increment represents a set of parcels, each with information as to tons, grade, and other attributes. Other input information includes capital; fixed, variable, and terminal costs; prices; and metallurgical recoveries.
Objective function	Determine the allocation of material in the sequence of increments to stockpiles, waste dumps, or available processing streams on a period-by-period basis. Determine also the reclaiming of material from the stockpiles to feed to the available processing streams on a period-by-period basis.
Decision variables	The period in which each parcel is mined and the allocation of that parcel to a defined waste, process, or stockpile. The decision may be implemented by way of one or more cutoffs.
Constraints	Constraints always include maximum rates for mining and processing through each processing stream and product sales. Often, additional constraints are available.
Optimization method	Cutoff and stockpile optimization may be achieved as a separate process applying dynamic programming methods. Alternatively, it may be combined with schedule optimization and solved using linear programming or mixed-integer programming methods.

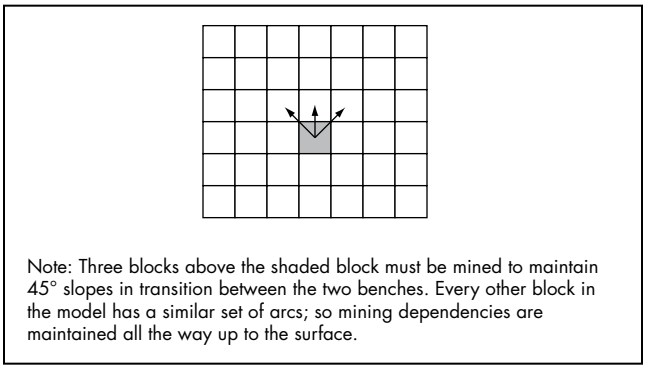
Resource Models

Most open-pit mine planning is conducted with a regular block model. A *block model* is one in which the area of interest is represented in a set of discrete rectangular blocks, each of which accounts for a subset of the total three-dimensional space. Together, these blocks represent the whole area of interest. A *regular block model* is one in which all blocks in the framework have identical dimensions.

A block model should include all mineralized material in the area of interest plus the topography and the surrounding barren ground such that the most optimistic pit could be contained within the model framework. An example of the extent of the model's framework is shown in Figure 10.2-19.

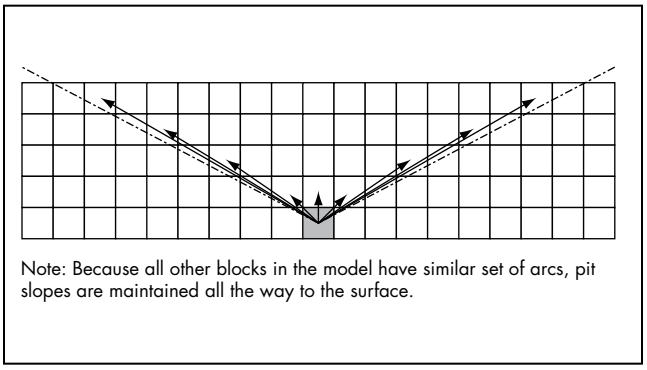
A set of attributes is assigned to each block in the model during the resource modeling process. These attributes include tonnages, grades, and other information about the material in the block. Ideally the set of attributes will include all characteristics that vary with position in the model and all characteristics that are expected to be significant drivers for one or more of the following:

- Units of product produced
- Price achieved per unit of product



Note: Three blocks above the shaded block must be mined to maintain 45° slopes in transition between the two benches. Every other block in the model has a similar set of arcs; so mining dependencies are maintained all the way up to the surface.

Figure 10.2-17 Two-dimensional illustration of the use of arcs (arrows) to define block dependencies



Note: Because all other blocks in the model have similar set of arcs, pit slopes are maintained all the way to the surface.

Figure 10.2-18 Two-dimensional illustration of the use of arcs (arrows) to define block dependencies for a slope of 31° (dotted lines)

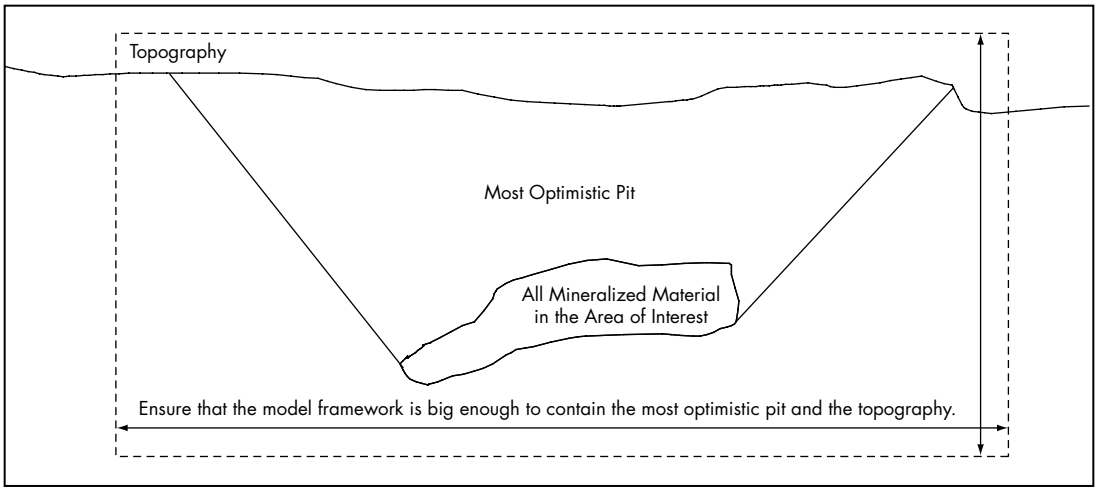


Figure 10.2-19 Extent of the model framework

- Costs (mining, processing, transportation)
- Time in a bottleneck process
- Assignment to an ore or waste stream
- Pit slopes

Examples of significant drivers are provided in Table 10.2-14.

In practice, the data available for inclusion in the block model will be less than ideal, with data collection and modeling constrained by budget, time, or technology. The gap between the information that is desired and the reality of available data should be examined at each phase of planning and recommendations made for exploration and geology so that sufficient modeling data is available for subsequent planning phases.

For planning purposes, blocks are treated as selective mining units (see Table 10.2-15) and also represent the smallest unit used to model pit slopes (smaller blocks lead to more precise modeling of pit slopes). If the original model received does not already have appropriate block dimensions, it may be necessary to split or combine blocks in it to achieve a block shape and size that is suitable for planning. Guidelines for a suitable planning block size are as follows: Set the block height to the likely bench height for mining. Set the block length and width in ratio to block height according to the likely pit slopes. For pit slopes in the range of 45° to 55°, block widths and heights should be approximately equal to the block height. For pit slopes outside that range, apply Equation 10.2-11. (Although the equation will yield a precise answer, it is only intended as an approximate guide.)

The ideal width (W) or length of a block can be calculated using the following equation:

$$W = \frac{1.2H}{\tan\theta} \qquad (10.2\text{-}11)$$

where

H = block height
θ = average pit slope

The selective mining unit is equal to the block dimensions, but the selective processing unit may be smaller.

Individual blocks in a model may include many sub-blocks or parcels, each representing a fraction of a block. A sub-block is a subset of a block with its own grade and tonnage attributes and with a defined position and volume in the parent block. In the modeling process, sub-blocks are produced to more closely follow geological boundaries. Figure 10.2-20 illustrates block models with and without sub-blocks.

Table 10.2-14 Example drivers to consider for inclusion in the block model

Key Parameter	Example Drivers (Various Commodities)
Product produced	• Zinc grade determines maximum zinc available for recovery. • Copper in gold ore may reduce gold recovery. • Ore texture can affect liberation, which in turn impacts separation and recovery.
Price per unit	• Diamond size affects price per carat. • Sulfur in coal reduces the price achieved per ton.
Processing costs	• Clays affect consumption of reagents. • The Bond work index affects comminution costs.
Mining cost (including rehabilitation)	• Geotechnical characteristics affect the need for blasting. • Acid-producing pyrites affect waste management and reclamation costs. • Broken rock texture affects erosion and slope characteristics of dumps.
Time in the bottleneck process	• The Bond work index affects resident time in the mill. • Ore texture affects best screen sizes to achieve target liberation.
Stream assignment	• Any or all of the above-mentioned drivers will contribute to the assignment of the material to an ore or waste stream.
Pit slopes	• Geotechnical characteristics affect the maximum pit slope angles. • The water table affects ground conditions and maximum pit slope angles.

Table 10.2-15 Definitions for selectivity

Term	Definition
Selective mining unit	The smallest unit that can be added to or removed from a pit outline
Selective processing unit	The units of material that, once mined, can be selectively assigned to a process or waste dump

Like a sub-block, a *parcel* is also a subset of a block with its own grade and tonnage attributes. However, unlike a sub-block, a parcel does not have a defined position within the block. In the modeling process, a set of parcels is produced for each mineralized block to represent a grade distribution. Figure 10.2-21 provides an illustration of parcels in a block. In planning, sub-blocks and parcels represent selective processing units (see Table 10.2-15).

The person who created the model and the person who will use the model in the planning process should discuss the use of these artifacts to represent processing selectivity and satisfy each other that the modeling is sufficiently accurate. The previously described methods are general guidelines only. Success in the preparation of a model for use in mine planning is indicated by satisfactory answers to the following questions:

- Is there enough information in the model (block attributes) to allow for sufficiently accurate simulation of cash flows, recoveries, and throughputs?
- Do the block dimensions allow for effective modeling of mining selectivity and pit slopes?
- Are the parcels or sub-blocks sufficiently representative of processing selectivity?

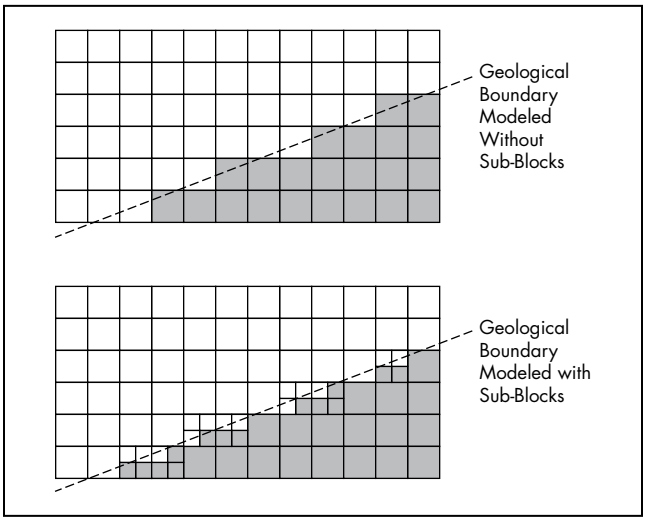


Figure 10.2-20 Sub-blocks may be used to more accurately model geological boundaries. Sub-blocks are only included in blocks that require greater resolution.

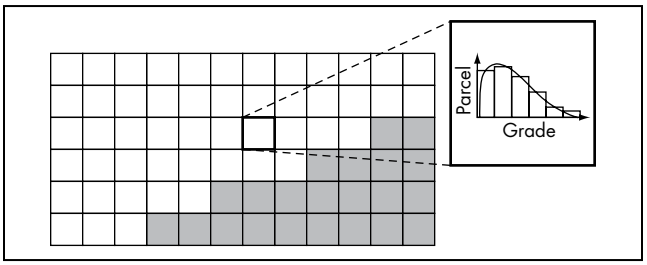


Figure 10.2-21 A set of parcels represents a grade distribution in a block. Each parcel has its own tonnage and grade.

Stockpile Models

The following definitions relate to the purpose of the stockpile and are represented as three different types. Although any given stockpile is usually of a single type, it is certainly possible for a single stockpile to serve more than one purpose.

- **Grade stockpile:** A grade stockpile is a stockpile maintained for the purpose of deferring the processing of low-grade material until later in the mine life. This is done principally to increase project NPV. Grade stockpiles are large (on the scale of years of production). Deposition and reclamation occur over many years. Grade stockpiles are usually associated with precious metals and base metals mines. They are less commonly associated with bulk products (e.g., coal, iron ore, and bauxite) or industrial minerals (e.g., limestone, mineral sands, and phosphates).
- **Blending stockpile:** A blending stockpile is a stockpile maintained for the purpose of storing material with particular grade characteristics, until such time as it can be blended with other material (either from stockpiles or run-of-mine material) so as to achieve a desired blended characteristic. Blending stockpiles are common for many bulk products and industrial minerals. They may also be used to facilitate the feed to an extractive process for precious or base metals, to improve processing recovery or throughput, or to reduce costs.

- **Buffer stockpiles:** Buffer stockpiles are designed to deal with short-term mismatches between the output of one process and the input to another. They are relatively small, representing a few hours or days of production. A common location for a buffer stockpile is on the front end of a comminution circuit to ensure that the process continues without being affected by fluctuations in deliveries of ore from the mine.

Cost Models for Optimization

To perform the optimization, the impact of decision variables on revenues and costs must be incorporated into the model. Revenues are positive cash flows and are associated with product output. Costs are negative cash flows and must be associated with activities in such a way as a change in decisions made in the optimization model will be reflected by an appropriate change in costs.

The following basic modeling definitions are required to conduct the cost modeling effectively:

1. **Capital costs:** These are costs that are incurred once to build or unlock productive capacity. For example, the cost to build an additional processing plant is a capital cost associated with the productive capacity of the new plant.
2. **Variable costs:** These are costs that vary in proportion to activities such as mining, processing, and shipping finished product. There is a generally accepted set of categories used in open-pit mine planning for variable costs, and these are described in Table 10.2-16.
3. **Fixed costs:** These are costs that are incurred regularly (periodically) and are associated with maintaining a level of productive capacity. Fixed costs are sometimes called time costs. For example, the cost to maintain a mining camp for 500 employees and their families is a fixed cost—it does not increase or decrease with the rate of mining, processing, or product produced on a monthly basis, and it only changes with the number of employees.
4. **Terminal costs:** These are costs that are incurred at the cessation of normal mining operations (typically mine closure and rehabilitation costs).

These four cost definitions and the more detailed variable cost categories in Table 10.2-16 are all that are required to build a complete cost model for mine plan optimization.

It is important to make a distinction between the definition for capital cost shown previously and a range of other costs that use "capital" in their title. For example, terms such as *replacement capital* and *sustaining capital* refer to expenditure to maintain the productive capacity of capital assets and should be treated as fixed costs or variable costs.

Capital costs for equipment that has a productive life that is shorter than the expected mine life can also be treated as a variable cost. For example, in a long-life mine, the truck fleet may be replaced several times and at the end of the mine life, the remaining fleet will be sold at a price that relates to the trucks' remaining productive life. In this case, the capital can be prorated into the variable cost of mining.

If a contractor is used for mining, then the capital costs remain in the contractor's accounts, and the only costs that are relevant to the contractee are the ones charged to the contractee. These are framed primarily as variable costs.

Table 10.2-16 Common variable cost categories used in open-pit mine planning

Common Name	Description and Application
Mining costs	This category is applied to all material mined as a dollar amount per unit of mass (e.g., dollars per ton) or volume (e.g., dollars per cubic meter). There may be different mining costs for different areas in the mine, and the cost may be varied by depth, distance from the pit rim, or by rock type. Mining cost is inclusive of drilling, blasting, loading, and dumping material. This cost category should be applied to any cost that varies with the quantity of material mined, including continuous rehabilitation of mining areas. In the event that ore costs more to mine than waste (e.g., because it requires grade control and closer drilling), then the difference should be included in the processing cost.
Processing costs	This category is applied to processed material in addition to the mining cost. Processing cost is applied as a dollar amount per unit of mass (e.g. dollars per dry metric tons) or volume (e.g., dollars per cubic meter) input to the process. A different processing cost will be used for each available process. The cost may be varied by rock type or by one or more grades. Include in this category any costs that vary with the amount of material input to processing, including the rehabilitation of tailings.
Selling costs	This category is applied to the units of product output from the process as dollars per unit of mass (e.g., dollars per gram for gold) or dollars per product unit present (e.g., dollars per Btu for coal). Include in this category costs that vary with the quantity of product output, including freight, freight insurance, royalties, and commissions.
Rehabilitation costs	This category is applied to all waste mined as a dollar amount per unit of mass (e.g., dollars per dry ton) or volume (e.g., dollars per cubic meters). Include in this category any costs that vary with the quantity of waste mined, including continuous rehabilitation of waste dumps.

Sometimes costs can be modeled as either variable costs or fixed costs, and the best method may depend on whether analysis is being done over a very long term or a very short term. As an example, when the cost of labor in mining is considered, as the rate of mining increases and decreases over the life of the mine, labor can be increased and decreased to meet the needs—over the long term, labor is a variable cost. Over short periods—for instance, from one month to the next—if the need for mining labor changes, there is very little that can be done to increase or decrease the supply of labor and its consequent cost—over short periods, labor is a fixed cost.

As it is for the example just presented, so it is for many other costs—in long-term modeling, many costs behave as variable costs, but these same costs may behave more like fixed costs over the short term.

A blend of these two behaviors is also possible: the semi-variable cost. However, rather than thinking of this as a separate category of cost, it can be thought of as a cost in two parts: one part is variable and the other is fixed.

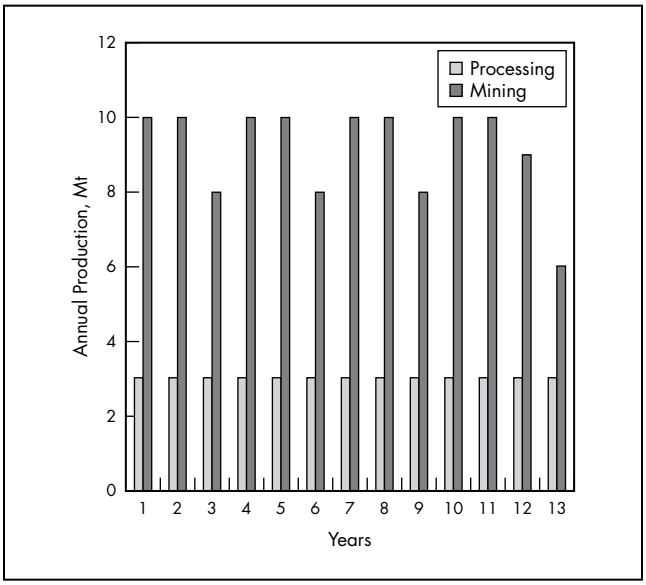


Figure 10.2-22 Example of mining schedule in which the mining fleet (10 Mt annual capacity) is not fully utilized in every period. Using a fully variable cost model for mining in this case may not be appropriate.

Table 10.2-17 Example data to use with Equations 10.2-14 and 10.2-15

Case	Description	X	Y
1	To operate the fleet at full capacity costs $20 million/yr	10 Mt	$20 million
2	To operate the fleet at 80% capacity (mining only 8 Mt) costs $18 million/yr	8 Mt	$18 million

If a mining fleet is considered capable of mining 10 Mt/yr at an annual cost of $20 million, provided that the fleet mines exactly 10 Mt/yr, it does not matter whether the cost is modeled as a variable cost ($2/t) or a fixed cost ($20 million/yr). However, it is not always possible to schedule full utilization of a mining fleet throughout the mine life. The annual mining schedule shown in Figure 10.2-22 can be considered. In this case, the mining fleet cannot be fully utilized in every year. To extend the example, assume that for two common production rates, the following statements are true:

- To operate the fleet at full capacity (10 Mt/yr) costs $20 million/yr.
- To operate the fleet at 80% capacity (i.e., mining only 8 Mt/yr) costs $18 million/yr. (These data are summarized in Table 10.2-17.)

The fixed and variable parts of the cost can be calculated using simultaneous equations as follows:

$$Y_1 = aX_1 + c \tag{10.2-12}$$

where
Y_1 = annual cost for Case 1
a = variable cost
X_1 = annual throughput for Case 1
c = fixed cost

$$Y_2 = aX_2 + c \tag{10.2-13}$$

where
Y_2 = annual cost for Case 2
X_2 = annual throughput for Case 2

$$a = (Y_1 - Y_2)/(X_1 - X_2) \tag{10.2-14}$$

$$b = (Y_1 - X_1) \times (Y_1 - Y_2)/(X_1 - X_2) \tag{10.2-15}$$

By substituting the Table 10.2-17 data into Equations 10.2-14 and 10.2-15, it is seen that the variable cost for the example is $1/t and the fixed cost is $10 million/yr.

The previous discussion focused on a model for NPV maximization and included fixed and variable costs. However, pit optimization, as described in Table 10.2-11, can only use variable costs and, in some special cases, capital costs, as follows:

- Fixed costs cannot be used in pit optimization. Any costs that would ordinarily be modeled as fixed costs should be recast as variable costs for the production activity that determines the life of the mine. When a fixed cost is modeled as a variable cost in this way, it is often referred to as an implicit fixed cost. For example, for a mining operation in which production is constrained by the processing plant capacity, the fixed costs for the entire operation (mining, processing, and shipping) total $20 million/yr, and the processing capacity is 5 Mt/yr. To render the $20 million fixed cost as a variable cost, $4 (i.e., $20 million/5 Mt) would be added to the variable cost of processing (i.e., add $4/t).
- Passive attempts should not be made to build capital costs and terminal costs into the pit-optimization model, as there is no way of doing it without introducing bias to the optimization. There is an exception to this rule: A special technique is available to include the capital cost associated with moving infrastructure, a road, or a river (refer to the "Allowing for Topographical Features" section earlier in the chapter).

The implication of the second point is that pit optimization should proceed with an incomplete cost model, one in which capital costs and terminal costs are ignored. This is correct—the pit optimizer is best thought of as an engine to maximize the difference between the variable revenues and variable costs of the mine. The optimized value of the pit yielded by the pit optimizer only represents the sum of the variable revenues and costs. To get a final pit value, the capital and terminal costs are deducted. This is best done in the context of schedule optimization.

Alternate Method for Modeling Costs for Bottlenecks

In the "Cost Models for Optimization" section, all costs are modeled as capital, fixed, variable, or terminal costs. A fixed cost varies only with time, and a variable cost varies in proportion to activities such as mining, processing, and shipping of finished product. One of these activities will constitute the bottleneck for the business, and for the bottleneck, instead of applying a variable-cost model, it is possible to apply a special kind of fixed-cost model. This is particularly useful when the throughput rate for the bottleneck depends on the type of material sent to it. For example, in a processing plant

with a nominal capacity of 3 Mt/yr and an annual operating cost of $30 million, at its nominal rate, the processing cost is $10/t. However (in this example), soft rock can be processed at 3.5 Mt/yr, whereas hard rock can be processed at only 2.5 Mt/yr. A simple variable cost will not work for both rock types, nor will a simple fixed cost, because the capacity depends on the rock type fed to the process. The alternative is to use a time-consumption model.

To build on the example, assuming the processing plant has 95% availability, there are 8,322 processing hours available in a year ($0.95 \times 365 \times 24$). That means the processing cost is $3,605 per hour ($3 million/8,322). The actual throughput depends on the type of rock type. Soft rock can be processed at a rate of 421 t/h, and hard rock can be processed at the rate of 300 t/h. Another way of stating this is that 1 t of soft rock consumes 0.0024 hours of processing time (about 9 seconds), whereas hard rock consumes about 0.0033 hours of processing time. If processing costs are then applied to the rock based on the time they consume rather than their tonnage, the costs will be modeled correctly, and the throughput will also be modeled correctly. In summary, the time-consumption model for this example is as follows:

- The processing capacity is 8,322 hours.
- The processing cost is $3,605 per hour.
- Soft rock quantity (processing time consumed) is calculated by multiplying the block tonnage by 0.0024, and the hard rock quantity is calculated by multiplying the block tonnage by 0.0033.

Dealing with Tax

Cost modeling for the purposes of optimization is done on a pretax basis. The reason is that optimization software uses simple cash-flow models, whereas a more complicated accrual accounting model must be used to properly model taxation liabilities. This is not to say taxation should be ignored—it is a significant cost. However, it cannot be adequately dealt with in a simple cash-flow model. The solution is to maximize pretax NPV in the optimization software and then to take the pretax cash flows and reanalyze them in an appropriate spreadsheet to calculate tax liabilities and post-tax NPV. The post-tax NPV will, of course, be lower than the pretax NPV, and only projects with positive NPV should proceed. The fact that optimization proceeds on the basis of pretax NPV maximization does not generally present problems because there is a positive relationship between pre- and post-tax NPV—decisions in mine planning that maximize pretax NPV will also (usually) maximize post-tax NPV.

Workshop for Framing

Many techniques are available for value-focused thinking and framing of decisions and analysis. The fundamental concepts are described by Keeney (1992).

One such technique for framing is presented here. The following are two basic parts of the technique:

1. Define the problem/opportunity statement. What is to be achieved?
2. Determine key decisions to be made and the risks and uncertainties to be managed. How is this achieved?

The problem/opportunity statement has already been preempted in this chapter in discussion of the primary objective—to maximize NPV. NPV maximization should feature strongly in most, if not all, open-pit mine planning problem/opportunity statements. Although everyone involved in the mine planning process may think they know and understand the objective of the task, it is worthwhile conducting a workshop in which the key stakeholders state their belief of the objective or objectives of the task. These viewpoints should be collated and a consensus reached on exactly what it is that the team must achieve. The objectives should be made explicit in writing—stating not only what the process seeks to achieve but also a statement of what is in and out of scope for consideration. This exercise clarifies and reinforces for all what the value metric is and focuses all efforts on optimizing that metric within the bounds specified. The following are examples of problem/opportunity statements:

> **Example 3.** "To develop open-pit mine A and maximize NPV. Identifying ore feed from other source mines is not part of this study."
> **Example 4.** "To explore different product mixes by including the low-grade ore up-dip of the current open-pit to maximize NPV. Modifications to the current processing system are not included in this study."

When the problem/opportunity statement is clear, the next task is to determine key decisions to be made, the risks, and the uncertainties to be managed. This part of the exercise is also best carried out in a workshop setting with the key stakeholders participating, and it has the following three steps:

1. **Present and catalog issues.** This step involves reviewing all issues that are considered important and relevant to any of the stakeholders and then cataloguing them as being factual information or known or unknown risks, opportunities, uncertainties, or decisions that need to be made either now or in the future.
2. **Decipher key decisions and generate alternatives.** From the list of decisions in the catalog from Step 1, focus should be drawn to the key decisions that are within the scope of the study and that could have a significant impact on the value metric. Unconstrained thinking or brainstorming should then be conducted regarding all possible alternatives for these key decisions. The alternatives should include known alternatives or choices, as well as new creative ideas. An efficient way of recording this information is in table form, with the key decisions listed as column headings and alternatives for each key decision listed below. An example strategy table is shown in Table 10.2-5.
3. **Develop the hypotheses to be evaluated.** The final step is to design a few hypotheses for the best set of decisions to achieve the objectives. For example, Hypothesis 1 could be to "minimize time to first ore" and Hypothesis 2 could be to "minimize upfront capital investment amount." Each hypothesis is comprised of a choice of an alternative for each key decision consistent with the premise of the hypothesis. Each hypothesis, along with relevant risks, opportunities, and uncertainties, is then further assessed in mine planning. The hypotheses are not a new set of objectives but simply a way of mapping the entire field of possible decision alternatives. Each hypothesis must be measured against achieving the objective set out at the start. Analysis will reveal one hypothesis that most optimally meets the objective. When the preferred hypothesis is found, it can be further refined (e.g., increasing the mine production rate from 6.0 to 6.7 Mt may further increase NPV).

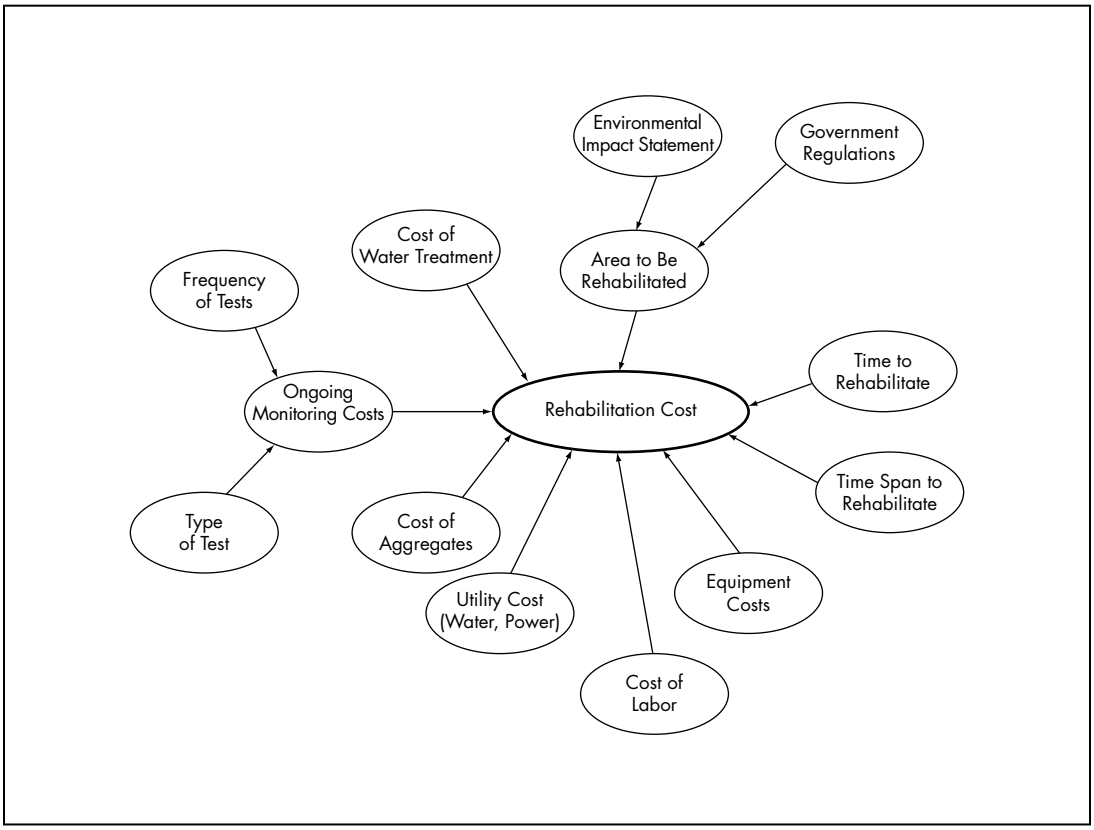


Figure 10.2-23 Influence diagram

Workshop for Building a Model of Highly Uncertain Data

Although most of an individual's education and training drives the person to arrive at a single point estimate for parameters, often the experience with outcomes will show the estimates to be incorrect. With perfect hindsight, a person will look for possible justifications for why the outcome was different from the estimate. Often it will be concluded that because the parameter was dependent on a number of factors, any of which could have played out, the confidence in the single estimate should have been low. In other words, the parameter is highly uncertain. The most common approach for treatment of such uncertain parameters has been to conduct a sensitivity analysis. The parameter is tested for (typically) ±10% variation in value, and the impact on the results is reported. This approach, however, does not consider the possibility that there could be a lot of factors that may drive the value lower than the factors that could drive the value higher (or vice versa). Neither does it answer the question of how much confidence should be placed in the single point estimate as compared to the sensitivities.

This section provides a technique to model parameters that are uncertain. The technique teases out the full range of possible outcomes and also assigns likelihood of occurrence.

The exercise is best conducted in a workshop setting with subject-matter experts participating. It consists of the following steps:

1. **Define the parameter.** The parameter under investigation should first be clearly defined along with the units in which it is to be measured. This is to ensure that no ambiguity arises when using the parameter later. For example, if the parameter is "rehabilitation cost," it should foremost be established (as the case may be) that the cost is in U.S. dollars in real terms; that it is cost per hectare of area rehabilitated and not per volume of material rehabilitated; that it does not include the cost of releveling the waste dump, which is covered under waste dump maintenance cost; and so forth.
2. **Draw influence diagrams.** It is important to recognize and understand what factors, events, risks, and uncertainties could impact the outcome of the parameter. For example, factors that would impact rehabilitation cost are what the rehabilitation involves (e.g., mixing other aggregates with the waste rock, purchasing that aggregate, providing drainage, reestablishing flora, and providing irrigation), the cost of equipment required for the job, fuel costs to run the equipment, time span in which work is to be completed, total area to be rehabilitated at any one time, and so forth. An influence diagram may be used for representing this information in a pictorial format. An example of an influence diagram is shown in Figure 10.2-23.
3. **Explore the maximum and minimum possible values.** The workshop participants should be asked to consider a very unfavorable situation in which all the factors that negatively impact the parameter of interest are present. Participants should be asked to narrate such a situation and work toward exploring the extreme end of the range. The questions posed can include: "How bad can it get?" "Is it remotely possible?" Individual factors on the influence diagram should be assessed, and an overall minimum value derived. A brief narrative should be written

and agreed to. The participants should then be asked to consider a very favorable situation in which all the factors go extremely well. Again, the intention is to explore the extreme end of the range, this time on the optimistic side with questions such as: "How good can it really get?" "Is it remotely possible?" Individual factors on the influence diagrams are assessed, and an overall maximum value is derived. Again, a brief narrative should be written and agreed to.

4. **Revisit the best estimate.** Having completed the exercise of exploring the extreme limits, the participants should then be asked to consider what they think the *best estimate* of the parameter would be after consideration of Step 3.
5. **Explore other point estimates.** The workshop participants should then be asked to estimate a couple of other point estimates: an optimistic estimate and a pessimistic estimate. Again, a brief narrative should be written about how these estimates were determined.
6. **Assign likelihoods.** Likelihoods or confidence levels may be assigned as continuous probability distributions or discrete probabilities of occurrence. Assigning likelihoods is often a partially objective and partially subjective exercise. For parameters that have a lot of information and data available, statistical analysis can be conducted to determine likelihoods. This is then overlaid with subjective information based on the experience and knowledge of the experts. For example, there is a detailed database available for rehabilitation cost in the area based on which statistically determined likelihood is estimated. However, it is anticipated that the environmental regulations will get more stringent with a new incoming government (which may require extra rehabilitation activities). There is also a shortage of labor because of new projects initiated in the neighboring areas, and labor rates are likely to be higher. These pieces of information will mean that there is a likelihood of costs increasing over their statistically determined value.

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CHAPTER 10.3

Mechanical Extraction, Loading, and Hauling

James D. Humphrey and Joshua D. Wagner

This chapter discusses the characteristics and applications of the following common systems used in modern mining operations to extract, load, and haul waste and ore:

- Dragline systems
- Bucket-wheel excavator systems
- Loader and hauler systems
- Other systems: bottom-dump coal hauler, trolley-assist mining truck, wheel-tractor scraper, and in-pit crushing and conveying system

Whether a mining operation must select a new system or optimize an existing system, it is important to consider far more than just cost or discounted cash flow. Different systems have different operating characteristics and risk exposures. These characteristics and the capability to manage risk rather than cost more often separates the best option from the rest.

An initial evaluation of operational requirements will narrow the choice of systems. However, inevitably the project feasibility comes down to a capital- and ownership-cost evaluation by means of one of several discounted-cash-flow methods. Significantly, two key components of that evaluation, production and operating costs, are likely to be estimates, and the sensitivity of the system to operational variables affects the accuracy of these estimates and hence the project feasibility.

In the sections that follow, the following operating variables are discussed as characteristics of each system:

- Mobility: the capability of the system to relocate
- Flexibility: the capability of the system to change direction or work irregular patterns
- Operational range: the variety of application methods in which the system can be used to address requirements for operational change
- Sensitivity to geologic variance: the impact of unexpected changes in material characteristics or unexpected structural or stratigraphic variations

Each of the systems discussed is a well-established system for good reason, and any can be the best choice for a particular application.

DRAGLINE SYSTEMS

Draglines (Figures 10.3-1 and 10.3-2) are self-contained systems that load and transport material to a dump point. They are highly productive, comparatively low in operating costs and labor requirements, and extremely robust, and subsequently have very long lives, commonly 30 to 40 years.

For purposes of comparison, the dragline system is a high-capital-cost, low-operating-cost system that is moderately flexible and can operate through a moderate range of applications with low sensitivity to geologic variance.

Because of their high productivity and capability of direct disposal of material, draglines are favored for area mining in areas of flat-lying tabular geology with high production requirements. The most common application for large draglines is overburden removal in coal mining. However, because they excavate below their working level and exert very low ground-bearing pressure, they are uniquely suitable for digging very wet materials, occasionally even below water.

A large dragline can operate through a range from about 50 m (170 ft) above to 65 m (210 ft) below its working level. This means that, with advanced techniques, the dragline can handle overburden depths of about 80 m (260 ft).

Although the largest dragline ever built had a 170-m^3 (220-yd^3) bucket, for the last few decades the largest draglines built have had 125-m^3 (160-yd^3) buckets. Draglines of this size are capable of moving 30–35 million BCM (bank cubic meters) (40–45 million BCY [bank cubic yards]) per year.

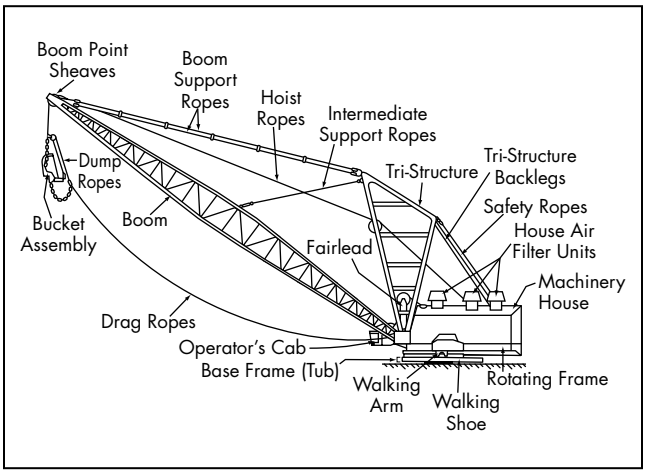
Dragline applications and operations are generally determined by two major factors: the placement of spoil material in the space available and the three basic bucket controls (hoist, drag, and swing). The patterns of tub positions, dig location, digging sequence, and dump locations are best optimized by a pit design that considers spoil placement and the characteristics and interrelationships of the bucket controls. To a lesser extent, the digging peculiarities of the bucket and the bench space required for operations are also considerations for pit design.

In most basic dragline operations, the dragline removes overburden material to uncover ore that is the most recent in

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Figure 10.3-1 Typical dragline



Source: Humphrey 1990.

Figure 10.3-2 Dragline design

a series of parallel adjacent pits. Figure 10.3-3 shows a typical dragline pit. Overburden material from the current pit is placed in the previous adjacent pit, from which product has been removed by auxiliary equipment. Pits are narrow and relatively long:

- Pit widths are most commonly 25–60 m (80–200 ft). Widths for rehandle operations tend be on the large side, to reduce the percentage of rehandle. Width is influenced by the maneuverability of the product-removal equipment, depth of the overburden, blasting method, material characteristics, dragline advance rate, and dragline dump radius.
- Pit lengths vary greatly because of the influence of geology, topography, and artificial obstacles. They are most commonly 1,000–2,000 m (3,000–6,000 ft), although some operations have used pits as short as 300 m (1,000 ft) or as long as 3,000 m (10,000 ft). In shorter pits, the sequencing of product removal and blasting becomes complicated and frequent ramp construction is required. In longer pits, power distribution systems become expensive and complex, and dragline propel distances can be excessive.

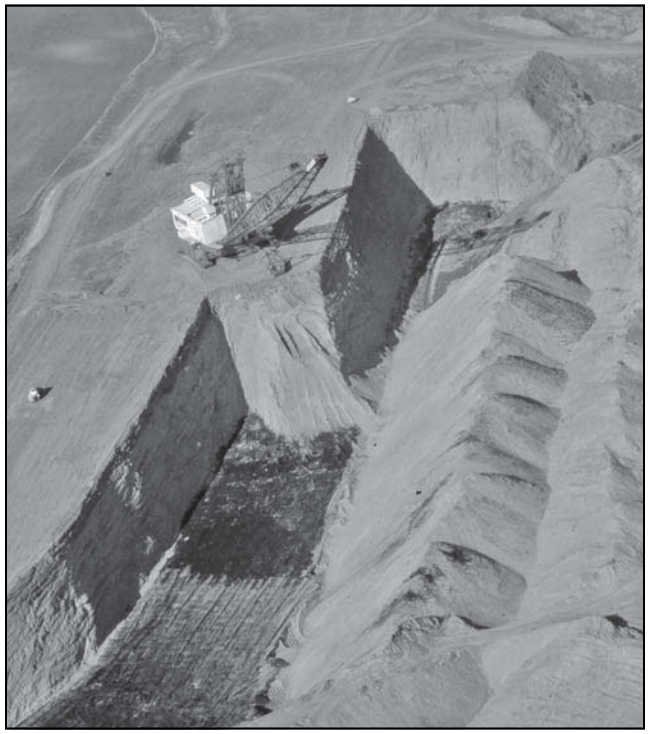
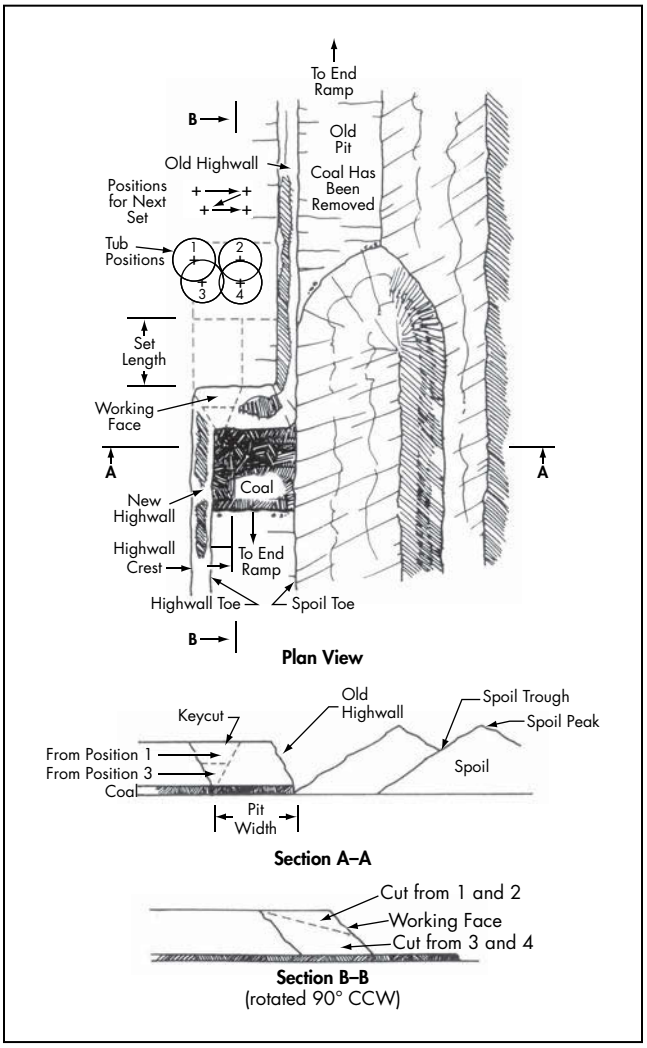


Figure 10.3-3 Typical dragline pit

The pad on which a dragline sits while it works must be clear of hard spots and protruding rocks, and must be relatively level, graded to a slope of ≤2% to provide drainage yet avoid damage due to swing-motor overheating and structural stresses. Modern draglines can propel up and down a ≤10% grade or across a ≤5% grade. When they transition between grades, it is important that they do so gradually, always distributing the load evenly across the tub (the dragline's circular base) and shoes. As a general rule, the rate of grade change should be ≤3% per tub diameter. For example, for a tub 20 m in diameter, the rate of grade change should be ≤3% per 20 m, so a change from 0% to 9% should take at least $9/3 \times 20 = 60$ m. Additionally, when the possibility exists of bridging the shoes, the pad material should be sufficiently compacted to prevent supporting the shoes by the endpoints only.

Draglines are designed to work in soft-underfoot conditions, and as such are designed with tub ground-bearing pressures on the order of 1.2–1.4 kg/cm^2 (17–20 psi). During propel, about 80% of the machine weight is transferred to the shoes, and the remaining weight is carried by the tub edge. This ratio can be changed by carrying the bucket or setting it on the ground. Reducing the tub-edge load by setting the bucket on the ground reduces the probability of pulling a roll under the tub during propel in soft-underfoot conditions.

Rather than remove material from a continuously advancing face as a shovel does, a dragline removes material from a specified length of the pit, called a *set* or *block*. The dragline swings approximately 90° and casts into a pile in the previous pit. Set lengths for larger machines are about 30 m (100 ft), or about 16 steps for the dragline. To remove the overburden from a simple set, a dragline may use from two to four tub positions before retreating to start a new set.



Source: Humphrey 1990.

Figure 10.3-4 Digging positions in a dragline pit

Commonly a dragline follows a pattern of digging positions to excavate a set (Figure 10.3-4). The first two rear positions are set back far enough to ensure that no material is too close to the fairlead to be reached. In shallow pits, these first two positions may suffice to reach the desired depth. In deeper pits, digging may soon reach the point where the drag ropes scour through the crest of the digging face, in which case the dragline must move forward to clear the drag ropes. Thus, from the rear positions, the upper part or *lift* of the set is removed. In the last two positions, the dragline has moved forward to the edge of the digging face to reach down for the lower lift of the set.

The lateral positions in a pit are also significant. From the positions along the highwall (1 and 3 at the bottom of Figure 10.3-4), the dragline can fix the alignment and slope of the new highwall with the *key cut*. This trench-like cut is confined to a bottom width of only a single bucket as it works down. Such a cut allows maximum lateral control of the bucket with minimal lateral strain on the boom. In addition, if the entire dig path is not directly radial to the dragline, production and mechanical availability can suffer. At the bottom of the lift in position 1, the dragline generally moves laterally to position 2. This *plug* position allows the dragline to spoil at maximum range. This move is made before completion of the key to reduce delay caused by hoisting clear of the key before beginning the swing. As excavation progresses, the plug is removed in lifts comprised of a series of cuts to an equal depth (about one-half the bucket height) made in a sweeping pattern. The sweeps normally progress from the spoil side to the key so as to minimize any hoisting required before swinging the bucket to the spoil. Then the dragline moves forward into positions 3 and 4 to excavate the lower lift in much the same fashion.

In Figure 10.3-3, the dragline has stepped back from the face for maintenance but has completed the first three positions of the set. The top of the set has been removed from positions 1 and 2 and the key cut has been finished from position 3. The dragline is now ready to move over closer to the spoil and finish the plug from position 4.

This description, although typical, should be considered general. Set lengths and digging positions will vary depending on operating conditions and machine capabilities.

Dragline Operating Methods

Draglines can operate by means of several operating methods, described in the following paragraphs.

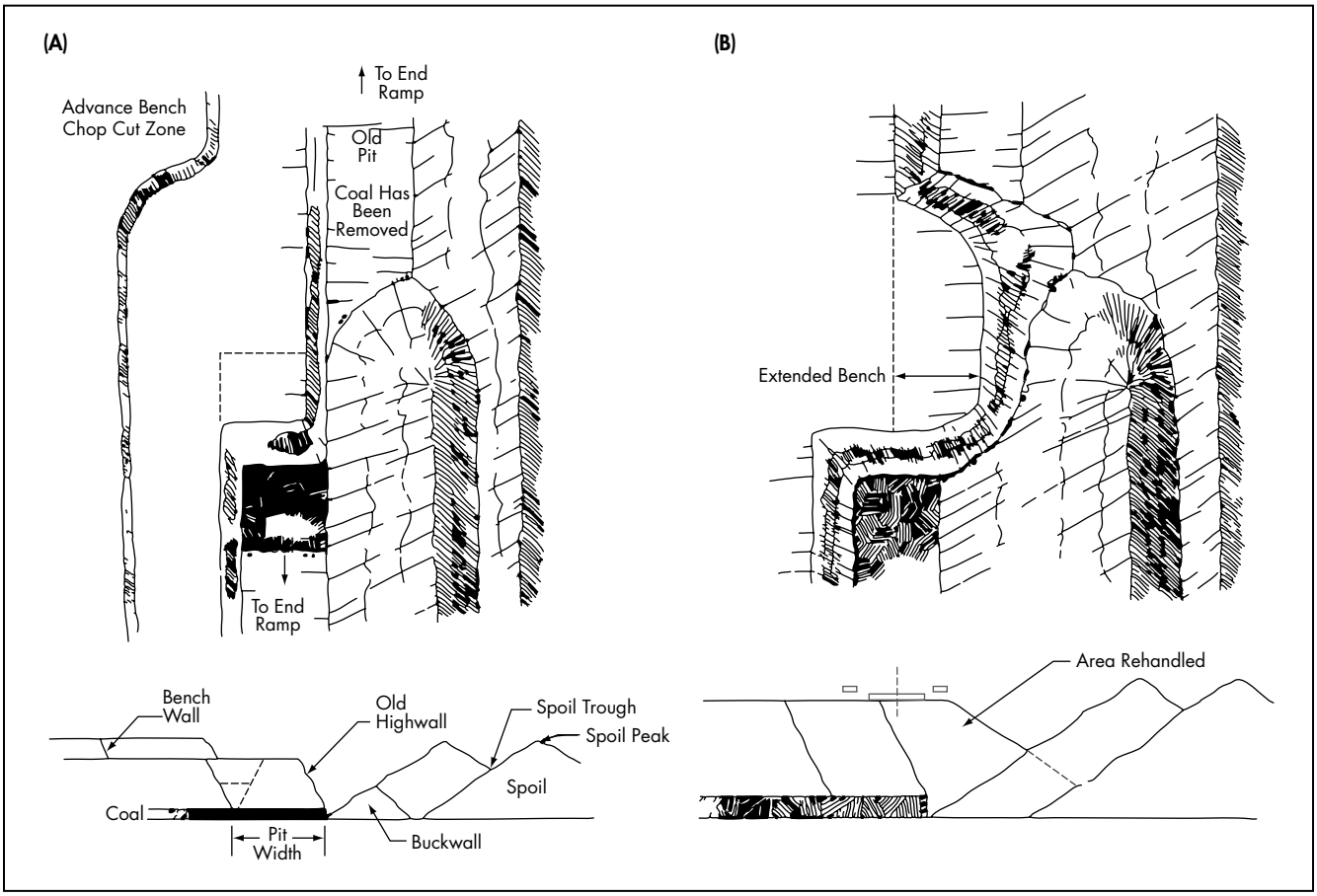
Simple Side Casting

The standard dragline method is simple side casting, used when the dragline has the required reach to move the overburden to its final place. With typical angles of repose and pit widths, the maximum overburden that can be handled by this method is a little less than half the effective radius, discussed later in the "Dragline Selection" section of this chapter. Variations on simple side casting are common.

Advance Benching

Advance benching (Figure 10.3-5A) is useful in areas of uneven terrain or in overburdens where a top layer of unconsolidated material overlays competent rock. The set is split into an upper and lower bench. The lower bench is removed conventionally; the upper bench is removed by *chop cutting*, which typically means digging above the working level but also can mean engaging the bucket at the dump radius. Either way, the bucket is at least partly pulled down a face rather than up or across it. The bucket is usually held in a dump position, teeth down, then lowered onto the face and dragged in. Chop cutting is sometimes used instead of a key cut in spoil-side operations to clean the highwall. However, chop cutting is hard on the rigging, ropes, and bucket, and can increase downtime and repair costs and decrease productivity. Productivity is further decreased by the lower fill factor and increased drag-to-fill time. Advance benching generally requires a longer swing angle as well. Although efficiency varies, typically a reduction of 10%–20% of the conventional rate should be used for initial estimates. When practical, chop cutting should not be done above the height of the fairlead, or else production and maintenance can be significantly impacted.

If the material in the advance bench is extremely unconsolidated, it is sometimes convenient to build a *buckwall* (visible in Figure 10.3-5A) out of dry competent material removed from another area of the set and placed as a retaining wall at the toe of the spoil. The unconsolidated material is then placed and contained behind the buckwall. A buckwall can be used to help stabilize any spoil pile.



Source: Humphrey 1990.

Figure 10.3-5 (A) Advance benching and (B) extended benching

Extended Benching

To extend operations in deeper overburdens, the alternative methods of extended benching and spoil-side benching (discussed in the next subsection) can be used to remove material to a depth of about twice that achievable by simple side casting. However, these methods come at the price of increased rehandle, slower cycles, and more complex planning, although the impact on rehandle and cycle time can be reduced by use of auxiliary equipment. These methods can also be used on a temporary or localized basis around ramps, spoil, or highwall slumps, at high spots in the overburden, or inside curves.

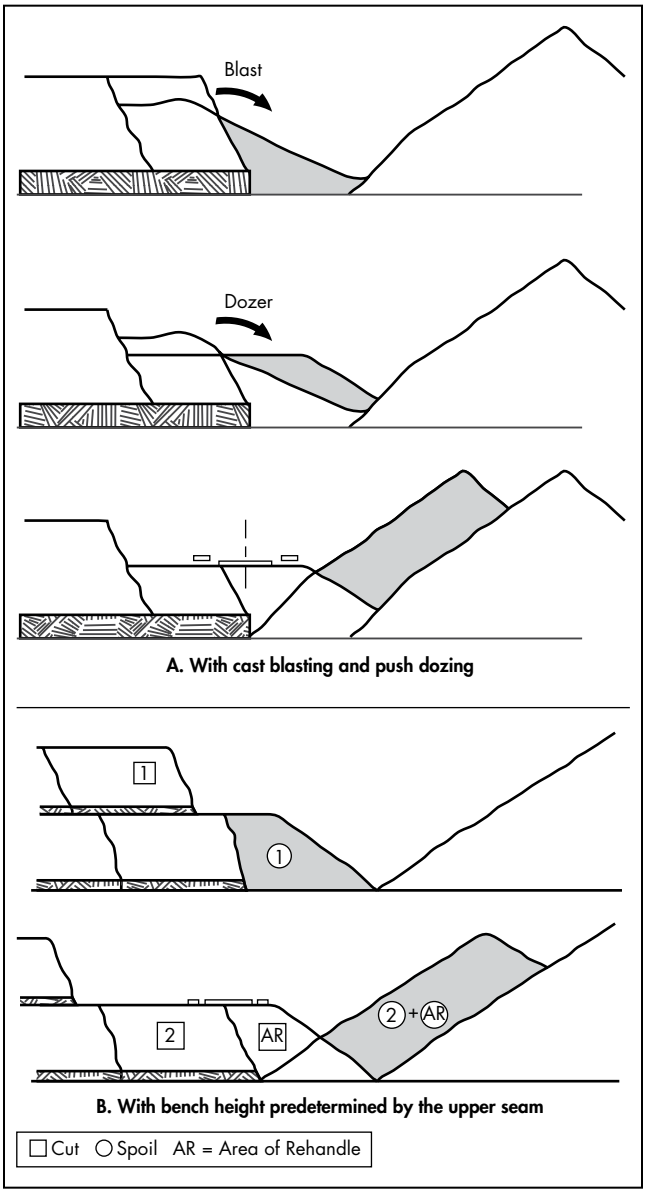
In extended benching (Figure 10.3-5B), the dragline places the driest, most competent material from the set against the old highwall. Enough material is placed so that, after leveling by dozers, it forms a bench. The dragline then moves out onto the bench in a position closer to the spoil. As excavation progresses, the bench is removed. This method can be used in two-seam operations as well. A disadvantage of extended benching is that the swing angle is long, lowering total production.

When calculating production requirements, it is important to remember that the rehandle material in the extended bench is loose material, and therefore has a different bucket factor.

For long-term applications, extended benching is frequently combined with cast blasting and push dozing (Figure 10.3-6A), both of which are effective for moving material short distances downhill. Blasting lowers the bench level, decreasing rehandle; it also moves some material to its final place, increasing production. The cast-blasting profile is then leveled by dozers to form the extended bench.

Bench height and width should be designed to take maximum advantage of dragline hoisting. However, although high hoisting can lengthen cycles, the benefit of a lower bench height with its lower rehandle is usually the determining factor in setting bench height. In multiple-seam operations, the bench height may be predetermined by the upper seam (Figure 10.3-6B).

Two positioning issues often arise in extended benching: cleaning the coal toe and cutting the key. For cleaning the coal toe, because the extended bench covers the coal toe, it is clearly advantageous, especially with thicker seams, to position the dragline just outside the edge of the coal, which is the best position from which to clean the spoil toe away from the coal toe and thus minimize rib loss. However, doing so may require pushing the extended bench out a little farther than is necessary to meet dump requirements. For cutting the key, a lower bench without a setback prevents the dragline from being positioned directly over the key. The dragline can be positioned no closer to the new highwall than the rear-end clearance radius. To keep the highwall clean and well defined, a presplit blast is commonly used. The dragline can be positioned out on the extended bench at a distance from the highwall equivalent to the dump radius and then the key can be chop-cut. Alternatively, the key can be developed by



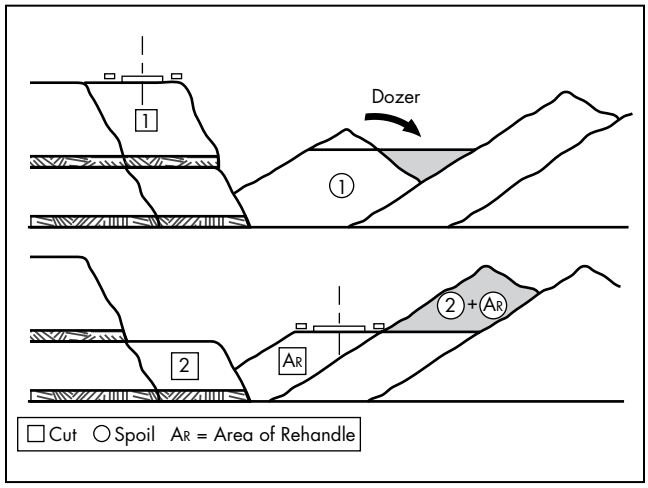
Source: Humphrey 1990.
Figure 10.3-6 Combinations of extended benching

auxiliary equipment, typically dozers but sometimes backhoes or smaller draglines (Ingle and Humphrey 2004). In the latter case, it is critical to consider in advance the finer blasting fragmentation required for auxiliary equipment.

Spoil-Side Benching

In spoil-side benching (Figures 10.3-7 and 10.3-1), also called *pull-back*, overburden is removed in two independent passes. This method is common in two-seam operations and is virtually required in three-seam operations.

On the first pass, from the highwall side, material is moved by standard side casting. Spoil from that pass is allowed to ride up the highwall, and then the peak is leveled to form a pad for the second pass. The first pass is generally completed for an extended length of pit, whereupon the dragline bridges across to the spoil bench. On the second pass, the spoil bench can be removed in either direction, depending on pit



Source: Humphrey 1990.
Figure 10.3-7 Spoil-side benching

sequencing, spoil-bench development, and cable layout. On the spoil side, the dragline is positioned so that the key can be chop-cut with the boom perpendicular to the highwall.

The design height of the spoil bench, which also determines its width, is based on the reach requirement to chop-cut the key on the spoil-side pass, the dig depth, and, if the coal toe must be cleaned, the dump-height limitations of the dragline and tub position. If the first pass does not generate enough material to achieve the necessary spoil-bench height, material can be removed from the spoil-side position and placed one or two sets behind the dragline, much like an extended bench.

It is best to develop the spoil bench with auxiliary equipment; however, the amount of material to be moved may justify assistance from the dragline. And since the bench is developed in advance (the dragline is chopping in the direction of travel), it is better to develop a finish grade for only the road width of the dragline along the spoil edge, leaving the outer edge of the bench for the auxiliary equipment to finish-grade.

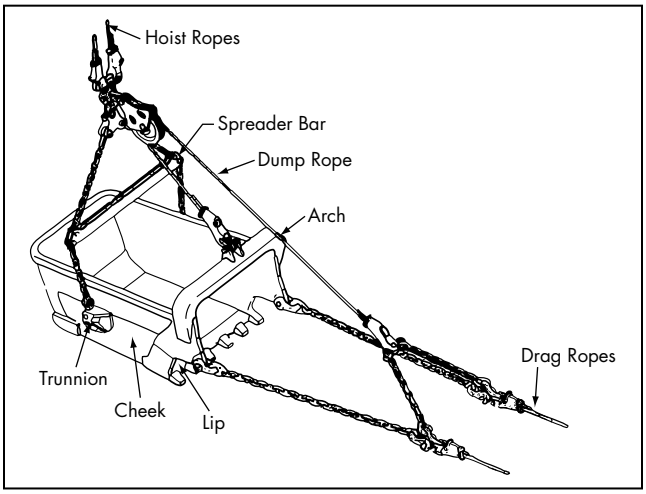
Spoil-side benching requires carefully managed cable moves and layouts, particularly when raising the bench level from the spoil side. With the cable on the bench and the dragline progressing toward the cable, the bench must be raised in halves. This requires moving the cable from side to side and sometimes also swinging over the cable. Swinging over the cable should be done only with a protective covering and carefully controlled operator technique.

Spoil-side benching enables operation of multiple draglines in a pit (Figure 10.3-1). In a fairly short pit, this can afford a very high production rate. However, it is difficult to schedule multiple draglines, primarily because of the complexity of matching the advance rate of draglines working on separate benches. Although production requirements can be proportioned, short-term variability in production rates invariably causes inefficiencies. Very few operations run tandem draglines in this manner for any length of time.

Dragline Production

The amount of material moved by a dragline is determined by the following basic parameters:

- Bucket capacity: how much material is put in the bucket
- Cycle time: how fast the bucket is cycled



Source: Humphrey 1990.
Figure 10.3-8 Dragline bucket rigging

- Operating hours: how many hours per year the dragline is kept digging

Bucket Capacity

Bucket capacity was historically measured in accordance with SAE standard J67 (1998). This standard calls for a struck-top and front-face calculation and then subtracts 10% of the calculated volume to account for the slope of the front face. It also uses estimating factors to assist with the calculations of the complex curves of a typical bucket. However, the final so-called "rated capacity" in no way represents the behavior of material in the bucket. Rather, it simply provides a uniform method for comparing bucket capacities. For calculating production, the rated capacity must be adjusted for material swell and fill characteristics.

The material in a bucket is loose, so the rated capacity of a bucket can be thought of in terms of loose cubic meters (LCM) or loose cubic yards (LCY). Material swell changes with handling and varies within a bucket or pile of material; within a large pile, it also varies with time. Additionally, it can differ dramatically from the swell in a shovel dipper or truck body.

To account for swell and fill, two factors have been introduced:

1. The *swell factor* F_s is affected most noticeably by fragmentation and material composition, but also by bucket design.
2. The *fill factor* F_f, although more complicated to determine, generally has a larger impact on production variability and should be studied carefully. Fill is affected most noticeably by fragmentation and operator technique, but also by bucket-rigging configuration (Figure 10.3-8) and design.

F_s and F_f depend on material characteristics, digging method, rigging configuration, and bucket design; they do not remain constant if any of these variables change. They are thus best determined from field data, optimally by starting with bucket count and block volume, and then—for a particular material, operating method, and rigging setup—calculating the bank volume moved per bucket.

F_s for material in a dragline bucket is typically about 1.3 loose volume per bank volume. F_f is typically about 0.90 of the SAE J67 rated capacity. F_s and F_f can be combined into a single bucket factor, typically about 0.70, meaning that a bucket rated at 100 m^3 typically carries about 70 BCM. The typical distribution of payloads is about ±10%.

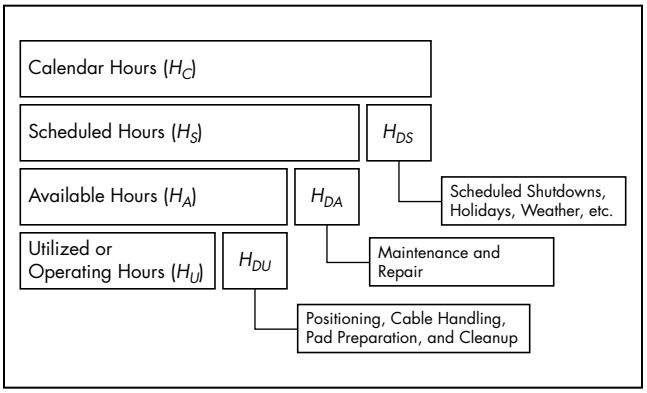
Cycle Time

The digging cycle of a dragline is comprised of five main components: (1) drag to fill, (2) hoist and swing, (3) dump, (4) return swing and lower, and (5) bucket-spot. The time required for each varies depending on a number of factors, most notably dig depth, hoist height, and swing angle. Other variables include material characteristics, dragline performance speeds, and operator proficiency. Because of the diversity of these factors, even machines of the same model and design have different cycle times. Typical designed cycle times for larger machines are in the range 50–60 seconds for a 90° swing with a low dump. A typical cycle is dominated by components 2 and 4; about 70% of a typical cycle is required to get the bucket over to the spoil and back again.

Component 2 of the cycle (hoist and swing) is actually three independent movements: swinging, hoisting, and paying out drag. Each has a specific time requirement. For almost any dump point, one of these movements takes more time than the others. Therefore, two movements are retarded intentionally so that the slower dependent movement has time to coincide at the dump point. However, drag pay speeds are so rapid that they are seldom the dependent movement. It is convenient to think in terms of the curve that the bucket follows at maximum hoist and swing speeds. This swing–hoist coincidental curve represents the points in space that make maximum use of the time available for both functions. Cycles dumped below the curve are *swing dependent*; cycles dumped above the curve are *hoist dependent*. For example, if a particular cycle is a long swing with a short hoist, then it is below the curve and swing dependent. Hoist distance and swing angle can be controlled by pit and digging pattern design, and performance speeds can be affected by operating technique. Thus it becomes obvious that a dragline operator should dump each bucket not at the peak of the spoil but rather near the swing–hoist coincidental curve. Although this is not practical for every cycle, the more coincidental cycles that occur, the more efficient the operation.

Hoist distance and swing angle are minimized by optimizing the bench height and the digging positions of the machine while minimizing the number of relocations (which cause nonproductive propel time). Obviously, machine positioning is limited by the key cut, drag-rope clearance, tail clearance, and reach requirements.

Bucket speed during these independent movements is very dependent on operating technique. Hoist speed is a function of the load in the hoist rope, which is directly related to bucket position. Because hoist acceleration is extremely quick and represents a small proportion of the hoist time, acceleration time does not affect hoist time significantly. To keep the bucket in the carry position, tension must be maintained on the drag rope, which increases the load in the hoist rope. Typically, the hoist load is about 120% of the bucket and payload weight. The closer the bucket is carried to the boom, the greater the hoist load, which ranges from about 110% to 140%. The change in hoist speed is directly proportional to the change in hoist load. So the further out the bucket is carried, the faster it hoists. However, carrying the bucket too close to the dump radius generally causes material to slough off the front of the bucket, reducing the fill factor.



Source: Humphrey 1990.

Figure 10.3-9 Relationship between calendar hours and operating hours

Table 10.3-1 Operating efficiency factor

	Utilization			
Availability	**Excellent**	**Good**	**Fair**	**Poor**
Excellent	0.83	0.77	0.70	0.65
Good	0.78	0.72	0.66	0.61
Fair	0.74	0.68	0.62	0.58
Poor	0.69	0.64	0.59	0.54

In contrast, swing time is very dependent on acceleration. Acceleration to full speed and deceleration to stop requires about 60° of the swing, which is about 85% of the swing time for a swing angle of 90°. Acceleration and deceleration are relative to the rotational inertia, which is a function of the mass times the square of the radius of the center of mass, so it is greatly affected by bucket location. On longer swings, where hoist is not a factor, keeping the bucket in tight as long as possible improves swing time.

Of bucket capacity, cycle time, and operating hours, inefficiency in cycle time is the most difficult to diagnose and improve.

Operating Hours

Ostensibly the easiest performance parameter to measure is time. In reality, there are more ways to categorize delays and define losses than there are mines around the world. The ubiquitous "hour" is possibly the single most misleading term used in mining. The basic goal is to reduce the number of hours we have to work with down to the number of hours actually worked. Clearly understood definitions are critical to production reporting and estimating.

Figure 10.3-9 shows the relationship between calendar hours and operating hours. Although there are about 8,760 calendar hours in a year, draglines typically operate for 6,000–7,000 hours per year. For initial estimating, it is adequate to combine availability and utilization values into a single operating efficiency factor (Table 10.3-1). Typical availability and utilization values are each about 85%, which provides an operating efficiency of about 72% of the scheduled hours. Availability and particularly utilization can be affected significantly by application. The same dragline on a highwall pass will see a significant difference in operating efficiency on a spoil-side pass where it is chop-cutting and pad-building, typically due to increased propel time, decreased dump-rope life, increased bucket maintenance, and so on.

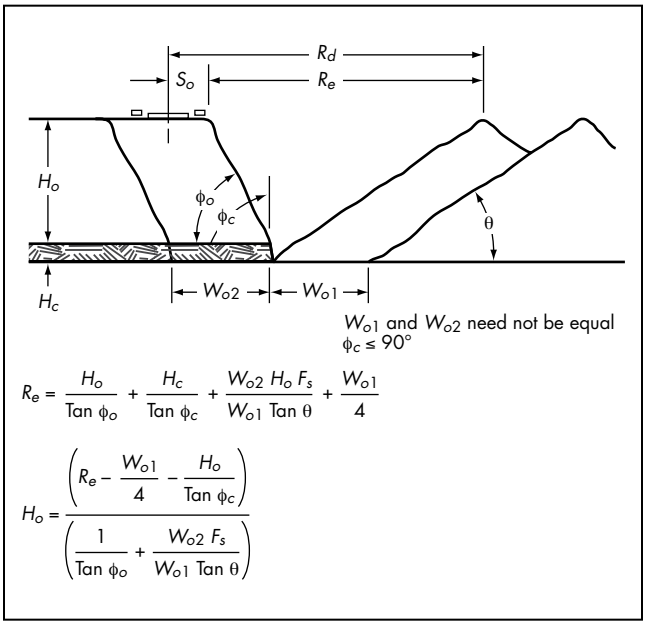
Apart from the obvious minimization of maintenance and utilization delays, for maximum production it is key to use the available hours efficiently. Most dragline operations can realize significant production gains by reducing cycles that do not contribute directly to ore production. The most significant contributors to inefficient operation are rehandle, nonproductive cycles, and poor planning. To address these contributors, the following are important:

- **Effective management of rehandle.** The use of auxiliary equipment and careful planning are the best tools to manage rehandle. The cost of rehandle should be thought of in terms of incremental costs of ore production, rather than the difference in cost per unit of production for dragline operation compared to that for auxiliary system operation. In addition, in certain applications, a slight increase in rehandle (e.g., changing a bench level or pushing out an extended bench) can improve production due to the effect on cycle and propel time.
- **Productive digging.** Nonproductive cycles are often difficult to measure but can decrease production significantly. There probably isn't an operation at work today that couldn't reduce the amount of time the dragline does pad preparation and cleanup work by better use of auxiliary equipment. Draglines should not waste time heeling the bucket to level the pad or pushing down the roll. In addition, efficient coordination of the ground crew during cable layout and pad preparation can minimize move times.
- **Quality operation planning and coordination.** The amount of detailed planning required to operate a dragline most efficiently should not be underestimated, particularly for nonroutine digging. Specific tub positions, bench levels, and material placement need engineered plans. It is important to involve dragline crews in planning to ensure their understanding and consensus. In addition, it is a mistake to assume that everyone can easily create, from a two-dimensional drawing, a three-dimensional (3-D) operation. Rather, it is wise to use 3-D software or sandbox models to work through operations in advance, especially complex operations such as ramp crossings. In complex operations, the use of "playbooks," with a diagram for every tub position to describe the digging and dumping points, can be extremely valuable.

Dragline Selection

An important concept to keep in mind when sizing or selecting a dragline is to select the dragline for the mine plan, not the mine plan for the dragline. Draglines are engineered systems, generally customized for an application; even when purchased used, they can be modified during reassembly to better fit an application. The two major parameters used to select a dragline are dump radius and allowable load. Occasionally, other parameters such as ground-bearing pressure of the tub, rear-end swing clearance, dump height, and dig depth may also affect selection.

Dump radius R_d is the horizontal distance from a machine's center of rotation to the hoist rope when the bucket is vertically suspended. Part of this radius is consumed by the stand-off distance S_o (Figure 10.3-10), which is the distance from the center of rotation to the crest of the old highwall. The



Source: Humphrey 1990.

Figure 10.3-10 Dragline geometry

remaining dump radius is the effective radius R_e; thus $R_d - S_o = R_e$. Stand-off distance varies depending on machine size, operational history, and overburden conditions. In the absence of field data, for planning purposes the minimum stand-off distance is commonly considered to be 50% of the width of the dragline from the outside of the shoes or 75% of the tub diameter.

The allowable load, sometimes called the maximum suspended load or rated suspended load, is the maximum weight of bucket, rigging, and material for which the dragline is designed to provide optimum performance. Standard-duty buckets including rigging weigh about 1.2 t (metric tons) per rated m^3 (2,000 lb per rated yd^3) of capacity, although they may vary from 1.1 to 1.4 t (1,800 to 2,300 lb), depending on the application. Overburden densities are site specific, but 1.8 t/LCM (3,000 lb/LCY) is a common approximation. The combined weight of a standard bucket and material load is then about 3.0 t per rated m^3 (5,000 lb per rated yd^3). Therefore, an operation that requires a dragline with a 46-m^3 (60-yd^3) bucket requires the dragline to have about a 138-t (300,000-lb) allowable load. Allowable load is calculated for a 100% full bucket (peak bucket load), even though the average fill used for production calculations is less (e.g., 90%).

For a given dragline model, or more correctly a given frame size, the allowable load can be varied by changing the dump radius. Basically, a shorter reach means a larger bucket. This design change is limited by the hoist load that the gearing and motors can handle (maximum allowable load) or, conversely, the longest boom that the frame can handle. The general rule is: the longer the reach, the deeper the depth that can be handled without rehandle and the less rehandle. However, the best choice is almost always the largest bucket at a shorter reach, despite the higher rehandle. The larger bucket more than makes up for the additional rehandle with additional production.

By calculating bucket requirements using a standard range of production values—8,400 scheduled hours, 60-second cycles, 72% operating efficiency, and a 70% bucket factor—it can be calculated that a dragline produces about 250,000 BCM/yr per m^3 (250,000 BCY/yr per yd^3) of rated bucket capacity. This unit of production per unit of bucket capacity is a convenient general factor called the production factor or digging index (Humphrey 1990), and can be used to quickly estimate the annual capacity of a given bucket, or conversely the bucket required for a given production.

The production factor is commonly used to compare differences in dragline operations, either one application vs. another or one dragline vs. another. With historical data, production factors for specific applications can be developed over time and used for planning and forecasting purposes. Operating mines commonly have production factors of 200,000–300,000, depending on application, cycle time, and efficiency. Production factors are often calculated on an hourly basis, with care given, of course, to which kind of hour is used. The hourly production factor can be useful for measuring the efficiency of a specific operation within an application (Kennedy 1990).

The application of production factor presumes that overburden production requirements are known and have been adjusted for any ore losses in the pit and plant and any rehandle expected for the scenario.

Caution should be used with volumetric terminology. Some operations, mostly outside of North America, report production in terms of *prime* (also called *virgin* or *in-situ*) volumes but label them "bank" volumes. The prime volume is the actual volume of overburden above the coal that was uncovered and does not include rehandle. In fact, most mines typically experience 5%–10% of additional operation rehandle for ramps, bench fill, and the like. A production factor based on prime volume is thus smaller than can be expected in actual operation. Prime volume is adequate for comparing machines using the same method at the same mine; however, it is not a true measure of individual dragline productivity. It is advisable to use the terminology *total (including rehandle) bank cubic meters* (TBCM) and *prime (not including rehandle) bank cubic meters* (PBCM), or for cubic yards, TBCY and PBCY.

Draglines as Loaders

Although draglines are normally used to direct-cast material, they are also sometimes used as loaders. In that case, the normally imprecise dumping technique of the dragline must be altered to something as precise as is used when spotting the bucket in the dig. Most experienced operators have little difficulty adapting to point dumping, and the method has been used to load hoppers, trucks, and barges. Generally, it is most useful when the pit bottom is unsuitable for truck traffic and too deep for a hydraulic excavator. For example, it is common in Florida phosphates for a dragline to mine the phosphate matrix and dump it into a slurry sump on the highwall. The sump is about the size of a large mining truck body.

Contrary to the initial mental image of this method, the bucket does not swing over the target. Rather, the swing is stopped with the bucket in the carry position and the drag is payed out to dump. Trucks are positioned facing away from the dragline so that the bucket enters over the tailgate. The



Figure 10.3-11 Typical bucket-wheel excavator

precision required to determine the truck position (i.e., to spot the truck) is no different than is required for any other loading tool. However, because the spot cannot be marked by the bucket as in shovel operation, position markers may have to be provided. As the dump target is generally on the highwall side, it is easy to locate the target so that the dragline has a very short swing angle and a short cycle time.

To reduce the dump distance and place the material in a smaller area, the dump ropes can be shortened. This has the adverse effect of increasing the load in the dump rope and decreasing the dump-rope life. Alternatively, the dump target can be elevated (such as to an advance-bench level) to shorten the dump distance. Of course, elevating the dump target much above the operator's line of sight compromises hoist time and visibility.

Typical dragline operations are viewable by satellite-image software in the vicinity of these coordinates:

- Simple side casting in a lignite mine: 47° 5.19′ N 101° 19.63′ W
- Cast blasting in a thick-seam coal mine: 43° 41.36′ N 105° 18.80′ W
- Cast blasting with spoil side in a multiple-seam coal mine: 21° 28.85′ S 148° 22.60′ E

BUCKET-WHEEL EXCAVATOR SYSTEMS

The bucket-wheel excavator (BWE) (Figure 10.3-11) is one of the grand machines of the mining industry and traces its origins to drawings by Leonardo da Vinci. The original concept from the late 1800s was technologically challenged by advances in the steam shovel, had its practical beginnings in the early 1900s, but had its first real mining applications in German lignite mines during World War I (Rasper 1975).

For purposes of comparison, the BWE system is a high-capital-cost, low-operating-cost system that has limited flexibility and can operate through a limited range of applications with sensitivity to geologic variance.

BWEs are highly customized and vary in design more than do any other mining machines, to the extent that nearly every machine is not just unique but almost dissimilar. The machines are very robust in design and consequently very long lived. Their most common mining applications are mining unconsolidated overburdens and lignite, handling bulk materials such as stockpiles and load-out facilities, and heap leaching pad construction and removal.

Three major types of BWE exist in mining operations:

1. Systems that direct-feed into a shiftable conveyor system that connects to a series of other conveyors and a discharge system; these machines can weigh up to about 12,500 t (14,000 st) and cut material from a bank height of more than 50 m (160 ft) at rates of more than 10,000 m^3/h (13,000 yd^3/h)
2. Systems with long discharge booms that direct-place material into the spoil
3. Compact systems, generally two crawler machines used for bench heights of about 10–15 m (30–50 ft) with production rates of 1,000–2,000 m^3/h (1,300–2,600 yd^3/h)

When the BWE is used to reclaim stockpiles for rapid loading of trains and ships, it is often rail-mounted. Because the shape of the stockpiles is known, this application is ideal for semiautonomous control.

The numbering system for BWE models is probably the most logical of all mining-equipment numbering systems. The format of a typical German BWE model number is (ThyssenKrupp Fördertechnik 2005)

$$\text{Sch Rs}\frac{1{,}600}{2.5}-28\times 12$$

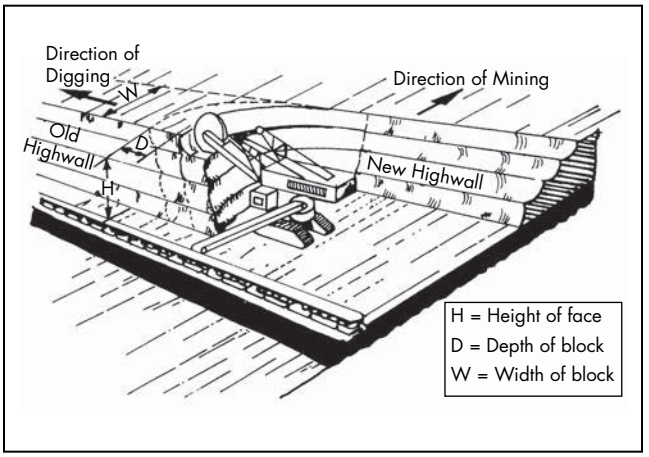
where

Sch = *Schaufelradbagger* (or simply *bagger*), the German name for the BWE; German manufacturers thus lead their model numbers with S, SH, SR, or Sch
R = *auf Raupen*, meaning that the system is on crawlers
s = *schwenkbar*, meaning slewable
1,600 = bucket size, L
2.5 = cutting depth, m
28 = cutting height, m
12 = crowd length, m

Other countries may order the components differently, as, for example, in the Russian ER $1{,}600\frac{28}{2.5}\times 12$. Compact or nonstandard BWEs generally have simpler model numbers, such as C500, which is a compact BWE with a 500-L bucket.

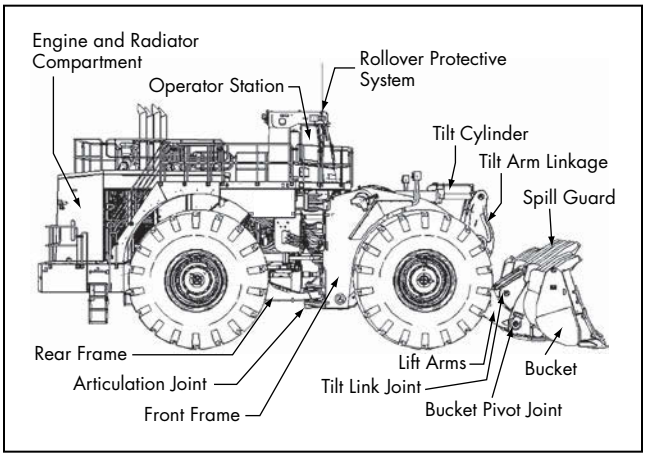
BWE productivity is highly specific to design and application. Cutting forces, wheel-rotation speed, number of buckets, slewing speed, and material characteristics all affect the production rate. As a (very rough) general rule, the peak capacity in cubic meters per hour is from 3 to 6 but usually about 4 times the capacity of a single bucket in liters. So, for example, the German "Sch Rs" example above has a bucket size of 1,600 L so would have a capacity of about 4,900 m^3/h (6,400 yd^3/h), where hours are operating hours, not scheduled hours (for more information, see the "Operating Hours" section).

BWEs are used primarily for unconsolidated materials. Even the largest-capacity BWE has a relatively small bucket, ≤5 m^3 (3.5 yd^3), and so has relatively limited capacity for rocky materials. Additionally, because of the associated conveyors, BWEs require linear, flat-floored mining faces that advance in straight or radial patterns. Thus their main application has historically been large lignite mines, although their largest U.S. application was in the 1960s and 1970s in the Illinois (United States) coal basin where they were used to direct-spoil glacial till covering the sedimentary overburdens.



Source: Atkinson 1992.

Figure 10.3-12 Bucket-wheel excavator used for block digging a lateral terrace cut



Courtesy of Caterpillar, Inc.

Figure 10.3-13 Wheel loader (smallest-capacity bucket)

BWEs can operate in linear pits with a spoiling process operating in parallel to fill the void a short distance away, or in multiple benches in deep pit mines with outside spoil dumps. Pit-face lengths vary from 800 to 4,000 m (2,500 to 13,000 ft), but are usually 1,500 to 2,500 m (5,000 to 8,000 ft). The BWE advances along the face, taking a cut width consistent with its design, up to 90 m (300 ft). The digging action is either a lateral terrace cut (Figure 10.3-12) or a vertical dropping cut, depending on machine design. At the end of the pit, the BWE can cut perpendicularly to the face conveyor for a short distance by means of its slewable discharge boom and or a belt wagon. This plunge cut enables the BWE to establish a face to work in the opposite direction, eliminating the need to deadhead.

Crawler systems for BWEs have as many as 12 crawler sets, commonly set up in an asymmetric arrangement with independent steering to enable larger machines to steer. Systems for use in soft-underfoot conditions can exert ground-bearing pressures much lower than for shovels, about 1.0–1.5 kg/cm^2 (14–21 psi). Tramming speeds are 4–12 m/min (13–40 ft/min) and a typical turning radius for a larger machine is 50–100 m (150–300 ft).

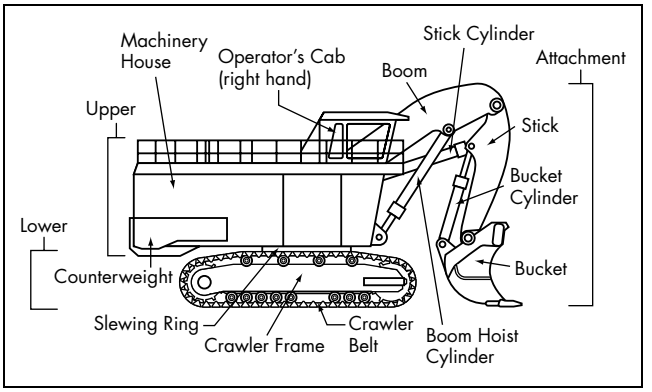
Typical BWE operations are viewable by satellite-image software in the vicinity of these coordinates:

- Compact BWE with multiple benches in advance: 39° 46′ 26″ N 111° 15′ 25″ E
- Large BWE prestripping in advance of dragline: 26° 43′ 14″ S 27° 57′ 37″ E
- Large BWE in large open pit: 50° 54′ 38″ N 6° 30′ 17″ E

LOADER AND HAULER SYSTEMS

More material is moved by loaders and truck haulers than by all other excavation systems combined. The deciding factors in the selection of this system are typically the qualitative characteristics of flexibility and the probability of achieving production and cost targets.

Loaders and truck haulers excel in flexibility. They are not dimensionally constrained by operating method, and so are able to move in any direction for any distance. They can thus work in constrained or irregular geology and terrain, and can be added incrementally, both of which make them virtually the only choice for use in very deep pit mines. Their flexibility also enables a mining operation to adapt quickly to changes in commodity prices, geology, and other influences that cause the original mine plan to change, as it inevitably does.



Courtesy of P&H Mining Equipment, Inc.

Figure 10.3-14 Hydraulic shovel (middle-capacity bucket)

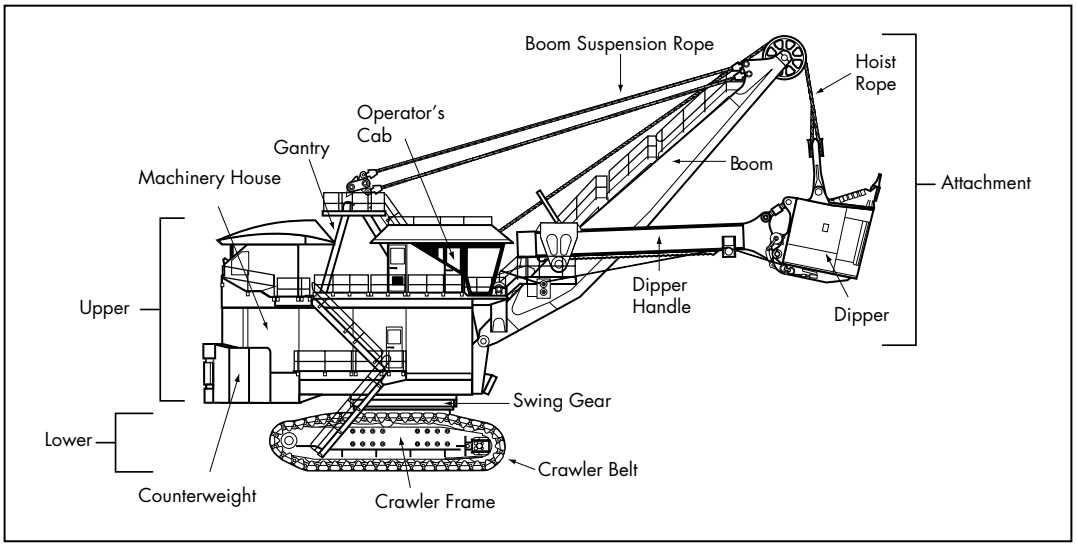
Because haulers always operate in parallel—and, in large systems, loaders do so as well—the impact of adverse performance on the part of any individual unit is minimized. Additionally, the system can operate through a large range of geologies and climates. This all provides for a dependable system with little unpredictable variation in efficiency, production, or cost.

For purposes of comparison, the loader and hauler system is a low-capital-cost, high-operating-cost system that is very flexible and can operate through a broad range of applications with low sensitivity to geologic variance.

Loading Tools

Loading tools are a specific class of excavator that depend on a separate independent haulage system. The most common types of loading tool, from small to large capacity, are wheel loader (Figure 10.3-13), hydraulic shovel (Figure 10.3-14), and mining shovel (Figure 10.3-15). In mining they are used in conjunction with haulers, most often off-highway mining trucks.

In the past 30 years, and particularly the past 10 years, the development of larger, more reliable wheel loaders and



Courtesy of P&H Mining Equipment, Inc.

Figure 10.3-15 Mining shovel (largest-capacity bucket)

hydraulic shovels has encroached into what was previously the exclusive territory of mining shovels: the loading of large off-highway trucks. These trucks, also called large mining trucks, generally have capacities of >135 t (150 st). The growth in wheel loaders and hydraulic shovels has lead to a new delineation in the loader market, with wheel loaders predominant at the lower end, hydraulic shovels in the middle range, and mining shovels at the upper end of bucket capacity (Figure 10.3-16).

Selection

Because all loading tools perform the same basic function (i.e., they load trucks), the differences among them lie in other characteristics. Likewise, due to design differences and resulting differences in capital costs and operating costs per unit of production, evaluations of net present value can prove useless. Deciding factors are more likely the qualitative characteristics of capacity, mobility, flexibility, life, and support requirements.

Capacity. The range in capacity of the three types of loading tool is a differentiating characteristic. For operation with large mining trucks, their capacities are as follows:

- Wheel loader: 27–45 t (30–50 st)
- Hydraulic shovel: 27–81 t (30–90 st)
- Mining shovel: 54–110 t (60–120 st)

These differences in capacity spread even further when compared on an annual production basis, because of cycle-time and operating-hour differences. Generally, when comparing machines of similar size, the mining shovel cycles more times per year than does a hydraulic shovel, which in turn cycles more times than does a wheel loader. This is due primarily to differences in operating hours but also, to a degree, in cycle time. Production is most heavily influenced by the degree of utilization (i.e., the extent to which a tool is kept in use when it is mechanically available). It is also influenced by consistency and efficiency of application, with problems arising when the pit layout is poor, resulting in long swing angles, excessive moves, and other workarounds. Table 10.3-2 shows the influence of swing angle on cycle time and hence production level of a loading tool.

The optimal production factors for loading tools are approximately as follows (the number in units of BCY per year per cubic yard is the same as the number in units of BCM per year per cubic meter; production factors are discussed in the "Dragline Selection" section):

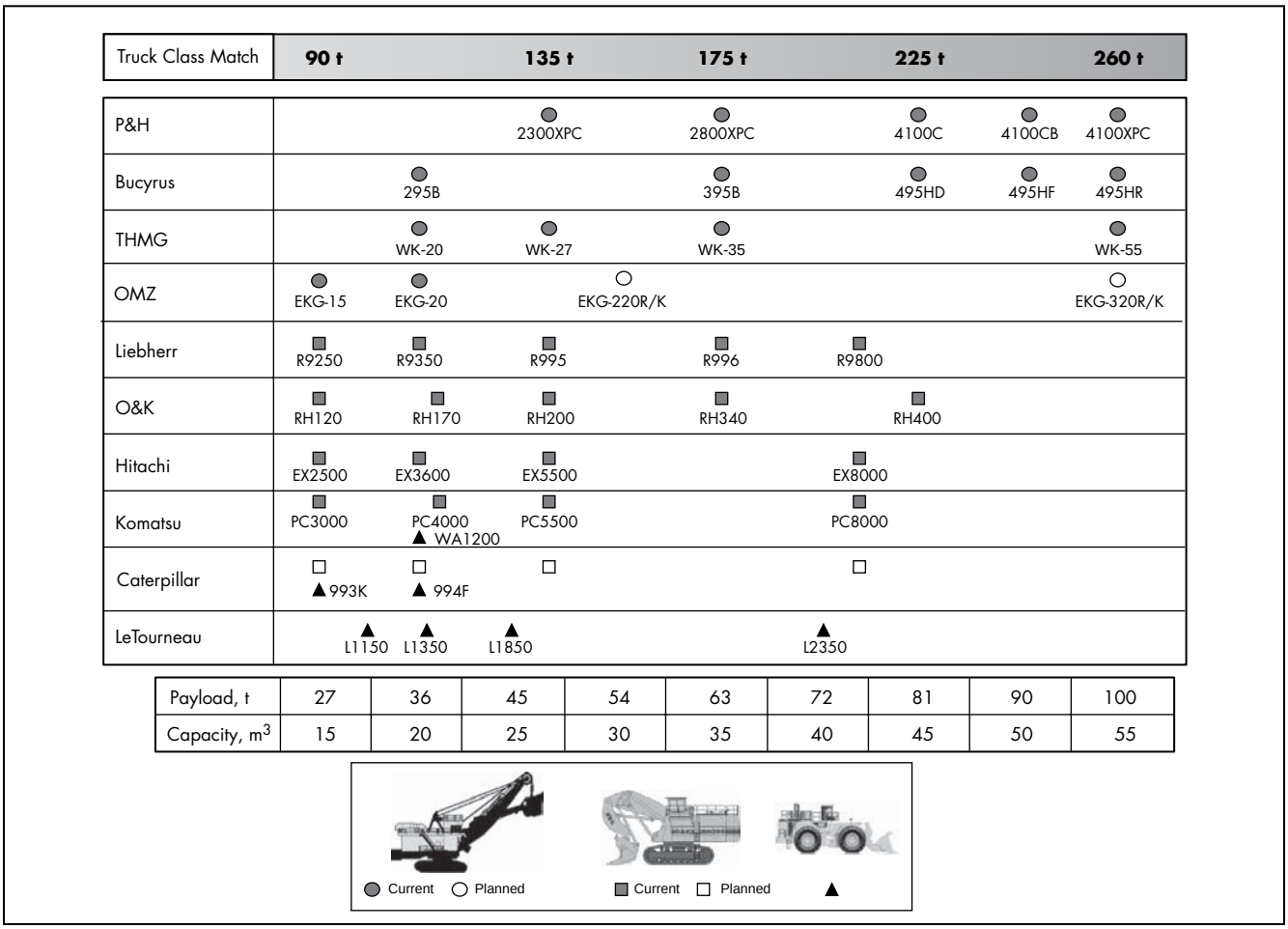
- Wheel loader: 330,000 BCM/yr per m^3 of dipper capacity
- Hydraulic shovel: 350,000 BCM/yr per m^3 of dipper capacity
- Mining shovel: 400,000 BCM/yr per m^3 of dipper capacity

Mobility. If mobility is critical to an operation, the best choice is usually a wheel loader. In operations having multiple faces that require frequent relocations or want a backup unit for multiple loaders, the wheel loader is uniquely capable of rapid relocation. However, recent developments in larger low-boy or float transporters has extended the capability of medium to large hydraulic shovels to rapidly relocate.

Flexibility. The capability to work in faces of different heights or to dig at different levels of a face is an advantage. Wheel loaders are most productive at face heights of at least three times the bucket height, although at shorter face heights they can drive forward through unconsolidated or well-broken material with only slight impact on productivity. In less-consolidated material, hydraulic shovels can penetrate a bank at different levels to separate material at the face with only slight impact on productivity.

Most hydraulic shovels can also be configured as backhoes or mass excavators for digging below grade and loading trucks below or at operating level. Loading trucks below operating level allows use of a number of spotting techniques and shortens cycle times. Loading trucks at operating level, although slower, is obviously desirable when the pit bottom is wet. Backhoes are generally limited to use where face heights are about equal to the stick length.

Mining shovels require a face height of about 50% of their point sheave height, basically about the height of the teeth when the dipper stick is horizontal. However, they can operate at higher bench heights than can wheel loaders and hydraulic shovels, which reduces other operating costs. A higher face



Courtesy of Caterpillar, Inc.

Figure 10.3-16 Loading tool industry offerings by manufacturer and class

Table 10.3-2 Impact of swing angle on production level for a loading tool

Swing Angle, degrees	Impact on Production, %
45	126
50	116
60	107
70	100
100	88
130	77
180	70

Source: Adapted from P&H Mining Equipment, Inc. 2006.

means fewer benches, fewer relocations, and lower drill-and-blast costs.

Life. Economic lifetimes for loading tools are generally as follows:

- Wheel loader: 5 to 7 years
- Hydraulic shovel: 7 to 10 years
- Mining shovel: ≥15 years

Of course, with enough replacement components, the life of any unit can be extended. Longer life is arguably a desirable feature: it is certainly necessary to justify a higher capital cost, but a system with a shorter life but lower operating costs and high resale value can be an equally good or better choice.

Support requirements. Several factors affect support requirements for a loading tool:

- **Drive system.** Large mining shovels are currently available only with electric drives. Wheel loaders and hydraulic shovels generally have diesel drives, although very few wheel loaders and some hydraulic shovels are optionally available with electric drives. Electric drives have lower and more consistent operating costs but require in-pit electrical reticulation systems comprised of electrical substations and power distribution cables involving specialized support equipment, personnel, planning, and operations. Diesel drives require fuel transport for refueling in the field but the equipment involved is usually common to mining operations using large equipment.
- **Digging profile.** Wheel loaders and hydraulic shovels, more so than mining shovels, can flat-pass and thus require minimal cleanup assistance. However, mining shovels, with their greater reach, can stand back farther from the face and so keep the truck back farther from the toe—although one could argue that this class of loading tool should not waste time doing cleanup work that is better left to auxiliary equipment.

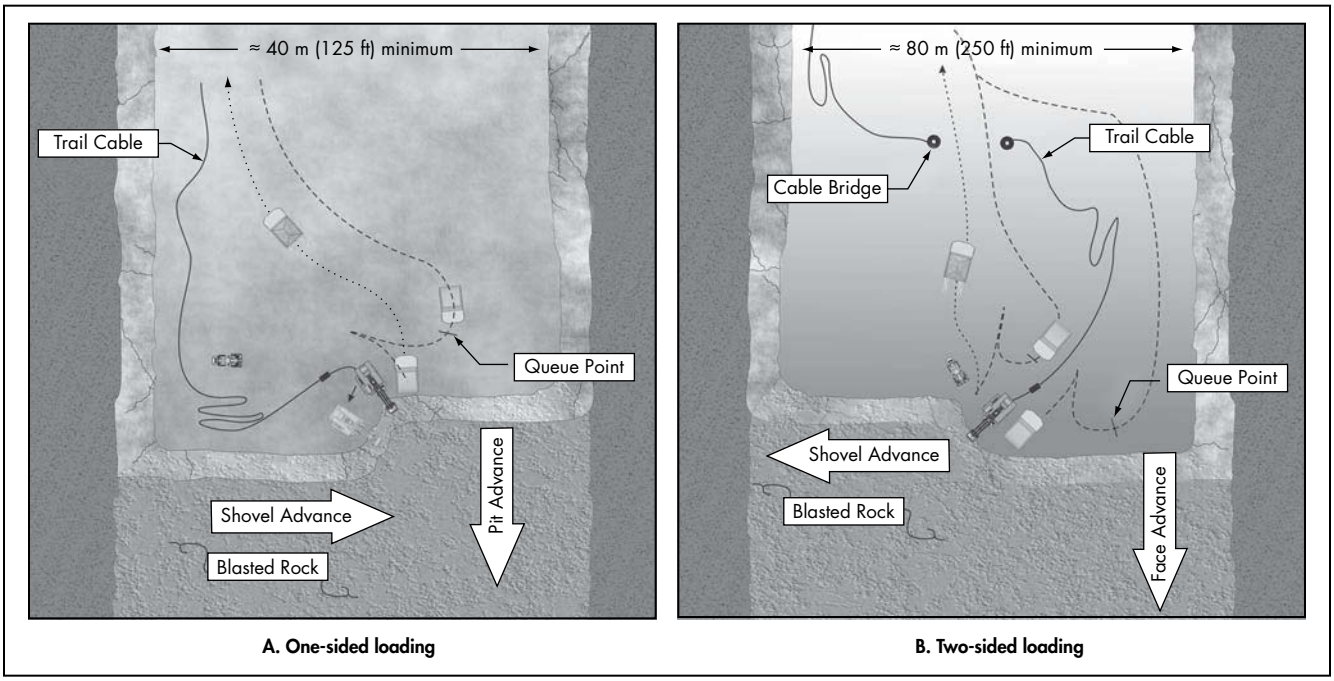


Figure 10.3-17 Loading-tool operating methods

- **Maintenance.** Larger hydraulic excavators and mining shovels have limited mobility and transportability, so all maintenance on these machines must be conducted in the field.
- **Material conditions.** The condition of the floor and bank affects the different types of loading tool differently. A wet or soft floor causes traction problems for wheel loaders, which can significantly impact tire costs and productivity. For soft floors, track shoes can lower the ground-bearing pressure exerted by a shovel. For very soft floors, extra-wide track shoes can minimize the ground-bearing pressure, although they reduce maneuverability due to the increased turning forces required.

Operating Methods

The most common loading-tool operating method is one-sided loading (Figure 10.3-17A), also called single-sided loading. This method requires minimal pit-support coordination and minimizes variations in traffic patterns and truck movements, the latter of which often significantly influences safety. The method also has a relatively small footprint, so it can easily be implemented in benches only 30–40 m (100–125 ft) wide. In the most common variation of the method, the truck stops or queues in a position where it can observe the loading operation, which allows the truck operator to ensure that the area is clear after the previous truck has been loaded and moved off. The location of the queuing and reversing points is generally left up to the truck operator, as it changes so frequently. However, choice of location can significantly influence cycle time, so training on efficient techniques is important. Road constrictions, pit obstacles, poor spot selection, cleanup, and cable handling are common causes of unnecessary delay. Traffic patterns are best designed to allow the truck to reverse with the shovel on the truck operator's left and thus always visible to the operator either directly or in the left side mirror.

One-sided loading has several disadvantages. Mining shovels and hydraulic shovels cycle in 30–35 seconds and spotting takes 45–60 seconds, so the shovel must wait for the truck, reducing production levels. In addition, the time required for a cleanup dozer to work in the spot area can delay operation.

These disadvantages can be addressed to some extent by two-sided loading (Figure 10.3-17B), also called double-sided loading, which reduces the shovel delay between trucks. Although this method appears to be just a doubled-up single-sided loading method, it actually adds some complexity. The road circuit is more complex because it requires a Y intersection, a much larger working face, up to three queuing points, and twice the cleanup and road-maintenance support. Electric-powered loaders also require a cable bridge and have more confined turnaround areas. All of these factors dictate a larger footprint for the loading area, so the method is more suitable for benches at least 80 m (250 ft) wide. Furthermore, near the ends of the working face, the work area becomes restricted and the shovel generally must revert to single-sided loading, so two-sided loading favors wide working faces. Two-sided loading can increase production by 5%–10%, with the general caveat that the more passes per truck, the less the benefit, since improvements come from reducing time between trucks.

Wheel loaders spot trucks somewhat differently than do the other types of loading tool (Figure 10.3-18). From a wheel loader, trucks are best spotted at a 45° angle, which allows the loader to approach the face and the truck at right angles with minimal turning between the two. A good setup requires less than one tire rotation of travel distance in each direction to load the truck.

A less-common method, drive-by loading (Figure 10.3-19), is used with bottom-dump trailers or backhoe loaders. It does not require reversing; rather, the truck merely drives alongside the loader. It requires a very narrow face with a long, clear

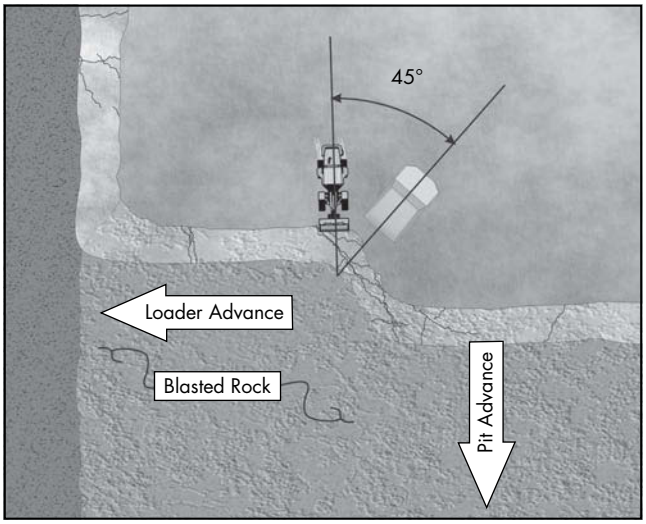


Figure 10.3-18 Wheel loader spotting a truck

bench adjacent so that the truck can approach along the bench and the loader need never move far from the bench.

Backhoes offer a few variations for loading trucks on the floor because the truck can back to the face at virtually any angle. The truck spot can be set up to bring the material over the side or through the tailgate, depending on the bench configuration. Some hydraulic shovels without independent pumps for hoist and swing can benefit from positions that emphasize either hoist or swing rather than working both together.

Matching Loading Tools to Haulers

Selecting the best loading-tool size to hauler size involves analyzing the number of passes required to load a truck and the number of trucks needed to match the shovel. The primary goal is to optimize the total loading and haulage cost.

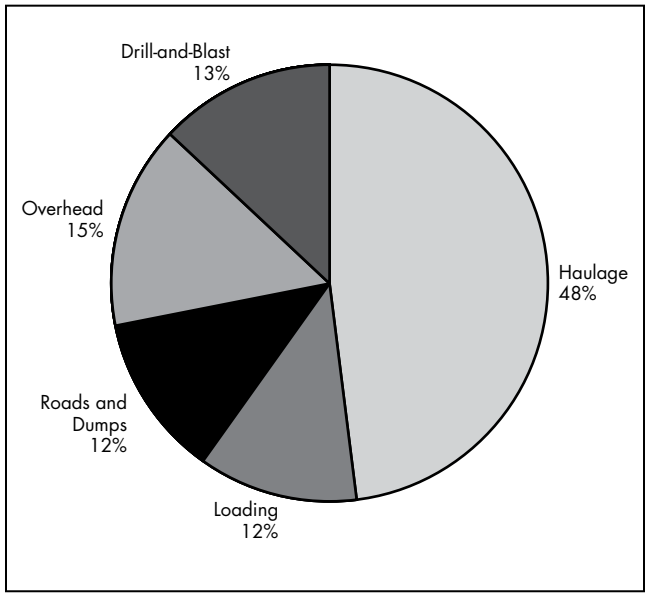
Maximum production is determined by the loading tool, not the hauler. However, pit production costs are most significantly influenced by the hauler (Figure 10.3-20). Specifically, haulers account for nearly 50% of the total system cost, and loading tools only about 10%. Thus the following bears repeating: in general, *the loading tool drives production and the hauler drives cost.*

The strategy in matching loading-tool size to hauler size is to consider but not be ruled by the concept of matching passes and minimizing the number of passes per truck. The variety of loaders and trucks available make it virtually impossible to always achieve a perfect match. Another complication is that variations in material density and bucket fill ensure that no two dipper loads will be exactly the same; in fact, the distribution of dipper and truck-load sizes is nearly normal. This distribution pattern is considered in loader design, and some latitude exists in matching dipper loads to truck loads. Target payload should be changed only after consultation with the manufacturer.

Given that spot time cannot be less than shovel cycle time, it follows that the more passes per truck, the more the shovel (and therefore the system) produces. The downside of this premise is that it favors selection of smaller trucks, which of course means higher operating costs and more congestion. The premise is therefore best considered as suggesting that



Figure 10.3-19 Drive-by loading



Courtesy of Caterpillar, Inc.

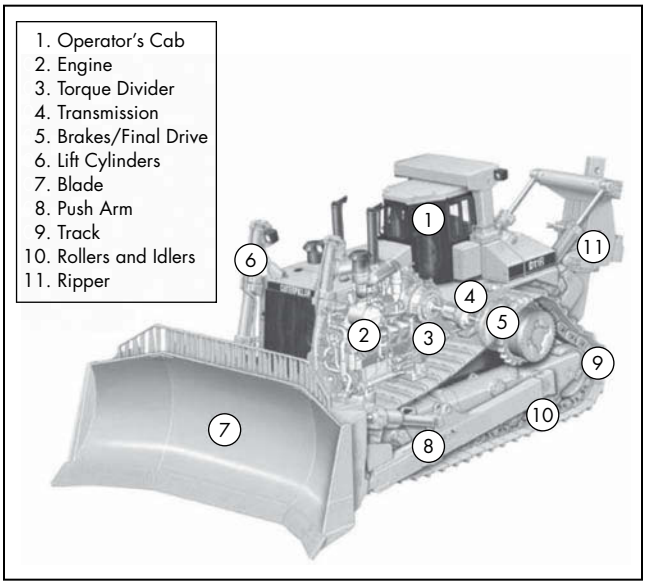
Figure 10.3-20 Relative influence of haulers on open-pit production costs

truck size, not shovel size, determines the number of loading passes.

In reality, variations in production or fleet costs caused by pass match are not as significant as the consequences of undertrucking or overtrucking a fleet. Undertrucking a typical fleet by only one truck offsets the production advantage of the additional pass per truck; likewise, overtrucking a fleet by only one truck offsets the cost advantage of a larger truck with fewer passes per truck.

Track Dozers

Large track dozers (Figures 10.3-21 and 10.3-22) are extremely common in all mining operations. They are designed to move the greatest amount of material in the most efficient way, and generally used for both utility and production work. Utility work includes tasks that support a mine's main production fleet, such as dump-site preparation and cleanup, bench preparation, road creation, stockpile work, and reclamation. However, the focus of this section is their use for production



Courtesy of Caterpillar, Inc.
Figure 10.3-21 Track dozer



Courtesy of Eric C. Orlemann.
Figure 10.3-22 Track dozers working in a pit

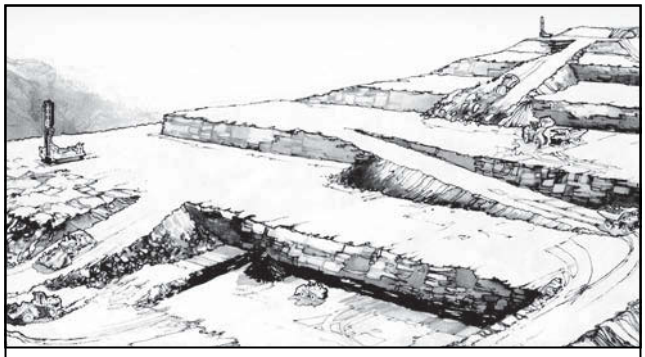
work, specifically mass excavation, for which they excavate, push, and rip in-situ or blasted material from one area to another. Examples of this use include assisting primary production loaders and slot dozing (Figure 10.3-23).

For purposes of comparison, the track dozer is a low-capital-cost, high-operating-cost system that has moderate flexibility and can operate through a limited range of applications with moderate sensitivity to geologic variance.

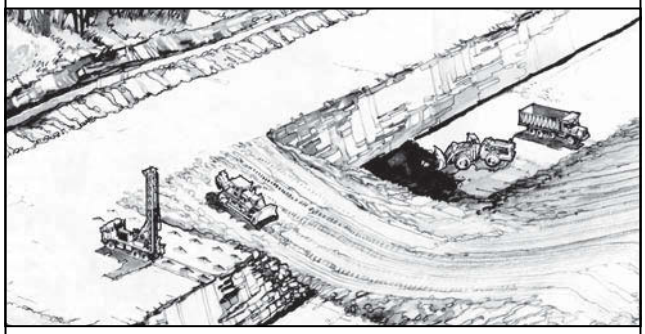
Currently, two major manufacturers build track dozers for the mining industry: Caterpillar and Komatsu. Table 10.3-3 compares their specifications for typical large track dozers used for mining production.

The specifications are very similar between manufacturers, except for the Komatsu D575 Super Dozer (SD), whose high-end size class is an anomaly. Most of the industry uses the smaller size classes because of their lower operation costs and flexibility.

Track dozers are complex machines, because of their variety of mechanical, electrical, and hydraulic systems all fitted into a compact design that protects against the elements.



A. Assisting a primary production loader



B. Performing slot dozing

Courtesy of Skelly and Loy, Inc.
Figure 10.3-23 Track dozers

Operating conditions are unlike those for other material-excavation equipment that loads statically and advances as the face moves. Rather, slot-dozing and rip-and-push operations to assist large loaders require the dozer to push material at varying distances and gradients in poor ground conditions. Therefore, machine design must be very robust. To this end, the mainframe is rigid and consists of multiple fabrications and castings. All major components and systems are mounted to the frame. The radiator, engine, torque divider, transmission, brakes, and final drives are housed in the frame and body, and in most cases are modular for ease of removal and installation.

The components that typically require extensive forethought for maintenance are those that engage the ground, commonly referred to as ground-engaging tools (GETs). GETs include blades and wear plates, rippers (if fitted), rollers and idlers, and tracks. All dozer manufacturers provide a large variety of options for these components, except for rollers and idlers. The blade and associated wear plates are customized for each application; for instance, Caterpillar provides five types of blade (semiuniversal, universal, reclamation, coal, and carry-dozer). Two types of ripper (single-shank and multiple-shank) are available. And both regular-size and wide track shoes are available.

The goal is to maximize GET life and productivity by matching the machine configuration and options to the site characteristics. Thorough site analysis by the manufacturer is required to evaluate the production cycle and material characteristics. The longer the push distances, potentially the higher the replacement frequency. Material characteristics such as

Table 10.3-3 Track-dozer specifications by manufacturer and model*

	Caterpillar			Komatsu			
Specification	**D10T**	**D11T**	**D11T CD**	**D375**	**D475**	**D475 SD**	**D575 SD**
Operating weight, kg (lb)	66,451 (146,500)	104,590 (230,580)	113,000 (249,120)	71,640 (157,940)	108,390 (238,960)	113,198 (249,560)	152,597 (336,420)
Flywheel engine, kW (hp)	433 (580)	634 (850)	634 (850)	455 (610)	664 (890)	664 (890)	858 (1150)
Blade, m^3 (yd^3)	18.5–22 (24.2–28.7)	27.2–34.4 (35.5–45)	43.6 (57)	18.5–22 (24.2–28.8)	27.2–34.4 (35.6–45.0)	45 (58.9)	68.8 (90)

*Specifications vary by manufacturer.

Table 10.3-4 Dozer case study of larger loads slower versus smaller loads faster

Parameter	**Dozer 1 (larger loads slower)**	**Dozer 2 (smaller loads faster)**
Average doze time, min	0.95	0.83
Average doze speed, km/h (mph)	2.4 (1.5)	2.9 (1.8)
Average return time, min	0.40	0.38
Average return speed, km/h (mph)	5.9 (3.7)	6.1 (3.8)
Average cycle time, min	1.35	1.21
Cycles per hour	44.4	49.6
Average blade load, LCM (LCY)	34.4 (45)	27.5 (36)
Average push distance, m (ft)	40 (130)	40 (130)
Production, LCM (LCY)	1,492 (1,952)	1,418 (1,855)
Productivity difference, %	+5	Base

fragmentation (size), abrasiveness, and cohesiveness must be evaluated to determine a dozer's replacement life.

Dozer Productivity

The first step in determining track-dozer productivity is to calculate how much material a particular dozer can push. This narrows the number of suitable size classes and defines some viable configurations before detailed productivity calculations are made.

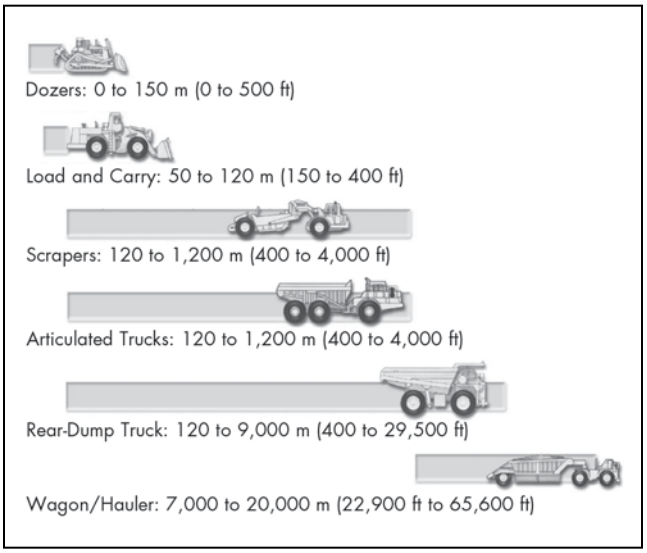
The following factors are involved in determining dozer capability:

- **Weight.** The dozer cannot push more than its weight.
- **Coefficient of traction.** This is the percentage of the dozer's weight that can be pushed for a given material before the track shears or slips. For most materials, this value averages 60%. For loose sandy material, it can be as low as 30%. Multiplying the coefficient of traction by the dozer's weight gives the weight that a dozer can push.
- **Material density.** The denser the material, the smaller the volume of material that the dozer can push. Blade capacity and selection are directly related to material density.
- **Carry-force ratio.** This is the energy needed to compensate for friction or drag and push material across itself. It is usually about 10% and is included in calculations of drawbar pull, which provides the corresponding machine speed.
- **Slope.** This is the percent grade downhill or uphill. It is included in calculations of drawbar pull, which provides the corresponding machine speed.
- **Push distance.** This is the same as the return distance, although push and return are traveled at different speeds. Both have a large impact on cycle time.

In production dozing, it is critical to plan dozer size and configuration so as to maximize productivity. A common tagline in the industry is "big load slow," which describes how to manage a machine for maximum productivity. Productivity is defined as work done over a unit of time. The industry continues to debate whether dozer productivity is higher when pushing larger loads slower or smaller loads faster. Table 10.3-4 shows a case study of two identical dozer configurations for a slot-dozing application. Cycle conditions are identical and the average push distance and gradient are approximately the same. The average cycle time is about 10% slower for dozer 1 than for dozer 2, in large part because the blade load is 25% larger. The end result is that productivity is 5% higher for dozer 1, pushing larger loads slower.

Among the factors that can improve the operating efficiency of a dozer are technique and technology. Technique has to do with how an operator approaches a particular job. The following technique tips reflect best practices for a variety of dozer operations related to production dozing and ripping, where the highest productivity gains can be made. In some cases, these adjustments can increase productivity by up to 25%.

- Slot: Front-to-back technique (most efficient)
  - Operator works the cut from front to back.
  - Push distance increases with each pass.
  - Efficiency is optimal due to downhill blade loading.
  - Creates the slot and uses it throughout the cut.
- Slot: Back-to-front technique (less efficient)
  - Operator works the cut from back to front.
  - Push distance decreases with each pass.
  - Efficiency suffers from uphill blade loading.
  - Does not fully use the slot throughout the cut.
- Slot: Back-each-pass technique (less efficient)
  - Operator starts each pass at the back of the cut.
  - Each pass uses the entire length of the cut at a uniform depth.



Courtesy of Caterpillar, Inc.

Figure 10.3-24 Haul distances for various types of surface haulers

  - Efficiency and productivity suffer because the machine travels the entire length of the cut in both directions with each pass.
- Berm: Criss-cross removal (most efficient for removing center berms)
  - Operator works the cut from back to front.
  - Push distance decreases with each pass.
  - Existing slots are used to hold in material and increase blade load.
- Berm: Management
  - Berm should not exceed blade height and should be high enough only to trap material for optimum loading.
  - Center berm width should be one-third the blade width.
  - For optimum productivity, the smaller the berm, the easier the disposal.
- Ripping
  - Operator should rip downhill when possible.
  - Operator should reduce speed in shock and impact conditions.
  - Operator should try cross-ripping if material does not free up.
  - Operator should pull the ripper tip forward after penetration.

The final piece to improving dozer efficiency concerns technology. Dozer manufacturers have developed software features that increase safety and efficiency. These features provide automatic control or even real-time data to allow the operator to adjust to conditions. One such feature is Caterpillar's AutoCarry (Caterpillar 2008), which automatically optimizes the blade lift-and-lower functions during the carry segment of push dozing by monitoring, calculating, and integrating data on power-train output, ground speed, track slip, and tractor attitude. Another such feature is an onboard Global Positioning System (GPS) that links directly to survey data in the mine office. The operator obtains needed data via a cab-mounted display rather than by reading surveying maps or looking for grade stakes. GPSs are available from the major manufacturers and from third-party suppliers.

Haulers

The hauling of material (such as coal, ore, sand, gravel, or topsoil) from one point to another in a safe, efficient, and cost-effective manner is critical for mining operations. Selecting the type of surface hauler requires a thorough understanding of the selected mining method and its associated advantages and disadvantages along with the machinery available to the industry.

A number of haulage options are available for mining, all of them with unique characteristics that can be optimal for a particular mine site and haul distance. Figure 10.3-24 compares haul distances for dozers, front-end loaders, wheel-tractor scrapers, articulated dump trucks, off-highway rigid-frame trucks, and belly-dump haulers, each of which has an economic advantage at certain haul distances, with some overlap.

On-Highway Trucks

On-highway trucks are not widely used in mining operations because of their lack of capacity, which is only 6–12 m^3 (8–15 yd^3), and their limited performance capability. However, some mine sites use these trucks for hauling coal (eastern United States), prestripping ttopsoil in contract surface mining operations (Australia), and hauling construction aggregates for road building. Their capability to travel long distances at relatively low cost while meeting local on-highway regulations and their overall flexibility in mining operations where landscape is limited provide a unique hauling alternative.

Three types of on-highway truck are commonly used for mining operations (DumpTrucksGuide.com 2006):

1. **Standard dump truck.** This truck typically has a two- or three- (1 front, 1–2 drive) axle truck chassis with a dump body mounted on the frame. The dump body is hoisted hydraulically by cylinders mounted between the cab and the front wall of the body. The small size of the truck allows for exceptional maneuverability in tight loading areas.
2. **Semitrailer rear-dump truck.** This tractor–trailer combination typically has a three-axle tractor and a two-axle trailer. The trailer body is hoisted hydraulically. Key advantages compared to a standard dump truck are faster unloading and increased payload.
3. **Semitrailer belly-dump truck.** This tractor–trailer combination typically has a three-axle tractor and a two-axle trailer with a C-shaped dump gate. The dump gate, mounted on the trailer, is hoisted hydraulically. A key advantage is the capability to unload material as a wind row.

The advantages of on-highway trucks in typical mining operations are few, but there are applications for which they can provide an adequate hauling alternative. Their small size allows for flexibility in operations where loading areas are small, as for eastern U.S. contour coal-mining operations. Their low fleet cost is advantageous when mining operations need additional hauling capacity for short-term use. They require substantially less investment in up-front capital and subsequent operating cost than do large off-highway mining trucks. They can also be used as a secondary fleet for special



A. With hydraulic cylinders



B. With ejector body

Courtesy of Caterpillar, Inc.

Figure 10.3-25 Off-highway articulated dump trucks

projects such as road construction and prestripping, which are typically contracted from local construction firms.

The disadvantages of on-highway trucks are based primarily on their relative performance compared to that of large off-highway trucks. Payload capacity is an obvious difference; on-highway trucks are smaller and thus have higher overall fleet costs per ton of material due to their lower productivity and the larger number of trucks required. Additionally, they are not designed for use in rigorous mining conditions. Their structural design, electrical and hydraulic systems, brake and steering performance, and power train are designed for highway use, not for 24/7 mining applications. In mining operations, haul road gradients plus rolling resistance can be as high as 20%. On-highway trucks can have difficulty in deteriorating road conditions, to the point where support equipment is needed to recover them, reducing fleet productivity. Thus, they are not considered to be a primary haulage solution for the mining industry, although they will always have limited use in mining applications.

Off-Highway Articulated Trucks

Off-highway articulated trucks are a hauling alternative primarily for middle-to-large-scale construction projects. They are often used for prestripping, road construction, and material hauling for ground preparation for buildings and other infrastructure. When used with small hydraulic excavators or wheel loaders, they can constitute an effective loader and hauler fleet for a mass excavation project. They are widely used for pre- and postmining construction in soft-underfoot conditions, in small loading and dump areas, or on steep (10%–15%) grades.

Table 10.3-5 Articulated dump truck specifications*

Specification	Size Class, t (st)			
	23 (25)	27 (30)	32 (35)	36 (40)
Gross engine, kW (hp)	224 (300)	261 (350)	298 (400)	335 (450)
Net engine, kW (hp)	212 (285)	250 (335)	287 (385)	324 (435)
Top speed, km/h (mph)	56 (35)	56 (35)	56 (35)	56 (35)
Tire size	23.5R25	23.5R25	26.5R25	29.5R25

*Specifications vary by manufacturer.

An articulated dump truck (ADT) (Figure 10.3-25A) is a three-axle machine with an articulation point between the front axle and the two rear axles. The articulation, which is unique to this truck type, is useful where there is limited area in which to operate. The three axles all provide power to ground. This all-wheel-drive capability provides an advantage over on-highway or off-highway rigid trucks whose one or more rear axles are the only source of power to ground. It also allows the ADT to operate well in soft-underfoot conditions, defined as rolling resistances of 10%–20% (Caterpillar 2000). An ADT can vary power to the wheels according to road and haul conditions. For example, a Caterpillar ADT has three operating modes: a standard mode (40/60 split between front and two rear axles), a low mode (50/50 split), and a high four-wheel-drive mode (all three axles have equivalent power to ground).

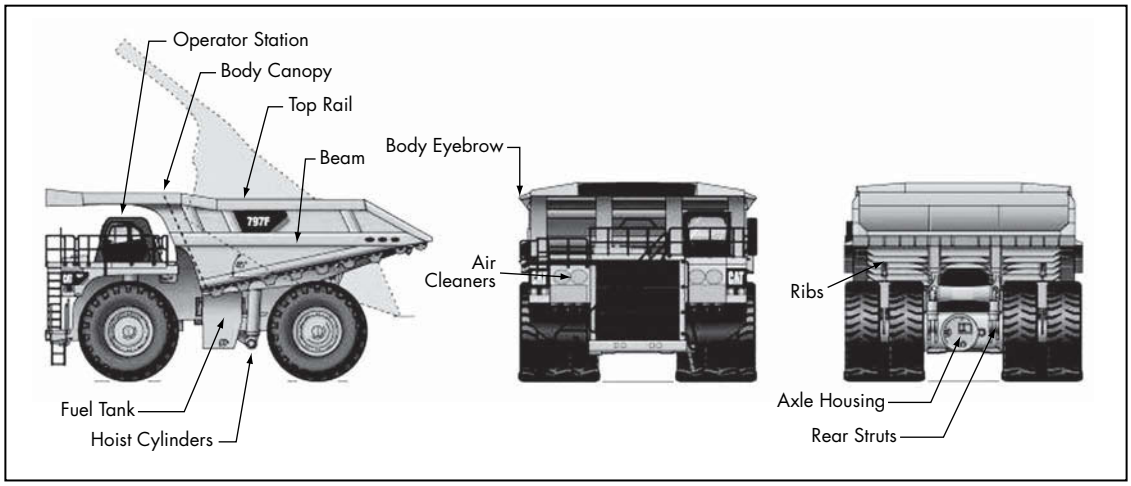
The dump body is mounted on the rear frame, with traditionally two options to dump material. The first option is similar to that for on-highway dump trucks: two hydraulic cylinders hoist the body, dumping material rearward. The second option is an ejector body (Figure 10.3-25B) with a hydraulically moveable front wall that runs on a rail system fixed to the side wall; the front wall pushes material back and dumps it rearward. Despite its added design complexity, an ejector body increases productivity by decreasing both dump times and the amount of carryback per load. It also allows the machine to safely dump in steep inclines or side slopes.

A number of major global construction and mining manufacturers provide ADTs for the mining industry, including Caterpillar, John Deere, Komatsu, Terex, and Volvo. In addition, a number of regional manufacturers in China, India, and elsewhere have offerings. ADTs generally range from 23 to 36 t (25 to 40 st) in four size classes that increment every 4.5 t (5 st). Table 10.3-5 highlights their specifications by size class (payload).

ADTs providing the surface mining industry a hauling solution that is adaptable to tough hauling conditions. However, because of their limited hauling capacity, they are not viable as primary production machines.

Off-Highway Rigid-Frame Trucks

The primary hauling machine in mining is the large off-highway rigid-frame truck (Figure 10.3-26). In the 1950s and



Courtesy of Caterpillar, Inc.

Figure 10.3-26 Off-highway rigid-frame truck

1960s, it was used as an alternative to locomotives and small dump trucks. Over the years, it has proven to be cost-effective, flexible across a variety of applications, and capable of handling the rigors of 24/7 operation. It continues to be a frequent hauling solution of choice for surface mining.

An off-highway rigid-frame truck has a rigid (unarticulated) frame constructed of multiple steel fabrications and castings. To this frame are mounted the truck cab, body, diesel engine, power train, front and rear suspension, front and rear wheels, tires, and more, all interconnected by sophisticated mechanical and electrical hardware and software. The two most distinguishing characteristics of the various makes and models are payload capacity and power-train type.

Payload capacity. The size or payload of an off-highway rigid-frame mining truck plays a significant role in determining the viability of a mining operation. For example, for a given mine plan, one could elect to operate either fifty 90-t (100-st) trucks or twenty-five 180-t (200-st) trucks. At first glance, assuming that machine performance is equal, the second choice, at half the fleet size, would seem to reduce costs dramatically. However, each fleet carries a different capital- and operating-cost footprint, and these footprints should not be assumed to be linear.

Trucks for surface mining currently have payload capacities of 90–360 t (100–400 st). These values have evolved over time, driven by the mining industry's desire to go larger in order to maintain or increase production while decreasing fleet size and operating costs. Within this payload range, there are five distinct classes, designated according to size:

1. 90-t (100-st) class
2. 135-t (150-st) class
3. 180-t (200-st) class
4. 220-t (250-st) class
5. >290-t (320-st) ultra-class truck (UCT)

Figure 10.3-27 shows current mining truck models by size class and manufacturer. Some manufacturers are not included because information was not available.

Product strategies regarding payload capacity clearly vary by manufacturer. Most mining truck manufacturers work closely with the industry to determine the appropriate size for an application, typically matching the rated payload capacity to the current and expected future loading tools, using three to five passes as the optimal level. For example, for a truck with a rated payload capacity of 220 t (250 st), an electric cable shovel with a capacity of 46 m^3 (60 yd^3), assuming a 90% fill factor, could load 1,780 kg/LCM (3,000 lb/LCY) of material in approximately three passes.

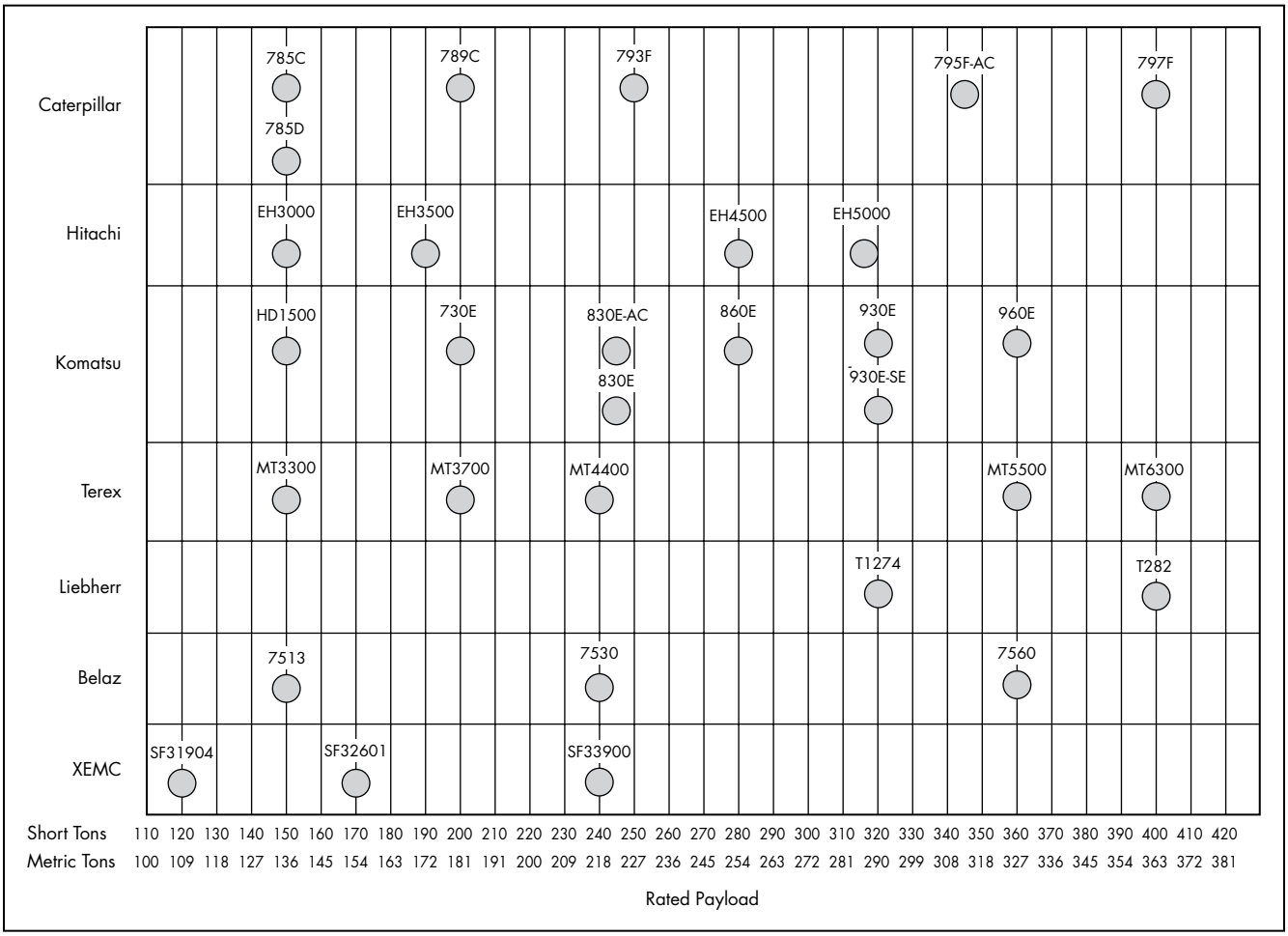
As important as payload is to a mining operation, the following points continue to be debated:

- Can we get more payload?
- Are our truck payloads at optimal levels?
- Should we upsize?

Some of these questions can be answered by considering the loaders and associated practices, but in some cases the trucks should be considered as well. Thus, after a decision is made about size class, the next decision should concern the type of truck body. There are now a myriad of truck body designs for any type of mining truck, and the choice of design depends on the material characteristics of the mine, which differ from country to country. Truck bodies today can be customized for each operation so as to maximize payload, reliability, and durability. For the purposes of such customization, a mine operation creates a profile listing the following information:

- Material type
- Material density (lightest), needed to ensure that payload is met, regardless of any fluctuation in material density
- Material fragmentation (size), needed to determine liner-plate thickness and dumping characteristics
- Material abrasiveness, needed to predict wear characteristics for determining the appropriate liner package
- Material cohesiveness, also needed to determine the appropriate liner package
- Loading tools (model and type, percent utilization, bucket size)
- Body/mining truck dimension limitations, needed to determine maintenance facilities, hoppers/crushers, loading tool dump heights, and more

Another point of discussion is payload management. The constant push to increase payload for an existing truck fleet is not a bad thing if trucks consistently perform, on average, at under their rated levels. Because pushing trucks beyond their



Courtesy of Caterpillar, Inc.

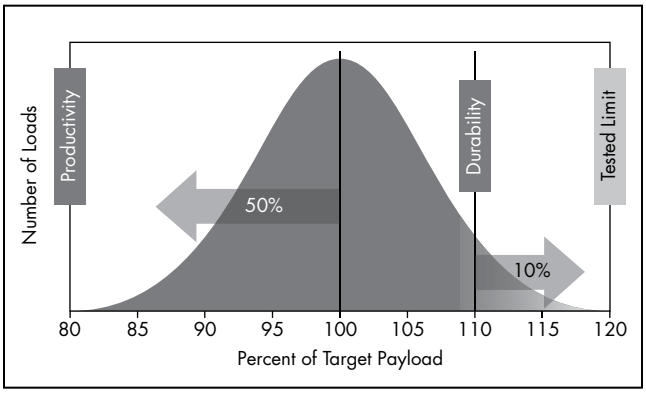
Figure 10.3-27 Off-highway rigid-frame truck models by size class and manufacturer

rated levels can be detrimental, most manufacturers have a payload policy that outlines the levels at which a truck can perform within the certification or design envelope of the machine. Standard practice today is a so-called 10/10/20 payload policy (Figure 10.3-28) that distributes truck payloads over a set period of time to address risks associated with overloading. The highest risk is overloading beyond 120% of target payload.

All mining trucks are designed to meet certain Society of Automotive Engineers (SAE) and International Organization for Standardization (ISO) design standards that address not only component and system functionality but also safety. Two particular standards are related to payload: ISO 3450 (1996) concerning brake certification and ISO 5010 (1992) concerning steering certification. The requirement for certification is that a truck loaded to its maximum payload (an overloaded state at the upper end of the normal distribution of payloads) should stay within requirements. Overloading beyond the payload policy risks, among other things, machine durability, as the structural components are designed to certain life targets and, if overloaded more frequently than recommended, become fatigued and prone to fail. Early failure increases maintenance and repair costs dramatically due to unplanned rebuilds and repairs. Payload management can be difficult to apply in practice, but it plays a large role in meeting production requirements and keeping a truck fleet operating safely and efficiently. It is important to check with the manufacturer for the tested limit specific to a truck.

Finally, when should a truck fleet be upsized to a new size class? Moving to a higher size class has many benefits and at least a few barriers, which become more daunting with increasing truck size. The following is a list of considerations (Caterpillar 2009):

- Mine operations
  - Mine design: Larger mining trucks can require changes in haul road design and load and dump area, especially when upsizing to UCTs.
  - Loading tools: It is important to have the proper loaders to meet production requirements with the new fleet.
  - Support equipment: Larger mining trucks place a stronger demand for support equipment to maintain haul roads and loading/dump areas.
  - Operator training: Curriculum and training tools must be changed.
- Mine maintenance
  - Facilities: Maintenance facilities (such as shop, lube islands, and parts and component storage) may need to be upgraded to handle the larger machines.



Courtesy of Caterpillar, Inc.
Figure 10.3-28 10/10/20 Payload policy

– Tooling: Tooling requirements must be upgraded to handle larger components and any specialized tools.
– Training: Significant training may be required when upsizing, regardless of system commonality.

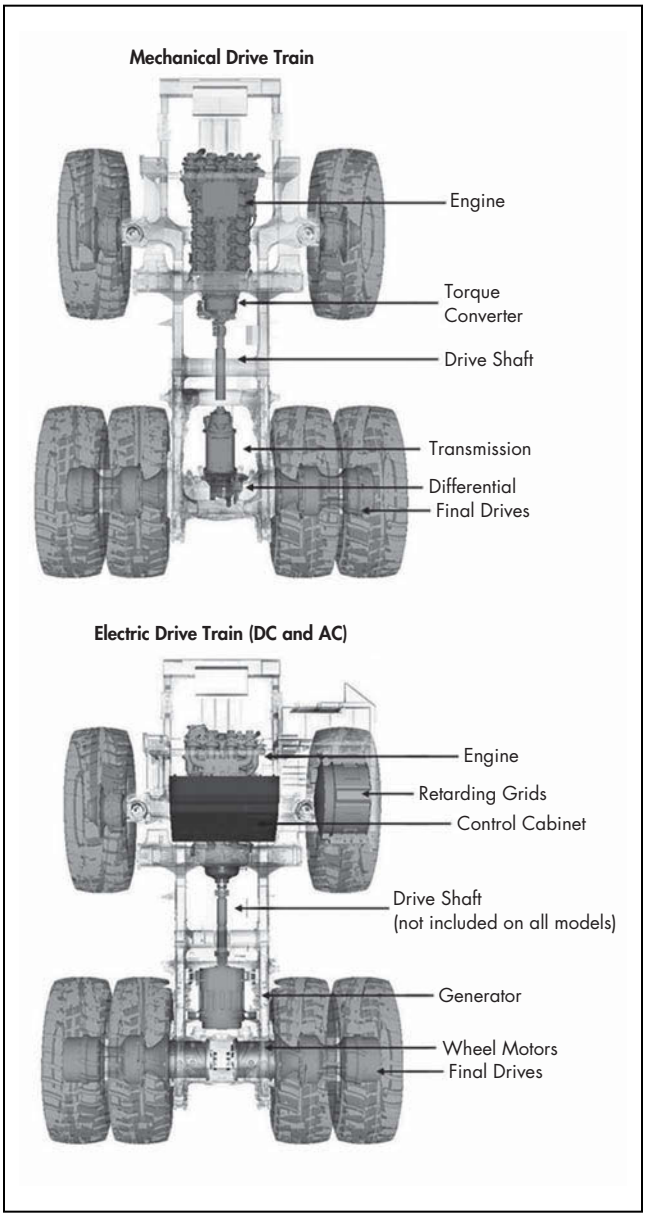
Drive-train type. Truck performance is the next key piece of the puzzle. Three types of drive train are currently in use:

1. Mechanical drive, similar to those used in on-highway automobiles and trucks
2. Direct-current (DC) electric drive
3. Alternating-current (AC) electric drive

Figure 10.3-29 shows a side-by-side comparison of the mechanical and electric drive trains.

The mechanical drive train contains five major components: engine, torque converter, transmission, differential, and planetary gear sets (wheels). The power source is the diesel engine plus torque converter; the latter transmits rotational power from the engine to the main driveshaft. The transmission controls machine torque and speed during operation. The differential transfers output torque to the wheels. Two hydraulic brake packs are mounted on each of the axle shafts on the two rear wheels. The entire system is activated by means of a variety of electronic control modules and hydraulic control systems.

DC and AC electric drive trains contain six major components: engine, generator, power converter, wheel motors, planetary gear sets (wheels), and retarding grid. The power source is the diesel engine plus generator; the latter converts mechanical power from the engine into electric power. AC current from the generator is then converted into useable form. In a DC-drive truck, a rectifier converts it into DC power; in an AC-drive truck, a rectifier converts it into DC power and inverters convert it back to a controllable version of AC power suitable for managing the amperes, volts, and frequencies of the wheel motors in order to create machine speed and torque. The DC or AC wheel motors receive the electric power and feed it mechanically to the planetary gear sets (wheels). The retarding grid—a bank of resistor elements—provides braking force by turning the wheel motors into generators, creating power rather than receiving it. This power is sent through a control cabinet and on to the resistor elements. The resistors impede the flow of the electric power, which causes the wheel motors to slow rotation. Heat generated is cooled by an electric fan.



Courtesy of Caterpillar, Inc.
Figure 10.3-29 Mechanical and electrical drive trains

The three types of drive train have unique performance characteristics that impact productivity and operating costs. The major points to compare are the following, examined in more detail in Table 10.3-6:

- System limitation on grade
- Speed on grade (propelling)
- Speed on grade (retarding)
- Top speed
- Fuel consumption
- Maintenance and repair costs

OTHER SYSTEMS

These additional systems deserve serious consideration:

- Bottom-dump coal hauler
- Trolley-assist mining truck

Table 10.3-6 Points of comparison for three types of drive train

Point of Comparison		Mechanical	Electric, DC	Electric, AC
System limitation on grade	Define	Stall torque gradeability (STG) is the maximum percent grade that a loaded mining truck can pull out at zero speed. Trucks can stall in load and dump areas with poor underfoot conditions, requiring them to be pulled out. In addition, open-pit mines continue to go deeper. To make doing so economically viable, ramp gradients have increased from 8% to ≥10%. When rolling resistance is added, mining trucks must overcome ≥12% total effective grades, causing unexpected wear on a number of power-train components.		
	Compare	Is best in class. The maximum STG is about 28%–30%. The design accommodates all mining operations that continually operate at gradients up to 15%.	Is the least effective. The standard system has a peak STG of 23% but can be modified to reach 25%. Has thermal limitations during long uphill hauls on gradients of >8%. A thorough site evaluation must be conducted.	Is superior to the DC electric drive system due to the higher power density of the wheel motor. STG is ≤26%, depending on configuration. Has thermal limitations during long uphill hauls on gradients of >10%. A thorough site evaluation must be conducted.
Speed on grade (propelling)*	Define	Performance is dictated by truck weight, engine horsepower, and power train efficiency, and can vary from one model to another depending on a manufacturer's design strategy. However, power train efficiency is a point of differentiation and is the cumulative efficiency of all of the major components. The higher the efficiency, the lower the required engine horsepower. Most surface mines have haul-profile segments at a positive grade; thus the higher the speed, potentially the higher the productivity.		
	Compare	Has the highest gross power train efficiency at its peak, 88%. At an engine horsepower equivalent to the other power trains, has a higher speed-on-grade performance.	Has the lowest gross power train efficiency at its peak, 81%.	Has a gross power train efficiency at its peak of 83%.
Speed on grade (retarding)*	Define	Performance is dictated by truck weight and braking/retarding power. In loaded downhill applications, this is a key performance metric. As long as a truck can travel safely at its optimum level, this point of comparison can optimize productivity.		
	Compare	Is significantly improved since its inception due to addition of another gear.	Has the lowest speed on grade in retarding. However, changes in retarding systems have closed the gap.	Is best in class. The wheel motor power is capable of high levels depending on truck class, and thus has high-speed capability in retarding.
Top speed*	Define	Top speed, both empty and loaded. This is typically a point of differentiation with applications that have long flat hauls. However, except for DC-drive trucks, the systems do not differ significantly.		
	Compare	Depending on truck model, either 61–66 km/h (38–41 mph) or 55 km/h (34 mph).	55 km/h (34 mph)	Is best in class, at 64 km/h (40 mph) regardless of truck-size class.
Fuel consumption	Define	Fuel consumption is one of the most debated performance characteristics of mining trucks. So much depends on the application, engine type and fuel efficiency, condition of the engine and other machine systems, and even how the operator handles the machine. Thus, for a proper evaluation, a site study is required. However, the functionality of a power train is focused on reducing fuel consumption.		
	Compare	Power train efficiency permits engine horstepower to be less than that of an equivalent electric-drive model. Less horsepower equals less fuel consumed for an equivalent speed on grade. Also has a feature that allows the truck to consume zero fuel while retarding (Caterpillar 2006). When the truck approaches a downgrade and the operator's foot is removed from the throttle, gravity takes over. With a mechanical linkage between the wheels and engine, no combustion (and thus no fuel) is needed to maintain engine rotation.	Possesses unique operating characteristics that save fuel. The engine does not cycle during acceleration, as does a truck with a mechanical drive train. Rather, it runs at constant speed, allowing control software to manage speed and torque. Also, the engine actively manages horsepower demand from the engine as it cycles through the engine power curve, matching requirements to operating conditions. Finally, the engine applies a partial-power feature on minimal grades, reducing power draw from the engine to the drive system.	
Maintenance and repair costs	Define	The second most debated performance characteristic is the cost for maintenance and repair, defined as the life-cycle operating costs associated with replacing parts and components, along with labor required for removal, installation, and repair. This characteristic is difficult to compare, as much depends on the severity of the application, maintenance practices, on-site support infrastructure, and more. A fleet-selection process should provide site-specific comparisons of the impact of these costs on the operating cost of a drive system.		

*Refer to the manufacturer's performance handbook for rimpull and retarding curves.



A. Truck–trailer configuration



B. Unibody configuration

Courtesy of Eric C. Orlemann.

Figure 10.3-30 Bottom-dump coal haulers

- Wheel-tractor scraper
- In-pit crushing and conveying system

Bottom-Dump Coal Hauler

A semipopular haulage solution for surface coal is the bottom-dump coal hauler. It has become a staple in thermal coal operations because of the need to haul coal from the pit to a nearby power plant.

The truck–trailer configuration (Figure 10.3-30A) of the bottom-dump coal hauler has a mining truck chassis as the tractor, modified with a hitch assembly to receive a trailer. The most common size class is the 90–136 t (100–150-st) standard truck chassis. The high-volume trailer has typically 1.5 to 1.7 times the payload capacity of the corresponding truck fitted with a rear-dump body. This high capacity is well suited to long hauls with few high-gradient segments, and the bottom-dump coal hauler provides a higher production rate than does a traditional mining truck, thus lowering haulage costs per ton for the truck fleet.

The main players in this market segment for the chassis are Caterpillar (777 and 785 models), Hitachi (CH120, CH135, and CH150 models), and Komatsu (785 and HD1500 models), all of whom provide the necessary chassis modifications from the factory. The trailers are designed and manufactured by smaller specialty firms such as Kador Engineering (Australia), Maxter-Atlas (Canada), and Mega (United States). These firms provide the trailer and hitch assembly, whereas truck OEMs provide the additional axle to be fitted to the trailer, in the interest of product consistency. One firm provides the complete package: Rimpull (United States) provides an entire lineup of tractor–trailer options with their CW160, CW180, and CW200 models.

The truck–trailer configuration of the bottom-dump coal hauler has limited application due to the design specifications of the chassis. A fully loaded rear-dump truck has a continuous rating that allows for total effective gradients of 10%–15%. When the additional weight of a trailer is added and payload is increased, performance drops, limiting the machine to effective gradients of just 5%–10%. This is suitable for a number of surface coal operations, especially where the coal seam is relatively shallow and there is a limited amount of cover or overburden. Another point of limitation is haul distance. The truck–trailer configuration is economically viable only for haul distances of ≥3.2 km (2 mi). For shorter distances, this configuration, with its added costs associated with the trailer, tires, higher fuel consumption due to an increase in cycles, and lower performance characteristics does not compete well against a traditional truck configuration. However, for one-way haul distances of 8–16+ km (5–10+ mi), all of the benefits associated with this configuration outweigh those for the rear-dump truck.

A variant of the truck–trailer version of the bottom-dump coal hauler is the unibody configuration, or unibody coal hauler (Figure 10.3-30B), currently manufactured only by Kress (United States). This machine has a built-in bottom-dump body (i.e., it is unitized). Among its advantages compared to the truck–trailer configuration, it has a significantly higher payload-to-weight ratio, a higher horsepower-to-weight ratio, and a 50-t- (55-st-) lower empty weight. It also has higher fuel efficiency (fuel consumed per ton) and lower metric-ton kilometers per hour (ton miles per hour), which improve tire life. Its higher horsepower-to-weight ratio can enable it to, for example, operate at higher effective gradients to increase productivity. In addition, its drive train is capable of achieving higher top speeds—typically up to 30% higher—than is the drive train for the tractor–trailer configuration, which is limited to the speed capability of the chassis. This can increase productivity measurably on long hauls.

Trolley-Assist Mining Truck

A trolley-assist mining truck (Figure 10.3-31) is a unique application for mining trucks, and strictly exclusive to electric-drive models. Its use for large-scale material transport dates back to the late 1930s in Italy's full-trolley systems; its use for mining began during the energy crisis of the 1980s. With upgrades in technology, it still has relevance in the mining industry. Currently, two suppliers offer a trolley assist on their mining trucks: Hitachi and Komatsu. Four operations use it today, all in Africa.

A trolley-assist mining truck draws its power from overhead power lines that are run on haul segments where the largest benefits can accrue, such as where the loaded truck operates on a positive grade. The truck is fitted with a pantograph that acts as a conduit between the line and the truck's electric-drive distribution system. As the truck approaches the line, the operator lifts the pantograph until it contacts the line. When the two engage, the operator removes his or her foot from the throttle and continues to steer while the truck draws power from the line. Power is fed to the wheel motors, temporarily replacing the diesel engine and generator.



Figure 10.3-31 Trolley-assist mining truck

Trolley assist has several benefits:

- Decreased fuel consumption, achieved by running the engine at idle for the length of the line. Depending on the length of the ramp, fuel savings can be as high as 50%.
- Increased productivity per cycle, achieved by using the excess power capacity in the wheel motors. The power rating of the wheel motor is almost twice the engine gross horsepower, in order to meet the technical requirements for continuous operation under diesel power. That is the primary reason for the significant speed-on-grade performance in retarding, when the wheel motors use their full potential. The same principle applies during trolley assist, when power from the diesel engine is replaced by power from the overhead line. The result is an increase of up to 80% in speed-on-grade performance. Depending on the haul cycle, this can translate into an increase in production of up to 10%.
- Increased diesel engine lifetime. The heaviest toll on an engine in a haul cycle normally occurs when the truck is fully loaded on a grade—the very point at which trolley assist kicks in. The life of a diesel engine is calculated in terms of the total quantity of fuel consumed during the design life. With the engine operating at idle on grade rather than at maximum, the life can be extended significantly, potentially eliminating one complete engine rebuild over the life of the truck.

Trolley assist also involves additional operational costs and constraints, including the following:

- **Relative costs of electric power and diesel fuel.** This is one of the single largest variables to consider when evaluating trolley assist. The cost for diesel fuel can be enormous for a medium to large fleet, but the ultimate question is whether the savings in fuel can offset the cost of electric power.
- **Capital cost of trolley wayside equipment.** This consists of mine power distribution, substations, masts, and wire.
- **Capital cost of truck trolley equipment.** This consists of a pantograph, auxiliary cooling, and truck controls.
- **Mine plan.** Trolley assist does not allow for operational flexibility. After the equipment is in place, it typically is not moved until doing so makes economic sense, often 5 to 10 years from the initial installment. Therefore, it is critical to evaluate the long-term mine plan and determine whether or not a permanent main haul road is possible.
- **Haul profiles.** Determining which haul cycle benefits from trolley assist is one of the most critical pieces to the evaluation. A long haul segment with a grade that the truck travels loaded is the best choice.
- **Capital cost for additional mine-support equipment.** Costs associated with additional motor graders or wheel dozers may need to be included. Haul roads where trolley assist is used must be kept in pristine condition, since spillage and rutting can cause the truck to lose connection with the overhead line.

The industry will continue to support trolley assist. Technology improvements now under consideration include concepts such as auto-control when ascending a grade and regeneration of power when retarding during a return cycle.

Wheel-Tractor Scraper

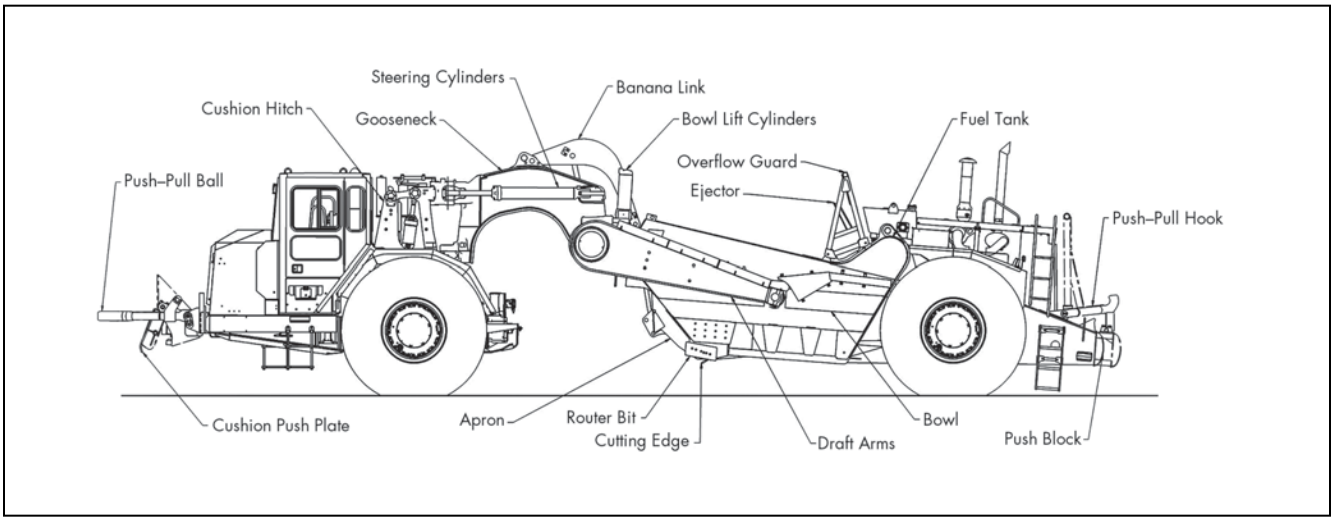
One of the oldest concepts of bulk material handling is the wheel-tractor scraper (WTS) (Figure 10.3-32), which traces its roots back to horse-drawn slip scrapers in the late 1800s (ASME 1991). Today the WTS is the only machine that can self-load, haul, and dump with a single operator. For purposes of comparison, the WTS is a low-capital-cost, high-operating-cost system that is very flexible and can operate through a limited range of applications with high sensitivity to geologic variance.

Mobility and flexibility are key characteristics of the WTS, which makes it ideal for small, short-life mining projects. Its capability to remove and place material in controlled lifts makes it the machine of choice for topsoil relocation in reclamation operations.

WTSs are available with three types of loading: pan, elevator, and auger. The pan uses the motion of the machine to force material in to the bowl; the elevator and auger have mechanical apparatus that assist the material into the bowl. The pan is slower to load but is better suited for blocky materials. The largest units have a bowl capacity of $\leq$34 m^3 (44 yd^3) for earth and rock densities, and can be larger for lighter materials such as coal.

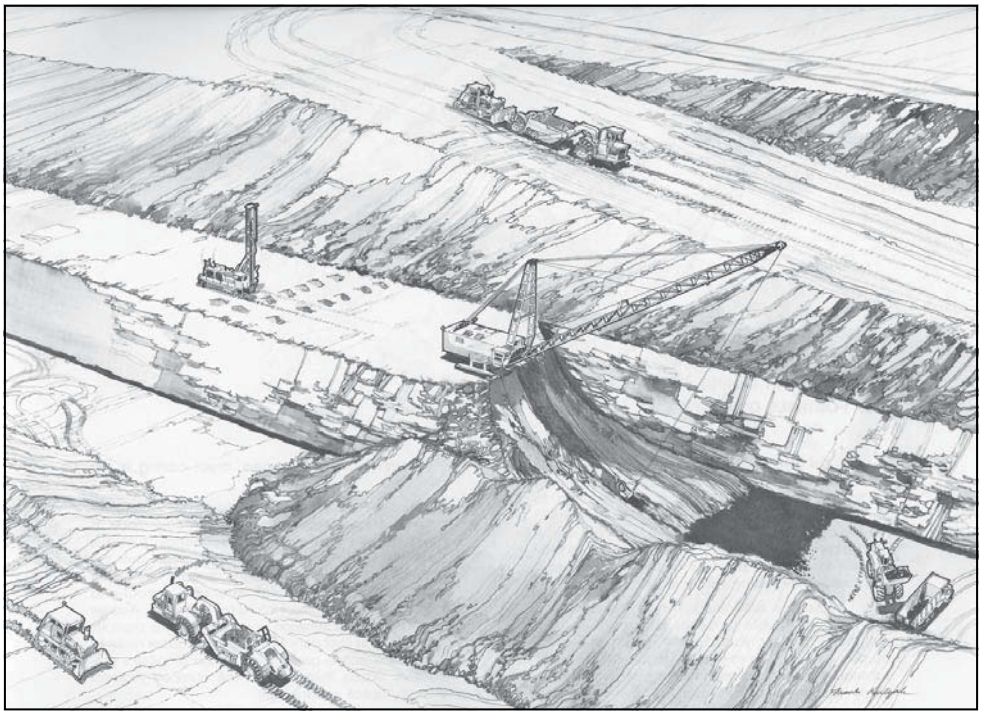
WTSs are either single-engine or twin-engine systems that can be pushed, usually by a dozer, to assist with loading. Larger units can also configured in a push–pull arrangement for connecting two scrapers during loading, thus putting the horsepower of two machines on one cutting edge. Because of its cutting mechanics, the WTS best suited to unconsolidated materials.

Production is obviously affected by haulage distance, but for a moderate haul of 450 m (1,500 ft) a large WTS can produce at a rate of about 300 BCM/h (400 BCY/h). It is capable of speeds >50 km/h (30 mph). Optimal one-way haul lengths are 200–1,200 m (400–4,000 ft). The basic haul cycle is similar to that for a truck: load, haul, dump, and return. Because the load and dump components of the cycle can be 50–100 m (150–300 ft) long, it is most efficient to set up the haulage route so that the loaded haul is shorter than the return. The effect of haul distance on production rate is not quite 1:1; for example, doubling a mid-range haul distance decreases the production rate by about 40%. Properly designed and well-maintained roads are as critical to a WTS as to any other hauler, although this fact is often overlooked, possibly because the load and dump areas are relatively rough. A smooth haul



Courtesy of Caterpillar, Inc.

Figure 10.3-32 Wheel-tractor scraper



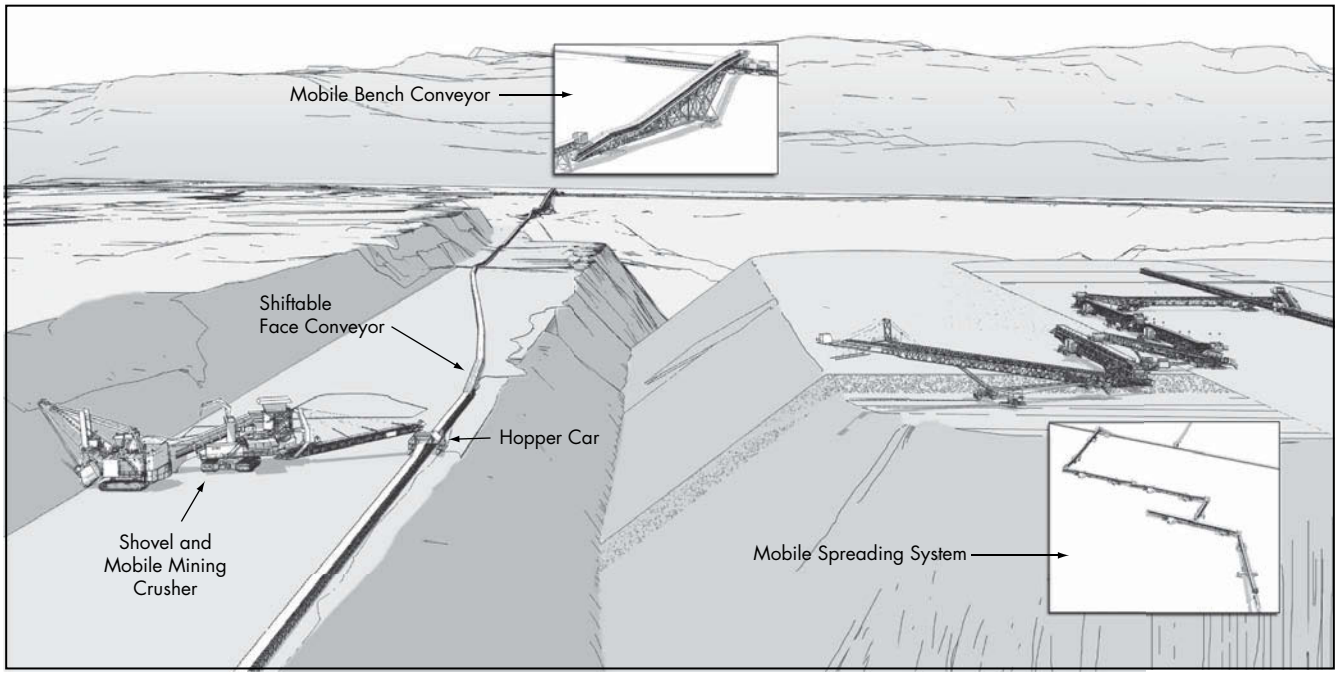
Courtesy of Skelly and Loy, Inc.

Figure 10.3-33 Wheel-tractor scrapers performing typical operations, including dozer-assisted downhill loading

not only lowers rolling resistance, it also makes for a smooth ride with reduced loping and therefore higher speeds.

Under most conditions, WTS load times are in the range of 30–60 seconds. Production can be enhanced by downhill or assisted loading (Figure 10.3-33). A dozer assist or a push–pull system delivers additional horsepower, increasing the production rate by about 10%. Having a ripper-equipped dozer handy can help to loosen hard-packed material between scraper loads. Loading is accomplished by lowering the bowl until the material flows steadily. Cutting too deep may take longer, resulting in higher fuel consumption and spinning tires with their associated costs.

WTS dump or spread time is about 20–30 seconds and the total fill time is 35–45 seconds (Caterpillar 1998). A significant feature of the WTS is its capability to spread a load in a controlled manner. Laying material down in thin lifts improves compaction on multiple lifts or allows material to be spread to a specified thickness.



Courtesy of P&H Mining Equipment, Inc.
Figure 10.3-34 In-pit crushing and conveying system

In-Pit Crushing and Conveying System

A relatively recent and increasingly popular entry into large surface mining systems is the in-pit crushing and conveying (IPCC) system (Figure 10.3-34). This system uses a crusher/sizer unit to process material from a cyclic loader to a size that is suitable for conveyor transport, extending the application of around-the-pit conveyor systems to include consolidated waste and overburden.

For purposes of comparison, the IPCC is a high-capital-cost, low-operating-cost system that has limited flexibility and can operate through a limited range of applications with moderate sensitivity to geologic variance.

In-pit or near-pit crushing of ore has always been common, primarily because the location of the crusher station has only a very small impact on the total comminution costs. Crushing of waste, on the other hand, until the advent of this technology, has been difficult to justify on the basis merely of enabling greater use of conveyor transport.

Although IPCCs have been used since the 1950s, the first large-scale mining systems capable of working with feed direct from a mining shovel were implemented only in the 1980s. With the implementation of compact twin-roller sizers, in about 2002, the system finally gained wider acceptance.

The IPCC is a mobile crusher/sizer machine. Historically "mobile" meant relocatable, usually with significant cost, infrastructure, and time requirements. However, the IPCC is self-propelled or easily transportable with no fixed infrastructure requirements and travels with its own loading tool.

For crushing, the IPCC crushes material to a size suitable for feeding to a conveyor belt: in all three dimensions, the material can be no more than about 30% of the width of the belt, and a significant percentage of the material must be smaller than that, in order to cushion the belt and prevent damage to it. Although the IPCC's crushers and sizers can handle materials with extreme rock strengths, to date, the IPCC (for waste) is most cost-effective for use with materials whose compressive strengths are 50–90 MPa (7,000–13,000 psi) and somewhat less cost-effective for harder or softer materials.

For conveying, the IPCC relies on a series of components that feed material from one to the next. A cyclic loading tool such as mining shovel feeds into a mobile sizer that follows along with the shovel. The sizer feeds into a face conveyor, much as for a BWE. The face conveyor feeds into a series of other conveyors, eventually leading to a discharge conveyor at the dump.

Because of the multiple conveyors, the IPCC is suitable for pit geometries that favor use of draglines and BWEs (i.e., those with long linear faces). Because its loading tool is a shovel, it can handle materials with rock strengths greater than a BWE can handle. At the time of this writing, however, IPCC systems are still finding their niche application but show favor for waste transportation in linear pits with geometries that require long hauls with relatively limited elevation change.

The digging face of the IPCC has a large footprint, similar to the case for the BWE, which needs to be considered when planning other pit operations such as blasting or pit access. Conversely, the conveyor route has a relatively small footprint, which can reduce costs for ramps, earthen bridges, and the like. Likewise, the dump area can be configured with some flexibility and material can be placed in final form with very little rework required for rehabilitation.

The IPCC has an electric drive system. Although this type of system requires electrical infrastructure in the pit, it does not depend on diesel fuel, whose costs tend to fluctuate independent of the product market. It also does not have tires; certainly the tire shortage of 2006–2008 had a significant influence on a number of IPCC purchases.

The IPCC generally requires minimal operating labor per unit of production. Operational and maintenance labor requirements are cyclic, with significantly higher demand during

conveyor relocation. These cycle demands are relatively easy to manage where maintenance contractors are available. The operating method lends itself to semiautonomous operation of some components, further decreasing labor demand.

The newest IPCCs have been matched to the large mining shovels with peak capacities of about 10,000 t/h (11,000 st/h). Because the system components are arranged in series, overall efficiencies are similar to those for a BWE system, about 55%–60%, resulting in an average production rate of about 6,000 t/h (6,600 st/h).

Typical IPCC operations are (or in the second case will be) viewable by satellite-image software in the vicinity of these coordinates:

- IPCC prestripping in advance of dragline: 21° 43′ 33″ S 147° 59′ 23″ E
- IPCC open-pit application with truck and shovel: 22° 43′ 23″ S 147° 38′ 08″ E

ACKNOWLEDGMENTS

Information in the "Bottom-Dump Coal Haulers" and "Trolley-Assist Mining Truck" sections is taken from Moore 2007 and Hutnyak Consulting 2004, respectively.

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CHAPTER 10.4

Selection and Sizing of Excavating, Loading, and Hauling Equipment

Eric N. Berkhimer

INTRODUCTION

Taking a very simplified approach, mining consists of four distinct steps: rock breakage (if required), excavation, loading and hauling (as necessary), and processing. This applies to any type of mineral from precious metals to coal, sand and gravel, and crushed stone. In this very basic reasoning, it is easy to see the importance that the mine must place on the sizing and selection of the excavation, loading, and haulage equipment as this can have a tremendous impact on the economics of the mining operation. For example, according to Blackwell (1999), the operating costs for haul trucks alone may account for one-third to one-half of the total mine operating costs.

Prior to equipment selection, mine planning should have defined the ultimate pit limits based on economic conditions, pit geometry, and production rates. These have a major impact on equipment sizing and selection, which in turn can also affect the design parameters in a circle of refinement throughout the process. Therefore, even during the mine planning stage, considerable attention must be paid to equipment sizing.

For much of the 1990s, the mining industry ran under the assumption that "bigger is better," and this led original equipment manufacturers (OEMs) to develop larger loading and haulage equipment in order to take advantage of economies of scale. As noted by Bozorgebrahimi et al. (2003), there were even several mining industry conferences on this subject, such as the World Mining Equipment Haulage 2000 Conference. But sometimes theory does not match reality; only a few years later, the same forum rearranged the words in the title and asked, "Is bigger better?" The answer coming out of the conference was: "It depends." Based on the application and operating conditions of the mine, the economies of scale anticipated may not be realized because other costs, such as dilution of the ore, complexity of the operating system, and production delays when a larger unit is down, have increased more than enough to offset the savings in operating labor.

Therefore, the issue of selection and sizing of surface excavation, loading, and haulage equipment is more critical than ever. Poor selection can result in higher costs and significantly lower economic performance of the mining operation. After the mining method is defined, it is important to consider the selection of equipment and advantages/disadvantages of one mining "system" over another.

LARGE EXCAVATION EQUIPMENT SELECTION

As defined here, *excavation equipment* refers to equipment that digs material and dumps it directly into an area with limited additional handling. Equipment in this category includes walking draglines, stripping shovels, and bucket-chain and bucket-wheel excavators. Walking draglines and stripping shovels are considered "cyclic" excavators; chain and bucket-wheel excavators are considered "continuous" excavators.

In past mining practices, excavation equipment often played a significant role in the mining operation. However, as modern legal and environmental practices have increased in importance, their application and usage has diminished greatly as mining companies find it difficult to meet modern environmental standards and regulations while using excavation equipment such as stripping shovels and bucket-chain and bucket-wheel excavators. Walking draglines are still used in some geographic areas, but to a lesser degree than in the past. Because of the reduced use of excavation equipment, the following information is provided for introductory purposes.

Walking Draglines

Walking draglines are used most frequently to cast material directly because they have higher bucket capacities and greater reach abilities than other excavation equipment. Designs of

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tubs to provide lower ground bearing pressures and various walking mechanism systems have been developed over the past 30 years to provide a longer life at a lower operating cost. A dragline with a 170-m^3 (220-yd^3) bucket, a 122-m (400-ft) boom, and a mass of 12,700 t (14,000 tons) had been in use in the United States. However, because of environmental considerations, most of the largest draglines, particularly in the United States, have been removed from service.

Good bench preparation for walking draglines requires that the material be well blasted for good bucket loading yet remain sufficiently stable for the dragline to operate and move without the highwall collapsing. It is common for the dragline bench to be prepared so that any point loading that can damage the tub is kept to a minimum. The costing of equipment, such as tracked bulldozers, used for this bench preparation needs to be included when calculating system costs.

Stripping Shovels

A stripping shovel with a bucket size similar to a dragline will be more productive because of its crowd and breakout ability, shorter cycle times, and its ability to handle dense rocks. Although a competent floor is necessary (and may require some preparation), stripping shovels do not require the more extensive bench preparation necessary for draglines, so this additional cost is saved.

The largest stripping shovel used to date had a 138-m^3 (180-yd^3) bucket with a 65-m (215-ft) boom and installed power of 22,500 kW (30,000 hp). As with draglines, the largest stripping shovels have been taken out of service in favor of truck/shovel operations largely because of environmental considerations.

Size Selection

The primary steps in selecting a cyclic excavation machine involve (1) bucket capacity selection, (2) determination of machine geometry, and (3) reassessment of the first two steps into a standard model.

The first step in loading tool selection should be to determine (from the mining method planned) the planned production rate—in whatever units are applicable—and then convert these units into a loose volume to be moved per hour. Table 10.4-1 shows applicable factors for material densities and swell factors for a list of common materials. This will allow the calculation of the required bucket size using the following equation:

$$Q = \frac{P \times T}{3{,}600 \times B \quad \text{FE} \times A} \tag{10.4-1}$$

where Q is the bucket (dipper) capacity, P is the required production (loose volume per hour), T is the theoretical cycle time, BF is the bucket fill factor, E is the job operating efficiency, and A is the mechanical availability expected over the period of operation.

Theoretical Cycle Time

Values for theoretical cycle time (T) can be obtained from information supplied by the OEM or from time studies of similar machines in similar conditions. Table 10.4-2 provides average cycle time for a range of bucket capacities for draglines and stripping shovels.

Bucket Fill Factor

The bucket fill factor (BF) is a factor of the material sizing condition and the ease or difficulty of filling the bucket. Ideally, this factor is best determined by field measurements. However, if this information is not available, typical bucket fill factors can be found in Table 10.4-1.

Job Operating Efficiency

Job operating efficiency (E) is the percentage of time that a machine will actually operate (as dictated by maintenance, scheduling, and operating practices). Good management of the operation, typified by excellent management and supervision, planned maintenance programs, high availability, and so forth, will result in a better operating efficiency. Conversely, poor management, as typified by sloppy maintenance practices, low machine availabilities, and so forth, will result in a lower operating efficiency. Average operating efficiency figures should range between 0.75 and 0.90 (0.83 average).

Mechanical Availability

Mechanical availability (A) is the percentage of time that a machine is running and available to work. In good operating conditions with good maintenance practices, availability will be higher. As conditions deteriorate (extreme temperatures, dusty conditions, etc.) and maintenance practices become poorer, availability will be reduced. Availabilities of 85% to 95% should be achievable in most operating conditions during at least the first few years of equipment operation.

One further adjustment that needs to be made for cyclic excavation machines is to divide the calculated bucket capacity by a propel factor to make allowance for time required for the machine to move. The typical propel factor for a walking dragline is 0.94; a typical propel factor for a stripping shovel is 0.96.

Machine Geometry

Now that the bucket capacity has been selected, the next step is to consider machine geometry—primarily the dumping radius and the dumping height. Normally, the cut width is equal to the pit width and all exposed material is loaded out. In a typical dragline pit, the minimum cut width is determined by the mineral loading and transport equipment requirements. For small stripping shovels, cut width can be as narrow as 15 to 18 m (50 to 60 ft) with larger shovels requiring a pit width of 24 to 30 m (80 to 100 ft). In practical considerations, a narrow pit width allows more efficient use of the spoil space and reduces the dragline cycle time. Allowances will also need to be made if the overburden to be removed has widely variable thickness.

The geometry of cyclic excavation machines can be detailed more effectively by drawing or computing average and extreme pit sections and plans showing both the stripping and loading operations. Multiple iterations can be done to finalize the geometry.

Final Selection

After bucket size and operating geometry (dumping radius and dumping height) are determined, the optimal machine can be selected from OEM literature. Although it may be impossible to match exactly with calculated numbers, the nearest model can be adapted. For example, a reduction in dumping

Table 10.4-1 Bulk density, swell factor, and digability of common materials

Rock	Bank Density (t/m³)	lb/yd³	Swell Factor	Fillability	Digability*
Asbestos ore	1.90	3,200	1.40	0.85	M
Basalt	2.95	5,000	1.60	0.80	H
Bauxite	1.90	3,200	1.35	0.90	M
Chalk	1.85	3,100	1.30	0.90	M
Clay (dry)	1.40	2,400	1.25	0.85	M
Clay (light)	1.65	2,800	1.30	0.85	M
Clay (heavy)	2.10	3,600	1.35	0.80	M–H
Clay and gravel (dry)	1.50	2,500	1.30	0.85	M
Clay and gravel (wet)	1.80	3,000	1.35	0.80	M–H
Coal (anthracite)	1.60	2,700	1.35	0.90	M
Coal (bituminous)	1.25	2,100	1.35	0.90	M
Coal (lignite)	1.00	1,700	1.30	0.90	M
Copper ores (low-grade)	2.55	4,300	1.50	0.85	M–H
Copper ores (high-grade)	3.20	5,400	1.60	0.80	H
Earth (dry)	1.65	2,800	1.30	0.95	E
Earth (wet)	2.00	3,400	1.30	0.90	M
Granite	2.41	4,000	1.55	0.80	H
Gravel (dry)	1.80	3,000	1.25	1.00	E
Gravel (wet)	2.10	3,600	1.25	1.00	E
Gypsum	2.80	4,700	1.50	0.85	M–H
Limonite	3.20	5,400	1.40	0.85	M
Iron ore (40% Fe)	2.65	4,500	1.40	0.80	M–H
Iron ore (+ 40% Fe)	2.95	5,000	1.45	0.80	M–H
Iron ore (+ 60% Fe)	3.85	6,500	1.55	0.75	H
Iron ore (taconite)	4.75	8,000	1.65	0.75	H
Limestone (hard)	2.60	4,400	1.60	0.80	M–H
Limestone (soft)	2.20	3,700	1.50	0.85	M–H
Manganese ore	3.10	5,200	1.45	0.85	M–H
Phosphate rock	2.00	3,400	1.50	0.85	M–H
Sand (dry)	1.70	2,900	1.15	1.00	E
Sand (wet)	2.00	3,400	1.15	1.00	E
Sand and gravel (dry)	1.95	3,300	1.15	1.00	E
Sand and gravel (wet)	2.25	3,800	1.15	1.00	E
Sandstone (porous)	2.50	4,200	1.60	0.80	M
Sandstone (cemented)	2.65	4,500	1.60	0.80	M–H
Shales	2.35	4,000	1.45	0.80	M–H
Taconite	3.68	6,210	1.70	0.80	H

Source: Atkinson 1992.
*E = easy, M = medium, M–H = medium–hard, H = hard.

Table 10.4-2 Approximate cycle times for cyclic excavation machines*

Bucket or Dipper Size		Cycle Time, seconds	
m³	yd³	Dragline	Stripping Shovel
6–27	8–35	62	52
28–45	36–59	65	53
46–150	60–200	68	56

*These values are based on a 120° swing for a dragline and a 90° swing for a stripping shovel, which approximates average field conditions.

radius will result in a reduction in dumping height, and vice versa. Because these units often involve customization, the mining company should work closely with the OEM on final design specifications.

Cost Estimation

Figure 10.4-1 can be used to calculate the estimated hourly ownership and operating costs for the various machinery and equipment referenced in this chapter based on information available from the OEMs and/or actual experiences in similar applications.

1 Machine Sales Price (less tires) Including All Options, Taxes, etc. _____
2 Freight and Insurance (to site) _____
3 Import Duties (if applicable) _____
4 Subtotal _____
5 Assembly/Erection Costs _____
6 Insurance During Assembly/Erection _____
7 Subtotal _____
8 Operating Information
a. Years of Operation
b. Hours/Year
c. Total Lifetime Hours
9 Machine Depreciation and Amortization

$$\text{Cost/Hour} = \frac{\text{Subtotal 7}}{\text{Item 8c}} = _____$$

10 Average Investment Formula

$$= \frac{\text{Subtotal 7} \times (\text{Item 8a} + 1)}{2 \times \text{Item 8a}}$$

11 Interest Rate %
12 Insurance %
13 Taxes, etc. %
14 Total %
15 Interest, Taxes, Insurance, etc.

$$\text{Cost/Hour} = \frac{(\text{Item 14}) \times (\text{Item 10})}{\text{Item 8b}} = _____$$

16 Trailing Cable Costs (if electrically powered)
Capital Cost + Import + Insurance Up to Start of Production _____
17 Trailing Cable Life
a. Years
b. Hours/Year
c. Total Hours
18 Trailing Cable Depression

$$\text{Cost/Hour} = \frac{\text{Item 16}}{\text{Item 20c}} = _____$$

19 Trailing Cable, Average Investment

$$= \frac{\text{Item 16} \times (\text{Item 17a} + 1)}{2 \times \text{Item 17a}} = _____$$

20 Trailing Cable, Interest, Taxes, Insurance, etc.

$$\text{Cost/Hour} = \frac{(\text{Item 14}) \times (\text{Item 19})}{\text{Item 17b}} = _____$$

21 Total Ownership Costs/Hour: Item 9 + Item 15 + Item 18 + Item 20 _____
22 Electric Power Consumption/Hour × Cost/kWh _____
23 Diesel Fuel Consumption/Hour × Cost/Liter (U.S. gallon) _____
24 Preventive Maintenance Supplies & Filters _____
25 Undercarriage (if calculated separately) _____
26 Tire Costs
a. $/Tire
b. Number of Tires
c. $/Set of Tires Item 26a × Item 26b
d. Average Tire Life, Hours

$$\text{Tire Cost/Hour} = \frac{\text{Item 26c}}{\text{Item 26d}} = _____$$

27 Repairs Costs (labor + parts) _____
28 Total Operating Costs Item 22 + Item 23 + Item 24 + Item 25 + Item 26 + Item 27 _____
29 Operating Labor/Hour (include social benefits, taxes, insurance, etc.) _____
30 Total Ownership + Operating Costs/Hour Item 21 + Item 28 + Item 29 _____
31 Cost/Metric Ton (Ton) _____

$$= \frac{\text{Item}}{\text{Metric Ton(Tons)/Hour}} = _____$$

Figure 10.4-1 Estimating ownership and operating costs

Bucket-Chain Excavators

A bucket-chain excavator (BCE) excavates material below the grade of the main house unit and transports it upward and away from the unit. It can mine high outputs in weak, unconsolidated ground. Its primary advantages are its excellent downward-digging ability and its moderate ability to dig upward. Its greatest disadvantages are its inability to dig hard ground or to excavate materials selectively.

Bucket-Wheel Excavators

Bucket-wheel excavators (BWEs) are very effective for mining large volumes of unconsolidated material, although they also offer some productive ability in harder formations. They are also more selective than a BCE and can accurately cut bands as narrow as 100 mm (4 in.), albeit at a lower output. One primary disadvantage is their inability to achieve downward-digging action without special modifications.

The BWE digs with a series of evenly spaced buckets attached to the circular wheel at one end of the unit. The excavated material is fed via a transfer point in the wheel to the belt conveyor system of the excavator for discharge.

Advantage of Continuous Excavation

Continuous excavators have reduced dynamic stresses, lower service weights, reduced maintenance costs, and lower power consumption when compared to cyclic excavation tools. Because of the way they operate and transfer the load, BWEs have a lower ground-bearing pressure, allowing more efficient operation in softer underfoot conditions.

Output of Continuous Excavators

Continuous excavators are normally rated in terms of their theoretical output where

$$Q_{th} = \frac{60Fs}{\text{swell factor}} \tag{10.4-2}$$

where Q_{th} is the theoretical output in bank cubic meters per hour (cubic yards per hour), F is the capacity of each individual bucket, s is the number of buckets discharged per hour, and the swell factor is that of the material being excavated (from Table 10.4-1). It is necessary when comparing multiple models or OEMs to define bucket capacity in the same way in all cases.

The annual output capability can be calculated by

$$Q = Q_{th} \times OH \times SF \tag{10.4-3}$$

where Q is the annual output in bank in m^3 (yd^3), Q_{th} is the theoretical output calculated in Equation 10.4-2, OH is the annual scheduled operating hours, and SF is a service factor reflecting operating efficiency and machine availability. Common service factors range from 0.5 to 0.8, depending on ground conditions, climate, managerial efficiency, and so forth.

Bucket-Chain Versus Bucket-Wheel Excavator

Factors favoring the BCE, provided selective mining is not required, include

- Soft, nonabrasive rocks with high, deep cuts;
- When the initial box cut is already opened;
- Wet pit operations;
- Where transport gradients must be reduced;
- Where cuts must be taken below grade;
- Where specific slope profiles are required; and
- Where there are large undulations in the surface of the mineral bed.

Factors favoring the BWE include

- When selective operation is required;
- Where harder ground and/or some boulders are encountered;
- When higher availabilities and lower machine maintenance costs are of critical importance;
- When high upward digging capability is needed; and
- Where sticky materials are encountered.

LOADING EQUIPMENT SELECTION

A number of different types of loading tools are available for use in the mining industry, including front-end loaders, hydraulic excavators, and electric cable shovels. These are used to directly load material into a haulage unit for transport to a dump or processing facility.

Size Selection

As with cyclic excavation machines, the first step in loading tool selection should be to determine (from the mining method planned) the planned production rate—in whatever units are applicable—and then convert these units into a loose volume to be moved per hour. Equation 10.4-1 can be used to calculate an estimated bucket capacity using the same parameters previously defined.

Theoretical Cycle Time

Again, values for T can be obtained from information supplied by the OEM or from time studies of similar machines in similar conditions. In the absence of other data and as a basic assumption to calculate the required bucket capacity, a cycle time of 30 to 40 seconds per pass can be used.

After a specific loading tool is selected, a reiteration of this calculation can be performed to calculate a bucket (dipper) capacity more specifically. This range is based on truck positioning on the same level as the loading tool and an 80° to 100° swing. If this is not the case, allowances should be made to use a slower cycle time.

Bucket Fill Factor

This is a factor of the material sizing condition and the ease or difficulty of filling the bucket. Table 10.4-1 can be used to approximate the bucket fill factor.

Job Operating Efficiency

As previously defined, average operating efficiency figures should range between 0.75 and 0.90 (0.83 average).

Mechanical Availability

Availabilities of 85% to 95% should be achievable in most operating conditions during at least the first few years of equipment operation.

After this calculation is completed, one additional adjustment should be made to allow for time lost to exchanging trucks. To allow for this lost time, if the calculated figure is divided by 0.80 if using a front-end loader or single-side loading and 0.9 if using a shovel and double-side loading (alternately loading on either side of the shovel), a final bucket capacity requirement can be defined.

The next step in loading tool selection is to consider the type of loading equipment. Figure 10.4-2 shows the applicable range (by bucket capacity) for front-end loaders, hydraulic excavators, and electric cable shovels.

Type of Loader

From the chart in Figure 10.4-2, one or two of the loading tool types may be eliminated in this step. For example, if the required bucket capacity is 15 m^3 (19.6 yd^3), then no electric cable shovels fall into this classification, so the choice comes down to a front-end loader or a hydraulic excavator. Likewise, if the required bucket capacity is 60 m^3 (78.5 yd^3), then the only choice (unless using more than one loading tool to achieve the necessary production) is an electric cable shovel.

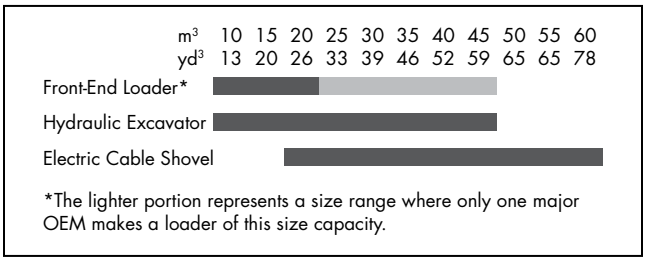


Figure 10.4-2 Loading tool bucket capacities

Table 10.4-3 is a comparison of key points to be considered when selecting among front-end loaders, hydraulic excavators, and electric cable shovels.

If a hydraulic excavator is selected, then a further choice needs to be made between a backhoe front and a loading shovel front. A hydraulic front shovel offers many of the same advantages as the electric cable shovel—high digging forces, good operator visibility—along with the additional advantage of better selectivity because the operator can easily see and control where the bucket is placed into the face. A backhoe front offers the advantage of less required bench preparation, excellent operator visibility into the truck bed to place the load, and the ability to reach more material without moving the unit. The choice between front shovel or backhoe front is often driven by local preferences. Most hydraulic excavator manufacturers also offer machines, particularly the larger size machines, with electric power (rather than diesel engine power). This allows the mine to take advantage of less expensive electric costs (where applicable), but some mobility is also lost.

Productivity Estimation

After a specific loading tool has been selected, the productivity estimate can be recalculated using Equation 10.4-1 but rearranged to yield the production. If truck loading, the calculated production rate should be adjusted by multiplying the

Table 10.4-3 Comparison of loading tool types

Front-End Loaders	Hydraulic Excavators	Electric Cable Shovels
Lowest capital cost	Moderate capital cost	Highest capital cost
Highest operating costs (particularly if tire wear is high)	Moderate operating costs	Lowest operating costs (dependent on costs for electricity)
Very mobile, particularly useful where blending of different material types is required	Mobility to move between faces, but not enough for blending operations	Lack of mobility due to connection to electric grid system
Most efficient bench height at 75% of maximum digging height (lowest of loading tool options)	Most efficient bench height at 75% of maximum digging height	Most efficient digging height with bench height equal to boom point height
Tires make unit susceptible to soft underfoot	Lowest ground pressure for use in soft underfoot conditions	Good ground pressure (but higher than hydraulic excavator) because of much higher operating weight
Bucket must be lowered to ground level to begin dig cycle	Ability to dig from the top down by penetrating the loading face at some level aboveground	Bucket must return to ground level to start dig cycle
Lowest lifetime operating hours; easy transportation and assembly so loader can be moved between mining operations	Lifetime hours 60,000 to 100,000; can be disassembled and reassembled on a different site relatively quickly	Highest lifetime operating hours; requires longer-life mine to take advantage of this
Relatively simple technical training and tooling required	Some specialized training and tooling required	Requires specialized technical training and tooling
Requires very good blasting in order to allow bank penetration and good productivity	Ability to load from the top down provides ability to handle more poorly blasted material	Best suited to work with well-blasted material
Typical cycle times of 30 to 45 seconds	Typical cycle times of 25 to 35 seconds	Similar cycle times to hydraulic excavators
Wide bucket and the need to return to ground level to dig limit ability to mine selected layers or pockets of material	Can selectively mine layers or pockets of material	Difficult to mine selectively because of requirement to return to ground level to begin dig cycle
Mobility provides excellent loading area cleanup	Can use reach of front attachment to keep floor of loading area clean	Typically uses a loader or rubber-tired dozer to keep floor of loading area clean
Material in bucket discharges quickly into truck bed, creating a higher shock load to be transferred onto frame and other components	Allows measured discharge of material from bucket to truck to ease forces acting on truck frame and other components	Allows measured discharge
Develops more greenhouse gases as a result of diesel engine	Develops more greenhouse gases as a result of diesel engine (eliminated with electic-powered hydraulic excavator)	More environmentally friendly because power supplied by electric trail cable
Very strong used market for good quality front-end loaders	Good used equipment market for hydraulic excavators	Limited used equipment market for electric cable shovels due to transportation and reassembly requirements

Table 10.4-4 Passes to load typical articulated truck size*

Truck Size, t (tons)	2.3 m^3 (3.0 yd^3)	3.4 m^3 (4.5 yd^3)	4.6 m^3 (6.0 yd^3)	5.7 m^3 (7.5 yd^3)
18.1 (20)	4			
22.7 (25)	4–5	3		
27.2 (30)	5–6	4	3	
36.3 (40)		5	4	3
45.4 (50)		6	4–5	4

*Based on 100% bucket fill factor and 1.8 t/m^3 (3,030 lb/yd^3) material.

Table 10.4-5 Passes to load typical rigid-frame haul truck size*

Truck Size, t (tons)	15 m^3 (19.5 yd^3)	20 m^3 (26.2 yd^3)	25 m^3 (32.7 yd^3)	35 m^3 (45.8 yd^3)	45 m^3 (58.9 yd^3)
136 (150)	5	4	3		
181 (200)	6	5–6	4	3	
217 (240)		6	5	3–4	
290 (320)			6	4–5	3–4
360 (400)				6	4–5

*Based on 100% bucket fill factor and 1.8 t/m^3 (3,030 lb/yd^3) material.

result by 0.8 or 0.9 (based on operating conditions) to allow for truck exchange time (as described previously).

HAULAGE TRUCK SELECTION

The three types of haulage trucks used in mining operations include on-highway trucks, articulated trucks, and rigid dump trucks. The selection will depend on operating conditions and production required.

On-Highway Trucks

On-highway trucks in mining applications have a fairly limited application and are most often used to transport materials such as coal (commonly in the eastern United States) and construction aggregate (sand and gravel, crushed stone) for further processing and/or delivery to a customer's site. These trucks may be mine owned or (more likely) are contracted from a local trucking company. The size of such units is limited to a maximum of 18 to 20 t (20 to 22 tons). Selection is normally driven by factors such as availability and cost (if leased/contracted) rather than on productivity or size.

There is, however, at least one manufacturer of off-highway style trucks that manufactures up to a 36-t (40-ton) truck for use within the mine site. They purport lower tire costs and lower fuel consumption, resulting in a lower operating cost truck than either articulated or rigid-frame trucks. Depending on operating conditions and longevity of the mine, though, this type of truck could experience a much different pattern of maintenance costs. If a mine is considering a truck of this size-class, a detailed economic analysis should be undertaken to better understand the economics of the different types of trucks. Figure 10.4-1 can be used to calculate estimated ownership and operating costs. A haul study, usually done with the assistance of the various OEMs, will assist in calculating production rates. Several commercially available haul simulation programs are also available; however, these programs do not normally include on-highway trucks in their fleets.

Articulated Trucks

Like on-highway trucks, articulated trucks have a limited application in mining. Articulated trucks have a significant advantage over other haulage equipment in areas with soft underfoot conditions and on distances ranging from 120 to 1,200 m (400 to 4,000 ft). In longer-distance applications and with solid haul road construction, rigid dump trucks are the most likely common choice. Articulated trucks can also be used on steeper grades than rigid dump trucks.

Most OEMs manufacture articulated trucks with capacities from 22.7 to 36.3 t (25 to 40 tons) with a few now manufacturing 45.4-t (50-ton) trucks. Articulated trucks are most commonly loaded by construction-class hydraulic excavators with bucket capacities ranging from 2.3 to 4.6 m^3 (3.0 to 6.0 yd^3). The excavator size selection would be made in accordance with the production requirements, as shown previously in Equation 10.4-1.

At that point, an acceptable guideline is that a truck should be loaded in four to seven passes. Table 10.4-4 provides a pass-match chart assuming various-sized articulated trucks are loaded by various-sized hydraulic excavators.

The number of trucks required can then be calculated by using the number of passes and the pass cycle time to calculate a load time, adding "fixed" times such as positioning at the loading tool and dump time as estimated from experience or from information obtained from the OEM, and finally adding estimated travel times as calculated using either a haul simulation program or the rimpull-speed-gradeability curves and retarder charts available from the OEM. Rolling resistance—a factor related to how far the truck tires penetrate the haul road surface—must also be considered in this calculation when using the charts indicated.

Rigid Dump Trucks

Rigid dump trucks are the backbone of haulage equipment for the worldwide mining industry. Available primarily from five global OEMs—Caterpillar, Komatsu, Bucyrus, Hitachi, and Liebherr—along with other locally manufactured and supported units, capacities range from 36 to 360 t (40 to 400 tons) with the smaller trucks of 36 to 90 t (40 to 100 tons) considered as construction/quarry trucks (also used in smaller metal/nonmetal mining operations) and the larger trucks—greater than 90 t (100 tons)—considered as "mining-class" haul trucks.

Size Selection

Mine operators typically target to load a haul truck in three to five passes from the loading tool. With hydraulic excavators and front-end loaders, the range is four to seven passes; with electric cable shovels, the range is three to four. If a truck is loaded in fewer passes, the large amount of material dumped at one time can cause excess stress on tires and structural components of the truck. If it takes more passes to load, the truck waits excessively in the loading area, producing inefficiencies.

Table 10.4-5 shows the number of passes required to load different haul truck sizes with various loading tool bucket sizes ranging from 15 to 45 m^3 (19.5 to 58.9 yd^3). Major electric cable shovel manufacturers have started to rate buckets by the amount of material carried in each (i.e., 90-t [100-ton] load). The bucket capacity is sized to match the mine's material density, making it an easy calculation to determine the number of passes required to load a truck.

Mechanical Versus Electric Drive

One of the primary decisions to be made in haul truck selection is whether to use a mechanical-drive truck or an electric-drive truck. Some OEMs offer both mechanical- and electric-drive trucks in the mining class (greater than 136 t [150 tons]) while others offer only electric drive. Within the electric-drive option, there is also a choice between direct current (DC) and alternating current (AC), although AC-drive offers additional advantages over DC-drive and is becoming the more common choice in today's market.

Electric-drive trucks typically travel at higher speed on grades ranging from 4% to 10%, have potentially lower maintenance costs, offer slightly better fuel economy, have a smoother operator ride, and offer better retarding capacity to stop the truck. However, they do have a higher capital cost and require more specialized technical training and capabilities. Most OEMs also offer the ability to convert the electric-drive truck to a trolley-assist truck, resulting in much higher speeds on steep grades.

Mechanical-drive trucks can more effectively travel on steeper grades (greater than 10%), have a larger market presence resulting in more knowledge in the field, have a lower capital cost, and require a lower level of technical specialization. They are also lighter-weight vehicles.

Productivity Estimation

After a loading tool and truck capacity are selected, the productivity of the "system" should be calculated in order to determine the number of trucks necessary and to ensure that the estimated productivity of these discrete units operating together will still meet the required production.

By far, the most efficient way to calculate this productivity is to utilize either one of several commercially available simulation programs or to utilize an equipment manufacturer's program. These programs will estimate a production based on the unit load time, fixed times for the loading tool and the truck, and other data input. At least one haul road profile will be required so that the simulation can be performed based on a specific site condition. Alternatively, the only way to estimate fleet productivity is to calculate travel speeds for a given grade using an OEM's rimpull-speed-gradeability chart and add in the applicable fixed times for the loading tool and truck. Use of computer programs will also allow much quicker evaluations of different loading tools and truck fleet sizes.

OTHER EXCAVATION TOOLS

Many mining production systems incorporate some combination of excavation and transportation of material. These include wheel loaders used in a load-and-carry application, tractor scrapers, bulldozers, loading draglines, and surface miners. These are often "auxiliary" production systems or used in applications with lower production requirements.

Load-and-Carry Wheel Loaders

In addition to truck-loading applications, wheel loaders can also be used in load-and-carry applications, either to dump directly into an adjacent mined-out area (such as in an overburden stripping application) or to dump into a portable crusher (and conveying system). The key here is to minimize the distance the loader must travel, as longer distances result in slower cycles and increased tire wear, a significant cost factor in load-and-carry operations. Load-and-carry applications should be limited to less than 120 m (400 ft).

Productivity of a load-and-carry operation can be calculated as follows:

$$t_c = \text{time to load bucket} + \text{travel loaded} + \text{dump time} + \text{travel empty} \quad (10.4\text{-}4)$$

where t_c is the loader cycle time in seconds. Travel time is calculated by dividing the distance to be traversed (meters) by the travel speed (meters per second). Travel speeds can be determined from OEM literature and specifications.

$$P = (3{,}600 \times Q)/t_c \quad (10.4\text{-}5)$$

where P is the productivity in cubic meters per hour, Q is the bucket capacity in cubic meters, and t_c is the cycle time in seconds calculated in Equation 10.4-4.

Tractor Scrapers

Another excavation tool found in some mining applications, particularly overburden stripping (with haul distances ranging from 120 to 1,200 m [400 to 4,000 ft]) is the tractor scraper. They may also be found in some medium hardness rock applications that fragment well after blasting. Many models of scrapers are available, with most capacities ranging from 15 to 34 m^3 (20 to 44 yd^3). As noted in the *Caterpillar Performance Handbook*, four primary types of scrapers are available: single-engine conventional scrapers, tandem-powered units, elevating scrapers, and auger scrapers (Caterpillar 2008).

Single-engine conventional scrapers have the widest range of applications and commonly require the use of a pusher tractor (bulldozer) to be loaded most effectively and economically. Conventional scrapers operate most effectively with lower haul grades, lower rolling resistances, and better floor conditions.

Tandem-powered units offer higher tractive effort capabilities, meaning they can be used in softer underfoot conditions and in higher rolling resistance applications. Normally, tandem-powered units are also pushed by a pusher tractor to assist in loading.

Elevating scrapers are self-loading and best used in applications with short to medium haul distances. They also do not perform well in adverse grades and with high rolling resistances. Elevating scrapers do not perform well in sticky materials (such as some clays) or material containing rock.

Auger scrapers are also self-loading units and, like elevating scrapers, do not work well with sticky or rocky materials. The auger scraper does offer improved tire life and excellent ejection characteristics, and is suitable for a range of conditions.

Type Selection

As detailed in *A Reference Guide to Mining Machine Applications* (Caterpillar 2001), there is some overlap between applications of the different types of scrapers, but the following can be used as a basic guideline:

- Short distances, lower grades—elevating or auger
- Short distances, higher grades—tandem-powered
- Longer distances, moderate grades—conventional
- Low rolling resistances, short distances—elevating or auger
- Moderate rolling resistances, longer distances—conventional
- Higher rolling resistances, variable distances—tandem

Productivity Estimation

Production with a scraper can be estimated using a methodology similar to that used previously with other types of equipment, by considering both fixed times (time to load and dump the scraper) and variable times (travel times loaded and empty), utilizing OEM rimpull-speed-gradeability charts.

$$\begin{aligned}\text{total cycle time (TCT)} = {} & \text{fixed time} \\ & + \text{travel loaded} \\ & + \text{travel empty}\end{aligned} \tag{10.4-6}$$

Fixed time for scrapers will include the load time (typically 0.5 to 1.0 minutes, depending on type of scraper and actual conditions) and the time to maneuver and spread or maneuver and dump the material (typically 0.6 to 0.7 minutes).

Scraper productivity can then be calculated as

$$P = \frac{\text{scraper heaped capacity} \times \text{fillability} \times 60 \times E \times A}{\text{swell factor} \times TCT} \tag{10.4-7}$$

Bulldozers

Bulldozers are not usually considered as primary excavation tools but can be used to supplement the primary excavation tools. One particular application is in coal mine overburden stripping where cast blasting is employed and bulldozers are used to push a significant amount of material into an already-mined adjacent pit. Excavation with a bulldozer should be limited to less than 100 m (328 ft).

To calculate the production that can be achieved by a bulldozer, the blade capacity of the bulldozer under consideration can be obtained from the OEM literature. This would provide a volume per pass in loose cubic meters and can be converted to bank cubic meters by dividing the blade capacity by the swell factor (from Table 10.4-1).

The cycle time can be calculated as

$$t_c = \frac{\text{distance}}{\text{dozing speed}} + \frac{\text{distance}}{\text{reverse speed}} \tag{10.4-8}$$

Generally, dozing speeds of 1.5 to 2.5 km/h (0.9 to 1.5 mph) will be the most economical. Reverse speeds (generally in third gear) can be found in OEM literature.

When considering bulldozer production, the following should be noted:

- If a ripper is used, the production (when dozing) will be increased, but the time required to rip will need to be accounted for.
- Any available slope should be taken advantage of to doze downhill.
- Steady dozing pressure should be maintained.
- Speeds should be reduced in areas of heavy shock and impact conditions.

Loading Draglines

Crawler-mounted draglines used to load directly into haul trucks have limited application in modern mining methods for many reasons: (1) they have limited crowd and breakout action compared to shovels, backhoes, and front-end loaders; (2) cycle times are longer when compared to other loading tools, resulting in lower production rates; and (3) they have limited ability to spot the bucket and dump into a truck.

Several areas in which loading draglines do find an application are in deeper wet pits (such as underwater sand and gravel operations) and in secondary roles such as excavating box cuts, digging sumps, final pit cleanup, and other areas where they are not the primary production tools.

Estimating production and operating costs of loading draglines can be completed using previously introduced methods adapted to this type of equipment.

Surface Miners

Relatively new to use in surface mining excavation, surface miners can be used to excavate material where drilling and blasting is prohibited (such as in some quarry applications) or where selective mining is required or where materials have a relatively lower compressive strength. Surface miners can be used in either a discontinuous system (where the surface miner cuts the material and loads it into a haul truck) or in a continuous system (where the material is wind-rowed behind or to the side of the machine for later loading (usually by a front-end loader into a haul truck). Combinations of a surface miner with a conveyor system can also be achieved.

In a discontinuous system, minimum operating cost is achieved where the truck body is sized so that the truck remains still while the surface miner moves forward and loads the truck uniformly over the full length. The maximum volume of each cut can be calculated as

$$\begin{aligned}0.6 \times {} & \text{bucket-wheel diameter} \\ & \times \text{width of the bucket-wheel head}\end{aligned} \tag{10.4-9}$$

In a discontinuous system, the overall productivity will be limited by the number of haul trucks to be used and the distance and haul profile over which the material can be moved.

According to information from Wirtgen (2000)—one of the leading surface miner OEMs—in continuous cutting operations, the surface miner can achieve outputs up to 1,400 m^3/h (1,831 yd^3/h) (bank) depending on material compressive strength. Maximum productivity is achieved in materials with a compressive strength up to 40 MPa (6,000 psi). Reduced productivity is achieved in materials up to 80 MPa (12,000 psi), and in special cases, such as small lenses of material or thin layers of up to 120 MPa (18,000 psi). Some information sources have shown a range of five to seven times higher productivity in material with a compressive strength of 10 MPa (1,500 psi) versus a material with a compressive strength of 80 MPa (12,000 psi).

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CHAPTER 10.5

In-Pit Crushing

Ronald W. Utley

To establish and maintain competitiveness in international markets for mineral, coal, and stone products, it is necessary to adopt the latest proven technology and economic systems in open cast mining. In today's markets, overburden of increasing thickness has to be stripped, transported, and dumped; distances to the stockpile are becoming longer; depths of mines and quarries are increasing; ore grade is decreasing; and costs for energy and labor are continuously escalating.

Trucking of waste and ore from mines, quarries, and pits is a flexible materials-handling transportation system. Mine planners, especially at the start of a greenfield project, find that trucking is the easiest transportation system to design and plan for. As the pit or quarry becomes deeper or farther away from the delivery points, mine planners and designers should perform trade-offs between cost and flexibility of transportation systems. This will ensure that their operations will continue to have the best and most economic materials-handling systems for their operations.

As pits have become deeper and their capacities increased, in-pit crushing and conveying (IPCC) has become the comminution and transportation method of choice for most mine planners. In response to the IPCC option, truck manufacturers developed larger trucks. When long-term planning is possible, IPCC is preferred for the materials-handling transportation system.

There are three main steps in designing an excellent crushing plant: (1) process design, (2) equipment selection, and (3) layout. The first two are dictated simply by production requirements and material characteristics, but the layout can reflect the inputs, preferences, and experience of a large number of parties. These can include the owner's engineering staff, operations and maintenance personnel, equipment manufacturers, and especially the mine planners.

The types of in-pit crushers usually reviewed by mine planners for hard ore are fixed plants mounted at the rim of the pit and semimobile and fully mobile plants within the pit. The gyratory crusher is the crusher of choice for capacities over 2,500 t/h (metric tons per hour) [2,755 stph, or short tons per hour]). For soft-rock applications, including overburden, coal, and oil sands, fully mobile continuous mining systems are becoming increasingly attractive. The crushers most often selected for these applications are low-speed sizers and double-roll crushers. Two types of conveyor systems being selected for these systems are the conventional conveyor and high-angle conveyor. The high-angle conveyor system has yet to be put in operation in a high-tonnage mining operation.

IPCC DEVELOPMENT

In 1956, the first *mobile*, or self-propelled crusher, was installed in a limestone quarry in Hover, Germany. Use of these earliest mobile crushers solved the major problem of wet, soft ground conditions that did not permit the use of haulage trucks because of the high cost of building and maintaining haulage roads. Quarry operators also wanted to take advantage of continuous belt conveyor haulage systems and the resulting cost savings.

European mines, particularly in the coalfields, are frequently characterized by soft materials, allowing the use of bucket-wheel excavators, followed by a continuous haulage system of belt conveyors. In the late 1950s, contemporary quarry operators with background knowledge of belt conveyors in mining easily accepted the concept after in-pit mobile crushing solved the problem of run-of-quarry size reduction required for conveyor application.

Large-capacity in-pit crushing started as early as 1969–1970 in Alcoa's bauxite mines in Western Australia. In the late 1970s the first fully mobile gyratory crushers 1,370 × 1,880 mm (54 × 74 in.) were installed in South African open cast mines. All of these crushing plants were fully mobile (with walking feet), although one was used with dump trucks and stayed in position for several years while the others were working at the face in conjunction with shovels and were moved several times a shift.

Three events took place in the early 1980s that laid the foundation for the worldwide acceptance of IPCC:

1. The oil embargo of 1979 put mine operators on notice that they should not tie their future to conventional truck haulage systems. Approximately 50% of truck operating costs are related to fuel, lubrication, and tire consumption, which are either directly or indirectly dependent on

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the cost and availability of petroleum products. In the late 1970s and early 1980s, petroleum costs (and thus the cost of their derivative products), as well as availability, were unpredictable because of cartel pricing policies and an extremely volatile world market. Severely increased truck haulage costs, coupled with potential fuel shortages, forced greater emphasis on the development of haulage systems that either eliminated the need for trucks or at least greatly reduced truck haulage distances.

2. Duval Corporation installed a movable, indirect feed crushing and conveying system that incorporated a 1,500 × 2,260 mm (60 × 89 in.) primary gyratory crusher with a rated capacity of 3,629 t/h (4,000 stph) in the company's Sierrita copper-molybdenum open-pit mine, located near Green Valley, Arizona (United States).
3. The U.S. Bureau of Mines presented its highly published treatise, "Movable Crusher Systems—Concepts and Application" at the American Institute of Mining Engineers In-Pit Crushing and Conveying Symposium in Salt Lake City, Utah (United States) in 1980.

This combination of need, visibility, and credibility built on 25 years of related experience and was the springboard for change. In the 30 years since the Sahuarita IPCC in Arizona became operational, the mining industry has made great strides in improving efficiency. IPCC systems have significantly contributed to increased profitability through reduced production costs. By limiting truck haulage to short distance material transport between the face and the crusher, and, in some cases, by completely replacing trucks, the installation of IPCC systems achieved a marked reduction in overall mining costs. Due to the economies provided by IPCC, mines have been able to operate economically with lower ore grades, greater pit depths, and at higher capacities.

A series of events have taken place since 2006 which is again changing IPCC:

- The larger hard-rock mines of the world are operating at or approaching 10,000 t/h (11,000 stph). Overburden removal for some of the hard-rock mines is exceeding 10,000 t/h and soft-rock mines, including coal and oil sands, are approaching 14,000 t/h (15,432 stph).
- It appears that truck sizes have reached their maximum of 363 t (400 st) while shovels have continued to get bigger. Truck manufacturers have stated that construction of a 500-ton truck is possible, but to date none has been built. The development of an efficient mine plan can only be achieved through an analysis of the integration of material removal, loading, hauling, and dumping. With the larger shovels, the largest trucks are now the limiting factor in optimizing load-and-haul production. For large production mines with long-haul distances, IPCC has become more attractive.
- The hard-rock crusher of choice, the gyratory crusher, has either reached or is close to its maximum practical size. From the outset, in 1985, the 1,500 × 2,260 mm (60 × 89 in.) size was the crusher of choice for IPCC. The 1,500 × 2,970 mm (60 × 117 in.) size followed in 1990. Manufacturers have been offering 1,800 × 3,250 mm (72 × 128 in.) size machines for IPCC, but, to date, no orders have been placed.
- Double-roll crushers and low-speed sizers have been developed for soft rock, coal, and oil sands for capacities of 12,000 t/h (13,228 stph) and designed for 14,000 t/h (15,432 stph).
- The world went through another round of oil price increases, which, in turn, caused another round of increases in mining costs based on fuel for truck fleets—a reminder that the best way to ensure stable power costs is with electric power rather than oil.
- Deliveries of new truck tires for haul trucks increased to more than 2 years (briefly from 2006 to 2008), threatening the shutdown of some operations and increasing the cost of tires.
- The concern over global warming has heightened the awareness of minimizing carbon dioxide (CO_2) emissions. These requirements may favor IPCC in general and continuous mining in particular. However, the plant designer needs to consider the delivered cost of electric power when comparing electric power–driven systems with diesel truck systems.
- IPCC plants have been required to add dust suppression and/or dust collection systems.

CONTINUOUS IPCC SYSTEMS DEVELOPMENT

The use of continuous mining systems is primarily dependent on the type and properties of the ore being mined. In the case of light and loose earth, bucket-wheel excavator technology, combined with a system of conveyors, offers the advantages of a continuous mining system. In order to take advantage of continuous mining in harder ore, such as minerals and hard coal, crushers are required to reduce the run-of-mine (ROM) ore to a conveyable size.

The objective of the fully mobile continuous crushing system is to totally eliminate the need for trucks by having the shovel feed the ROM ore directly to a continuous materials-handling system. The crushing needs to be fully mobile, such that it follows the movements of the shovel, and has to be designed to suit the movement of the shovel boom and bucket, and match the operating capacity of the shovel.

The ore is fed directly from the shovel to the crushing plant, is crushed to conveyable size, and discharges to a system of shiftable and fixed mine conveyors. To increase flexibility, a short mobile transfer conveyor can be added to the system. In most cases, existing cable shovels or hydraulic excavators can be used if a mobile crushing plant replaces the truck fleet. The hopper height and geometry are similar to the truck box, so the shovel operation is similar whether loading trucks or the mobile crushing plant.

A characteristic feature of continuous open-pit mining technology is its long life expectancy. The large number of such examples includes the in-pit crushing system at the Morenci open-pit copper mine in Morenci, Arizona, which commenced operations in the late 1980s. Another example is the fully mobile primary gyratory crusher system at the Foster Yeoman limestone quarry in England, which also dates to the mid-1980s.

Advantages of IPCC

The main reason for the implementation of semimobile and mobile crushing plants instead of fixed crushing plants is the optimization of material transport around and out of the pit on its way to the waste dump or processing plant. In this case, *optimization* means the overall cost comparison of truck and conveyor transport in combination with crushing plants, stockpiles, and dumping equipment.

The basic design of crushing plants has not changed much in the last few years. Only the environmental protection requirements for mining companies to install dust suppression and/or dust collection equipment have been added. (Almost all IPCC gyratory crusher plants built in the past years are of the semimobile direct-dump design.)

The selection and design of a crushing plant depend on evaluation and consideration of the following:

- Kind of material to be crushed
- Tonnage of material to be conveyed
- Area, depth, and development of the open pit
- Space availability (height, area) at the favorable crusher locations, taking into consideration the mine design and especially the ore that becomes inaccessible underneath the installation
- Conveying options out of the pit
- Type of downstream material flow in connection with the utilization of the entire crushing and conveying system
- Comparison of maintenance for feeding the crushing plant by direct dump or with an apron feeder

Advantages of belt conveyor haulage as compared to truck haulage include the following:

- Stationing the crusher in the pit reduces cost by shortening the haulage distance between the loaders and crushing plant.
- Operating costs associated with fuel, tires, and lubricants are reduced. These products tend to increase in price at a rate that exceeds the rate of monetary inflation.
- Labor costs are reduced. Although most in-pit systems, either operating or planned, use truck haulage, the haulage distance is shorter and the number of trucks can be partially or totally reduced. This reduction produces a corresponding decrease in operators and maintenance personnel.
- Compared with truck haulage, safety risks are reduced.
- Because mining ventures are long term, in-pit crushers and conveyors offer greater predictability for future costs.
- Dependence on the availability of fuel is greatly reduced.
- Dependence on rubber tires is greatly reduced.
- Conveyors can traverse grades of up to 30° versus approximately 10–12° for trucks. This ability facilitates shorter haulage distances and reduces haulage road construction.
- Conveyors can easily cross roads, railways, waterways, and other obstructions.
- With the reduction of haulage costs, lower-grade ore bodies can be mined economically. This is particularly important because many established ore bodies are decreasing in grade with an increase in depth.
- CO_2 emissions are greatly reduced.
- Downhill conveyors can produce regenerative electrical power instead of dissipating heat, as is the situation with trucks braking.
- Conveyors are more energy efficient than trucks.
- Conveyors require less skilled labor for maintenance than trucks.
- IPCC equipment can achieve maximum operational availability because of greater independence from weather conditions such as fog, rain, snow, and frost.
- The cost of haulage road maintenance is significantly reduced by using conveyors.
- Continuous flow of material can be maintained by using conveyors.
- With the availability of technologies such as finite element analysis and computer simulation, in-pit crushing stations have been refined to a point where their performance and integrity is equal to that of traditional crusher stations. By allowing for future relocation as the mine expands, long-term mine planning is more flexible.

Disadvantages of IPCC

The principal drawbacks of belt conveyor haulage as compared to truck haulage are the following:

- Short-term flexibility is reduced. The great mobility of trucks allows mine managers extreme flexibility in the mine plan. Once an overland conveyor is installed, it is prohibitively expensive to move as part of a mine plan change.
- Upfront costs are higher.
- While the semimobile system is being moved, truck capacity may be insufficient to feed the process or to strip waste.
- Remote sites are particularly susceptible to parts shortages. Truck haulage offers a range of capacities, while conveyor systems offer no capacity if parts are not available.
- Capacity increments are easier to achieve with trucks compared to large IPCC systems.
- In mines where ore blending is important, truck flexibility provides an added advantage.
- Lump size is limited. Once blasted, ore and waste in hard-rock mines can be loaded directly into a truck and hauled out of the pit. Generally, for conveyors, it is necessary to crush the blasted ore or waste.
- A key issue with the IPCC concept has been the inability to maintain projected production over extended periods and maintenance problems, since the shutdown of any one piece of equipment in an IPCC system would shut down the whole system.
- The production of the IPCC system is dependent on how it is loaded. If the excavator/shovel is not working in harmony with the rest of the IPCC system, maximum production cannot be obtained on a consistent basis.
- Maintenance of the complete IPCC system has to be monitored on a continuous basis. The purpose is to minimize the downtime potential of the entire IPCC system while any one individual component is being maintained.

DESIGN PARAMETERS

The principal design parameters that drive IPCC selection and configuration include

- Production requirements,
- Truck sizes,
- Capital and operating costs,
- Ore characteristics,
- Ore-body geometry,
- Reserve life,
- Estimating infrastructure and equipment,
- Availability of power and diesel,
- Country risks,
- Safety and environment,
- Project location (climate, geography, terrain),
- Life-of-mine/expansion plans,
- Operational considerations, and
- Maintenance requirements.

Table 10.5-1 Requirements for process design

Process	Requirements
Process description	General
	Primary crushing
General material characteristics	Maximum rock size
	Material specific gravity
	Material bulk density
	Material hardness
	Material abrasiveness
	Clay content
	Material moisture, wet season
	Material moisture, dry season
	Angle of repose
	Angle of withdrawal
Operating schedule	Days per year
	Hours per day
	Nominal annual throughput
	Mining shifts per day
	Crushing plant shifts per day

Production Requirements

The process design criteria define the project's production requirements. Typical requirements are shown in Table 10.5-1.

Crusher station–receiving hoppers have to be designed to handle, as quickly as possible, ore that is delivered by the largest trucks (of loading equipment) in the fleet. The typical design capacity of the feed hopper is two truckloads and in some cases up to three trucks' capacity. The discharge chamber below the crusher has typically been designed to hold a minimum of 1.25× the capacity of the receiving hopper to prevent the crushed ore from backing up into and damaging the crusher. As the standard size of today's mine trucks are 300- to 400-ton capacity, the size of the crushing stations must be very large and high just to contain the ore. Some of the semimobile crushing stations now being designed are direct-dump stations with high-speed crushed-ore-receiving conveyors capable of removing the material at rates in excess of the crusher's maximum capacity. The crushed rock is then taken directly to the out-of-pit transport conveyor or into an in-pit surge bin. Use of the high-speed belts has reduced the discharge chamber capacity by as much as 50%. The initial semimobile gyratory crushers were fed with an inclined apron feeder, which allowed the overall height of a station to be contained within two bench heights and allowed for instantaneous dumping of material into the hopper.

Capital Costs

Large semimobile primary crushing plants can be very costly, especially if they include the inclined apron feeder. It is unwise to estimate crusher installation costs based simply on equipment price plus a contingency allowance for other costs. The following direct costs, including installation labor hours, must all be taken into account:

- Earthworks and civil engineering
- In-pit construction planning, in order to prevent the interruption of ongoing operations
- Concrete
- Structural steel
- Architectural
- Mechanical
- Electrical and instrumentation

Indirect costs can be at least half as much again as direct costs and include

- Engineering, procurement, and construction management;
- Start-up and commissioning;
- Construction equipment;
- Spare parts;
- Freight;
- Taxes;
- Escalation; and
- Owner's costs (relocation, hiring and training personnel, permits, licensing fees, etc.).

Ore Characteristics

Ore characteristics are a critical element in both crusher and conveyor selection. Dry ores require greater provisions for dust suppression and collection. Wet, sticky ores can plug chutes and crushers, reduce surge capacity, and misalign belts.

For mines at which ore characteristics change over time, it can be costly to initially design a plant without the necessary flexibility. Some owners stipulate that initial capital investment be kept to a minimum, with design modifications paid for out of the operating budget. This is not always easy to achieve.

Project Location

A project's geographical location, topography, remoteness, and climate all affect the crusher plant design. Construction costs are generally much greater at high altitudes, in cold climates, and at remote sites. Modular construction and subsequent transportation to the site can improve the economics of such a project.

Geography dictates what material can be best used economically in a particular region. A flat quarry operation lends itself to having the conveyor installed in one position for long periods of time. A deep copper pit will sometimes require that the crushing station and receiving conveyor need to be moved. Naturally, it would be best to find a wall that requires no more setbacks. The conveyor could then be installed either up this face with a high-angle conveyor or in a slot designed to install a conventional conveyor. Another alternative would be similar to the setup at the Island Copper installation on Vancouver Island in British Columbia (Canada) where the operator installed a conveyor in a tunnel up at 15° out of the pit.

Life-of-Mine/Expansion Plans

The life of the mine is a key element in the design of any crushing plant. The selection of a fixed crusher versus a semimobile plant is an important design consideration in the overall life of a mine. Moving a crushing plant and adding feed conveyors to the takeaway conveyor can be expensive.

Any expansion plans for most IPCC systems should be built into the crusher and conveyor systems at the start of a project. A conveyor system's tonnage can be increased in the future simply by speeding up the conveyor and, if required, adding additional drives.

Operational Considerations

It is important to provide a comfortable, well-ventilated workspace with drinking water and restroom facilities nearby. Also, the operator should be able to see all the main parts of the crushing facility under his control, either through a good window or by means of TV cameras/monitors. Vibration and

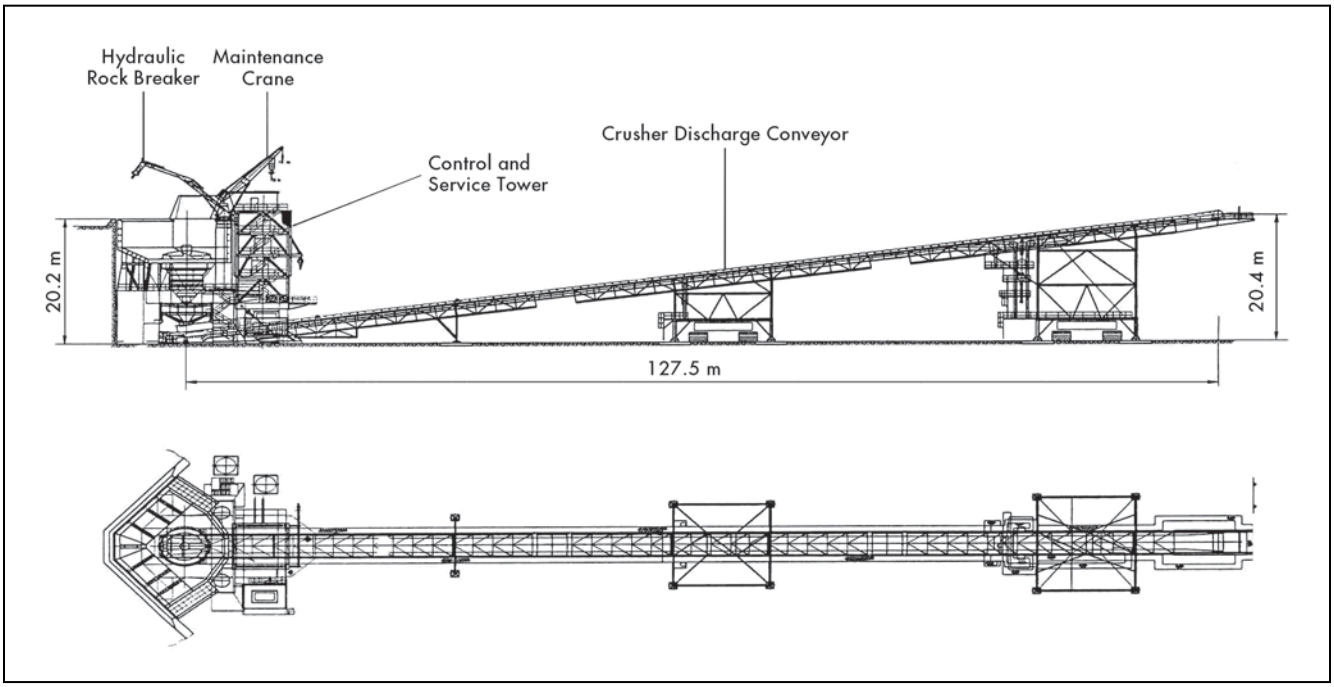


Figure 10.5-1 Typical IPCC semimobile crushing plant layout

noise at any crusher station must be kept to a minimum. The conveyor should have vehicle access along at least one side.

Maintenance Requirements

Keeping maintenance requirements to a minimum helps achieve higher overall operating availability. Scheduled preventive maintenance at the crusher station and conveyor involves a number of elements, including

- Crusher wear parts,
- Feeder wear parts,
- Conveyor skirting and adjustment,
- Oil and lubrication,
- Conveyor belt repair,
- Electrical and instrumentation adjustments, and
- Visual inspections.

Provisions must be made for either jib or mobile cranes to remove and replace crusher wear parts, concaves, and the main shaft. Trolleys, jib cranes, and pull-points should be designed to facilitate equipment maintenance. Oil and lubrication systems should be centralized and designed for easy, automatic changes, with provisions for well-ventilated, centralized lubrication rooms, where possible.

PROCESS DESIGN CRITERIA

Typically, the information required to develop IPCC system design criteria include

- Geographic data,
- Climatic data,
- Process design data (process description, ore characteristics, etc.),
- Civil design criteria,
- Structural design criteria,
- Mechanical design criteria, and
- Electrical/instrumentation design criteria.

PLANT LAYOUT AND DESIGN

A carefully designed layout can save significant investment dollars, because structures and infrastructure (rather than major equipment items) represent the major cost element of the crushing plant (Figure 10.5-1). The mine planner and plant designer must prepare a layout that meets the needs of the design criteria and flow sheet, as well as selecting the equipment in the most economic possible configuration. It is important to keep structural costs down, to design for ease of maintenance and operation, and to combine best practices with advances in fabrication and erection. Most in-pit crushing plants are designed by crusher manufacturers, so it is imperative that the designer works closely with the selected equipment supplier. The manufacturer must remember that the production, process, economic, safety, and operational design needs come first.

Three-dimensional (3-D) computer-aided design (CAD) systems tied up with the mine planning 3-D modeling assists greatly in being able to visualize the finished and phased effects of any IPCC-installed materials handling transportation system.

PRIMARY CRUSHER SELECTION

The crusher is the key to success with any IPCC system. The in-pit crushing plant can be provided with almost any type of primary rock crusher. Selection of the primary rock crusher is based on three fundamental considerations:

1. Type and characteristics of the ore, which determine the type of crusher required
2. Plant capacity, which determines the size of the crusher
3. Plant layout and design

As the term *primary* implies, primary crushers are used in the first stage of any size reduction cycle. The gyratory crusher is the workhorse of the hard-rock crushing industry. Primary

Table 10.5-2 Primary crusher capacity

Crusher	1,500 stph	3,000 stph	6,000 stph	9,000 stph	12,000 stph	15,000 stph
Gyratory						
Double-toggle jaw						
Single-toggle jaw						
Double-roll						
Low-speed sizer						
Hybrid roll/sizer						
Impactor						
Hammermill						
Feeder breaker						

gyratory crushers are capable of taking blasted ROM and run-of-quarry feed in size up to 1,500 mm (60 in.) and producing products ranging in size from 50 to 300+ mm (2 to 12+ in.). This type of crusher can sustain production at rates between 318 to 9,072 t/h (350 to 10,000 stph), depending on the feed characteristics, crusher setting, and crusher size of the application.

The primary gyratory crusher is only one of a family of primary crushers that include the following:

- Gyratory crusher
- Gyratory jaw
- Double-toggle (DT) jaw crusher
- Single-toggle (ST) jaw crusher
- High-speed double-roll
- Low-speed sizer
- Hybrid roll sizer
- Impactor
- Single-shaft hammermill
- Double-shaft hammermill
- Feeder breaker

All primary crushers can be used in mobile plants. Impactors and hammermills are compact and generate a high reduction ratio. The high speed of the machines requires special attention to dynamic forces.

The jaw crushers are good for small-tonnage plants, and the ST jaw crusher has the advantage of being lighter in weight than the DT jaw crusher. Large-capacity jaw crushers result in large crushing plants.

Double-roll crushers are large machines in which the two rolls rotate inward so the out-of-balance forces are minimized. These machines are limited to relatively soft and nonabrasive materials.

However, since the return on investment for crushing and conveying systems in the mining industry is heavily dependent on both high capacity and the ability to handle hard and abrasive ores, the gyratory crusher has been the reducer of choice throughout the evolution of IPCC (Table 10.5-2).

PRIMARY CRUSHER TYPES FOR LARGE CAPACITY

Gyratory Crusher

Primary gyratory crushers (Figure 10.5-2) are typically furnished with radial feed openings of 1,065 mm (42 in.), 1,370 mm (54 in.), and 1,500 mm (60 in.). The largest radial feed opening of any primary gyratory crusher operating in the world is a 1,800-mm (72-in.) Traylor crusher. The capacity of even the smallest standard unit, the 1,065-mm (42-in.) gyratory crusher, can be sustained at about 2,500 t/h (2,572 stph). The 1,500-mm (60-in.) gyratory crusher can crush up to 10,000 t/h (11,000 stph), depending on the crusher design, ore characteristics, and desired product size.

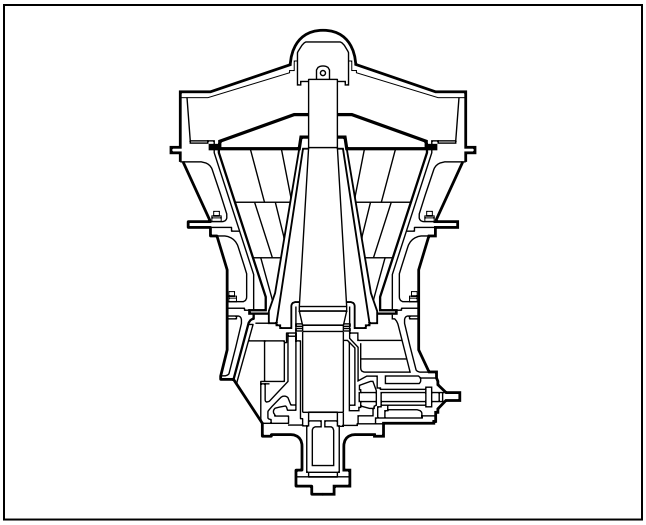


Figure 10.5-2 Gyratory crusher

The first 1,500-mm (60-in.) gyratory crusher was manufactured by the Traylor Engineering & Manufacturing Company in 1919. At that time, the largest haulage trucks available had a 34-t (35-ton) payload, and shovels were manufactured to match. In 2009, the operator had at his disposal 90-yd^3 shovels and haulage trucks with a 363-t (400-ton) payload. Truck manufacturers have advised that 500-ton trucks are a possibility. Tires are the only limitation. The result of larger haulage trucks is an obvious mismatch between the top size of ore fed to the crusher and the largest radial feed opening available. The consequence of this mismatch is the bridging of two or more large lumps that have been fed to the crusher at the same time. Bridging has been partially compensated for by the use of hydraulic rock breakers installed on pedestal-mounted booms. In some installations, the hydraulic rock breaker is employed up to one-fifth of the total time the crusher is in operation with 20% spent breaking oversize rock and 80% spent breaking bridges. Bridging has also been mitigated by improvements in blasting technology.

The largest standard size gyratory crusher, the 1,500 × 2,970 mm (60 × 117 in.), typically operates at the upper limit of the capacity range. These machines have been optimized. With improvements in gearing, bushing materials, and lubrication systems, the speed has been increased. Greater horsepower is available from the manufacturers. The crushing chambers have been optimized with CAD. Larger-size

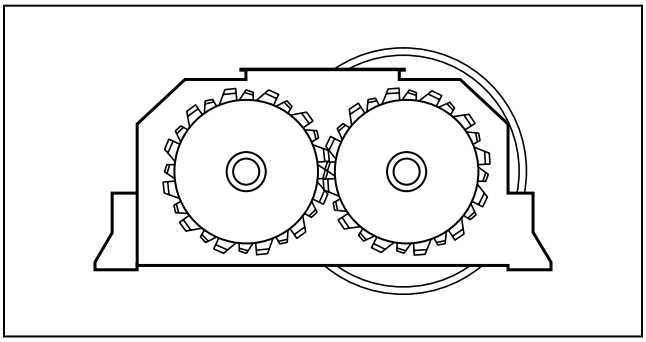


Figure 10.5-3 Double-roll crusher

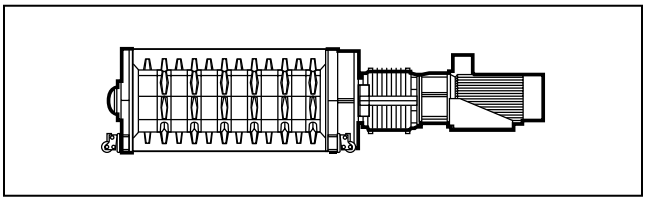


Figure 10.5-4 Low-speed sizer

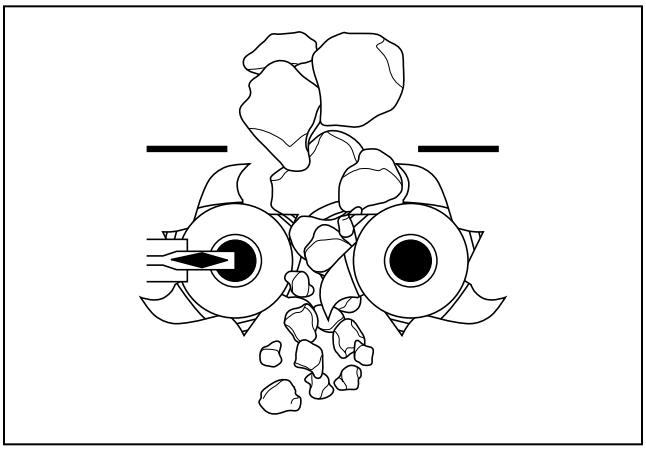


Figure 10.5-5 Hybrid roll sizer

crushers are available, and the next size will most likely be a 1,800 × 3,250 mm (72 × 128 in.). However, the increased capacity of between 7% and 10% does not seem to provide a cost-effective benefit because of the increase in weight, height, and cost for both the crusher and the auxiliary equipment to support the crusher.

Double-Roll Crushers

In the past few years, the high-speed double-roll crusher (Figure 10.5-3) has been gaining respect in many circles, primarily oil sands and overburden.

The capacity of the double-roll crusher can exceed 14,000 t/h (15,432 stph) because of its size and method of processing the ore. At this capacity, the product size is generally –400 mm (–16 in.), which is acceptable for oil sands because secondary processing and ablation during slurry transport will reduce the lump to a mixture of sand, bitumen, clay, and water. This degree of size reduction is also acceptable for waste handling, as the ore needs to be crushed to a size suitable for conveyor transport.

The double-roll crusher is suitable for ore that is wet, sticky, compactable, or has a high silica content—indeed every set of properties that causes problems for just about every other type of crusher.

The wear elements are the crusher teeth, which can be easily replaced. In fact, the double-roll crusher operates best with a mixture of new and worn teeth, so a "worn" crusher is still a performing crusher.

The double-roll crusher is capable of processing massive lumps which, in oil sands, can exceed 7 m (23 ft) in one dimension. The shovel operators try to mine selectively, but in the dead of winter, the steam rising from the working face usually obscures the operator's view of the face, especially at night, and, as a result, selective mining is not very accurate.

Low-Speed Sizers

In the early 1980s, low-speed sizers were introduced (Figure 10.5-4). This represented one of the only fundamental developments in primary crushers in three-quarters of a century.

The main technical feature of the low-speed sizer, which can broadly be considered a variety of toothed roll, is that it exploits the fact that the ratio of compressive strength to tensile and shear strength in the majority of rocks is approximately 10:1. The low-speed sizer breaks the rock in tension or in shear by snapping and chopping action rather than in compression as conventional crushers do.

Additionally, the positioning of the teeth on the rolls allows undersize to fall directly through the machine, resulting in high throughputs at very low rotational speeds, which means greatly reduced wear, energy savings, better control of discharge size in three dimensions, and greatly reduced fines.

Low-speed sizers are used for soft- to medium-hard nonabrasive, sticky types of materials up to 200 MPa (29,000 psi; for example, coal, oil sands, medium hard limestone, kimberlite, gypsum, clay, shale, schist, and gold ore. These sizers are also used to crush bauxite and overburden where the host rock is relatively soft and the inclusions range up to 400 MPa (60,000 psi). The low-speed sizer is not particularly sensitive to abrasion if the reduction ratio is low.

Low-speed sizers can be fabricated so that the frame can accommodate material larger than 2,000 mm (70 in.) as truck capacities continue to grow and ROM feed continues to increase.

Hybrid Roll Sizer

The hybrid roll sizer (Figure 10.5-5) features a compact design as a result of the roll diameter made possible by the aggressive tooth geometry. Thus, minimum space requirements and the distinctive advantages of a classic double-roll crusher, such as hydraulic gap adjustment and overload protection, are combined in one machine. Furthermore, the hybrid is able to compensate peaks with energy storage in the flywheels.

The advantages of a hybrid roll sizer are

- Compact design,
- Minimum generation of unusable fine material,
- High throughput capacities up to 11,000 t/h (12,125 stph),
- Hydraulic gap adjustment system,
- Overload protection, and
- Processing of wet and sticky materials.

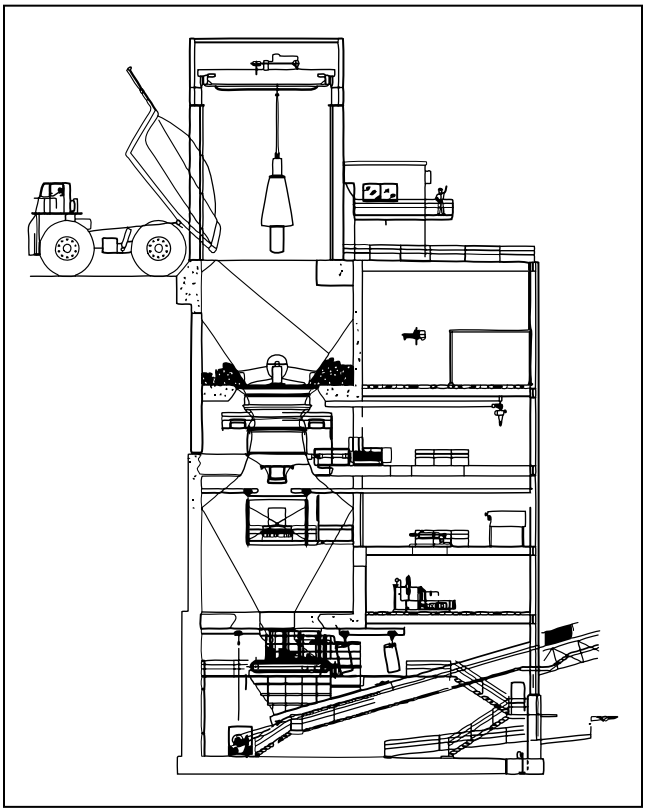


Figure 10.5-6 Fixed crusher: stationary rim-mounted crushing plant installed in a concrete structure

The disadvantages are

- Unsuitable for very hard and extremely hard materials,
- Low reduction ratio, and
- Not economic for low tonnages unless the material is difficult to handle.

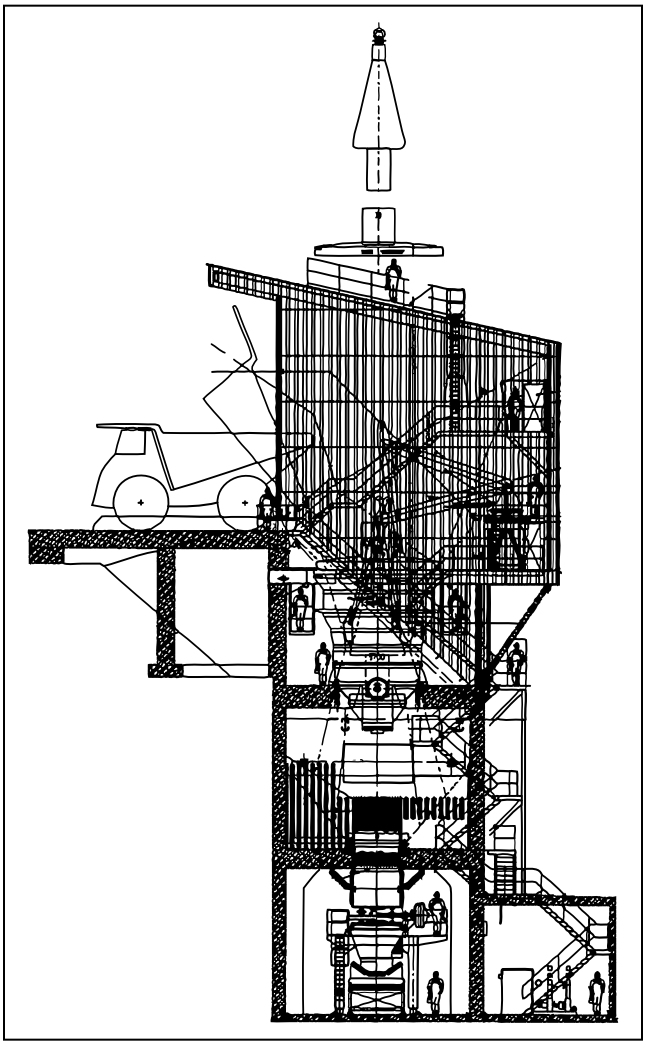


Figure 10.5-7 Rim-mounted structural steel station

TYPES OF IN-PIT CRUSHING SYSTEMS

The in-pit crushing systems developed and operated to date have varying degrees of mobility ranging from fully mobile units to permanently fixed plants, which resemble traditional in-ground crushing plants. The crushing plants can be stationary (mounted on concrete foundations) or semimobile style, supported on steel pontoon feet. As the mining operation progresses, the semimobile crushing plants can be relocated within the mine using multiwheeled trailers or transport crawlers. Typically, shovels load the ROM ore on to heavy-duty haul trucks that transport the ore to the crushing plant, thus relocating the crushing plant as the mine expands and reducing the distance that the large trucks need to haul the ore from the working face. The following terms are presented to help distinguish the range of mobility within the generic term of *in-pit crushing systems*.

Fixed Crushers: Stationary In-Ground or Rim-Mounted Crushing Plants

The stationary in-ground crusher is installed in a concrete structure below grade. The crusher is usually located external to the pit and is never moved.

The stationary rim-mounted crusher is typically installed in a concrete structure, which is part of or attached to the bench wing wall (Figure 10.5-6). A portion or all of the structure may be fabricated steel and could be disassembled and moved (Figure 10.5-7). The stationary rim-mounted crusher is usually installed for 15 or more years.

Semifixed Crushing Plant

The semifixed crusher is mounted on a steel structure that rests on a concrete foundation (Figure 10.5-8). The structure houses some or all of the auxiliary equipment and subsystems to operate the crusher. The crusher is located at or near the edge of the pit. Some degree of disassembly is required to move the structure. The planned frequency of moves for a semifixed crusher is no less than 5 to 10 years.

Semimobile Indirect Feed Crushing Plant

The semimobile indirect feed crushing plant is an all-steel structure (Figure 10.5-9). The plant typically consists of three major modules: the apron feeder, the crushing plant with the crusher, and a separate tower that houses the control room. The control room module is bolted to the crusher module when the plant is moved. The civil work required for retaining walls is relatively simple and offsets the cost of the apron feeder. However, the work is massive because

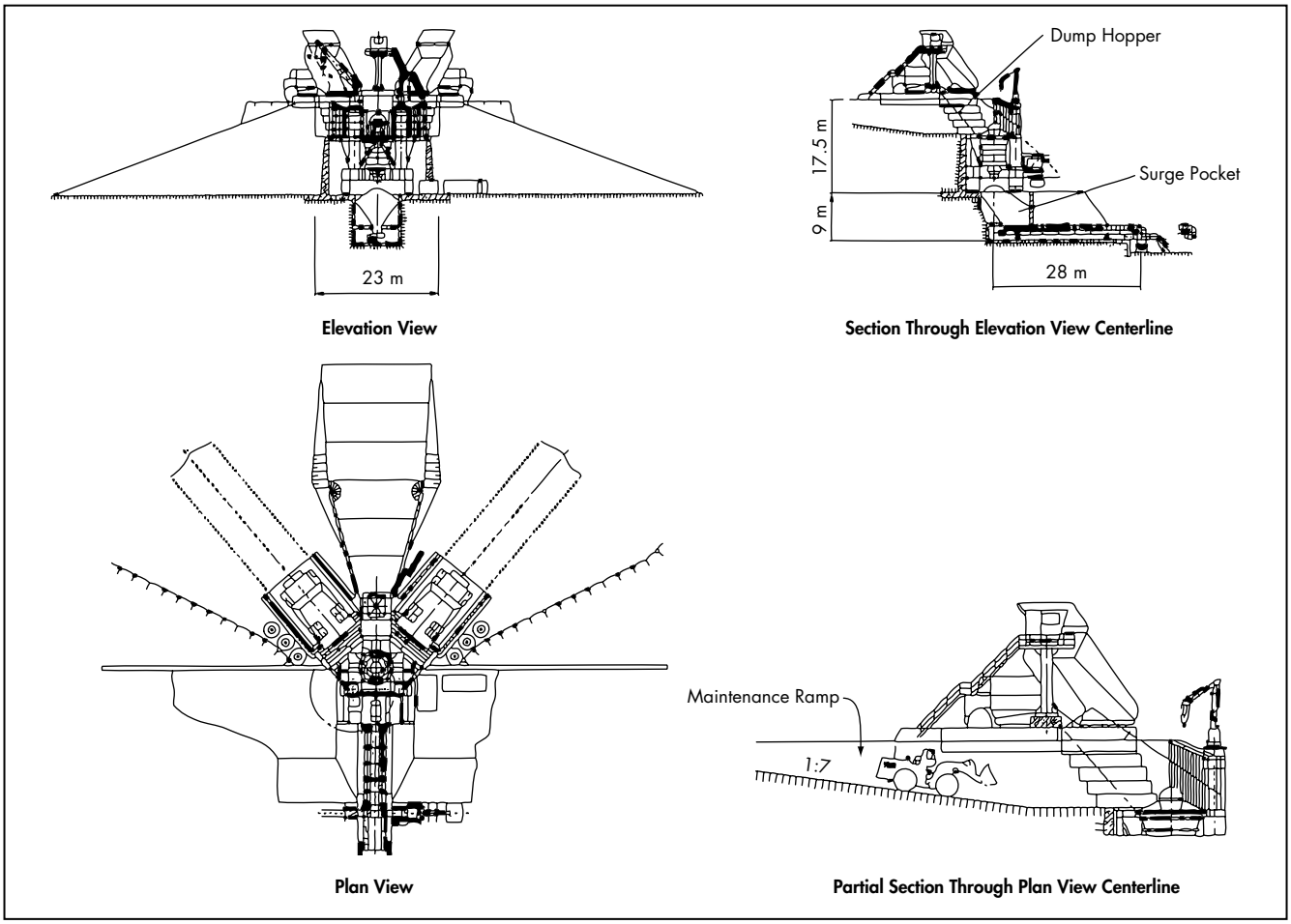


Figure 10.5-8 Semifixed crushing plant

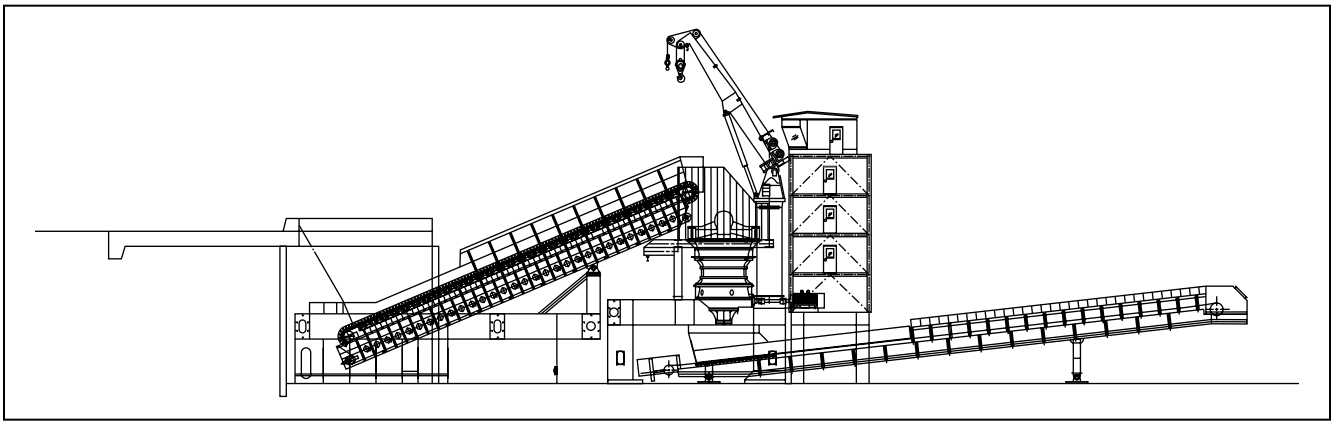


Figure 10.5-9 Semimobile indirect feed crushing plant

the structures, often in excess of 25 m (82 ft) in height, are required to support the load of the 400-ton trucks. The crusher is typically located near the centroid of the working portion of the mine to minimize truck haul distance. To allow for movement of the structure by commercially available transport equipment, bulkheads are built into the structure. The planned frequency of moves for a movable crusher is between 3 and 5 years.

Semimobile Direct-Dump Crushing Plant

The semimobile direct-dump crushing plant is mounted on a steel structure that houses all of the auxiliary equipment and subsystems to operate the crusher (Figure 10.5-10). The structure is self-supporting and rests on the mine floor, either with or without footers. The plant design allows for two or three dump points. To minimize truck haul distance, the crusher is typically located near the centroid of the working portion of

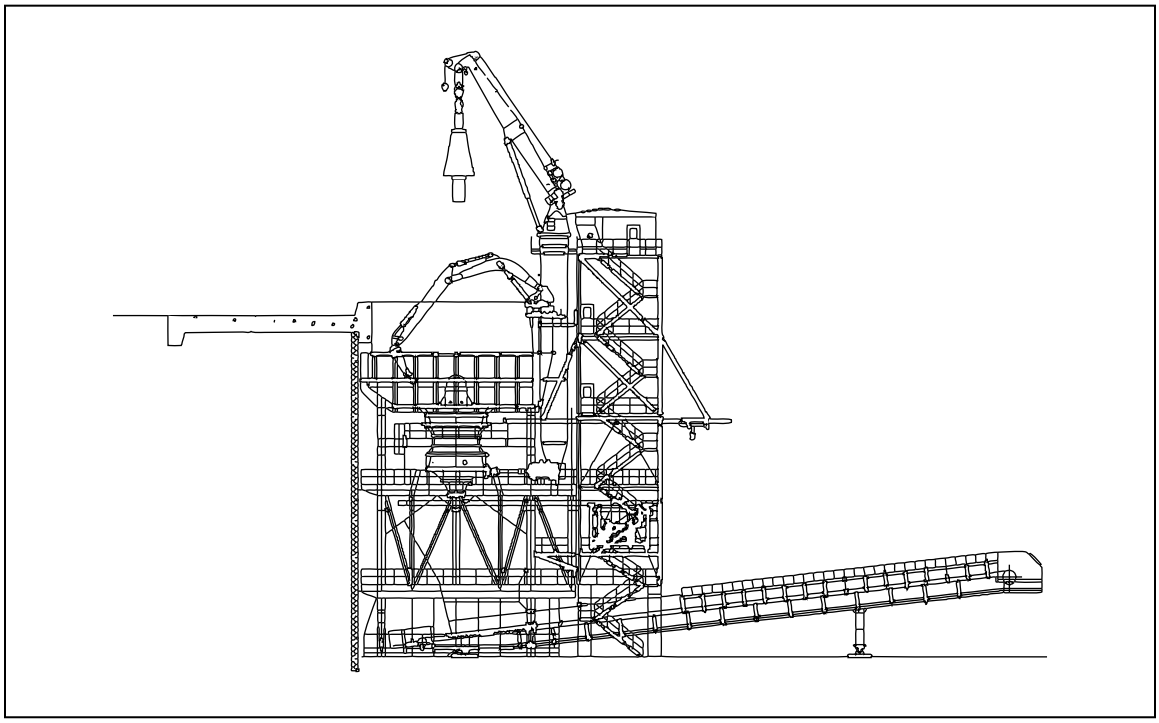


Figure 10.5-10 Semimobile direct-dump crushing plant

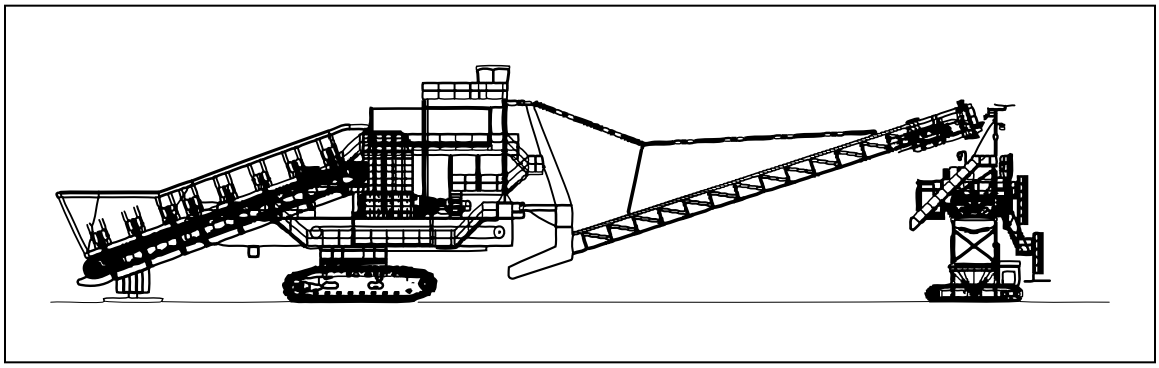


Figure 10.5-11 Fully mobile crushing plant

the mine. Bulkheads are built into the structure to allow for movement of the structure by commercially available transport equipment. The planned frequency of moves for a movable crusher is between 1 and 10 years.

Fully Mobile Crushing Plant

The fully mobile crusher is mounted on a steel platform and is self-propelled (Figure 10.5-11). The platform houses all auxiliary equipment and subsystems to operate the crusher and is self-supported and rests on the mine floor. To minimize truck or front-end loader haulage, the crusher is located at the working face. Wheels, crawlers, or pneumatic pads are integrated into the platform, and drive power to move the equipment is included on board. The planned frequency of moves for a fully mobile crusher is between 1 day and 1 week.

Fully Mobile Continuous Crushing System

The fully mobile in-pit continuous crushing system is mounted on a steel platform and is self-propelled (Figures 10.5-12 and 10.5-13). The platform houses the apron feeder, crusher, discharge conveyor, and all auxiliary equipment and subsystems to operate and propel the crushing plant. The feed hopper needs to accept ROM material that is dumped from shovels and/or draglines. The platform is self-supported and rests on the mine floor. The crusher is located at the working face for direct feed. Crawlers are integrated into the platform, and drive power to move the equipment is included on board. The plant moves in tandem with the shovel. The conveying system moves in tandem with the in-pit continuous crushing system.

OPERATION OF IPCC CRUSHING PLANTS

Fixed Rim-Mounted Crushers

Fixed rim-mounted crushers are increasingly incorporating traditional direct-dump arrangements. With these designs, the hopper above the gyratory crusher is designed to hold 1.5 to 2 times the capacity of the largest truck that will dump into the crusher during operation. Discharge surge bins have

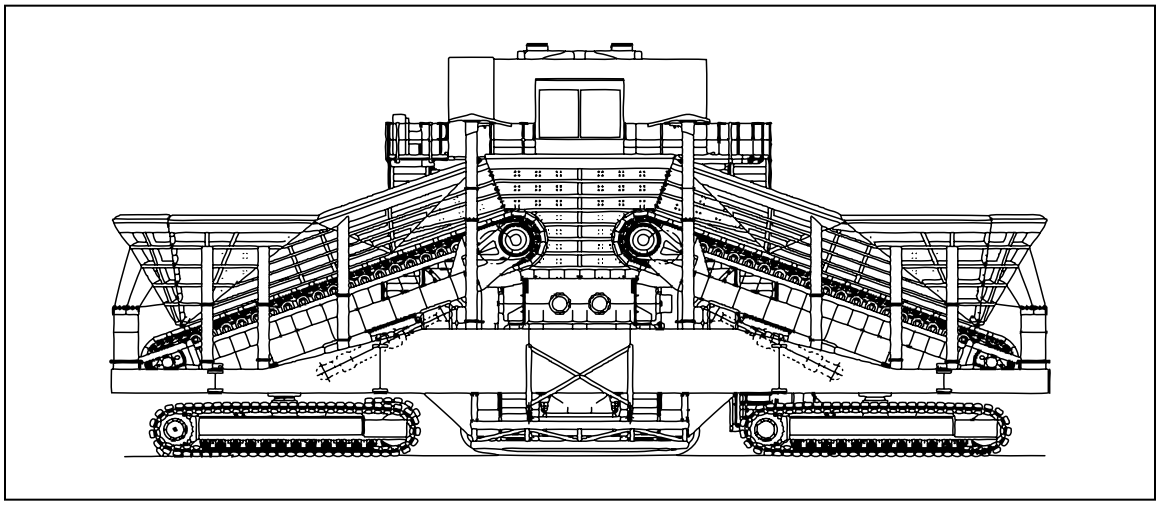


Figure 10.5-12 Fully mobile continuous crushing plant—triple track

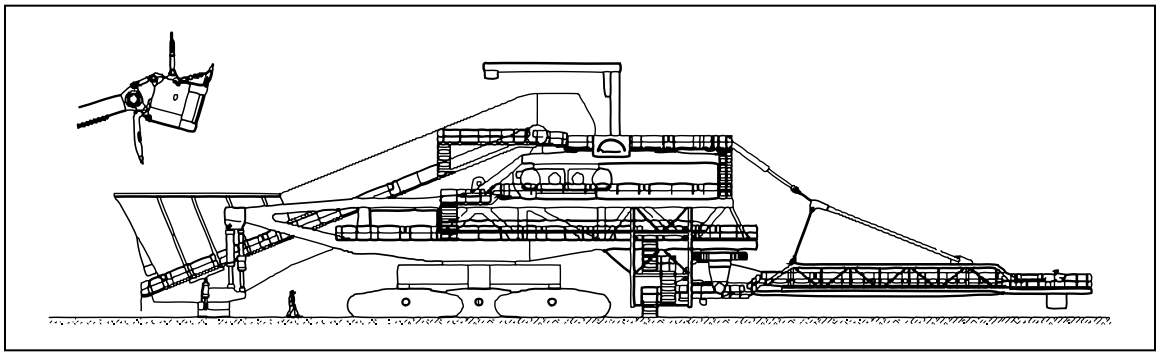


Figure 10.5-13 Fully mobile continuous crushing plant—dual crawler

traditionally been sized slightly larger than the feed hopper to accommodate any unusual fines condition. In order to reduce overall height, and thus capital costs, discharge apron feeders have been replaced by impact-resistant discharge belt conveyors. The trend away from discharge apron feeders to discharge belt conveyors has allowed for wider belts with greater capacity. In conjunction with high-capacity discharge belt conveyors, the typical capacity of the surge bin below the crusher has decreased dramatically.

Even with removing the discharge apron feeder and reducing surge bin capacity, direct-dump arrangements result in tall structures. With rim-mounted in-pit crushers, this tall overall height requires wing walls to support and reinforce the structure. Traditional fixed crushers are installed below grade and fed at grade. Recent in-pit installations such as at the Cripple Creek and Victor gold mine in Colorado (United States), which is an all-concrete structure, and Buxton Lime in Buxton, England, which is an all-steel structure, are partially below and above grade to accommodate a single bench height. The dump pockets can be arranged for one, two, or three dump positions. With a two-position dump pocket design, the two dump points are set 90° apart from each other. The spider orientation is either in-line with the centerline between the two dump positions or at 90° to the centerline between the two dump positions. Either position is mechanically satisfactory to the gyratory crusher. With a three-position dump pocket design, it is universally accepted that the spider is orientated in-line with the centerline of the central dump position.

Advantages of fixed rim-mounted crushers with direct feed arrangements include

- Traditional plants with simple configurations easily adapted for in-pit crushers,
- Reduced maintenance costs due to no longer needing an apron feeder,
- High crushing chamber throughput,
- Reduced capital costs due to limited degree of mobility,
- Reduced maintenance costs due to a greater amount of crushing in the upper portion of the chamber and decreased localized abrasive wear, and
- Greater capacity and finer product size due to the weight of the ore column above the crusher.

Disadvantages of fixed rim-mounted crushers include the following:

- Poured concrete design cannot be moved.
- Structural steel designs are typically not designed to be moved. If the structure were designed to be moved, an extensive substructure is required to support the plant for moving.
- Overall height is greater because of the higher dump point bench level.
- Greater height means extensive retaining wall structures.

Semifixed Crushers

Semifixed crushers are the mining industry's attempt to incorporate the advantages of limited mobility while eliminating the need for an extremely expensive inclined apron feeder. Semifixed plants have incorporated both indirect feed using a horizontal apron feeder as well as various forms of direct-dump arrangements. Due to the high capital and maintenance costs of apron feeders as well as the availability of high payload haulage trucks capable of sustaining high crushing capacities, the majority of semifixed crushing plants installed since about 2000 have incorporated direct-dump arrangements.

In a semifixed crushing plant, a portion of the station is fabricated from steel. The direct-dump feed hopper, crusher support structure, and control rooms are almost always steel fabricated. Differences in design are related to the degree where the lower portion of the plant is concrete or steel.

Typically, only the crusher and part (or all) of the dump hopper are mounted on a steel base, while the remainder of the station is of civil construction. The steel portion is moved by crawler transporter as self-propelled modular transporters (SPMTs). The station is moved to a new civil structure and the old station destroyed.

Advantages of semifixed crushers with direct feed arrangements include

- Traditional plants with simple configurations easily adapted for in-pit crushers;
- Reduced maintenance costs due to no longer needing an apron feeder;
- High crushing chamber throughput;
- Reduced capital costs due to limited degree of mobility;
- Increased long-term flexibility due to the limited mobility, which allows for future changes and modifications;
- Reduced maintenance cost due to greater amount of crushing in the upper portion of the chamber and decreased localized abrasive wear; and
- Greater capacity and finer product size due to the weight of the ore column.

Disadvantages of semifixed crushers with either indirect or direct feed arrangements include

- Only the crusher and part or all of the hopper are mounted on a steel base, and
- The balance of the station is civil construction. Greater overall height is due to the higher dump point bench level.

Semimobile Indirect Feed Crushing Station

The semimobile indirect feed in-pit crushing stations utilize an apron feeder to lift ore into the feed opening of the primary crusher. The use of an apron feeder allows for the crusher station to either operate at grade or to utilize a single low bench. The semimobile indirect feed in-pit crushing station is typically designed and built in three or more modules: apron feeder, primary crusher, and the control room. Some semimobile controlled feed in-pit crushing stations have a separate lubrication and hydraulic system module located adjacent to the main structure. New environmental controls for dust collection and/or dust suppression may add a dust collection module as a separate item. The control room is separate from the crusher structure to reduce crusher-initiated vibration in the control and electrical rooms. The design usually incorporates a feature that allows for the control room to be attached to the crusher module for moving.

Additionally, the use of a truck dump hopper at the apron feeder creates a large surge pocket between the mine and the crusher, making the flow of ore through the crusher more uniform and continuous.

Advantages of indirect feed using an apron feeder include

- Low bench height for dumping ore,
- Reduced truck queue time due to the surge pocket,
- Improved control of oversize material fed to the crusher, and
- Reduced crusher downtime due to bridging of large lumps.

Disadvantages of indirect feed using an apron feeder include

- Increased total capital cost,
- Increased maintenance costs associated with adding an apron feeder, and
- Increased maintenance costs associated with the crusher from using an apron feeder. Due to the nature of the feeder, ore tends to impinge upon small areas within the crushing chamber, causing premature localized wear of the concaves and mantles. The use of alloy steels has mitigated the problem, although the cost of alloy steel components remains higher than manganese steel, and availability is limited.

Semimobile Direct-Dump Crushing Plant

The semimobile direct-dump crushing plant has been the design of choice for IPCC since about 2000. This design incorporates all of the features of the traditional in-ground crushing station. The crushing plant incorporates the feed hopper, the crusher, and the lubrication and hydraulic systems to support the crusher, as well as all maintenance equipment, including a rock breaker and usually a crane with capacity to lift the mainshaft assembly with the oversize mantle. An operator's control room and electrical rooms are also included. Above the traditional gyratory crusher station with two dump points and an apron feeder discharge is the hopper, which can hold two times the capacity of the largest truck that will dump into the crusher during operation. For an operation with 400-ton trucks, this translates into 726-t (800-st) live ore capacity. Traditional design for the surge pocket under the crusher of a gyratory crusher plant would be 2.5 times the largest truck or 907-t (1,000-st) capacity to prevent backup of ore into the crusher chamber in case of an unusual fines condition. In order to reduce overall height, and thus capital costs, discharge apron feeders have been replaced by impact-resistant higher-speed discharge belt conveyors. The trend away from discharge apron feeders to discharge belt conveyors has allowed for wider belts with greater capacity. In conjunction with high-capacity discharge belt conveyors, the typical capacity of the surge bin below the crusher has decreased dramatically. Because the crusher discharge belt has a higher capacity than the haulage belt, a surge bin is usually added to the circuit.

The structure is self-supporting and rests on the mine floor either with or without footers. The plant design allows for two or three dump points. The crusher is typically located near the centroid of the working portion of the mine to minimize truck haul distance. Bulkheads are built into the structure to allow for movement of the structure by commercially available transport equipment. The planned frequency of moves for a movable crusher is between 5 and 10 years.

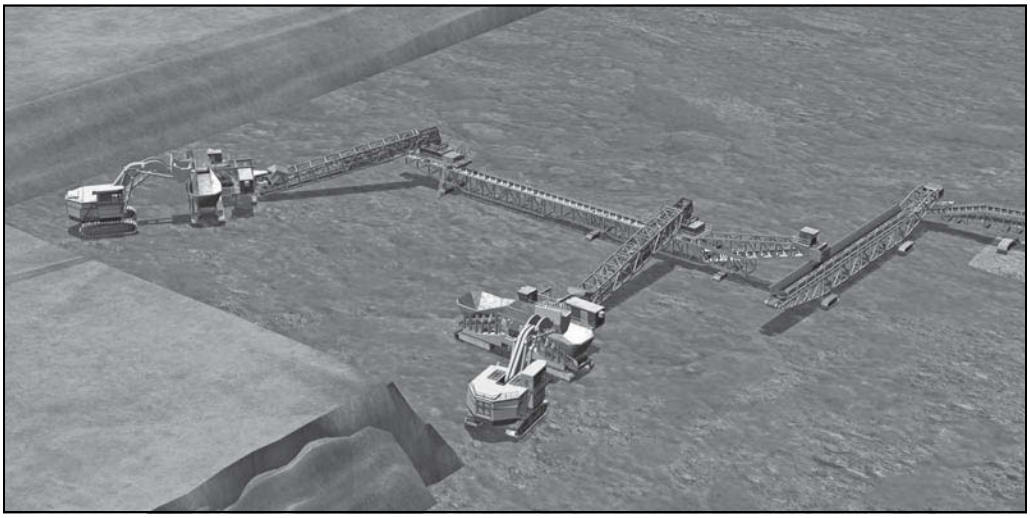


Figure 10.5-14 Fully mobile crushing plant

Advantages of semimobile direct feed arrangements include

- Traditional plant configuration,
- Reduced maintenance costs due to deletion of the apron feeder,
- High crushing chamber throughput,
- Reduced capital costs due to limited degree of mobility,
- Increased long-term flexibility due to the ability to move the complete station intact,
- Reduced maintenance costs due to greater amount of crushing in the upper portion of the chamber and decreased localized abrasive wear when compared to indirect feed designs, and
- Greater capacity and finer product size due to the weight of the ore column.

Disadvantages of semimobile crushers with either indirect or direct feed arrangements include

- Large and heavy structure requiring large transporters for moving; and
- Greater overall height due to the higher dump point bench level, which requires extensive bench-retaining walls.

Fully Mobile Crushers

The fully mobile crusher is mounted on a steel platform and is self-propelled. Wheels, crawlers, or pneumatic pads are integrated into the platform to move the station. The platform, which is self-supported and rests on the mine floor, houses all auxiliary equipment and subsystems to operate the crusher. To minimize truck or front-end-loader haulage, the crusher is located at the working face. The planned frequency of moves for a fully mobile crusher is between 1 day and 1 week.

The majority of fully mobile in-pit crushing stations utilize an apron feeder to lift ore into the feed opening of the primary crusher. The use of an apron feeder allows for the crusher station to operate at grade and move without the need for employing SPMTs. For hard rock, including aggregate and cement, the capacity has been limited to about 2,500 t/h (2,756 stph) with gyratory crushers. The 1,370 × 1,880 mm (54 × 74 in.) gyratory crusher is the largest unit used for this application.

Mobile crushing plants are favorably installed under the following conditions:

- Clear and undisturbed geological situation (almost even and horizontal coal and waste rock layers)
- Straight benches as long as possible for shiftable face conveyor installation
- Long-term mine planning for the design of face conveyors, collection conveyors at bench end (side slopes), and connecting conveyors to waste dumps or processing plants

Advantages of fully mobile crushers include

- Elimination of truck transport,
- Reduced number of personnel,
- Avoidance of high truck maintenance costs,
- Reduction of mine traffic, and
- Increase in overall safety.

Disadvantages of fully mobile crushers include

- Increased total capital costs,
- Increased maintenance costs associated with adding an apron feeder, and
- Increased maintenance costs associated with the crusher from using an apron feeder. Due to the nature of the feeder, ore tends to impinge upon small areas within the crushing chamber, causing premature localized wear of the concaves and mantles. The use of alloy steels has mitigated the problem, although the cost of alloy steel components remains higher than manganese steel, and availability is limited.

Fully Mobile Continuous Crushing Systems

The fully mobile continuous crushing system includes the fully mobile crushing plant as one component of the system (Figure 10.5-14). The components of the system are the shovel or dragline, the fully mobile crushing plant, and the conveying system that transports the crushed material to the next operation in the flow sheet. For fully mobile continuous crushing systems handling overburden, the conveyors take the material to a dump. For coal and oil sands, the material is transported to the processing plant.

The fully mobile crushing plant takes the material directly from the shovel or dragline. The full bucket load is discharged into the hopper of the fully mobile crushing plant. An apron feeder elevates the material and discharges it into the crusher, either a low-speed sizer, a high-speed double-roll crusher, or a hybrid roll sizer. The product from the crusher is transported to the overland conveyor system. All parts of the system are designed to work continuously. This requirement has led to the development of fully mobile crushing plants that have the ability to continue operation while moving. Older systems in oil sands in Canada and coal in Australia had to stop operations, lift the apron feeder off the ground, move to a new location, and set the apron feeder down before resuming operation.

Mobile crushing plants are able to work at one, two, or three benches with only one shiftable bench conveyor (multibench operation). An important design criterion for this effective technology is the transfer equipment to the bench conveyor, which has to bridge not only distances in length but also in height (crusher discharge conveyor, mobile transfer conveyors, and mobile transfer bridges). Ramps in the working face or in the side slope are necessary for equipment relocation from one bench to the other.

IPCC PROCESS CONTROL

Growing recognition exists in the industry of the effect that mining practices have on the efficiency of mineral processing operations. Among numerous variables, size distribution of ore is widely accepted as having a significant effect on the throughput and recovery achieved in mineral processing. Using available digital imaging technology, improved process monitoring and modeling provide the opportunity not only to identify these variables but also to monitor and control them throughout the entire mining process in real time.

The majority of mine designs employ crushers to reduce the size of the material originating from the mine. The primary crusher is the link between the chemical comminution (blasting) and the beginning of the mechanical comminution circuit (crushing and milling). As such, an in-pit crusher is not only a key point in the process to apply a measurement monitor but also a key resource to be optimized.

Feed to the primary crusher from the mine can be measured and monitored to establish blasting performance. Size information associated with each haul truck can be traced back to the location in the mine plan and used to help design future blasting practices. The crusher product is usually the beginning of the mineral processing circuit that involves more energy consumption to further reduce fragment size. In the short term, the crusher's performance is the responsibility of the crusher operator who, through the use of digital imaging, now has a record of the quality of the crusher product size and can make adjustments to the crusher to keep the product in specification, as required by the design, of the remainder of the comminution circuit. In the long term, archives of size trends of crusher feed and product as related to other key performance indicators, such as blast fragmentation, crusher/mill throughput, crusher reduction ratios, Bond work index, and energy consumption and efficiency, optimize management-level decisions. Opportunities for optimization include how to tailor blasting to feed the stationary mechanical comminution circuit, how to load the crusher, and how to establish better proactive maintenance of the crusher, as well as keeping tighter specifications on the feed and products of the various stages of comminution.

By applying digital imagery technology, a technology widely used in numerous other manufacturing industries, new innovative solutions are available to the mining industry. Applying imaging technology can generate volumes of size information, which were not previously possible, for use in long-term studies, as well as in short-term process control to create long-term operating savings as compared to manual sizing methods or, worse yet, no measurement at all. An engineer with more than 30 years of mining experience was quoted about image analysis: "At our mine we utilize digital image analysis systems to provide quality, quantitative fragmentation information on our blasting and integrate the fragmentation information into our operations database as a quality control mechanism within our ongoing continuous improvement program. An important step to controlling costs is controlling your process. Basically, any company that has a product that requires control of particle size and is concerned with profitability needs this valuable information."

CONCLUSION

IPCC systems with primary gyratory crushers have been the answer to escalating energy and labor costs from 1980 to the present and have the potential to continue to substantially increase the profitability of open-pit mining. The type and configuration for virtually any site-specific application have been designed and built by the world's leading manufacturers and have operated for sufficiently long periods of time to prove their viability.

As evidenced by recent installations such as Collahuasi in Chile and Freeport-McMoRan in Iran Jaya, Indonesia, which operate above 8,500 t/h (9,370 stph), there is a definite trend toward increased capacity for large primary gyratory crusher installations. Similarly, there has been a steady increase in the size of widely available haulage trucks during the same period, from 150 to 363 t (165 to 400 st). In contrast, the radial feed opening of the primary gyratory crusher has not increased since the first Traylor 1,500-mm (60-in.) crusher was built in 1919.

In-pit crushing stations have decreased in mobility from the fully mobile and movable designs dating from 1982 to 1985 to the semifixed and fixed rim-mounted stationary designs of 1998–2001. For example, the 1,370-mm (54-in.) movable indirect feed crusher at Minera Escondida in Chile was built in 1990. When the company installed a 1,500-mm (60-in.) crusher in 1995, a semimobile direct-dump design was chosen. Similarly, the 1,500-mm (60-in.) movable indirect crushers at Codelco's Chuquicamata mine in Chile dating from 1984 have been rebuilt, relocated, and reinstalled in new semifixed direct-dump installations.

The 90-yd^3 shovel and 400-ton haulage trucks have created a gross mismatch between feed size and feed opening with the primary gyratory crusher; a single bucket load weighs more than the total volume of ore that can be held within the largest primary gyratory crusher currently in operation. Further, a single truckload weighs three times the total weight of the common 1,065-mm (42-in.) primary gyratory crusher. It is physically possible for a shovel dipper to pick up a single rock 2,540 mm (100 in.) in diameter and deposit it into a crusher with a 1,065-mm (42-in.) radial feed opening.

Change is required in the comminution flow from fragmentation in the pit to size reduction in IPCC. The top size of ROM feed to the IPCC must be controlled, or else the industry needs to consider alternative crushing technologies

to compensate for the lack of top-size control and yet still be in a position to take advantage of the economies of scale with shovels, trucks, and IPCC.

The fully mobile concept can conceivably be utilized in all mining operations, where a shovel can excavate the ore directly at the face, with or without blasting.

For large open-pit ore mines, which because of the deposit characteristics mainly extend downward, potential applications are currently being examined. Compared to coal and oil sand mines, which typically have relatively wide benches, the planning and realization of a fully mobile concept in an iron ore mine is complicated by the chiefly vertical alignment of the deposits. The fully mobile system, however, promises considerable savings in capital and operating costs, and it is likely that a solution can be realized.

THE FUTURE

The standard 400-ton truck of 2011 and beyond will likely continue to feed direct-dump semimobile crushing plants with the standard 1,500-mm (60-in.) gyratory crushers with 1,220-mm (48-in.) feed. There is the ever-present possibility that the ore will be fed to the crushing plants utilizing 1,800-mm (72-in.) primary gyratory crushers or some other types of primary crusher capable of size reduction of ROM material larger than 1,500 mm (60 in.). As ore grade decreases, efficiency will increase. Capacities will increase as a result of the larger crushers, while, at the same time, operators will take advantage of the larger crusher to obtain the economy of scale.

In the period 1980 to 2001, the type of in-pit crushing plant has evolved from fully mobile to semimobile to semifixed and, in many cases, to rim-mounted stationary structures. This progression from fully mobile in the 1970s and early 1980s to less mobility throughout the balance of 1980s and into the 21st century would seem like a step backward, considering that the original premise of in-pit crushing was to reduce the dependence on truck haulage. The evolution of the in-pit crushing plant cannot be considered by itself but must be viewed in relation to other changes in the size reduction of ROM ore in the pit, as well as delivering to the secondary stage of comminution:

- The platform for the development of IPCC in 1980 was the typical multibench mine.
- The premise was that the design of the plant should be one bench high—that is, 13.7 m (45 ft) for slope stability, economy of dump pocket design, and safety.
- Inserting an apron hopper between the dump pocket and the primary crusher could attain the dump height.
- In 1980, the perceived wisdom was that the crushing plant would be moved every time the pit became deeper by one or two benches.
- The typical mine would be in a position to move the crushing plant in less than 5 years.
- The plant move could be accomplished in as few as 3 days.
- The truck size in 1980 was 150 t (165 tons).

The development of the 240-ton, 300-ton, 350-ton truck, and now 400-ton haul truck has had a profound effect on the in-pit crushing plant designer and operation of in-pit crushing systems:

- With the larger trucks, the dump pocket was no longer a simple structure.
- The economies of scale and efficiencies of the new larger haul trucks negated the original parameters for moving the plant for every one or two benches of depth, or every few years of operation.
- The operational advantages for the use of apron feeders to elevate the ROM ore largely disappeared because of the cost of the dump pocket and the limited number of moves required in the life of the mine.
- Maintenance costs showed that the operating costs of the inclined apron feeder was equal to or more than the maintenance cost for the gyrator crusher.
- The capacity of the gyratory crusher was actually less when operating with an apron feeder than when directly dumped into the crusher.
- The product from the gyratory crusher is finer when directly dumped than when the apron feeder is fed.
- Concave and mantle costs are higher when the crusher is fed with the apron feeder.
- Average capacities increased from 3,000 to 9,000 t/h (3,300 to 9,920 stph).

The net effect of these changes was the progression that evolved into semimobile direct-dump crushing plant designs:

- Plant designs that make use of the slope topography of the pit
- Plant relocation of 3 to 5 years or more
- Capacities to 9,000 t/h (9,920 stph)
- Direct-dump formats
- 1,500-mm (60-in.) primary gyratory crushers

This mismatch in shovel size, truck size, and crusher size will likely manifest itself in fundamental changes to the primary crusher and/or improved and innovative crushing plant designs rather than the installation of the larger and heavier gyratory crushers.

Four of the nine types of primary crushers have achieved capacities of 10,000 t/h (11,000 stph):

1. Gyratory crushers have been operating at or near these capacities on copper oxide ores in Chile. A 10% increase in capacity would be achieved with the next size of primary crusher, ensuring continuous operation at the 10,000-t/h (11,000-stph) level. The question for mine planners: Is the 10% increase in capacity and in feed size from 1,220 × 0 to 1,500 × 0 mm (48 × 0 to 60 × 0 in.) economically justifiable? Gyratory crushers can crush material with compressive strengths as hard as 600 MPa (90,000 psi).
2. Low-speed sizers have attained capacities of 14,000 t/h (15,432 stph) operating in oil sands projects in Canada as well as in coalfields in Australia. Low-speed sizers are crushing material with compressive strengths of 200 MPa (27,500 psi) on a continuous basis.
3. Double-roll crushers have also attained capacities of 14,000 t/h (15,432 stph) operating in oil sands projects in Canada. The giant double-roll crushers are currently limited to compressive strengths of 100 MPa (14,500 psi).
4. Hybrid roll sizers are rated at capacities of 14,000 t/h (15,432 stph) and compressive strengths of 100 MPa (14,500 psi).

The designers of the IPCC crushing plants are constantly working toward shorter, more cost-effective facilities. The low-speed sizer, with its inherent low-profile design and proven

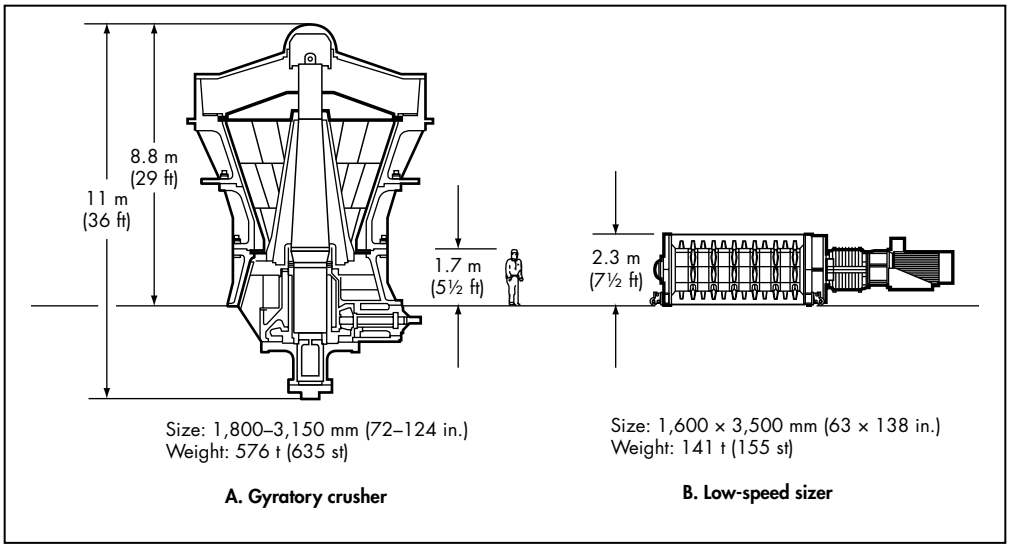


Figure 10.5-15 Comparison of gyratory crusher and low-speed sizer

high-capacity in crushing materials with characteristics similar to copper oxide, appears to provide the most realistic prospect of transfer of technology for future mining IPCC facilities.

A comparison of the next larger size gyratory with a low-speed sizer follows (Figure 10.5-15):

- ROM: 10,000 t/h (11,000 stph)
- Top size: 1,500 mm (60 in.)
- Product: 80% –23 cm (–9 in.)

There is an emerging market for high production, more than 9,000-t/h (9,920-stph) IPCC systems, especially for the removal of overburden. It no longer makes sense to use large volumes of assets such as trucks, tires, and labor to move a waste product that has no value. The ability to process high tonnages of soft overburden is made possible with the use of high-capacity crushers developed for soft material, usually less than 125 MPa (18,000 psi). These crushers have a more compact design and significantly lower capital cost than the typical hard-rock crusher. These soft-rock crushers, such as feeder breakers or sizers, make them perfect for overburden IPCC systems.

In the not too distant future, sizers will be crushing harder materials. Sizers are already crushing limestone with compressive strengths of 150 to 160 MPa (22,000 to 23,000 psi). The Lihir gold mine ore in Papua New Guinea is between 250 and 300 MPa (36,250 and 43,500 psi) and is crushed with a gyratory crusher. In addition to the hard ore, the mine also has wet, sticky volcanic ore that is crushed with a sizer. When the gyratory crusher is being maintained, the hard ore is run through the sizer. A bauxite mine in Australia crushes bauxite at the rate of 2,500 t/h (2,755 stph) with 100-MPa (14,500-psi) ore and 400-MPa (58,000-psi) dolerite inclusions. The history learned from these and other installations provides evidence that sizers will be able to handle overburden and ore with 50% of the feed 150–200 MPa (21,750–29,000 psi) material at rates of 16,000 t/h (17,640 stph).

ACKNOWLEDGMENTS

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CHAPTER 10.6

Design, Construction, and Maintenance of Haul Roads

Roger J. Thompson

INTRODUCTION

In truck-based hauling systems, the mine haul road network is a critical and vital component of the production process. As such, underperformance of a haul road will have an immediate impact on mine productivity and costs. Operations safety, productivity, and equipment longevity are all dependent on well-designed, well-constructed, and well-maintained haul roads (Thompson and Visser 1999). The mine haul road is an asset and should, in conjunction with the haul trucks using the road, be optimally designed and its routine maintenance managed accordingly. An ad-hoc or empirical approach to haul-road design is generally unsatisfactory because it has the potential for overexpenditure, both on construction and operating costs, arising as the result of the overdesign and over-specification of short-term, low-traffic volume roads; and the underdesign, leading to excessive operating and road maintenance costs, and premature failure of longer-term, higher-volume roads.

Economies of scale and the increase in haul truck payload have led the ultra-class truck (220 t [metric tons] and larger) population to rise to more than 40% of all mine trucks used (Gilewicz 2006). With this increasing size, haul road performance can be compromised, resulting in excessive total road-user costs, translating to an increase in cost per ton hauled but also indirectly to reduced production rates and vehicle and component service life. Truck haulage costs can account for up to 50% of the total operating costs incurred by a surface mine, and any savings generated from improved road design and management benefit the mining company directly as a reduced cost per metric ton of material hauled.

Central to the cost of truck hauling is the concept of rolling resistance (expressed here as a percentage of gross vehicle mass [GVM]). It is a measure of the extra resistance to motion that a haul truck experiences and is influenced by tire flexing, internal friction, and, most importantly, wheel load and road conditions. Empirical estimations of rolling resistance based on tire penetration specify typically a 0.6% increase in rolling resistance per centimeter tire penetration into the road, over and above the 1.5% (radial and dual wheel assemblies) to 2% (cross-ply or single-wheel assemblies) minimum resistance. In addition to tire penetration, road surface deflection or flexing will also generate similar results, with the truck tire running "up-grade" as the deflection wave pushes ahead of the vehicle. Taking an electric-drive rear-dump ultra-class truck of 376 t (GVM) as an example, on a ramp road with a basic rolling resistance of 2%, an additional 1% rolling resistance will reduce truck speed by 10% to 13%, whereas on a flat surface road, the truck speed will be reduced from 18% to 26%.

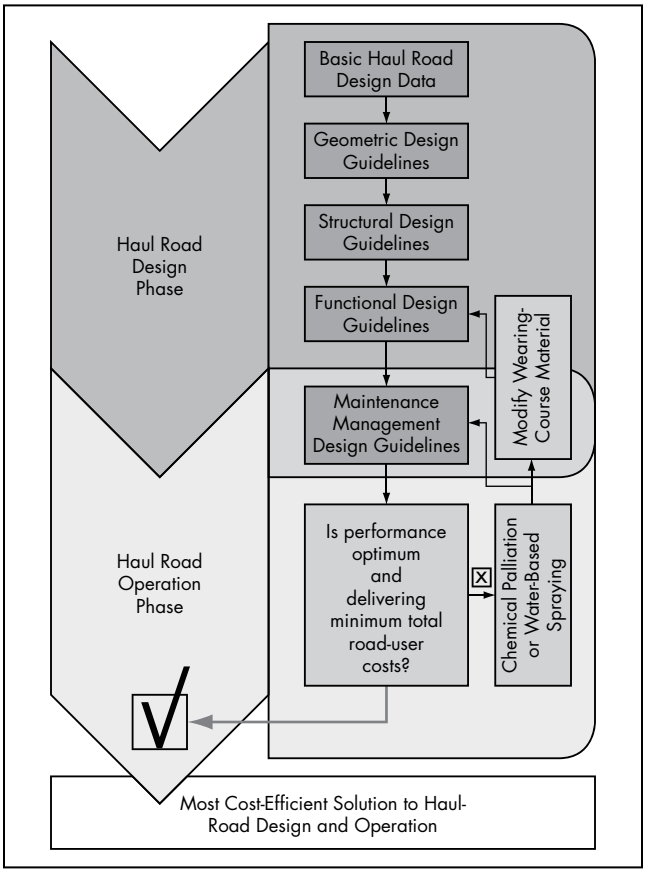
Although many concepts from highway engineering can be adapted to the design, construction, and management of mine roads, significant differences in applied loads, traffic volumes, construction material quality and availability, together with design life and road-user cost considerations, mitigate for a tailored design solution for mine haul roads.

COMPONENTS OF AN INTEGRATED MINE HAUL ROAD DESIGN

The operating performance of a mine road can be subdivided into four design components (i.e., geometric, structural, functional, and maintenance management); when designing and constructing a haul road for optimal performance, these design components are best addressed using an integrated approach. If one design component is deficient, the other components may not work to their maximum potential and road performance is often compromised. This will most often be seen as "maintenance intensive" or high rolling resistance roads, translating to increased equipment operating, downtime, and repair costs. The cure, however, is not necessarily just more frequent maintenance; no amount of maintenance will fix a poorly designed road.

Design and management of haul road systems should also be approached holistically, especially with regard to the benefits achieved from various solutions to enhance productivity. Although, for instance, trolley-assist may improve cycle times and reduce cost per metric ton hauled, it is first necessary to evaluate the extent to which an existing haul road network meets optimal design requirements before resorting to solutions that do not directly address the key deficiencies of the existing road system. The recommended approach is therefore to assess the extent to which the asset (the current road network) exhibits scope for improvement and, once optimized,

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Source: Adapted from Thompson and Visser 2008.

Figure 10.6-1 Integrated haul road design and management system components

then revert to resource supplementation to leverage these benefits through optimal asset and resource interaction.

Figure 10.6-1 illustrates the approach to mine road design, based on the geometric, structural (layerworks), functional (wearing course), and maintenance management design components, together with a dust palliative evaluation methodology. These design components form the basis for the following sections of the chapter.

The first component, that of geometric design, is commonly the starting point for any haul road design and refers to the layout and alignment of the road, in both the horizontal and vertical plane. The ultimate aim, to produce an optimally efficient and safe geometric design, can only be achieved when sound geometric design principles are applied in conjunction with the optimal structural, functional, and maintenance management designs.

The aim of a structural design is to provide a haul road that can carry the imposed loads over the design life of the road without the need for excessive maintenance. It is focused on the design of road layerworks and the response of construction materials in and under the road to the truck wheel loads.

The functional design is centered on the selection of wearing course (or surfacing) materials; the most suitable choice, application technique, and maintenance strategy is required. Commonly, the running surface of a mine road is a gravel mix, which lends itself to maintenance (blading or, over the longer term, rehabilitation). To improve performance of the material, palliation and/or stabilization is often considered, primarily to reduce both dust generation and material degeneration, the latter leading to increased rolling resistance and associated road maintenance.

The maintenance aspect of haul road design cannot be considered separate from the geometric, structural, and functional design aspects because they are mutually inclusive. Design and construction costs for the majority of haul roads represent only a small proportion of the total operating and maintenance costs. Although it is possible to construct a mine haul road that requires no maintenance over its service life, construction costs would be prohibitively expensive. The converse, an empirically designed and cheaply constructed road, would also incur excessive costs, in this case related to vehicle operating and road and vehicle maintenance costs. The use of an appropriate road maintenance management strategy will generate significant cost savings by virtue of a better understanding of the relationship between wearing course material degeneration rates (manifest as increasing rolling resistance on the road) and its influence on both cost per metric ton hauled and the cost of road maintenance itself.

A mine road network often comprises various roads, each with a specific function, traffic type (size of truck), traffic volume, service level (performance), and operating life. A road classification system should be developed, according to these parameters, as part of a mine-wide common framework for road design. This can be used as the starting point for design guidelines for construction personnel, to enable them to determine easily what design guideline is appropriate when constructing new roads or evaluating and rehabilitating existing mine roads. Clearly, not all roads are equal and thus the approach to design and management must be tailored to apply more resources to high-volume, long-term, and high-cost-impact road segments across the network. Figure 10.6-2 illustrates typical haul road design categories; the accompanying data forms the basic input to the four design categories previously discussed.

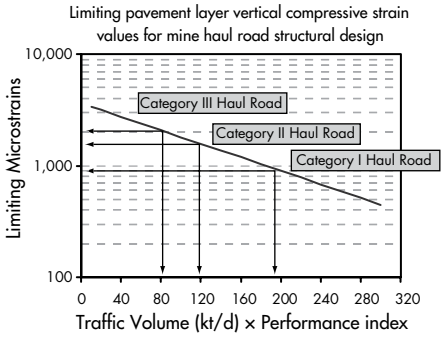
GEOMETRIC DESIGN

The geometric layout of a mine haul road is dictated to a great extent by the mining method used and the geometry of both the mining area and the ore body. Mine planning software enables various haul road geometric options to be considered and the optimal layout selected, both from a road design and economic (lowest cost of provision) perspective (MineMap 2008). Although these techniques often have default design values embedded in the software, it is nevertheless necessary to review the basic concepts of geometric design if any modifications are to be considered in the design of mine roads, either on the basis of economics or, more critically, from a safety perspective.

The road layout or alignment, both horizontally and vertically, is generally the starting point of the geometric design. Practically, it is often necessary to compromise between an ideal layout and what mining geometry and economics will allow. Any departure from the ideal specifications will result in reductions in both road and truck life.

Considerable data already exists pertaining to good engineering practice in geometric design (Kaufman and Ault 1977; USBM 1981; Tannant and Regensburg 2000), and forms the basis of the design criteria developed here. Broadly speaking, safety and good engineering practice require haul road alignment to be designed to suit all vehicle types using the road,

Haul Road Category	Typical Description	Range of Maximum Permissible Vertical Elastic Strains (microstrains)	
		Traffic Volumes >100 kt/d	Traffic Volumes >100 kt/d
Category I	Permanent life-of-mine high traffic volume main hauling roads and ramps in- and ex-pit Operating life >20 years	900	1,500
Category II	Semipermanent medium-to-high traffic volume ramp roads in- and ex-pit Operating life >10 years	1,500	2,000
Category III	Shorter-term medium- to low-volume in-pit bench access, ex-pit dump, or ramp roads Operating life <5 years (at >50 kt/d) or <10 years (at <50 kt/d)	2,000	2,500



Maximum permissible vertical strains can also be determined from (kt/d × performance index) where performance index is defined as

1. Adequate but fairly maintenance intensive,
2. Good with normal maintenance interventions, or
3. Outstanding with low-maintenance requirements.

Figure 10.6-2 Typical haul road design categories and associated design data

operating within the safe performance envelope of the vehicle, or, where this is not possible, at the speed limit applied. Ideally, geometric layout should allow the vehicles to operate at their maximum safe speed, but given that the same road is used for laden and unladen haulage, there is often the need to minimize laden travel times, through appropriate geometric alignment, while accepting compromises (generally in the form of speed limits) on the unladen return haul. The process of geometric design is shown in Figures 10.6-3 and 10.6-4 and discussed in the following sections.

Vertical Alignment

Vertical alignment considers the effect of road elevation changes on both sight and stopping distances of trucks, together with the effect of road grade on safety and performance.

Stopping Distance Limits

The truck manufacturer should confirm the distances required to bring the truck to a stop, following International Organization for Standardization (ISO) 3450:1996 standards or local requirements. The following equation can be used to produce a first estimate:

$$\text{stopping distance (m)} = \frac{v^2}{48 - 2.6 \times \text{grade}} \qquad (10.6\text{-}1)$$

where v is the speed of the truck (kilometers/hour) with GVM > 32 t and *grade* is the downslope (positive in direction of travel) (%).

The ISO 3450:1996 standard gives typically a 114-m stopping distance at 10% downgrade at 50 km/h and a 73-m stopping distance at 40 km/h. Although this satisfies most mine ramp road designs where rear-dump trucks are used, care should be taken when using the ISO approach for articulated dump trucks (ADTs). Steeper ramps are often used where ADTs are employed, because they commonly have better hill-climbing ability. With a ramp steeper than 10%, the ISO stopping distance would not necessarily apply. In general, and including driver reaction times, practical retard unassisted (emergency) distances can be determined from the following equation:

$$\text{stopping distance} = \frac{1}{2}gt^2\sin\theta + v_o t + \left(\frac{(gt\sin\theta + v_o)^2}{2g(U_{\min} - \sin\theta)}\right) \qquad (10.6\text{-}2)$$

where g is the acceleration due to gravity (m/s^2), t is the driver reaction time (s), θ is the grade of road (degrees) positive downgrade, v_o is the vehicle speed (m/s), and $U_{\min}$ is the coefficient of friction between the tire and the road, taken as 0.3. This first estimate for stopping distance is based on "ideal" braking and vehicle conditions (dry road, good skid resistance, etc.). When conditions under braking vary (wet roads, poor and slippery wearing course, spillage, etc.), a longer stopping distance would need to be considered.

Sight Distances

At least 150 m is required based on typical stopping distance requirements. On a curve or bend in the road, this could be difficult to achieve, as shown in Figure 10.6-5. Length of vertical curves (L) can be determined from consideration of the height of the driver above the ground (h_1[m]), an object of height h_2 (m) (usually 0.15 m to represent a prostrate figure in the road), SD as the minimum stopping distance (m), and ΔG as the algebraic difference in grades (%).

Where stopping distance is greater than the length of a vertical curve, then

$$L = 2SD - \left(\frac{200\left(\sqrt{h_1} + \sqrt{h_2}\right)^2}{\Delta G}\right) \qquad (10.6\text{-}3)$$

Where stopping distance is less than the length of the curve, then

$$L = \left(\frac{\Delta G \cdot SD^2}{100\left(\sqrt{h_1} + \sqrt{h_2}\right)^2}\right) \qquad (10.6\text{-}4)$$

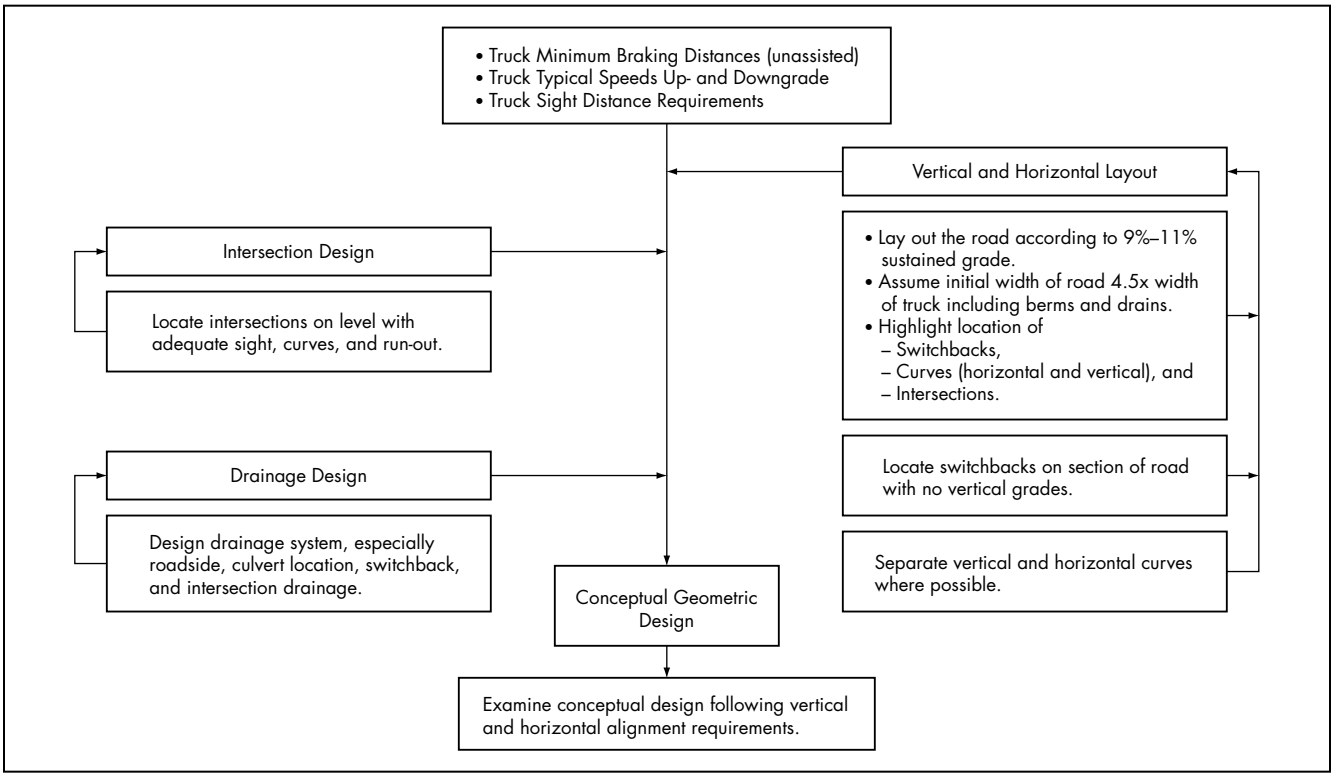


Figure 10.6-3 Conceptual geometric design

When the road curves around a bench edge, to maintain sight distance, a "layback" (LB) is used to keep the road away from the sight obstruction. The layback distance (defined in Figure 10.6-6) is found from consideration of the truck minimum stopping distance and curve radius R [m]):

$$LB = SD\left[1 - \cos\left(\frac{28.65R}{SD}\right)\right] \qquad (10.6\text{-}5)$$

Where sight distance is reduced below the stopping distance, speed limits should be applied or sight distances increased.

Optimal and Maximum Sustained Grades

Although maximum gradients may be limited by local regulations, ideally the gradient should be a smooth, even grade, not a combination of grades. Laden trucks running against the grade work best at total (i.e., grade and rolling resistance) values of between 8% and 11%. However, each truck engine and drive system combination has a characteristic "optimal grade curve" and it is a good geometric design starting point to determine the optimal gradient for the selected truck in use at the mine. Although travel times (laden) are sensitive to grades against the load, care should also be taken when selecting the grade, from the perspective of truck retard limitations on the unladen downward leg of the haul. This aspect becomes critical in the case of downgrade laden hauling when retard capacity would be the limiting design criteria.

Horizontal (Longitudinal) Alignment

Haul road width, curve layout, and associated superelevation and run-out, together with cross-slope or camber and intersection layout, are incorporated under longitudinal geometric design considerations.

Width of Road

The width of the road should allow enough room for the required number of lanes and all the associated safety and drainage features. The widest vehicles proposed determine the roadway width; the dimensions of the safety berms and drainage channels are also added to the roadway width to determine the construction width required. Table 10.6-1 summarizes these design roadway widths.

On single-lane roads, if the sight distance is less than the stopping distance, sufficient space must be provided for moving vehicles to avoid collision with stalled or slow-moving vehicles.

Curvature and Switchbacks

Any curves or switchbacks should be designed with the maximum radius possible and be kept smooth and consistent. Changes in curves radii (compound curves) should be avoided. A larger curve radius allows a higher safe road speed and increased truck stability. Sharp curves or switchbacks will increase truck cycle times and haul cost as a result of rear dual tire wear due to tire slip.

Minimum radius (R [m]) can be initially determined from

$$R = \frac{v_o^2}{127(e + U_{\min})} \qquad (10.6\text{-}6)$$

where v_o is the vehicle speed (km/h); e refers to the superelevation applied (m/m width of road); and $U_{\min}$ is the coefficient of friction between the tire and the road, taken as 0.3 (soft, muddy, rutted road surface) to 0.45 (partially compacted gravel surface). Where the pit layout does not permit these minimum radii, speed limits need to be applied.

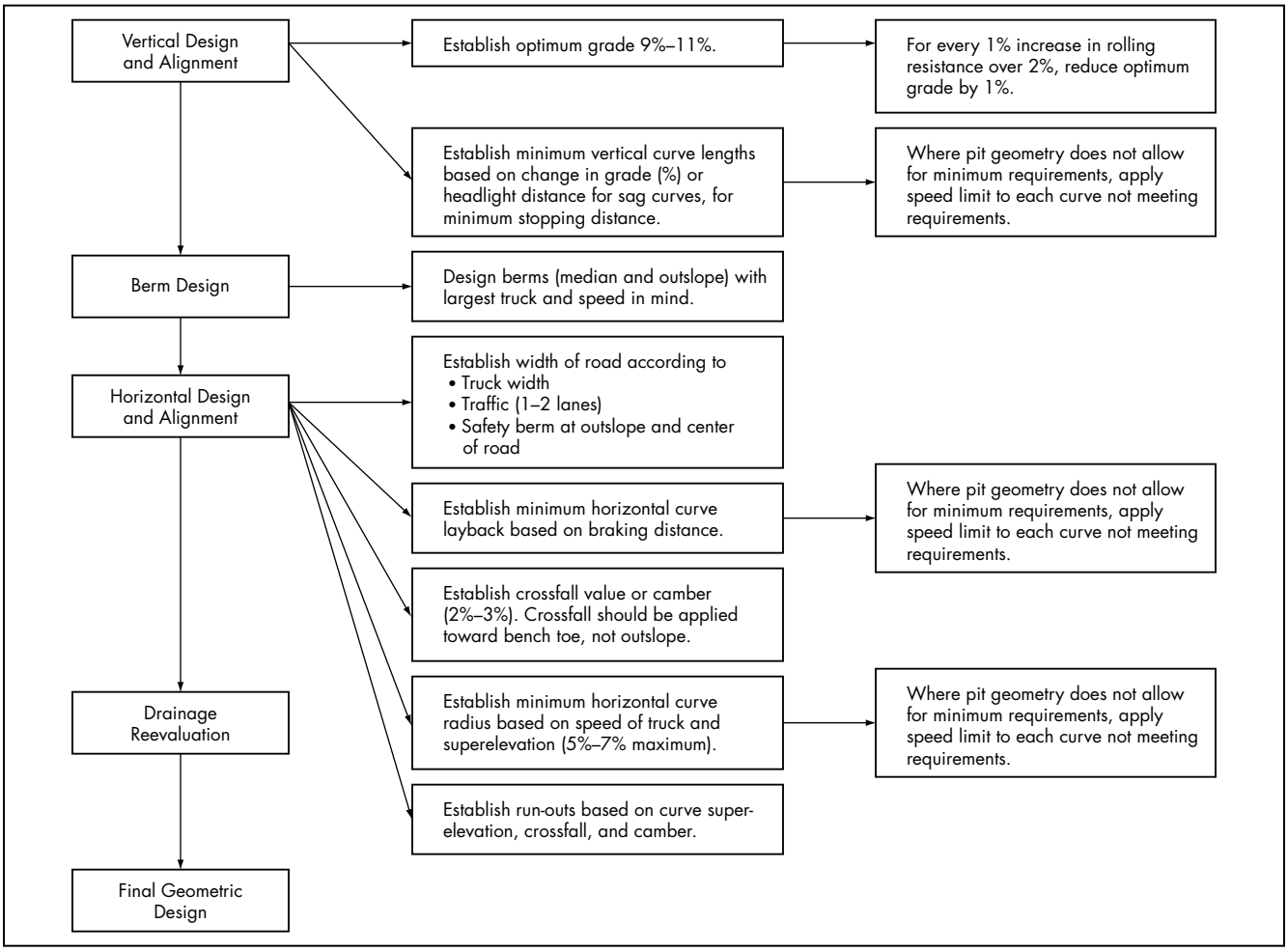


Figure 10.6-4 Final geometric design

Curve Superelevation (Banking)

This is the amount of banking applied on the outside of a curve to allow the truck to run through the curve at speed. Ideally, the outward centrifugal force experienced by the truck should be balanced by the side friction between tires and road. Superelevations should not exceed 5% to 7%, unless high-speed haulage is maintained and the possibility of sliding minimized. Table 10.6-2 shows typical superelevation rates based on speed of vehicle and radius of curve. Where tighter curves are required or truck speed is higher on approach to the curve, a speed limit should be applied.

Cross Slope or Camber

This is critical to the design and successful operation of mine roads. Applying a crossfall or camber ensures that water does not gather on and penetrate into the road surface. Standing water on or in a road will cause rapid deterioration of the road.

Two options exist: either a cross slope from one edge of the road to the other edge or a camber (crown) from the center of the road to both sides of the road. Whatever option is adopted, at the point where the road edge and camber or cross-slope downslopes meet, a drainage ditch must be provided. The two options are illustrated in Figure 10.6-7 (with safety berms omitted, but including drain locations).

A constant camber or cross slope of 2% to 3% is ideal, providing adequate drainage without incurring adverse truck tire and strut loading. A preference may exist for cross slopes because of the envisaged equalized load sharing and tire scrub (i.e., loss of tire tread [rubber]) because of shear contract with the road wearing course. A cross slope should be used with caution, with the slope falling toward the inside of the bench (toe position) as opposed to the outslope side. Where a camber or crown is selected and where this leads to the possibility of trucks sliding in the direction of the bench crest or outslope, or toward a large vertical drop, it is only recommended when large deflection berms are placed at road crest edge.

Special consideration must be given to determining when to use the maximum and minimum rates. Lower cross slopes are applicable to relatively smooth compact road surfaces that can rapidly dissipate surface water without the water penetrating into the road surface. In situations where the surface is relatively rough, a larger cross slope is advisable.

Run-out

Run-out should be used when a section of haul road changes from a cross slope or camber into and out of a superelevated section. The change should be introduced gradually to prevent excessive twisting or racking of the truck chassis. The run-out

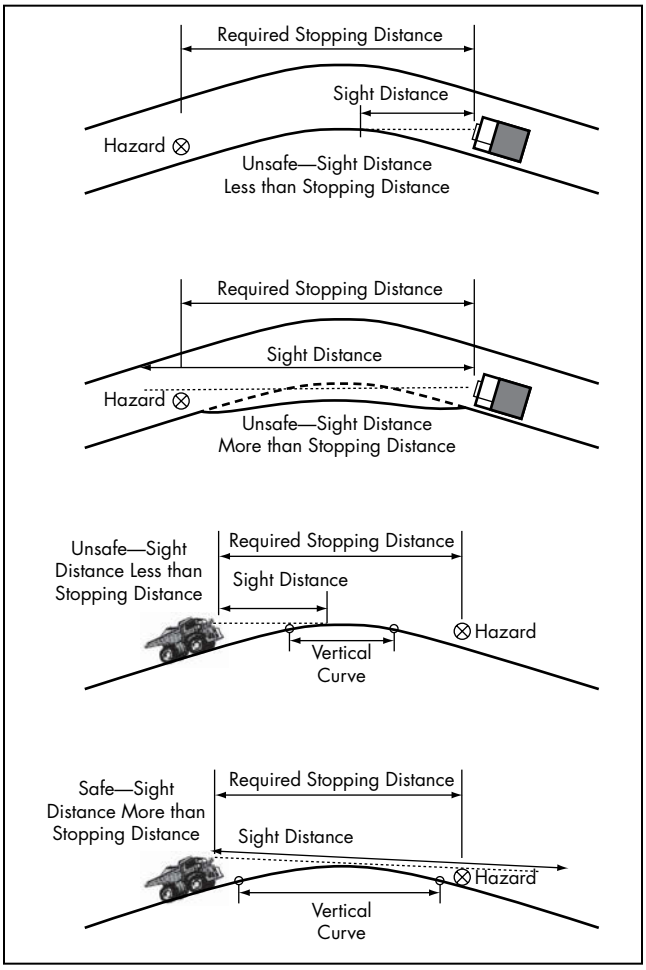


Figure 10.6-5 Horizontal and vertical curve line of sight

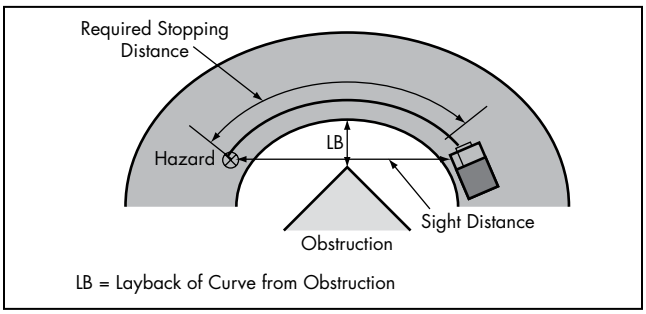


Figure 10.6-6 Layback requirements on obscured horizontal curve

length is typically apportioned 25% to 33% to the curve and 66% to 75% to the tangent or run-up to the curve. Run-out lengths vary with vehicle speed and total crossfall change and can be estimated as

$$CSx = 15.65 - 5.67 \log_{10}(v_o) \qquad (10.6\text{-}7)$$

where CSx is the maximum change (m) in elevation per 30-m tangent road length and v_o is the speed of the truck (km/h). Run-out is best incorporated in a mine road design "by eye" rather than by calculation. Where the run-in or run-out is at 0% (i.e., the road is flat), there should be a slight grade ideally to prevent water ponding on the road at this point.

Table 10.6-1 Minimum haul road width

Number of Lanes	Factor × Width of Largest Truck on Road
1	2
2	3.5
3	5
4	6

Note: For switchbacks or other sharp curves and/or roads with high traffic volumes or limited visibility, a safe roadway width would be designed with an additional 0.5 times the vehicle width. A four-lane road is recommended where trolley-assist systems are in use.

Intersection Layout

This often entails smaller curve radii than that required for maintaining high-speed hauling. Junction priorities should be maximized for nonyield traffic flows. The design of the intersection run-in and run-out should be a smooth spiral progression from one cross slope or camber to the curve superelevation and out again, as shown in Figure 10.6-8. Drainage at the side of each intersection leg must be adequate to keep water from ponding at the roadside and, where possible, intersection segments—or at least nonyield segments—must be placed on flat (not inclined) areas.

Where feasible and safe to do so, junctions can be designed on a herringbone layout, with traffic flow in the direction of the primary destination (i.e., the largest radius of curvature out of the junction), which will facilitate higher-speed hauling and reduce tire scrub, but consideration should be given to maintaining sight distances in all four quadrants.

Combined Alignment

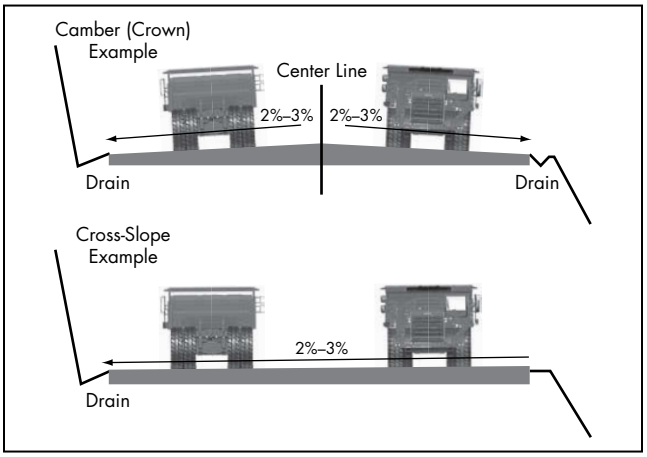
When laying out a haul road, some additional considerations arise with combined alignment:

- Sharp horizontal curves at or near the top of a grade section should be avoided. If a horizontal curve is necessary, it should start well in advance of the vertical curve.
- Switchbacks should be avoided where possible, but if mine plan dictates their use, the radius should be as large as possible and should not be placed on grade.
- Sharp horizontal curves requiring a (further) speed reduction following a long sustained downgrade where haul trucks are normally at their highest speed should be avoided.
- Short tangents and varying grades should be avoided, especially on multilane roads. Grades should be smooth and of consistent grade percentages.
- Intersections should be avoided near the crest of vertical curves or sharp horizontal curves. Intersections should be as flat as possible with sight distances being considered in all four quadrants. Where an intersection lies at the top of a ramp, 100–200 m of flat road should be considered before the intersection, and stopping and starting a laden haul truck on grade should be avoided.
- Drainage design at intersections should stop any ponding of water against intersection superelevated curves.
- Sections of road with no camber or cross fall should be avoided. Often encountered at curve superelevation run-in or run-out, these flat sections should preferably be at a 1% to 2% vertical grade to assist drainage.

Table 10.6-2 Superelevation rates*

	Speed and Superelevation, m/m Width of Road								
Curve Radius, m	15 km/h	20 km/h	25 km/h	30 km/h	35 km/h	40 km/h	45 km/h	50 km/h	55 km/h
50	0.035	0.060	0.090						
75	0.025	0.045	0.070	0.090					
100	0.020	0.035	0.050	0.075	0.090				
150	0.020	0.025	0.035	0.050	0.065	0.085			
200	0.020	0.020	0.025	0.035	0.050	0.065	0.080		
300	0.020	0.020	0.020	0.025	0.035	0.045	0.055	0.065	0.080
400	0.020	0.020	0.020	0.020	0.025	0.035	0.040	0.050	0.060
500	0.020	0.020	0.020	0.020	0.020	0.025	0.030	0.040	0.050

*Elevation rates in the shaded blocks should only be applied as a combined superelevation with a road (center) splitter berm used to separate slow and fast lanes of the road (each with its own speed-related superelevation), because of the possible instability of slow-moving vehicles negotiating higher rates of superelevation.



Courtesy of Caterpillar, Inc.

Figure 10.6-7 Cross-slope and camber options

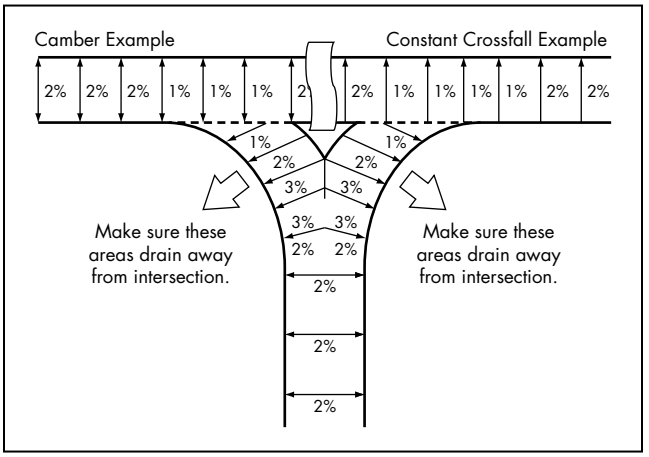


Figure 10.6-8 Junction layout and drainage details

Safety Berms

A crest (outslope) or road-edge berm (windrow) will not effectively stop trucks (especially high-speed laden or unladen trucks) from leaving the road. At best, they will provide limited deflection and warning to the driver that the truck path needs correcting. For large haul trucks, the berm height should be at least 66% of the truck wheel diameter. The slope of the sides of the safety berm should be preferably 3H:1V to ensure stability and maintenance of height.

Truck GVM has a significant deformation effect on the berm, which is typically constructed from unconsolidated material; as the truck climbs the berm, the high center of gravity in combination with a narrow width of the wheel track make trucks susceptible to overturn. Furthermore, large tire sizes and noncentering steering mechanisms reduce the tendency of the truck to redirect itself when encountering a berm. With 4×6- and 6×6-wheel drive ADTs, berm dimensions in excess of 66% wheel diameter are recommended, because of the truck's ability to climb smaller berms. Other factors such as inertial characteristics, sprung mass ratio, and suspension characteristics indicate significantly different response patterns for haul vehicles when encountering berms.

To arrest runaway trucks, median (center or arrestor) berms are often used. They are placed in the center of the roadway and a camber (crown) is required so that water drains to the edges of the road. To enable the driver to steer the truck onto the berm, the front edge is sloped at a 4H:1V run-in, up to the height required. This height is generally 2.5 times the truck ground clearance. Length of the berm is a minimum of 50 m, subject to spacing requirements. To allow the truck to run up, spacing between the berms is required. The berm spacing (Equation 10.6-8) is a function of maximum safe vehicle speed at entry to the berm (v_{max}, m/s), speed at point of brake failure or other hazard (v_{fail}, m/s), and θ, the grade of road (degrees) positive downgrade with a mean coefficient of rolling resistance of 0.05 applied. Figure 10.6-9 illustrates the typical median (arrestor) berm layout.

$$\text{berm spacing} = \left(\frac{v_{max}^2 - v_{fail}^2}{2g \sin(\theta - 0.027)} \right) \tag{10.6-8}$$

Ditches and Drainage

A well-designed drainage system is critical for effective operation and safety; water on the road or in the road layers will quickly lead to poor road conditions. As part of the haul road geometric design process, contours in the vicinity of the proposed road should be examined prior to construction to identify areas of potential ponding, locations of culverts, and so forth.

Drains at the edge of the road should be designed to lead the water off the road without causing erosion. Drains should not be cut into the base layer and should preferably be lined

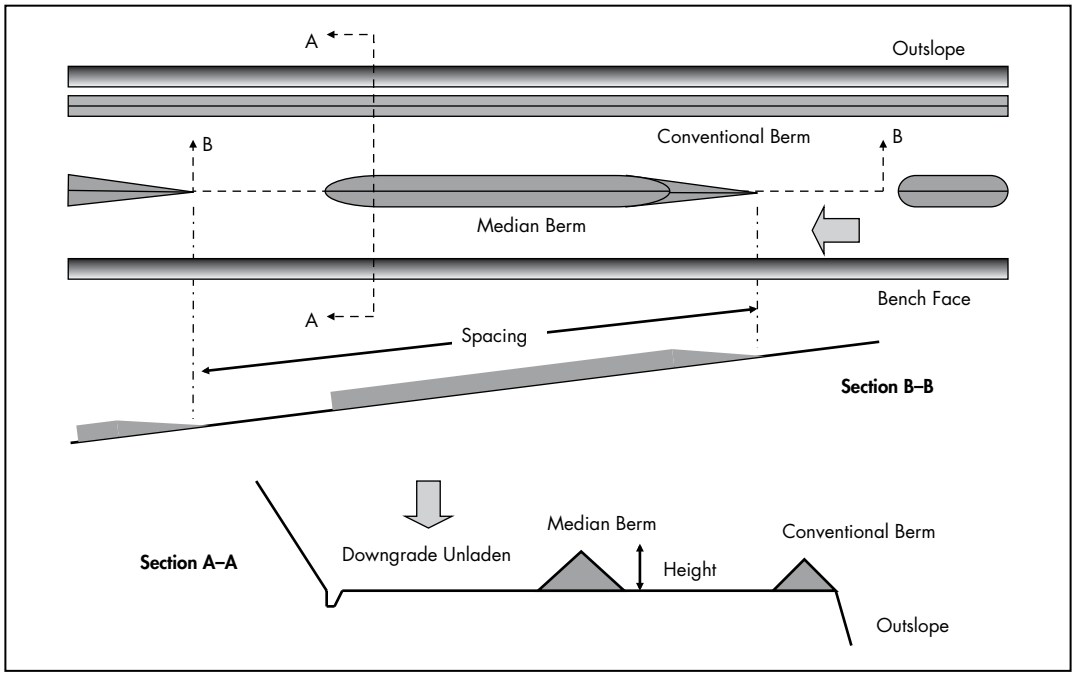


Figure 10.6-9 Median (arrestor) berm design

with compacted wearing course, thereby preventing water from seeping into the underlying layers.

V-ditches are recommended for nearly all applications, owing to their relative ease of design, construction, and maintenance. The ditch cross slope adjacent to the haul road should be 4H:1V or flatter except in extreme restrictive conditions. In no case should it exceed a 2H:1V slope. The outside ditch slope will vary with the material encountered. In rock, it may approach a vertical slope; in less consolidated material, it may be a 2H:1V slope or flatter.

In a cut/fill section, a cross slope toward the cut side should be used and drainage should run in a single ditch. In a total cut or total fill section, drainage should be carried on both sides with crown or camber from the road center line. Ditch lining is a function of road grade and in-situ material characteristics: between 0% and 4% grade, the ditch may be constructed without benefit of a liner except in extremely erodible material such as sand or easily weathered shale and silts; at grades greater than 5%, the lining should consist of coarse crushed waste rock placed evenly on both sides to a height no less than 0.3 m above the maximum depth.

Ditches must be designed to handle expected runoff flows adequately under various slope conditions. The primary consideration is the amount of water that will be intercepted by the ditch during a rainstorm. Typically, a 10-year, 24-hour storm chart should govern the design.

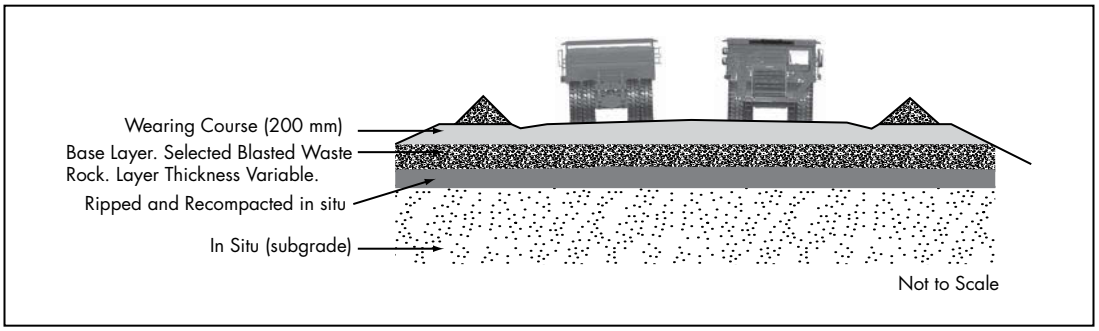
Culvert sections are used to conduct runoff water from drainage ditches under the haul road. If buried piping is used, they should be set at 3% to 4% fall and smooth-wall concrete pipes should be used in conjunction with a drop-box culvert of a size that will allow it to be cleaned with a small backhoe excavator. At all culvert inlets, a protective encasement or "headwall" consisting of a stable nonerodible material should be provided.

Typical culvert units are either portal and rectangular precast concrete culvert units or precast concrete pipe culvert units. Depth of cover after the culvert pipe is determined by the type of culvert in relation to the vehicles that will use the road. A minimum cover of 1,000 mm over the pipe is required in most cases. All prefabricated culverts should be constructed under trenched conditions after the road has been constructed. Concrete pipe culverts are laid on a layer of fine granular material, 75 mm thick, after the bottom of the excavation has been shaped to conform to the lower part of the pipe. Where rock, shale, or other hard material is encountered on the bottom of excavations, culverts should be placed on an equalizing bed of sand or gravel. After placement, the culvert trench is backfilled and compacted.

STRUCTURAL DESIGN

Structural design concerns the ability of a haul road to carry the imposed loads without the need for excessive maintenance or rehabilitation. Haul roads deteriorate with time because of the interactive effort of traffic load and specific subgrade and in-situ material strengths and structural thicknesses. The California bearing ratio (CBR) method (Kaufman and Ault 1977) has been widely applied to the design of mine haul roads in which untreated materials are used. However, when multilayered roads are considered in conjunction with a base layer of selected blasted waste rock, a mechanistic approach is more appropriate. When a selected waste rock layer is located under the wearing course, road performance is significantly improved, primarily because of the load-carrying capacity of the waste rock layer, which reduces the susceptibility of the soft subgrade and in-situ material to the effects of high axle loads. It also has the added advantage of reduced construction costs (by virtue of reduced volumetric and compaction requirements) compared with the CBR cover-curve design approach (Morgan et al. 1994; Thompson and Visser 1996).

A mechanistic design is based on a theoretical linear–elastic multilayer system model of the road. A limiting design criteria of vertical compressive strains in the subgrade or



Courtesy of Caterpillar, Inc.

Figure 10.6-10 Typical haul road structure and layer terminology

in-situ material is then used to assess the haul road under the specific loading conditions, thereby determining the adequacy of the structural design.

Vertical compressive strains induced in a road by heavy wheel loads decrease with increasing depth, which permits the use of a gradation of materials and preparation techniques, with stronger materials being used in the upper regions of the pavement. The road as a whole must limit the strains in the subgrade (in-situ) materials to an acceptable level, and the upper layers must in a similar manner protect the layers below. Using this premise, the road structure should theoretically provide adequate service over its design life. Figure 10.6-10 illustrates a mechanistically derived haul road structure.

In general terms, applied load, subgrade strength, and the pavement structural thickness and layer strength factors predominantly control the structural performance of a haul road. An upper limit of 2,000 microstrains is generally placed on subgrade or in-situ layer strain values. Strain values exceeding 2,500 microstrains are associated with unacceptable structural performance in all but the most lightly trafficked and short-term roads. Data from Figure 10.6-2 can be used to assist in selecting a limiting strain value, according to the category of road to be built and the associated operating life and traffic volumes. In addition, to prevent excessive damage to the wearing course, deformation at the top of this layer must be limited to no more than 3 mm.

As for trolley-assist roads, haul roads in general have a much higher degree of traffic channeling (in which trucks follow the same wheel paths), so structurally, no difference in the approach to structural design is warranted. Critical to the success of trolley-assist is the height of the road surface under the trolley lines; with a good structural design, slight loss of running surface or compaction of the road structure will generally be within the limits of the truck pantograph system. To prevent trucks being rejected from the trolley line, grade changes under trolley should be avoided as far as possible. However, in the case of wearing course material management, because of the high shear forces developed at the wheel–road surface interface as trucks are "accepted" under trolley and accelerate, severe wearing course raveling is often seen in these local areas. This can be ameliorated to some extent by the use of well-specified wearing course material and the use of dust palliatives, which assist in binding the wearing course material.

Mechanistic Structural Design Procedure

To determine the layer response to an applied load, a layered elastic model can be used to represent the various haul road layers in the design. Software is available that can be used to solve multilayer problems in road design, including ELSYM5A (FHWA 1985) and CIRCLY (MinCad 2008). Irrespective of the solution software used, the approach is similar.

The effective elastic (resilient) modulus (E_{eff}) and Poisson's ratio (μ, typically 0.35) define the layerworks material properties required for computing the vertical strains (ε_v) in a road. In addition to the material properties, a layer thickness (200 mm) is also specified for the wearing course (discussed later in the "Functional (Wearing Course) Design" section). By varying the thickness of the waste rock-layer, a solution for maximum strain in any pavement layer that is below the limiting strain criteria for that class of road is found. Generally, a three-layer model is sufficient where the road is built directly on subgrade fill (in-pit blasted rock) or in-situ material (ex-pit softs or weathered overburden). If the construction incorporates ripped and compacted in-situ material, this may also be added as another layer. For computational purposes, the layers are assumed to extend infinitely in the horizontal direction and the lowest pavement layer is assumed to be infinite in depth.

The applied load is calculated according to the mass of the vehicle and the rear dual wheel axle load distributions, from which the maximum single wheel load is found. The load application is determined from dual wheel geometry and, together with tire pressure, the contact stress is calculated. Figure 10.6-11 summarizes the layered elastic model.

The strains induced in a pavement are a function of the effective elastic (resilient) modulus values assigned to each layer in the structure. In order to facilitate a mechanistic design, some indication of applicable modulus values is required. Figure 10.6-12 gives recommended modulus value correlations to the Unified Soil Classification System (USCS) and (American Association of State Highway and Transportation Officials (AASHTO) classification system. To select suitable modulus values for in-situ materials, the associated range of CBR values (alternatively derived in the field from dynamic cone penetrometer probing, Tayabji and Lukanen 2000) are also given. Other values are published by Thompson and Visser (1996) and the South African Roads Board (SARB) (1994). Equation 10.6-9 can also be used in conjunction with layer CBR values to determine modulus values (E_{eff}, MPa), but in each case, care should be taken to ensure that the general correlations presented here are consistent with soil properties not directly assessed in the derivation of the equation.

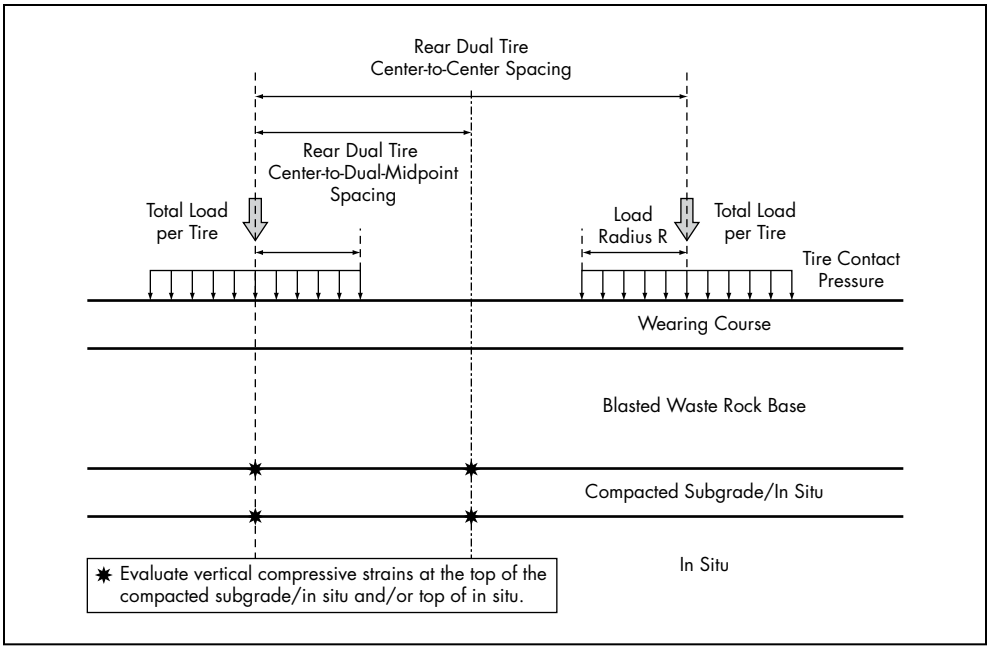


Figure 10.6-11 Multilayer model for structural design evaluation

$$E_{\text{eff}} = 17.63\text{CBR}^{0.46} \quad (10.6\text{-}9)$$

The modulus value adopted for the selected blasted waste rock layer is typically 3,000 MPa. This value is derived from consideration of a cement-stabilized layer in its precracked (large intact blocks with some shrinkage cracks) state, which corresponds closely to a well-compacted waste rock layer.

Method Specification Notes for Structural Design

The structural design is based on the use of selected blasted waste rock. This is subject to the material selected being blocky (maximum block size <50% layer thickness) and not containing excessive (minus-25-mm sieve) fines or clay contamination. Care should be taken when selecting blasted waste; rock from weathered overburden will contain excessive softs. To satisfy the layer specifications, either the blast burden and spacing can be reduced or powder factor increased to produce a finer fragmentation than a conventional production blast. Alternatively, a grizzly sizer can be used to remove the oversize material before placing in the road.

When constructing the various rock layers, the blasted selected rock should be placed in a single lift (if using impact roller compaction), or 300-mm lifts (if using vibratory roller compaction). By running haul trucks over the road during construction (end tipping) and dozing to profile, compaction of the layerworks will be improved and structural capacity increased. In addition, to improve compaction, all excessively large boulders (>50% of layer thickness) should be removed (if not scalped at the grizzly) from the upper portion of the compacted rock layer. The rock layer should be compacted (dry) by impact or vibratory roller compaction until no movement under the roller is visible. This improves the structural response of the road to the applied loads and provides an anvil for improved wearing course layer compaction. Following compaction, a final pass of a grid roller can be used to assist in creating a correction layer of crushed or blasted overburden rock, or similar can be placed if longitudinal or cross-alignment needs correction. Fines should not be used for this purpose.

Finally, a gravel wearing course layer of 200 mm is placed onto the selected rock layer. Design of this layer is discussed in the following section.

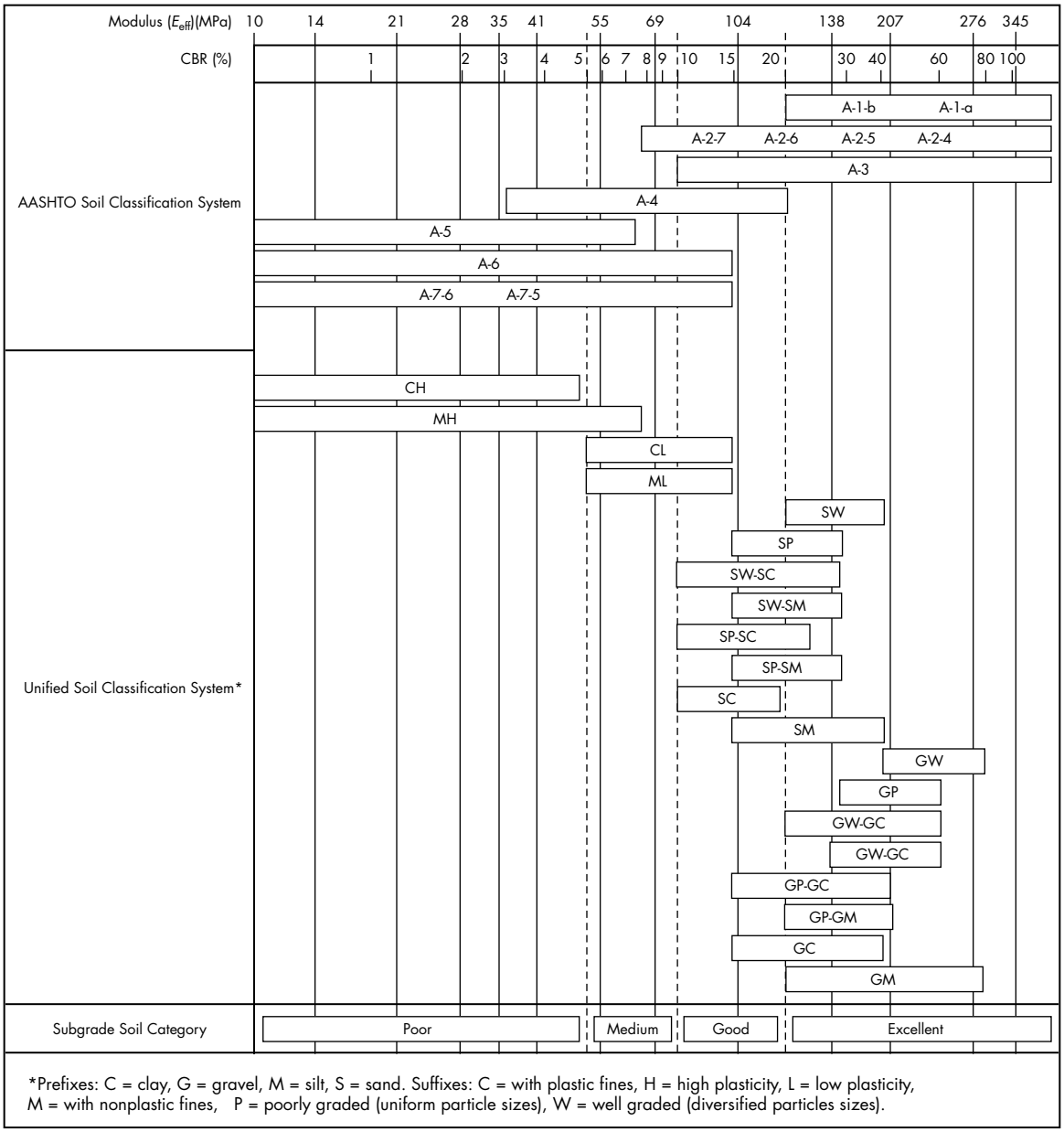
FUNCTIONAL (WEARING COURSE) DESIGN

Equally important as the structural strength of the design is the functional trafficability of the haul road. This is dictated to a large degree through the selection, application, and maintenance of the wearing course (or road surfacing) materials. Poor functional performance is manifest as poor ride quality, excessive dust, increased tire wear and damage, and an accompanying loss of productivity. The result of these effects is seen as an increase in overall vehicle operating and maintenance costs.

The functional design of a haul road is the process of selecting the most appropriate wearing course material or mix of materials, typically natural gravel or crushed stone and gravel mixtures that are commensurate with safety, operational, environmental, and economic considerations. The most common wearing course material for haul roads remains compacted gravel or gravel and crushed stone mixtures. In addition to their low rolling resistance and high coefficient of adhesion, their greatest advantage over other wearing course materials is that roadway surfaces can be constructed rapidly and at relatively low cost. As with structural designs, if local mine material can be used for construction, the costs are all the more favorable. This cost advantage is, however, not apparent in the long term if the characteristics of the wearing course material result in suboptimal functional performance.

An ideal wearing course for mine haul road construction should exhibit

- The ability to provide a safe and vehicle-friendly ride without the need for excessive road maintenance,
- Adequate trafficability under wet and dry conditions,
- The ability to shed water without excessive erosion,
- Resistance to the abrasive action of traffic,



Source: Adapted from NAPA 1994.

Figure 10.6-12 Typical elastic modulus correlations to empirical soil properties and classification categories

- Freedom from excessive dust in dry weather,
- Freedom from excessive slipperiness in wet weather, and
- Low cost and ease of maintenance (McInnes 1982; Thompson and Visser 2000a).

The defects most commonly associated with mine haul roads, in order of decreasing impact on hauling operational performance, are typically as follows:

- Skid resistance—wet
- Skid resistance—dry
- Dustiness
- Loose material
- Corrugations
- Stoniness—loose
- Potholes
- Rutting
- Stoniness—fixed
- Cracks—slip, longitudinal, and crocodile. "Crocodile cracks" are cracks in a road caused by high plasticity/high clay content material shrinking when drying. This is a well-known phrase among the pavement design fraternity.

By examining which wearing course material property parameters lead to these defects, a specification has been developed for wearing-course materials selection. The specifications are based on an assessment of wearing-course material shrinkage product (*Sp*) and grading coefficient (*Gc*), defined in Equations 10.6-10 and 10.6-11:

$$Sp = \text{LS} \times \text{P425} \tag{10.6-10}$$

$$Gc = \frac{(P265 - P2) \times P475}{100} \quad (10.6\text{-}11)$$

where

LS = bar linear shrinkage
P425 = percent wearing course sample passing 0.425-mm sieve
P265 = percent wearing course sample passing 26.5-mm sieve
P2 = percent wearing course sample passing 2-mm sieve
P475 = percent wearing course sample passing 4.75-mm sieve

A suitable wearing course material can be determined from the selection chart (Figure 10.6-13) in terms of two parameters that describe the material: the grading coefficient (*Gc*) and shrinkage product (*Sp*). If the three most critical haul road defects are considered, it appears that mine road-user preference is for much reduced skid resistance and dust defects. This defines the focus point of the specifications to an area bounded by a grading coefficient of between 25 and 32 and a shrinkage product of between 95 and 130 in which the overall and individual defects are minimized (Area 1). Extending this region to encompass poorer (but nevertheless operable) performance enables an additional area (Area 2) to be defined.

The specification stipulates individual parameter limits but also has predictive capabilities that contribute to an understanding of the consequences when materials outside the specified ranges are used as wearing course materials. If the only materials available for wearing course selection lie outside the parameter limits, a mixture of those materials can be evaluated using the preceding guidelines. The proposed mix ratio can be used to define a new "mixed" material specification proportional to the mix ratio, from which *Gc* and *Sp* can be derived. In a similar fashion, an existing haul road wearing course can be successfully rehabilitated by adding an appropriate material to restore the mix to specification.

The specifications should also be evaluated in the light of other material property limits identified as important in functional performance but not directly assessed in Figure 10.6-13. Table 10.6-3 presents a summary of these property limits, together with the type of road defects most often associated with departures from the recommended parameter ranges.

Method Specification Notes for Functional Design

The method specification for placement of the wearing course depends on whether the haul road is being built from new or being rehabilitated (where material is mixed with the existing wearing course to bring it back to specification). For rehabilitation, the existing wearing course layer should be ripped and scarified and any large lumps of compacted material broken down to a maximum size of one-third of the compacted layer thickness. During the processing, the scarified layer should be ploughed or bladed to bring large lumps to the surface. An offset disc harrow can be used for this purpose. Material to be mixed should be dumped, opened, and mixed with the existing wearing course. Placement of the wearing course should proceed in two lifts of 100 mm, each compacted to 98% maximum dry density to give a minimum CBR of 80% using a large vibratory roller and with moisture content at or slightly dry of optimum. Material volumetric requirements must be determined such that the final thickness of the wearing course does not exceed 200 mm.

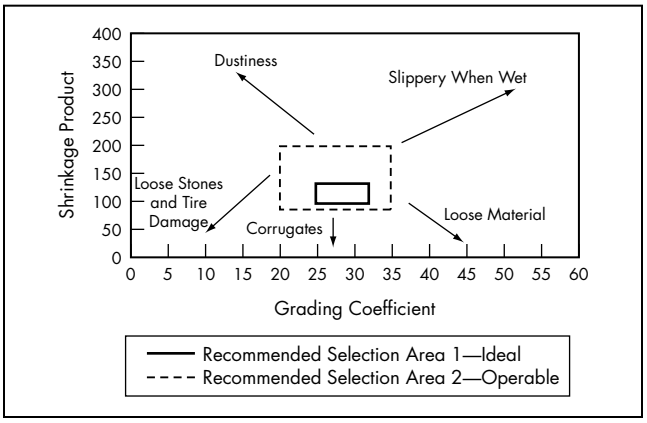


Figure 10.6-13 Haul road wearing course material selection

Estimating Haul Road Rolling Resistance

The rolling resistance of a haul road is primarily related to the wearing course material used, its engineering properties, and traffic speed and volume. These dictate to a large degree the rate of increase in rolling resistance. Ideally, road rolling resistance should not increase rapidly, which implies that those road defects (roughness defects) leading to rolling resistance should also be minimized. This can be achieved through careful selection of the wearing course material, which will minimize, but not totally eliminate, rolling resistance increases over time (or traffic volume).

To estimate rolling resistance at a point in time, an estimate of the roughness defect score (RDS) is required, and this can be determined from an initial estimate of the minimum and maximum roughness defect scores (RDSMIN, RDSMAX), together with the rate of increase (RDSI). Rolling resistance is then estimated from a minimum value (RRMIN) and the associated rate of increase (RRI). Equations 10.6-12 to 10.6-18 apply, together with the parameters and variables defined in Table 10.6-4. When using these equations, care should be taken to ensure the parameters limits are comparable to the values used in the original research (Thompson 1996).

$$RDS = RDSMIN + \left[\frac{RDSMAX - RDSMIN}{1 + \exp^{(RDSI)}}\right] \quad (10.6\text{-}12)$$

where

$$RDSMIN = 31.1919 - 0.05354 \cdot Sp - 0.0152 \cdot CBR \quad (10.6\text{-}13)$$

$$RDSMAX = 7.6415 + 0.4214 \cdot KT + 0.3133 \cdot Gc + 0.4952 \cdot RDSMIN \quad (10.6\text{-}14)$$

$$RDSI = 1.768 + 0.001 \cdot D(2.69 \cdot KT - 72.75 \cdot PI - 2.59 \cdot CBR - 9.35 \cdot Gc + 1.67 \cdot Sp) \quad (10.6\text{-}15)$$

$$RR = RRMIN + RDS \cdot \exp^{(RRI)} \quad (10.6\text{-}16)$$

where

$$RRMIN = \exp^{(-1.8166 + 0.0028 \cdot V)} \quad (10.6\text{-}17)$$

$$RRI = 6.068 - 0.00385 \cdot RDS + 0.0061 \cdot V) \quad (10.6\text{-}18)$$

Table 10.6-3 Recommended parameter ranges for mine haul road wearing course material selection

Impact on Functionality Below Recommended Range	Material Parameter	Range		Impact on Functionality Above Recommended Range
		Minimum	Maximum	
Reduced slipperiness but prone to raveling and corrugation	Shrinkage product	85	200	Increased dustiness and poor wet-skid resistance
Increased loose stones, corrugations, and potential tire damage	Grading coefficient	20	35	Increased raveling and poor dry-skid resistance
Reduced dustiness but loose material will ravel	Dust ratio	0.4	0.6	Increased dust generation
Increased loose stoniness	Liquid limit, %	17	24	Prone to dustiness, reduced raveling
Increased loose stoniness	Plastic limit, %	12	17	Prone to dustiness, reduced raveling
Increased tendency to ravel, loose stoniness	Plasticity index	4	8	Prone to dustiness and poor wet-skid resistance
Poor wet weather trafficability, churning, excessive deformation, and cross-erosion; maintenance intensive	Soaked CBR at 98% Modified AASHTO	80		Increased resistance to erosion, rutting, and improved trafficability
Ease of maintenance, vehicle-friendly ride, and no tire damage	Maximum particle size, mm		40	Poor surface finish following maintenance; potholing and potential tire damage

Table 10.6-4 Definition of parameters and independent variables used in the defect score progression model

Parameter	Description
RDS	Roughness defect score
RDSMIN	Minimum roughness defect score immediately following last maintenance cycle
RDSMAX	Maximum roughness defect score
RDSI	Rate of roughness defect score increase
RR	Rolling resistance, newtons/kilogram
RRMIN	Minimum rolling resistance at RDS = 0
RRI	Rate of increase in rolling resistance from RRMIN
Variable	**Description**
V	Vehicle speed, kilometers/hour
D	Days since last road maintenance
KT	Average daily tonnage hauled, kilotons
PI	Plasticity index
CBR	California bearing ratio of wearing course at 100% modified AASHTO compaction

Haul Road Dust Palliation

Dust generation is the process by which fine wearing course material becomes airborne. Such generation is termed a *fugitive* (or open) dust source. The amount of dust that will be emitted is a function of two basic factors:

1. The wind-erodibility of the material involved; and
2. The erosivity of the actions to which the material is subjected.

In broad terms, the effectiveness of any dust suppression system is dependant on changing material wind-erodibility or erosivity. The wearing course silt and fine sand fractions (i.e., between 2 and 75 μm) are a good indication of its erodibility.

The motivation for the use of some additional agent to reduce a material's inherent erodibility is based on increasing particle binding. The finer fraction, although contributing to cohesiveness, also generates much of the dust, particularly when the material is dry. The presence of larger fractions in the material will help reduce erodibility of the finer fractions, as will the presence of moisture but only at the interface between the surface and the mechanical eroding action. This forms the basis of the water-based dust suppression techniques used most commonly on mine haul roads.

The consequences of dust generation include

- Loss and degradation of the road pavement material, the finer particles being lost as dust and the coarser aggregates being swept from the surface or generating a dry skid-resistance defect;
- Decreased safety and increased accident potential for road users, because of reduced or obscured vision and reduced local air quality; and
- Higher vehicle operating costs (VOCs), with dust penetrating the engine and other components resulting in increased rates of wear and more frequent maintenance.

Many products available are claimed to reduce both dust and road maintenance requirements for mine roads. Often, however, no comprehensive, comparable, and controlled performance trials have been published. Additionally, incorrect application techniques and construction methods often result, which leads to considerable skepticism about such products and their overall cost-effectiveness.

From a mining perspective, the following parameters would define an acceptable dust palliative:

- Spray-on application with deep penetration (the ability to penetrate compacted materials) or, though less preferable, mix-in applications with minimal site preparation (rip, mix-in, and recompact)
- Straightforward applications requiring minimal supervision, not sensitive nor requiring excessive maintenance or closely controlled reapplications
- The road trafficable within a maximum of 24 hours (short product curing period)
- Availability in sufficient quantity at reasonable prices
- Adequate proven or guaranteed durability, efficiency, and resistance to deterioration by leaching, evaporation, ultraviolet light, and chemical reaction with wearing course or spillage on road
- Effective over both wet and dry seasons
- Evaluated against local and international safety standards and environmentally acceptable

Table 10.6-5 Dust palliative products and application parameters

	Hygroscopic Salts	Lignosulphonates	Petroleum-Based Products	Others (sulphonated petroleum, ionic products, polymers, and enzymes)
Climatic limitations	Salts lose effectiveness in continual dry periods with low relative humidity. Selection dependent on relative humidity and potential to water road surface.	Retains effectiveness during long, dry periods with low humidity.	Generally effective regardless of climate but will pothole (small diameter) in wet weather where fines content of wearing course is high.	Generally effective regardless of climate.
Wearing course material limitations	Recommended for use with moderate surface fines (maximum 10%–20% <0.075 mm). Not suitable for low fines materials or high shrinkage product/PI*, low CBR†, or slippery materials.	Recommended for use where high (<30% <0.075 mm) fines exist in a dense graded gravel with no loose material.	Performs best with low fines content (<10% <0.075 mm). Use low-viscosity products on dense fine-grained material, more viscous products on looser, open-textured material.	PI range 8–35. Fines limit 15%–55% < 0.075 mm. Minimum 98% maximum dry density. Performance may be dependent on clay mineralogy (enzymes).
Treatment maintenance and self-repair capability	Reblade under moist conditions. $CaCl_2$ is more amenable to spray-on application. Low-shrinkage product materials may shear and corrugate with high-speed trucks. Shear can self-repair.	Best applied as an initial mix-in and quality of construction important. Low-shrinkage product materials may shear and corrugate with high-speed trucks. Tendency to shear or form "biscuit" layer in dry weather—not self-repairing.	Requires sound base and attention to compaction moisture content. Slow speed, tight radius turning will cause shearing—not self-repairing but amenable to spot repairs.	Mix-in application—sensitive to construction quality. Difficult to maintain—rework. Generally no problem once cured.
Tendency to leach out or accumulate	Leaches down or out of pavement. Repeated applications accumulate.	Leaches in rain if not sufficiently cured. Gradually oxidizes and leaches out. Repeated applications accumulate.	Does not leach. Repeated applications accumulate.	Efficacy depends on the cation exchange capacity of the host material. Repeated applications accumulate.
Comments	A high fines content may become slippery when wet. Corrosion problems may result.	Generally ineffective if wearing course contains little fine material or there is excessive loose gravel on the road.	Long lasting—more effective in dry climates.	Generally ineffective if material is low in fines content or where loose gravel exists on surface. Curing period required.

Source: Adapted from Thompson and Visser 2000b.
*Plasticity index.
†California bearing ratio (%).

The broad classes of products available are described in Table 10.6-5.

The selection matrix in Table 10.6-6 can additionally be used to identify classes of palliative that would suit a certain application. However, the data does not specify the level of performance that could be expected, nor the average degree of palliation or degeneration rate expressed in terms of time from initial establishment and reapplication rates. This information would be required as a precursor to an economic assessment of the selected palliative benchmarked against the base case of water-based spraying.

A poor wearing course material cannot be improved to deliver an adequate performance solely through the addition of a dust palliative. The haul road wearing course material should ideally meet the minimum specifications presented earlier. If not, the inherent functional deficiencies of the material will negate any benefit gained from using dust palliatives. In road surfaces with too much gravel, dust palliatives do not appear to work effectively, more especially where a spray-on technique is used as opposed to a mix-in. The palliatives do not aid compaction of the surface because of the poor size gradation nor form a new stable surface. New surface area is created from exposed untreated material while, with a mix-in application, poor compaction leads to damage and raveling of the wearing course, traffic-inducing breakdown of the material, and eventual dust generation. With regard to water-soluble palliatives, rapid leaching may be problematic in some climates.

In compact sandy soils, tar and bituminous-based emulsion products appear effective where leaching of water-soluble products may be problematic. However, in loose medium and fine sands, bearing capacity will not be adequate for the tar/bitumen products to maintain a new surface, and degeneration can rapidly occur. In road surfaces with too much silt, it is unlikely that a dust suppression program will be effective. Excessive silt or sand fractions may lead to a slippery road, while poor bearing capacity leads to rutting and the need for road rehabilitation or maintenance, which destroys most products. Small-scale potholing has been observed on a number of pavements following spray-on application or reapplication, as a result of trafficking lifting fine cohesive material from the road. Again, where no depth of treatment has built up, this will lead to the creation of new untreated surfaces.

In general, spray-on applications do not appear appropriate for establishment of dust treatments, especially with regard to depth of treatment required. A spray-on reapplication or rejuvenation may be more appropriate, but only if penetration of the product into the road can be ensured; otherwise, it will only serve to treat loose material or spillage buildup, which will rapidly break down and create new untreated surfaces. A spray-on treatment is however useful

Table 10.6-6 Preliminary palliative selection matrix

	High PI (>10)	Medium PI (>5)	Sand	Wet Weather Trafficability	Ramp Roads	Heavy Traffic	Short Term	Long Term	Spray-On	Mix-In	Maintainable
Wetting agents		✓			✓		✓		✓I/R		
Hygroscopic salts		✓				✓	✓		✓R	✓I	✓M
Ligno-sulfonates	✓	✓	✓				✓	✓		✓I/R	✓SO
Petroleum emulsions	✓	✓	✓	✓	✓	✓	✓		✓I	✓I	
Polymer emulsions	✓	✓		✓	✓	✓		✓		✓I	✓SO
Tar/bitumen emulsions		✓		✓	✓	✓		✓	✓R	✓I	✓SR

Note: I = initial establishment application, R = follow-on rejuvenation applications, M = maintain when moist or lightly watered, SO = maintain with spray-on reapplication, SR = maintain with spot repairs.

to suppress dust emissions from the untrafficked roadsides, because it would be easier (and less expensive) to apply and, with the material typically being uncompacted, would provide some depth of penetration and a reduction in dust emissions because of truck-induced air turbulence.

When chemical-based dust suppressants are applied to an appropriate wearing course, the average degree of dust palliation and the period over which it is applied is often seen to be considerably better than that achievable by water-based spraying alone (Thompson and Visser 2002). However, in terms of cost-effectiveness, an evaluation is required with which to determine the extent of the cost benefits attributable to chemical-based dust suppression, together with an indication of those factors likely to alter the trade-off between water- and chemical-based dust palliation. A typical approach is illustrated in Figure 10.6-14.

Finally, it is worth noting that although tire chains are mostly limited to loading equipment, when dust palliatives are used, to maintain (treated) wearing course integrity, all tracked and tire-chained equipment should be kept off the road and either run along the road edge or be transported by low-bed trailers.

MAINTENANCE MANAGEMENT DESIGN

Design and construction costs for the majority of haul roads represent only a small proportion of the total operating and road maintenance costs; in particular, the use of an appropriate road maintenance management strategy has the potential to generate significant cost savings, particularly in the light of increases in rolling resistance because of the interactive effects of traffic volume and wearing course deterioration (Thompson and Visser 2003).

The management and scheduling of mine haul road maintenance has not been widely reported in the literature, primarily because of the subjective and localized nature of operator experience and required road functionality levels. In most cases, comment is restricted to the various functions comprising maintenance, as opposed to the management of maintenance to minimize total costs. Long (1968) suggests that adequate serviceability (functionality) can be achieved by the use of one motor grader (and water car) for every 45,000 tkm (metric ton–kilometers) of daily haulage. The U.S. Bureau of Mines' Minerals Health and Safety Technology Division (USBM 1981) in their report on mine haul road safety hazards confirm these specifications but without a clear statement as to what activities comprise road maintenance.

In addition to the lack of unanimous objectives in applying maintenance, the definition of maintenance as applied to mine haul roads varies from mine to mine. Table 10.6-7 summarizes these maintenance activities, modified for application to mine haul roads. The routine maintenance category is adopted here to describe the various activities envisaged for haul road routine maintenance. Table 10.6-8 summarizes the various routine maintenance systems that mines typically apply.

Routine maintenance is carried out on mine haul roads almost daily, depending on the functionality of the road and the traffic volume. The principal goals are to restore the road functionality to a level adequate for efficient vehicle travel with the aim of augmenting productivity and minimizing total road-user costs, and to conserve the integrity of the road wearing course by returning or redistributing the gravel surface.

Ad-hoc or scheduled blading is an inefficient means of road maintenance, with the potential to generate excessive costs due to overmaintenance or undermaintenance of the road. Ideally, an optimized approach is required with which to minimize total road-user costs. A maintenance management system (MMS) for mine haul roads is described here, following Thompson and Visser (2006) to address these needs.

Maintenance Management Systems

The ideal maintenance strategy for mine haul roads should be the one that results in the minimum total road-user cost because, in the case of mine haul roads, the agency maintaining the haul road network is directly affected by road-user operating costs. Two elements form the basis of road-user costs, namely road maintenance costs and VOCs. Both these cost elements are directly related to road condition or, more specifically, rolling resistance. The selection of a maintenance program for mine haul roads should be based on the optimization of these individual costs, such that total vehicle operating and road maintenance costs are minimized, as shown schematically in Figure 10.6-15.

Mine haul road maintenance intervals are closely associated with traffic volumes, with operators electing to forgo maintenance on some sections of a road network in favor of others. This implies an implicit recognition of the need to optimize limited road maintenance resources to provide the greatest overall benefit. This optimization approach is inherent in the structure of the MMS for mine haul roads. Two elements form the basis of the economic evaluation, namely

1. Haul road functional performance—rolling resistance increases with time/traffic, and
2. Vehicle operating and road maintenance cost increase with time/traffic.

The approach is suited to a network of mine haul road segments, as opposed to a single road analysis. For a number

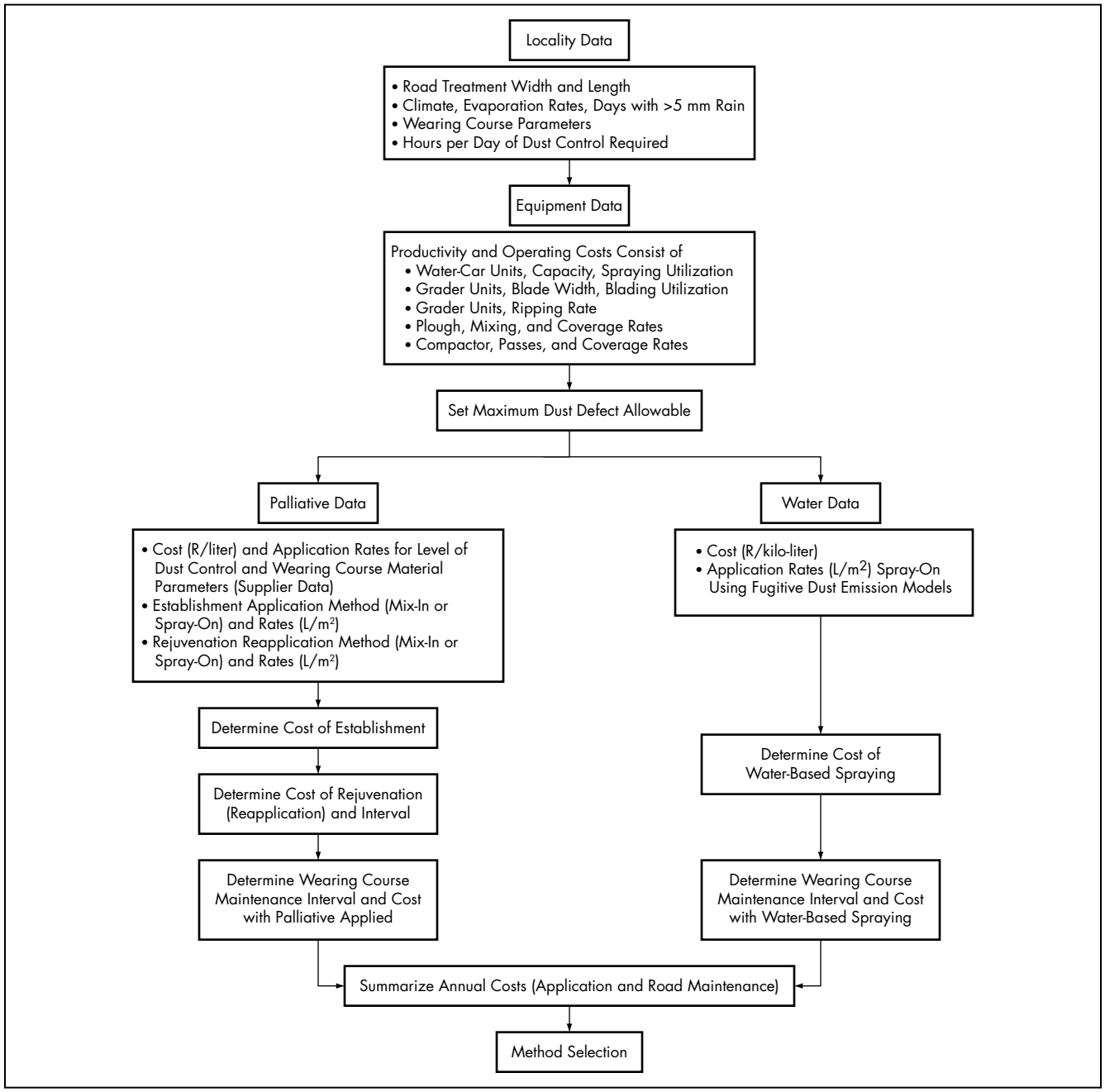


Figure 10.6-14 Basis of dust palliative cost–benefit evaluation

of road segments of differing functional and traffic volume characteristics, together with user-specified road maintenance and VOC unit costs, the model assesses

- Traffic volumes over network segments during the analysis period (as specified),
- The change in road as rolling resistance by modeling or on-site assessment,
- The maintenance quantities as required by the particular strategy,
- The VOCs (by modeling),
- Total costs and quantities, and
- The optimal maintenance frequency for specified segments of the network such that total road-user costs are minimized.

Figure 10.6-16 illustrates the MMS flow chart used to determine optimum maintenance interval for a mine road consisting of a number of road segments. Cost savings associated with the adoption of an MMS approach are dependant on the particular hauling operation, vehicle types, road geometry, tonnages hauled, and so forth.

The first element of an MMS for mine haul roads is based on modeling the variation of VOCs with rolling resistance. When combined with a road maintenance cost model, the

Table 10.6-7 Maintenance categories and activities for mine haul roads

Mode	Activity	Effect
Routine maintenance	Shallow blading Spot regraveling Drainage and shoulder maintenance	Fill minor depressions, reduce rolling resistance, exclude water. Reduce erosion and material loss, improve roadside drainage. Redistribute surface gravel.
Resurfacingt	Full regraveling Deep blading	Restore thickness of wearing course. Reprofile road and reduce rate of increase in rolling resistance. Remix wearing course material.
Rehabilitation	Rip, regravel, recompact	Improve, strengthen or salvage deficient wearing course.
Betterment	Rehabilitation and geometric improvement	Improve geometric alignment and structural strength.

Table 10.6-8 Routine mine haul road maintenance systems

System	Description
Ad-hoc blading	Reactionary maintenance management in response to poor haul road functionality. Typically managed by daily inspection of the road network and a subjective assessment of road segment functionality and maintenance priorities.
Scheduled blading	Road network is maintained according to a fixed schedule or frequency, irrespective of the actual functionality of the road segment being worked.
Maintenance management system	Road network is analyzed to determine rate of functional deterioration of individual segments, based on rolling resistance deterioration, traffic volumes, and so forth, and segment blading frequency is determined to minimize segment and network total road-user costs.

optimal maintenance strategy for a specific mine haul road commensurate with lowest overall vehicle and road maintenance costs may be identified.

Vehicle Operating Cost Model

The VOC model refers to the incremental cost of truck operation with changes in road rolling resistance. The cost model should consider the effect of increased rolling resistance on fuel consumption, tires, and vehicle maintenance. However, a reasonable approximation can be determined from fuel consumption alone. The prediction of fuel consumption variation with rolling resistance involves simulation with specific haul trucks to generate a speed model for various road grades. The speed model forms the basis of the fuel consumption model, derived from vehicle simulations coupled with vehicle torque/fuel consumption maps.

Road Maintenance Cost Model

The road maintenance operating cost per kilometer comprises both grader and water-car operating costs. Although not contributing directly to a reduction in rolling resistance, the incorporation of the watering costs in the maintenance costs model reflects (the ideal) operating practice in which, immediately before blading, the section of road is watered to reduce dust and erosion and to aid blading and recompaction.

Grader and water-car productivities of 0.75 and 6.25 km maintained road per operating hour for each machine, respectively, are typical and correlate with published figures of between 8 and 18 km of maintained road per 16-hour day. However, as the condition of the haul road deteriorates, maintenance becomes more time-consuming and the number of blade passes required to achieve an acceptable finish when the RDS exceeds 45 (equivalent to approximately 3% rolling resistance) increases. A maintenance productivity curve, shown in Figure 10.6-17, incorporates this reduction in grader productivity associated with excessively rough roads. RDS can be converted to rolling resistance through Equations 10.6-16 to 10.6-18. The road maintenance cost model is thus constructed from consideration of the average blade width per pass, road width, RDS before blading, motor-grader productivity curve, and hourly cost from which the motor-grader cost per kilometer is found. This cost is then combined with the cost per kilometer of the water-car and workshop costs to produce a total cost per kilometer for road maintenance.

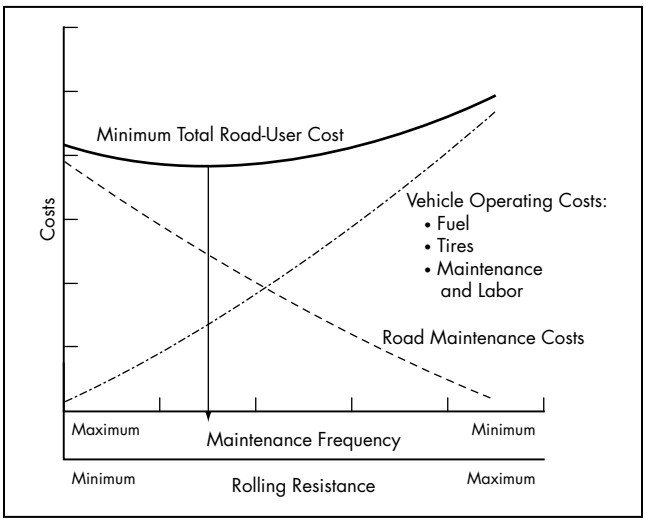


Figure 10.6-15 Minimum total cost solution and required road maintenance frequency from VOCs and road maintenance cost considerations for one segment of haul road network

Estimating Rolling Resistance from Road Visual Assessments

In addition to the estimation techniques for rolling resistance progression presented earlier in the chapter, rolling resistance can also be estimated from qualitative visual evaluation. A road defect classification system can be applied in which the key defects influencing rolling resistance are identified and the product of defect degree (measured from 1–5) and extent (measured

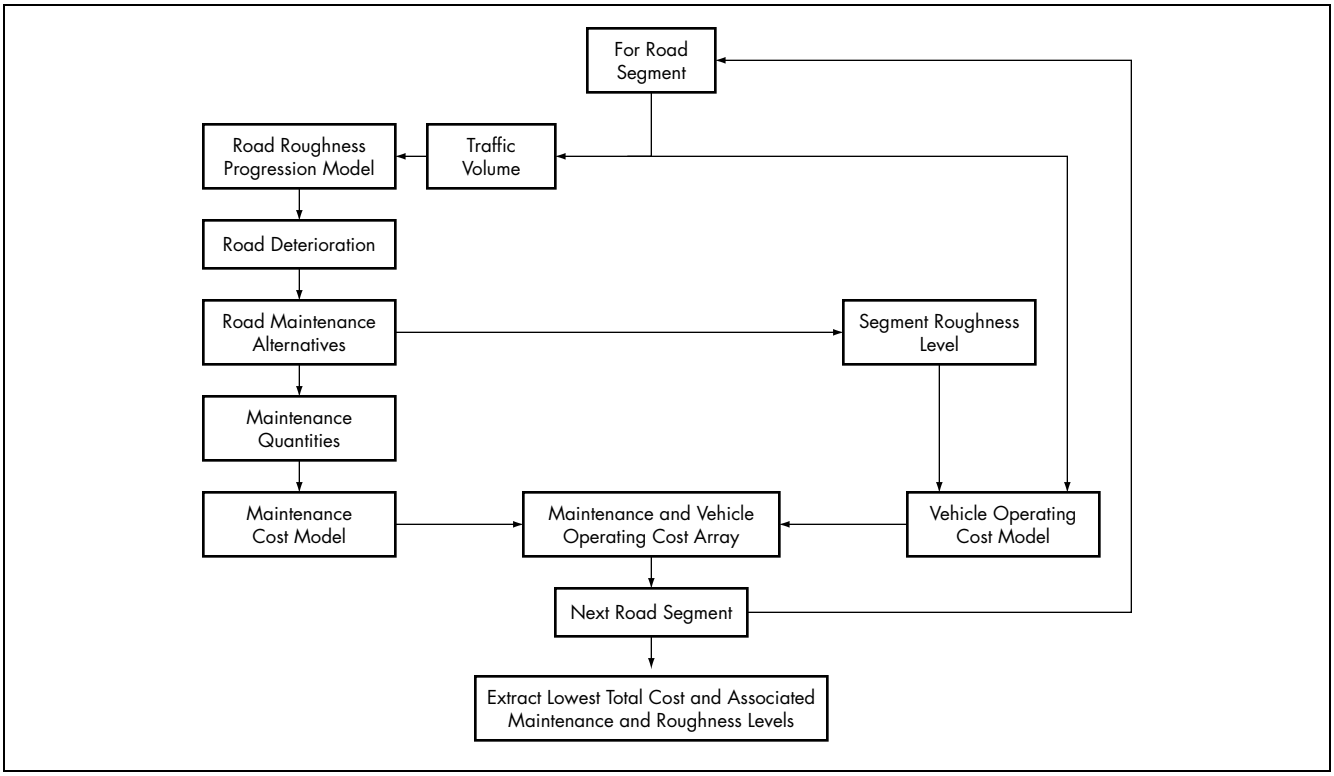


Figure 10.6-16 Flow diagram of MMS for mine haul roads (for a single maintenance strategy iteration)

from 1–5) are scored (following Table 10.6-9). The sum of the individual defect scores thus rated (equivalent to the RDS discussed previously) can be converted using Figures 10.6-18 and 10.6-19 to give a rolling resistance for the segment of haul road under consideration. More details of the approach and aids to visual evaluation of defects are given by Thompson and Visser (2006).

CONCLUSIONS

To design and operate optimally efficient and safe mine roads requires an integrated road design and management strategy. Specifically, through addressing the requirements of each aspect of road design (geometric, structural, functional, and maintenance management), rolling resistance can be optimized and total road-user costs minimized across a network. Implicit in the approach is the need to evaluate the contribution of various segments of the road network to the total cost of haulage, and prioritize resources to where they deliver the most benefit in terms of reducing cost per metric ton hauled.

Targeting haul road improvement strategies should be based on recognizing the root cause of underperformance. When geometric design is optimized and potential problems are identified and corrected, the structural design methodology can be used to reduce rolling resistance by minimizing deflection and deformation of the road structure beneath the vehicle wheels. Improving functionality of a haul road is based on the optimum wearing course material selection parameters. To achieve this optimum, an operation should evaluate its current or proposed wearing course and recognize what material deficiencies exist, as a route to determining which additional or new materials or palliative products should be added to bring the wearing course to specification.

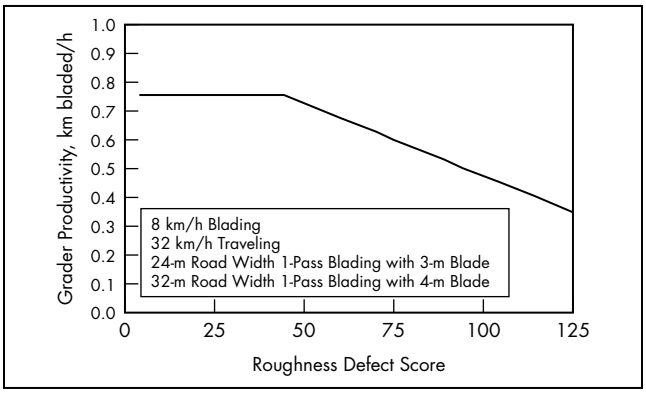


Figure 10.6-17 Productivity of a motor-grader during routine haul road maintenance operations

By using a modeling or visual evaluation system, both as a basis for rolling resistance evaluations and maintenance interventions, mines will be able to recognize recurring defects on the road and target improvement strategies, because the root cause for underperformance can be recognized. In the longer term, the approach can be used as a basis for planning strategic improvements by identifying the specification parameter and performance deficiencies across the network of roads. Segments of the network can then be selected for upgrade based on the costs of repair measured against the benefits derived from total road-user cost reduction over the operating life of the segment.

Table 10.6-9 Defect degree and extent scores for rolling resistance evaluation

	Description of Defect Extent or Degree				
Extent Score	**1**	**2**	**3**	**4**	**5**
Extent	Isolated occurrence, less than 5% of road affected	Intermittent occurrence, from 5% to 15% of road affected	Regular occurrence, from 16% to 30% of road affected	Frequent occurrence, from 31% to 60% of road affected	Extensive occurrence, more than 60% of the road affected
Defect Degree Score	**1**	**2**	**3**	**4**	**5**
Potholes	Surface pock marked, holes are <50 mm diameter	Potholes 50–100 mm in diameter	Potholes 100–400 mm in diameter and influence riding quality	Potholes 400–800 mm in diameter, influence riding quality, and obviously avoided by most vehicles	Potholes >800 mm in diameter, influence riding quality, and require speed reduction or total avoidance
Corrugations	Slight corrugation, difficult to feel in light vehicle	Corrugation present and noticeable in light vehicle	Corrugation very visible and reduces riding quality noticeably	Corrugation noticeable in haul truck and causing driver to reduce speed	Corrugation noticeable in haul truck and causing driver to reduce speed significantly
Rutting	Difficult to discern unaided, <20 mm	Just discernable with eye, 20–50 mm	Discernable, 50–80 mm	Obvious from moving vehicle, >80 mm	Severe; affects direction stability of vehicle
Loose material	Very little loose material on road, <5 mm depth	Small amount of loose material on road to a depth of 5–10 mm	Loose material present on road to a depth of 10–20 mm	Significant loose material on road to a depth of 20–40 mm	Loose material, depth >40 mm
Stoniness—fixed in wearing course	Some protruding stones, but barely felt or heard when traveling in light vehicle	Protruding stones felt and heard in light vehicle	Protruding stones influence riding quality in light vehicle but still acceptable	Protruding stones occasionally require evasive action of light vehicle	Protruding stones require evasive action of haul truck

Mine Haul Road Rolling Resistance Visual Evaluation

Date		Evaluator	
Road		Vehicle Speed km/h (V)	
Chainage		Traffic kt/d	

Defect	RDS (Rolling Resistance)			
	Degree (1–5)		Extent (1–5)	Defect Score
Potholes		×		=
Corrugations		×		=
Rutting		×		=
Loose Material		×		=
Stoniness–Fixed		×		=

Total Roughness Defect Score (RDS)

Estimated Rolling Resistance (%)
Refer to Graph for Rolling Resistance Percentages

Figure 10.6-18 Visual evaluation of haul road rolling resistance

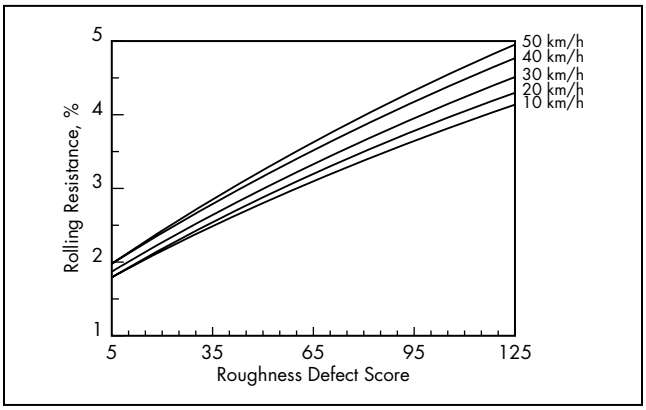


Figure 10.6-19 Rolling resistance estimation from RDS (visual) evaluations

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CHAPTER 10.7

Surface Ore Movement, Storage, and Recovery Systems

Sergio Zamorano

INTRODUCTION

After the mineral products are extracted from an underground or surface operation, they need to be moved, stored, and recovered before being treated further or sold to external clients.

Traditional and small operations are based on small-scale discontinuous movement of materials that are labor intensive. The trend of the industry is toward larger continuous systems with increased automation levels. While in the first part of the 20th century a 1,000-t/h (metric tons per hour; 1,100-stph [short tons per hour]) operation could have been classified as large, volumes of 10,000 t/h (11,023 stph) are quite common today. Some continuous operations reach 40,000 t/h (45,000 stph), and several mining operations reach levels close to 1 Mt/d (million metric tons per day [1.1 million stpd]) between ore and overburden removal. In most open-pit mines, the volumes of overburden and waste rock required to be moved are much larger than the ore volume. Ratios between waste and ore of 2 to 6 are quite common; they can get as high as 30 before an underground operation becomes a better option. The old paradigm of abundant and cheap labor in third-world countries is disappearing: Globalization brings common standards for environmental and safety requirements at the same time that economic development reduces the income gap between First and Third World countries. Thus, regardless of its location in the world, any operation needs higher productivity and continuous improvements to remain competitive.

This chapter will discuss the most common methods to move, store, and recover ore used in mineral operations, including newer developments and developments foreseen for the near future. The intent is to give a general overview in the limited space available, keeping in mind that a comprehensive and in-depth discussion goes beyond the intent of this handbook.

ORE MOVEMENT

This section reviews transport systems in which the payload is motionless in relation to the transporting media (truck haulage; rail, marine, and multimodal transportation); and belt conveyors and stationary systems in which the material is in motion during transport.

Truck Haulage

The loading and transportation of mined products by truck within the mine and to the processing plant are discussed in detail in other sections of this handbook that deal with surface mining and underground mining. Mine operations use underground or off-road-type trucks, for which design and size evolve continuously.

Large off-road trucks have reached capacities of up to 364 t (400 st) with gross machine weights around 625 t (688 st; Caterpillar, n.d.). In the past, large off-road trucks were mostly diesel-electric, with direct-current (DC) wheel drives; however, the newer, larger trucks have mechanical transmissions or alternate-current (AC) wheel drives. Recent developments include autonomous trucks for both underground and surface operations (Komatsu 2008) that are aimed at increasing reliability and safety while reducing labor requirements.

Trolley-assisted trucks have been in operation in southern Africa since the 1980s. A pantograph is used to connect an overhead trolley line, which feeds energy directly to the truck's electric wheel motors (Figure 10.7-1) while the diesel engine runs idle. Used on inclined ramps, the system allows savings in fuel consumption while the truck can climb faster and thus operate with higher efficiency (Alvarado 2009). The advantages of a trolley line are linked to the availability of cheap electrical energy and to high fuel costs. Initial designs had DC electrical feed; the advent of AC trucks has allowed the use of simpler and lower-cost AC distribution lines.

Off-road trucks are intended not for long-distance haulage, but for operations with complex deposits and multiple loading points. It is not unusual to find them operating on routes several kilometers long.

Long hauls are also common for overburden handling, where more effective transportation methods such as conveyors require the sizing of the mined material, as opposed to the trucks than can handle run-of-mine material without any further size reduction beyond blasting. As pits grow and become deeper, overburden has to be lifted more and more and deposited at ever-increasing distances as waste dumps are filled.

Transportation of mined products to market by road is less significant on a volume and distance basis than other

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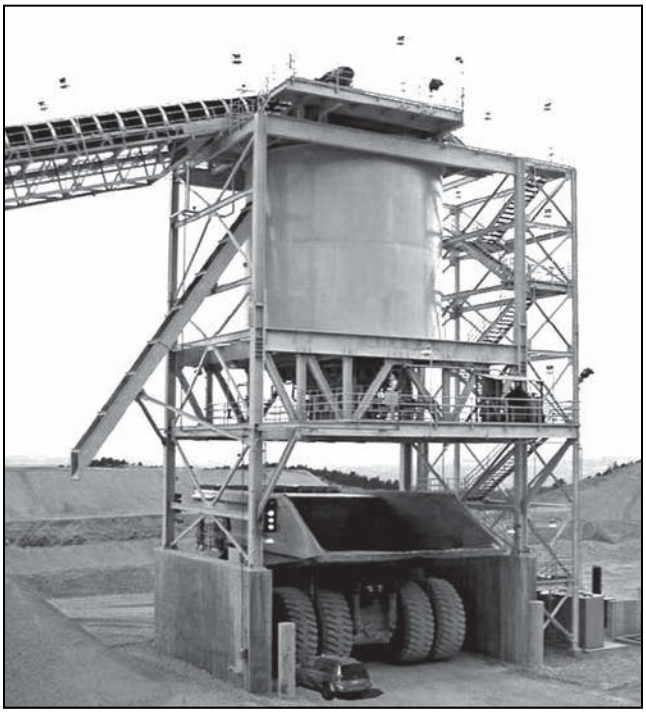
Courtesy of Exxaro Resources Ltd.
Figure 10.7-1 Trolley-assisted truck



Courtesy of Clement Industries.
Figure 10.7-2 Side-dump trailer

methods (e.g., rail or marine transportation). However, trucking is often involved at some stage of the transport chain between producer and final customer. The flexibility of trucks is diminished by the high cost per metric ton per kilometer, or per ton per mile, over longer distances.

Trucks designed for the movement of bulk minerals on standard roads are divided into three general classifications. The first and most common is the end-dump design. A variation of the standard design is the frameless trailer, in which the dump body is self-supporting. The objective with this custom design is to gain more payload while staying within the load limitations imposed by traffic authorities. Another alternative to increase payload is the use of aluminum construction in the trailer, normally employing extruded sections. The second classification is that of bottom-dump design. A special unloading facility is required, normally a long grated hopper over feeders and conveyors; the truck drives over the hopper to discharge the load. This design offers a fast unloading cycle, but at the expense of custom-designed truck bodies and significant civil works associated with underground structures. The third classification is the side-dump design (Figure 10.7-2). The advantage here is a faster unloading



Courtesy of FLSmidth Spokane, Inc.
Figure 10.7-3 Truck load-out station

cycle, although larger receiving hoppers are required than for the end-dump design.

The typical bulk-handling truck is composed of a tractor unit and a trailer or semitrailer attached to it. On long distances it is quite common to use longer combination vehicles that allow trucks to haul more on a single trip. They usually have a tractor unit that pulls two or three trailers. This type of truck is allowed in some states in the United States, mostly in the western part of it. They are also used in Canada, Argentina, and other countries, mostly in remote areas and where access roads are relatively flat. Australia allows the use of even longer vehicles; some Australian states allow the use of triple and AB-quad road trains that are up to 53 m (174 ft) long and weigh up to 170 t (187 st). The use of powered trailers allows for even larger combinations, with long-distance haulage systems reaching payloads up to 350 t (386 st).

The transport of small-size and dusty bulk materials is normally done using pneumatic trailers. Air pressure is used to discharge and transport the material through hose or piping directly into the storage container. The truck body is divided internally, so the bulk cargo can be gravity fed to the discharge ports. Airflow allows the material to move through the discharge pipe to its destination. Typical trailer capacity is 28 m^3 (1,000 ft^3). These trailers normally are used to transport fine and free-flowing material (e.g., cement and limestone) used as a reactive in mineral concentrators.

Dump trucks are loaded most of the time using front-end loaders or similar equipment. However, on high-capacity applications, loading bins are used at times (Figure 10.7-3), enabling 300-t (331-ton) trucks to be loaded in 60 seconds. Crushed material is loaded onto trucks to be placed on a heap and leached.

Rail Transportation

In smaller operations, common carriers generally handle rail transport, which is therefore outside the scope of the mining

company. However, the mine retains control of loading and unloading the trains and can influence the decision of the type of trucks to be used.

In a small to medium-size operation, partial train load can be used. The normal procedure is for the common carrier to drop off the required empty cars. Since the loading time is not a critical factor, the plant operator moves the empty cars to position them under a loading point with a car puller or a shifting locomotive. Most carriers charge per car, not per ton, of material transported; however, they do penalize the plant if the cars are overloaded. Therefore, controlling the loading accurately makes economic sense as it minimizes transport cost. This is accomplished by using weigh belt conveyors or track scales.

For throughputs ranging up to several million metric tons per year, it is normal to use unit trains. A unit train is a freight train composed of cars carrying a single type of commodity to the same destination, running nonstop between the loading and unloading points. This results in reduced transit time and lower transport cost. American railway companies introduced the unit train concept in the 1950s for the transportation of coal from mines to power plants. By the late 20th century, about 50% of the coal shipped in the United States was carried by these types of trains, typically in convoys of 100 cars. Other bulk cargoes transported by unit trains are cement and minerals such as iron ore. The largest train ever run in the world was a unit train carrying iron ore in Western Australia. The train consisted of 8 locomotives and 682 loaded wagons, carrying more than 80,000 t (88,185 st) of ore over a distance of 275 km (171 mi) to the port. The train was more than 7 km (4 mi) long. Today, trains with more than 200 cars and over 25,000-t (27,558-st) capacity transport iron ore daily in northwestern Australia, with more than one train per hour per line. Some of these trains are fully autonomous. Trains of comparable size are operated for iron ore transport in South Africa and Brazil as well as other countries. It is quite common in larger operations to have a dedicated railway and port owned and operated by the mine owner or a consortium of mine owners that handle the same commodity.

The need to load larger trains in an efficient and accurate way led to the development of specialized loading stations (Fletcher 1997). To load a train efficiently, it is necessary to ensure that the full unit train can be loaded—that is, that sufficient storage is required. A reliable, accurate, and fast loading system and a device to move the train through the load station are also required. The most common type of high-capacity train-loading plant is the flood loading system, in which the height of the material in the wagon is controlled by a profile chute while the discharge gate is controlled by photocells that monitor the position of the wagon. Two in-motion scales are installed, one before the loading system and one after, so the amount of material loaded in the wagon can be measured. Material loaded into the wagon is measured after the operation. The accuracy of the amount loaded depends on the dimensional and suspension consistency of the wagons.

The need to maximize the loading of each wagon without exceeding the maximum gross weight allowed on the tracks led to the development of the flask loading system in the 1970s in South Africa. In this system, the material is loaded onto a weigh flask first. When the required load is achieved, the gate from the storage bin is closed and the gate to the profile chute is opened, and the wagon is loaded with a predetermined load. Train load can be controlled accurately up to capacities of 8,000 t/h (8,818 stph).

In some installations, a silo with a capacity comparable to the train capacity is used. Lower-capacity systems fill the silo before the train arrives so that train-loading time is minimized. Typical silos have a capacity up to 12,000 t (13,228 st). Another option would be to use a smaller silo linked to a large storage system, such as a stockpile. In this instance, a larger loading system (operating at a capacity equal to the load-out rate) is required. This option is preferred when loading large unit trains having more than 100 wagons.

Losses of material due to wind can be reduced by applying crusting agents to the mineral surface. On fine and environmentally sensitive materials, such as mineral concentrates, railcars are covered.

When high tonnage rates are not required and capital expenditure needs to be minimized, the use of bottom-dump hopper cars is the logical choice. Normally, car movers are used to position the car over a hopper, and the bottom gates are opened manually to discharge the contents to the hopper. Feeders and conveyors or elevators are used to take the material to its final destination.

One way to obtain higher discharge rates is to use a trestle structure over a storage area. The loaded cars are tripped open by a device alongside the truck and are emptied while in motion. The disadvantages of this method are the additional capital and maintenance costs of the cars with special quick-opening doors and potential environmental dust problems when the material is dumped from the top of the trestle to the ground. At an installation recently built in Germany, the railcars are discharged over a 6.4-m- (21-ft-) wide belt that transfers the material at a rate of 17,000 t/h (18,739 stph). Of course, in this case, a higher discharge rate justifies the high capital and operational costs.

Extreme cold weather can cause the material to freeze if the cars are left to sit idle or the traveling time is long. This can cause serious operating problems when attempting to unload the cars. If this is a normal situation, a car-thawing system is required. Even then, large frozen lumps might form and will have to be broken up before going into the receiving system.

It can also be hard to unload fine materials after a long trip. This is particularly true with some mineral concentrates having a high moisture content that are loaded into the trains. Vibration and heat during the trip causes the material to consolidate, making car unloading very difficult. On relatively short distances and low throughputs, round "pots" are placed over flat cars, and the cars are overturned with the help of a crane during unloading. Vibrators or "beaters" are used to loosen up material stuck to the pot. When transporting mineral concentrates over long distances (several hundred kilometers), it becomes harder to unload the cars. One method that is used successfully is to place a backhoe excavator on a platform over the cars and excavate the material out. Throughputs up to 700 t/h (772 stph) are achievable in this fashion.

Unit trains are normally discharged using a rotary car dumper to achieve a high unloading rate. Each car is turned upside down and the contents discharged into a large hopper that feeds a conveying system leading to the storage location. Traditionally, it was necessary to uncouple each car so that it entered the dumper unattached. Newer, higher-capacity systems use rotary couplers that permit dumping without separating the cars. Also, high-capacity systems have twin dumpers that allow the discharge of two cars at the same time.

A high unloading rate is linked to the ability to move the cars in and out of the dumper quickly. To this effect it

is common to use a shifting locomotive with local or remote control. These units can handle a group of cars or an entire unit train. An alternative is the employment of a remotely controlled train positioner that can handle a full unit train. The device uses a pickup arm or arms to push the train one or two car lengths (depending on whether it is a single or twin dumper arrangement). While the cars are unloaded, the arm moves to the next car or cars. This type of operation requires the use of rotary couplers to prevent uncoupling of the cars.

Environmental regulations now require effective dust control by means of dust suppression or dust extraction systems. In most cases today, the installation for car dumping is entirely enclosed.

Marine Transportation

The vast majority of long-distance transportation of bulk minerals is done using marine transportation, coupled in most cases with some degree of overland transport. The capital and operational costs of loading and unloading equipment are outweighed by the low cost of water transport. The continuous growth of international trade has resulted in the transport of increasingly large amounts of bulk minerals down rivers and across the oceans. The largest transport volumes are associated with coal (steam and metallurgical) and iron ore, each representing about 800 Mt (882 million st) transported by sea every year (VK 2005; WSA 2009).

Inland marine transportation allows large amounts of low-cost minerals to be moved to final destinations at a cost per metric ton per kilometer (or per ton per mile) that is among the lowest for this process. In the United States, the Mississippi River system is used to transport about 50 Mt (55 million st) per year of coal—approximately 10% of the coal transported within the country. Barges are also used extensively in Europe for the inland transport of iron ore and coal. Barges can be self-propelled or towed by tugboats, or they can be pushed by towboats in a series of several barges at the same time. For example, a 10,000-hp towboat can push up to 30 barges down the open-river section of the Mississippi River to the Gulf Coast. Barge dimensions are limited by lock sizes in the area in which they operate. A typical barge is about 60 m (197 ft) long and 10 m (33 ft) wide with a loading capacity of approximately 1,350 t (1,488 st).

Ship Loading

Typical barge-loading systems include a fixed loading conveyor that discharges directly onto the barge. The barge is moved as necessary by a barge puller that is basically a system of winches and cables.

Various methods are employed to unload barges. The traditional method is the clamshell-type unloader. Other designs include continuous unloading systems with bucket-chain unloaders or bucket-wheel unloaders. In coal and iron ore applications, these systems can achieve capacities up to 5,000 t/h (5,511 stph).

For fine materials, pneumatic unloaders are used. They can be stationary, mobile within the dock area, barge mounted, or fully mobile (mounted on a truck). Materials such as cement are normally handled using pressure conveying; coarse and abrasive materials such as lime, alumina, and petroleum coke are handled using vacuum systems.

In locations without proper infrastructure, load from barges is transferred directly to ocean vessels. In relatively low-tonnage operations, this is done using the ship's cranes. Larger operations use custom-made ships to transfer the load using clamshell or continuous systems.

Ocean transport of minerals involves four main types of ships, namely Handy, Handymax, Panamax, and Capesize. Handy and Handymax sizes are the most common, with deadweight (dwt) ranging from 10,000 to 39,000 t (11,023 to 42,990 st) for the Handy type and between 40,000 and 59,000 t (44,092 and 65,036 st) for the Handymax type. They are used for multiple cargoes, either broken or bulk. In bulk materials they are limited to either short-distance transportation or the transport of low-volume, high-value cargoes such as copper concentrate. Panamax-size vessels are the largest ships that can go through the Panama Canal, and their deadweight fluctuates between 60,000 and 80,000 t (66,139 and 88,185 st). Their maximum draft is 14 m (45 ft), and they have less than 36 m (118 ft) beam. They are the most common type of vessel used for moderate and large volumes of bulk solids transported over long distances. Capesize vessels are not limited in size other than by port capacities. The most common sizes vary between 130,000 and 180,000 dwt. Some of the largest bulk carriers exceed 300,000 dwt, being as long as 340 m (1,115 ft) with a draft of 28 m (92 ft) and a beam of 60 m (197 ft).

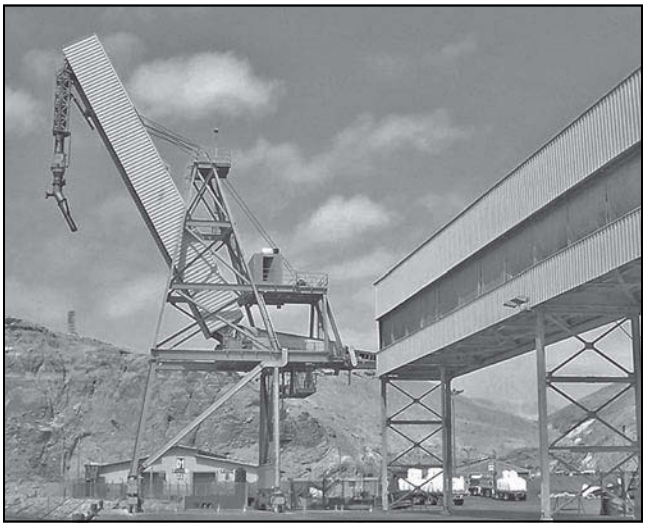
As volumes transported grow steadily, the tendency is toward using larger ships. In this way, both operating costs and carbon emissions are reduced.

There are several ways of loading bulk materials onto a ship. On low-capacity operations, fixed single-point loaders are used, and the ship is moved to load the different hatches. This is by far the simplest and cheapest way to load a ship. Telescopic heads, tilting telescopic chutes, and/or loading spouts are used to distribute the material inside the ship hull. On berths used for multiple purposes, mobile conveyors and loaders are also used. When loading is completed, the conveyors and loaders are removed to leave the area open for other uses.

As transported volumes increase, so do ship size and the number of hatches to be loaded. In these cases, the low efficiency of a single loading point doesn't make economic sense since the loading operation has to be stopped frequently to reposition the ship. Moving a large ship generally involves a tugboat or a large crew to operate the ship's winches. It is possible to reduce ship movement and loading interruptions by using a radial stacker that can rotate from the hatch.

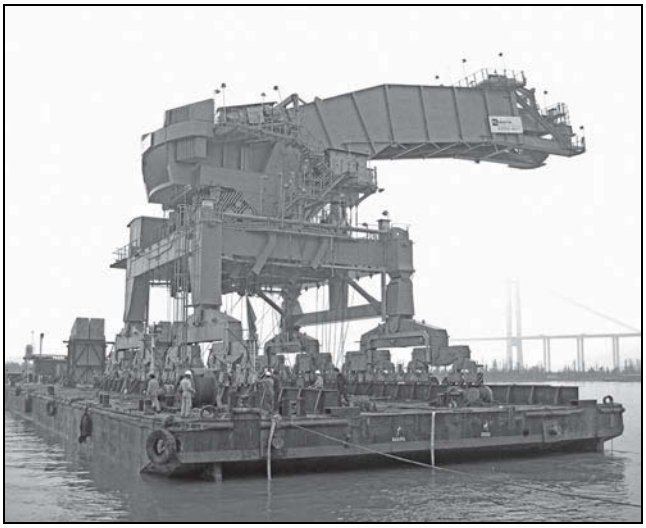
A traveling ship loader moves along a wharf, normally on rails, so it can move from one hatch to another, making it unnecessary to move the ship. The ship loader is normally fed by a conveyor parallel to the wharf and a tripper car. In most installations, the feed conveyor is installed inside an enclosed gallery (Figure 10.7-4). In this case, the boom conveyor can lift its boom, so it can move easily from hatch to hatch. This type of loader is relatively inexpensive; most operational costs are related to the civil cost of the continuous wharf and the structure of the elevated gallery. The loader depicted has a tilting telescopic chute and a rotating loading spout that allow for better trimming of the load in the ship.

Another type of traveling ship loader has a slewing boom. In some cases the boom is of fixed length; in others it is telescopic, which allows for an optimum distribution of the load on the ship. This factor is quite relevant with low-density materials such as coal, for which trimming the material is more complex, than with high-density materials such as copper concentrate and iron ore. In the case of a slewing boom, the machine needs to have an intermediate feed conveyor



Courtesy of FLSmidth Spokane, Inc.

Figure 10.7-4 Traveling and luffing ship loader



Courtesy of FLSmidth Spokane, Inc.

Figure 10.7-5 Traveling, slewing, luffing, and telescopic ship loader being transported to operating location

that connects the feed and the boom conveyors. Another variation uses a ground-level conveyor with a large tripper car that feeds a luffing, slewing, and telescopic loader that is parallel to the ship. This type of unit can load ships on either side of a dedicated wharf. Figure 10.7-5 shows a 14,000-t/h (15,432-stph) ship loader being transported fully assembled to its final destination. It is a common practice to assemble large port equipment on a barge while traveling and then move the equipment directly into its working position.

When the port is dedicated exclusively to the loading of bulk materials, the cost of the wharf can be minimized by using mooring dolphins to support the ship-loading system. A design that allows the loading of multiple hatches with a minimal civil cost is the so-called quadrant ship loader. It is formed by a fixed-span bridge that turns around a fixed point on the land side while traveling over a curved runway on the water side. A shuttle boom assembly travels over the bridge. The radial movement of the boom and the slewing bridge combine to provide access to the ship hatches. This type of loader can load ships from Handy to Panamax sizes without requiring movement of the ship during loading operations. The cost saving in civil structures is offset by the higher cost of the machine.

A variation of the quadrant ship loader is the linear loader, in which the runway is straight and the bridge moves back and forth over an inverted carriage mounted at the pivot point, thereby changing its span. A further development is the dual linear ship loader, in which a slew bearing and an inverted carriage are mounted on the carriage traveling on a runway parallel to the ship. The loading point is mounted over a fixed pivot and a shuttle boom is mounted over the bridge.

For larger ships (e.g., Capesize vessels), dual arrangements of quadrant or linear ship loaders are sometimes used (Dibben 2006).

Ship Unloading

Unloading ships is a significant part of the transport cost of minerals. The most common system used to unload ships is the clamshell bucket. In Handy- and Panamax-type vessels, it is quite common to use the ship's own deck crane to unload the material at smaller rates up to 300 t/h (331 stph). This type of ship is referred to as a geared vessel.

Dock-mounted clamshell unloaders can be classified into two main categories: crane and portal. Normally the unloaders are used in a fixed position and operate at capacities up to 700 t/h (772 stph). Some units are capable of moving along the wharf, but it is more common to use two cranes simultaneously to unload a Handy or Panamax ship. For larger capacities and ships (e.g., Capesize vessels with capacities up to 2,500–3,000 t/h [2,756–3,307 stph]), portal clamshell unloaders are frequently used. Again, using two unloaders simultaneously is quite common. The efficiency—or the actual average unloading rate achieved from start to finish divided by the free digging rate—is normally 50% or less. The ship's hold is cleaned by small earthmoving equipment that is lowered into the hold area. The need for extensive cleaning associated with clamshells is larger for smaller ships that have smaller holds, reducing the efficiency accordingly.

The need to improve efficiency and reduce ship turnaround times has prompted the development of continuous ship unloaders. A relatively lightweight continuous-unloading solution is the vertical screw system, which draws the material from the ship's hold and transfers it to a transfer conveyor belt. Fine materials can be handled at relatively low capacities in this fashion.

Higher capacities can be achieved using a bucket-chain continuous unloader. The lower demurrage cost and higher utilization of this type of unloader balance the higher cost when compared with the clamshell unloader. There are several variations to this design that include steel buckets on steel chains and steel buckets on rubber belting. Rubber pocket belts or sandwich belts loaded by bucket wheels are also used. Once the material is elevated above the ship's deck it is transferred to a belt conveyor, which transfers the material to a dock conveyor and port storage.

When fine and dusty materials are involved, pneumatic unloaders, either stationary or mobile, are normally used. Gantry-type unloaders can unload vessels up to 50,000 dwt.

Great Lakes, coastal, and Gulf of Mexico transport comprise a significant part of the mineral transportation market in North America. The basic differences between the ships used

in these environments and in oceangoing applications are size and draft. Although some self-unloading ships are used in ocean bulk transport, the majority of these vessels are found in the Great Lakes and the Gulf of Mexico. However, they are also used in Northern Europe and East Asia (OSC 2002) for transport distances up to 5,000 km (3,107 mi).

In most cases, the ship cargo holds have a W-shaped bottom with several hydraulically operated gates that discharge the material onto a belt-conveyor arrangement. The material is carried to deck level using sandwich-type belts or rubber pocket belts. A boom conveyor transfers the material to a hopper and conveying system on the dock or, in some cases, directly onto the ground. High unloading rates can be achieved with free-flowing materials.

The relatively short transport time and shorter unloading time offset the machinery cost and lost payload capacity on lake and coastal applications. Short travel time also reduces the probability of material consolidation and arching caused by long marine trips. Another alternative is the use of hybrid vessels that add a system of hoppers and conveyors to a conventionally outfitted ship. Most of the advantages of the self-unloading ship are obtained for a limited cost, but discharge rates are limited because of crane capacity.

Multimodal Transportation

Conventional transportation involves the use of a separate contractor or operator for each type of transport (i.e., trucks, railway, and marine). Multimodal transportation covers "door-to-door" delivery by a single transport operator, even when transport through several countries is involved. This method of delivery was developed during the container revolution of the 1960s and 1970s. Today, most multimodal cargo is containerized; however, it is different than typical container transport.

Most minerals are transported in bulk rather than in packaged or break-bulk shipments. However, some sophisticated or friable bulk minerals can be loaded in standardized marine containers or packed in flexible intermediate bulk containers (FIBCs) also known as Big Bags, Bulk Bags, or Super Sacks. The FIBCs have capacities from 500 kg (1,102 lb) up to 2 t (2.2 st), with volumes up to 3 m^3 (106 ft^3). FIBCs are normally made out of polypropylene or canvas fabric.

Regardless of transport method—container, bulk, or FIBC—it can be handled in a multimodal fashion.

Belt Conveyors

Since their introduction in the first half of the 20th century, belt conveyors, due to their simplicity, universality, and reliability, have become indispensable in the layout of facilities that store or process bulk solids. Belt conveyors allow optimal use of floor space and maximum flexibility in the design of the material flow scheme. While capital cost comparisons between conveyor and truck haulage are dependent on the application, operating costs of a conveyor system are far lower. In general, the larger the throughput of the system, the greater the operational savings when compared with an equivalent truck operation. Current technology allows single-flight belts with lengths up to 20 to 30 km (12.4 to 18.6 mi); the longest system in use is 100 km (62 mi) long and has been in operation since the 1970s.

Although most conveyors use 35-degree troughing idlers and three equal rolls, different troughing angles and idler configurations are employed in some operations. Speeds vary from less than 1 m/s (3.3 ft/s) on some short conveyors up to 8.5 m/s (28 ft/s) on selected overland conveyor applications, and even up to 15 m/s (49 ft/s) on spreading conveyors that handle up to 40,000 t/h (44,092 stph). Maximum belt width is limited to 3.2 m (10.5 ft) using conventional manufacturing techniques, although belts can be spliced along their axes to produce even wider belts. Belt width is normally determined by belt capacity, operating speed, and maximum lump size.

Conventional wisdom says that short conveyors handling large lumps are operated at low speed, whereas longer conveyors can be operated faster without compromising belt life and system availability. Belt width must be more than three to five times the maximum lump size of material to be handled. The higher the proportion of large lumps, the closer to five the factor should be. Lump size is defined as the maximum dimension of one side of a piece of material. It should not be confused with crusher opening or dimension of the screen mesh that the material passes through. On slab-like materials, the longest dimension can be up to five times the other dimensions or the opening that the lump has passed through.

The advent of bulk flow simulation techniques such as DEM (discrete element modeling) has made it possible to design transfers that reduce belt wear and tear from the beginning of a project and, therefore, limit the trial and error used in the past. That said, it is important to note that properties of bulk solids can vary widely, so proper testing is required. Even then, modeling can have limitations. Poor transfer design can lead to premature belt wear, poor tracking, material degradation, and dust generation.

There are several published guides for conveyor design. The Conveyor Equipment Manufacturers Association (CEMA) handbook is widely used both in North America and abroad to design conveyors (CEMA 2007). The CEMA book contains recommended practices from the North American conveyor equipment manufacturers and considers a simple calculation method for regular installations that has changed very little since the 1960s. The sixth edition, published in 2007, incorporated a universal calculation method in line with modern research and experience in large systems. Other publications such as DIN (Deutsche Institut für Normung) standards contain regulations about belt rating, component testing, and other aspects of conveyor design.

In long conveying systems, the aim is to reduce the number of transfers to the minimum possible; of course, this is limited by available technology and economic factors. With longer conveyors, vertical and horizontal curves are used to follow the terrain and bypass natural obstacles along the desired route (Figure 10.7-6). Currently, most long conveyors use steel-cord belts. In shorter conveyors, polyester/nylon fabrics (known as EP fabric) have replaced nylon/nylon and cotton fabrics used in the past.

Downhill conveyor applications offer the advantage of regeneration. The potential energy of the material is converted to usable energy as the conveyor drives generate electricity during operation. In many cases, the value of the energy generated is greater than the operating cost of the conveyor, resulting in an operating profit.

An alternative technology to conventional conveyor belts is the cable belt, which was developed more or less simultaneously with steel-cord belting during the 1970s. In this type of system, the belt is supported between two steel cables that transfer the driving forces and carry the belt and material. However, a limited number of installations use cable belts, whereas literally thousands use steel-cord belts. Most likely



Courtesy of FLSmidth Spokane, Inc.

Figure 10.7-6 Overland conveyor with vertical and horizontal curves



Courtesy of FLSmidth Spokane, Inc.

Figure 10.7-7 Pipe conveyor with carry and return transport installed underground

the reason for this is that cable belts are available through one source only, and their capital and operating costs appear higher than costs for conventional belts (Maton 2009).

Another variation of the conventional conveyor design is the pipe conveyor, in which the belt is forced into a pipe shape and encloses the material completely. Since the material is enclosed, spillage is eliminated. Pipe conveyors, therefore, have a lower environmental impact than conventional systems. The belt configuration allows for tighter vertical and horizontal curves in the design as compared with conventional belts; this reduces the civil works necessary with overland conveyors and makes the implementation of new conveyors easier in existing plants. Also, since the material is contained by the belt, pipe conveyors can operate on inclines approximately 50% greater than trough belt conveyors. In addition, the return side can easily be adapted to transport material simultaneously in the opposite direction of the main feed.

Another advantage of the pipe conveyor is its reduced space requirements as compared with a conventional belt. A dramatic example is given in Figure 10.7-7, which shows part of an 8.5-km (5.3-mi) pipe conveyor that transports cement on the carry side and clinker and coal on the return. Six kilometers (3.7 mi) of this conveyor is located in a tunnel beneath the city. A conventional conveyor would have required a wider tunnel and would have had serious difficulty following the complicated conveyor path, which involved dozens of tight vertical and horizontal curves.

Conveyors can negotiate steeper inclines than haul trucks, so the required ramps to go up or downhill are shorter. A pipe conveyor requires an even shorter distance to negotiate a slope since they can operate on steeper ramps. High, steep conveyors, such as the flexible sidewall belts, can go up to 90°. Flexible wall conveyors are quite popular and handle mostly free-flowing, low-abrasion materials. They are used sometimes on difficult applications, mainly when layout constraints do not allow the use of more conventional alternatives; however, their maintenance cost tends to be quite high. System cost is directly related to capacity in an exponential sense, with the belt being the driving factor. Most of these systems that are in operation have capacities less than 1,000 t/h (1,102 stph).

Another type of steep-angle system is the sandwich belt. In this design, a second belt rides above the carry belt in the steep section, holding the material in place with the assistance of inverted pressing idlers.

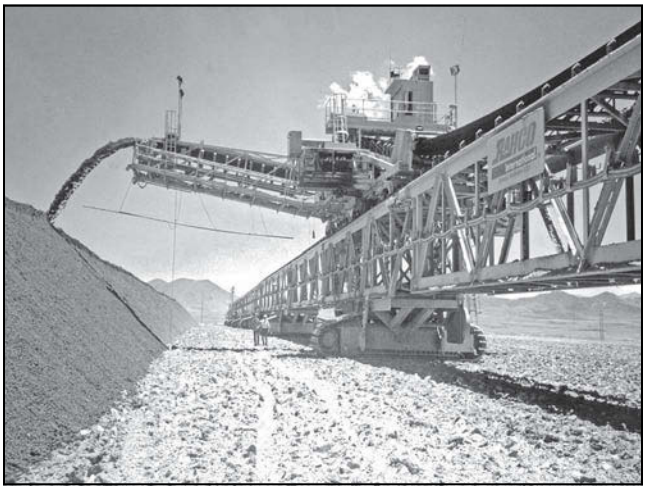
In a system that combines a flexible wall belt with cableway technology, a belt with sidewalls and integrated sets of wheels travels over carrier cables guided over supports. Long, unsupported spans are feasible, and environmental impact is reduced since civil works are minimal. This type of conveyor is appropriate in irregular terrain with great differences in elevation along the conveyor.

Traveling trippers are used to discharge material from any point along a conveyor belt. They offer great flexibility for storage layouts. Trippers can be mounted on tires, crawlers, or rails at ground level, and they can feed a traveling stacker to form a wedge-shaped longitudinal pile. Trippers can also be mounted on elevated structures and discharge the material to a pile below.

Shiftable conveyors can be used to follow an excavator or to feed a stacker that moves along the length of the conveyor. When the excavator or stacker needs to move to a new position, the shiftable belt is relocated. The individual conveyor tables are not connected to one another and are mounted on skids. Rails connect the skids longitudinally. When the time comes to shift the conveyor, belt tension is slackened and a bulldozer with a roller hook travels along the conveyor, pulling it sideways. The belt tables move snakelike in the direction of the pull 50–100 cm (1.6–3.3 ft) with each completed pass. Passes are repeated until the conveyor is shifted to its new location.

The mobile conveyor is also used in combination with excavators or stackers, or with its own hopper or cross conveyor, depending on whether the machine is used in a stacking or excavating/reclaiming operation. The conveyor is composed of a series of steel truss sections mounted on crawlers. A single conveyor belt moves along the top chord of the truss sections. The whole system moves as a unit because of a control system that monitors the movement and alignment of each section (Figure 10.7-8).

It is important that belt conveyors are designed so that material is not blown from the belt during operation. Covers



Courtesy of FLSmidth Spokane, Inc.
Figure 10.7-8 Mobile stacking conveyor

eliminate this concern and protect both the material and the belt and idlers from the elements. Vertical sideboards are often sufficient for wind problems; complete full covers, usually made from corrugated metal, enclose the entire troughed belt and components. Variations include three-quarter and doghouse covers that leave one side open so the belt can be inspected without having to remove the cover. Fully enclosed galleries are used when handling environmentally sensitive materials, a variation being large culvert-style steel tubes in which the conveyor is located inside the tube. Functionally, this is quite similar to the pipe conveyor discussed previously. Choosing one over the other depends on operational and cost factors.

Pipe Transport

Unlike the systems discussed to this point, pipe transport systems are stationary and the material is in motion. In truck, rail, marine, and conveying systems, the hardware is in motion and the payload is motionless relative to the transporting media.

Conveying solids in a pipe involves transporting solids using the drag resistance and pressure of a flowing medium, such as gas or liquid, to move the material. Pneumatic and hydraulic systems must be considered separately, as the characteristics of the carrier fluids are completely different. Drag resistance is proportional to the fluid density, the cross-sectional area of the solid particle, the square of the relative flow speed of the solid, and the Reynolds number.

Pneumatic conveying systems use a gas as the carrier medium. The low density of the gas (in the range of 1 kg/m^3 [0.062 lb/ft^3]) makes buoyancy nonexistent for practical purposes. The difference in density between solids and media must be compensated with high speeds to obtain the required carrying force, which results in high power consumption. High speeds create wear and attrition (degradation), so speed is limited in this regard. Since gases are compressible, the friction loss causes pressure loss. In other words, the gas expands along the conveying length, and unless the diameter is increased along the line, the speed, energy consumption, and wear increase. Therefore, pneumatic transport is generally restricted to distances less than 4 km (2.5 mi) and is used mainly in plant applications for materials that are dry, free flowing, and relatively small in particle size.

When the ratio between the mass of the transported solids and the mass of the gas is lower than 30, the system is said to be a dilute-phase system. In this case, the transport velocity is high, the mix is homogeneous, and the risk of blockage is minimal. When the mass ratio is much higher than 30, the system is called a dense-phase system. Plugs of material are transported at low speed but with high pressure gradient.

Calculating the design parameters of pneumatic systems is dependent on empirical information and varies depending on what materials are to be transported. Although published information about most common materials is available, it is recommended that appropriate laboratory tests be performed when designing a new system.

Hydraulic conveying, as its name implies, uses a liquid such as water as the carrier medium. The density of the mixture of liquid and solids results in significant buoyancy for the solids particles. Transport can be achieved at relatively low speeds. Far less energy is required for hydraulic conveying than for pneumatic conveying and other comparable transport methods, such as trucks. As the liquid is nearly incompressible, long transport distances—up to hundreds of kilometers or miles—are achievable. Typically, depending on the material handled, solids concentrations vary between 40% and 65% by weight.

In systems in which the solids particles are small enough (<30 μm) and can therefore be maintained in suspension at very low speeds, one can consider homogeneous conveying. The homogeneous suspension appears as a liquid with a higher density and viscosity than the carrier fluid and, in this case, the system can be designed in a similar fashion as for a pure liquid system.

If turbulence is required to keep the particles in suspension, the solids are no longer distributed homogeneously. However, up to a certain point, normally defined as a Reynolds number value of 2 (Weber 1981), it can still be considered pseudohomogeneous conveying, and conventional fluid formulas can be used for calculations. Also, a mixture with less than 20% variation in density between the pipe axis and the vicinity of the pipe wall can be considered homogeneous for practical purposes (Shou 1990).

With Reynolds numbers above 2, conveying takes place in heterogeneous suspensions in which the velocity of the solids and the liquid are different. The method used to design these types of systems was first proposed by Durand (1953) and is still used today with improvements and variations. The use of empirical data is fundamental to the design of systems having heterogeneous suspensions.

Increasing the density of the mixture will result in lower liquid requirements, but the flow properties of high-density mixtures are non-Newtonian, with a nonlinear relationship between shear forces and shear rate. Once again, empirical determination of the rheological behavior of the mixture is required before design is carried out.

Slurry transportation lines are used to transport diverse materials such as coal, iron ore and copper concentrate, bauxite, and phosphate mineral. They have a high capital cost, but operating costs are low in most cases. Slurry lines are used successfully in South America to transport copper concentrate from the Andes to sea level using the natural gradient and dissipation stations instead of pumps. The energy requirement is nil. They also offer the advantage of having a smaller footprint and lower environmental impact than other transport methods. A buried pipeline has almost zero environmental impact after it is built.

Another application of hydraulic transport is the disposal of tailings from concentrating plants. Because of the low grades involved in most concentrators, the volumes of tailings are several times larger than those of concentrates. Tailings lines involve large pumping systems and pipes up to 1,350 mm (53 in.) in diameter. As with any other hydraulic transport operation, the decision of proper solids-to-liquid ratio is driven by economic factors. A high concentration requires smaller pipes and less water but most likely more energy to operate. The tendency is to produce tailings with higher solids concentrations to the point at which the mixture is a paste, not slurry. In downhill operations, open troughs are sometimes used in lieu of closed pipes.

A hydraulic transport system requires a slurry preparation station, pumping stations (or dissipating stations in downhill applications), and a slurry receiving station. It might also include a dewatering plant to treat or reuse the water contained in the solids–liquid mixture.

ORE STORAGE

Since transport from the mine is not necessarily in sync with the processing, production, and consumption of materials, there is a need to store the ore at various stages. For trucking and marine shipping, ore is typically handled in batches; crushing and milling are continuous operations that do not lend themselves to discontinuity in the flow of material.

Storage Sizing

Traditionally, rules of thumb are used to size a stockpile. Typically, stockpiles are based on storage for one day or one shift, the size of a ship, or the duration of a shutdown. In coarse-ore stockpiles with gravity recuperation, a typical value for live capacity is one shift of production and for total capacity the duration of a long shutdown.

A more accurate way to determine sizing of a stockpile is through a dynamic simulation, which includes factors such as the availability of feed, the process plant, and the output system. For a given live capacity, it is possible to estimate the probable number of hours that auxiliary equipment would be required to reclaim dead material or the number of hours a plant would stand because of lack of process material. The net present value of the operating cost for the auxiliary equipment or production losses throughout the plant life can be compared with the extra capital cost of a system having larger live capacity. Operation below the live-capacity line requires the use of mobile equipment. The savings in operating costs and/or lost production might be enough to pay for the higher capital cost of a larger stockpile.

Dead capacity should be enough to accommodate probable shutdowns such as primary-crusher liner replacements, mobile-crusher relocations, or conveyor splices. This sort of shutdown usually lasts a few days. If space is available, additional dead capacity can be created using a bulldozer to spread the material. This kind of storage, usually known as a strategic stockpile, is often used on a coal-fired power station feed so the power supply is not interrupted due to lack of feed. In ship loading and unloading operations, the high cost of ship demurrage creates the need for large stockpiles.

The key word in sizing a stockpile is *probable*, as opposed to *possible*. A stockpile should be large enough that continuity of operations can be maintained under reasonably probable conditions. No matter how big the stockpile, it cannot accommodate all the possible situations. So when sizing stockpiles, it is important to balance the certain costs of a larger stockpile against the probable cost of an operational shutdown.

Storage Configurations

The conical shape is probably the most common configuration for stockpiles. A conveyor feeds material at a single discharge point, creating a cone. Although the natural instinct is to move the discharge point higher to achieve larger capacities, issues such as permissible ground loading, degradation of material when falling from the conveyor, energy cost to lift the material, cost of elevated structures, and reclaim considerations limit the maximum practical height. It is quite common also to use a fixed tripper or a second conveyor to create a dual cone when higher capacities are required. Longitudinal stockpiles can be created if the feed conveyor is elevated and has a tripper with a moving head or it feeds a shuttle conveyor to distribute the material along the axis. The elevated conveyor is supported by steel or concrete columns or by an A-frame structure. Radial stockpiles can be created using a radial stacker or a slewing-type stacker. Various stockpile configurations are discussed by Zamorano (2006).

Another option for forming a longitudinal stockpile is to use a wheel- or track-mounted traveling stacker. If the stacking boom can be raised and lowered (luffed), degradation of material when falling to the ground can be minimized. A boom that can rotate (slew) allows the formation of two parallel piles with one machine. Multiple stackers can be used to form multiple stockpiles in a stockyard operation.

Covered Storage

There are several reasons covered storage may be required; the most common is environmental regulation. However, protecting the material and preventing loss of fines due to wind are also reasons. For small tonnages, steel bins are satisfactory, whereas for larger volumes up to 10,000–12,000 t (11,023–13,228 st), concrete silos are commonly used.

In the case of conical or circular stockpiles, the covers can be conical or dome shaped. The conical shape is suitable for conventional steel structures constructed from steel trusses arranged radially, similar to a tepee. Domes are mostly designed as shell membranes with a variety of alternative designs. Conventional steel structures can be used to create a dome-shaped membrane-type structure that is covered with cladding. Another option is a space truss–type membrane formed with tubes and mechanical connections and covered with cladding. This type of dome is called geodesic and can be manufactured in steel or aluminum.

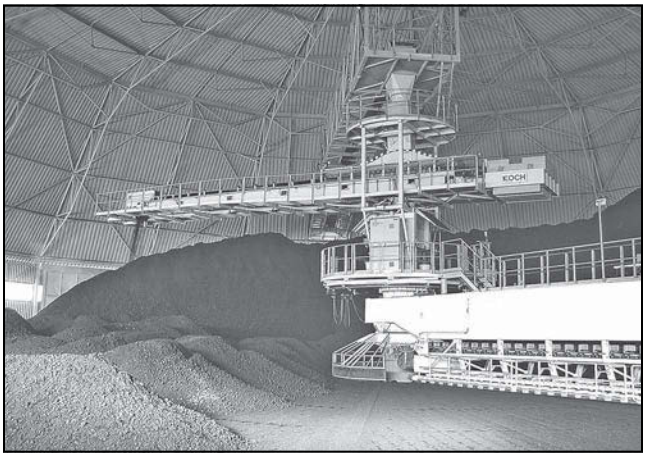
Other types of domes are made of thin concrete membranes, inflatable membranes (that require a blower in permanent operation), and covers using fabric cladding instead of metal or plastic. Covers for longitudinal stockpiles are usually a variation of the A-frame design. Closely spaced frames are used to support the cladding and the feed conveyor. Geodesic-type frames are suitable for stockpiles with rail-mounted traveling stackers or with a small, light feed conveyor supported by the cover.

ORE RECOVERY

Ore recovery comprises both reclaiming and blending the material.



Courtesy of FLSmidth Spokane, Inc.
Figure 10.7-9 Combined bucket-wheel/stacker reclaimer



Courtesy of FLSmidth Spokane, Inc.
Figure 10.7-10 Circular blending stockpile

Reclaiming Systems

In enclosed storage facilities, material is recovered via a gravity trough at the bottom of the bin or silo. If the material properties are known, a proper design can be developed and problems like arching and rat-holing can be avoided. Material can be recovered from stockpiles by means of mobile equipment such as front-end loaders and bulldozers that feed a reclaim hopper. Although this option presents the lowest capital cost, the operational cost is quite high, so this option can be justified only in small throughput operations.

Material can also be recovered by gravity using feeders and conveyors located underneath a stockpile. In this case, only a portion of the material—called live capacity—flows unassisted. The rest of the material, called dead capacity, is reclaimed with mobile equipment. In conical stockpiles or where free-flowing materials are involved, 20% to 25% of the storage is considered live. However, in large stockpiles or depending on material properties, rat-holing or arching may appear and the live capacity of the stockpile can be dramatically reduced (Cabrejos and Goodwill 1996) to a point that mobile equipment is required at all times.

A higher percentage of live storage can be attained using longitudinal stockpiles and several withdrawal points or rotary plow feeders. However, in this instance live capacity is 50% at best. Bucket-wheel reclaimers or other types of reclaimers (such as a portal scraper) can put live capacities in reach nearly all the time. These machines remove material placed by stackers when required (i.e., stockpiles, leach pads, etc.).

In some cases these machines are designed to operate as a stacker and a reclaimer; they are called combined machines (Figure 10.7-9). The additional capital expense of mechanized reclaiming systems is generally outweighed by the operational cost savings realized.

Another approach is the use of large, squat silos with assisted feed and reclaim to obtain a 100% live capacity within a small footprint (Nijhof and Ruijgrok 1986).

Blending

Variation of material properties in different areas of a specific mine or from different mines in a single processing facility necessitates blending the material to obtain a homogeneous product. This might involve a chemical component or even particle distribution.

There is a certain amount of natural mixing whenever a bulk material is handled. A crude level of mixing is achieved on stockpiles with the use of front-end loaders or clamshell buckets. The same can be said regarding boom-mounted bucket-wheel reclaimers and scraper reclaimers. Another approach is to use different bins to feed a common conveyor.

The blending effect is defined as the ratio of the standard variation of the control property at the store input and the standard deviation of the same property at the store output. Theoretically, the blending ratio is proportional to the square root of the number of layers reclaimed simultaneously. In reality it is somewhat lower, particularly in coarse materials.

In longitudinal stockpiles, when blending is not required, the preferred stacking method is the cone-shell method, in which the stacker discharges onto a single point. Once the maximum height is reached, the stacker moves a discrete distance and continues to discharge, forming contiguous cones. The chevron stacking method consists of depositing the material while the stacker moves to and fro over the center of the stockpile. To ensure proper blending, the material must be reclaimed from the face of the pile working across the entire cross section. This can be achieved with a bridge-scraper reclaimer that employs a harrow to guide the material into the scraper chain at ground level. The reclaimer recovers material from previously stacked material, while more material is stacked in the area that has been reclaimed.

The chevron method can also be used on a circular blending stockpile, as shown in Figure 10.7-10, where work is done on a continuous basis. A slewing stacker is mounted at a center pile and a bridge scraper reclaimer feeds a discharge point also on the center.

According to the windrow method, material is deposited over several positions across the full width of the stockpile. This method prevents segregation, and the reclaimer can operate on one part of the cross section at a time. A bridge bucket-wheel reclaimer can be used in this type of arrangement.

ENVIRONMENTAL CONSIDERATIONS

Environmental factors become more relevant every day in connection with ore transport and storage operations. Obtaining the relevant local environmental permits has become a key element for most mining operations, and compliance with environmental management standards such as ISO 14000 is

becoming a requirement for international trade. This section addresses concerns directly related to the environment.

The handling of bulk solids generates unwanted airborne dust. During transport by truck or train, it is quite common to use canvas covers to prevent contamination along the transport route. As has been discussed, covers are used along belt conveyors for the same purpose. When fine and troublesome materials are an issue, the entire conveyor is completely enclosed, or closed systems such as pipe conveyors are used. Loading and unloading activities and transfer points generate large amounts of dust, which is controlled by enclosure, suppression, extraction, control of fall speed, or by a combination of these.

Coal storage presents the risk of spontaneous combustion and can produce toxic fumes. Management systems on open stockpiles include the use of large spraying cannons to maintain surface moisture levels. This type of system requires a water management plan to control the acid water percolated from the storage area. Stormwater treatment facilities are required in open storage facilities for coal or other types of contaminating materials.

Enclosures can reduce the fugitive emissions. However, sometimes additional suppression or extraction systems are required. At times, negative-pressure systems are employed on enclosed stockpiles to capture dust.

Dust suppression involves the addition of water at a transfer or drop point to capture airborne particles. Chemical surfactants can also be used to increase the efficiency of a dust suppression system, as they facilitate the coalescence of the water droplets and dust particles. Another approach is the use of "dry fog," in which a special nozzle produces very small droplets. These droplets are approximately the same size as the dust particles and improve the chances of capture. At the same time, this minimizes water consumption and the addition of water to the ore. In open stockpiles, large spraying cannons are used to maintain superficial moisture and reduce dust emissions.

Dust-extraction systems use negative pressure to collect the dust from a transfer point or other dust source. The dust is then removed using a filter, cyclone, or scrubber, depending on the particular application.

Cascade chutes are used to reduce the speed of the material on large drops. The lower the speed, the less dust is generated.

Any water used for dust control or hydraulic transport is treated prior to discharge. However, in dry areas, conflicts can arise between the mine and local communities because water is a scarce resource and residents don't want it to be diverted for mining use. In addition, local communities are concerned that water contamination will result from the mining process.

The issue of noise, including its impact on endangered species, can be another area of conflict between a mine and local communities. If this is an issue, conveyors and pipelines have a clear advantage over trucks.

An aspect of mining that became an issue only recently is carbon emissions. Compared with truck haulage, continuous transport methods such as conveyors and pipelines can reduce the amount of carbon released into the atmosphere. Energy efficiency and the method by which power is generated also greatly influence carbon emissions.

Impacts on the landscape and surrounding communities are also relevant factors to be considered when making decisions about handling systems for bulk materials. Again, continuous methods tend to produce the lowest impact. Planners often neglect to consider the disruption caused by a great many trucks moving through communities.

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CHAPTER 10.8

Strip Mining

Gary Dyer and Ken Hill

INTRODUCTION

For minerals that lie within economic reach by surface mining, and when the deposit's specific geometry, either tabular or bedded, allows or dictates, an approach to extraction called open-cast, or strip mining as it is more commonly referred to, can be employed.

Although a range of commodities such as phosphate, bauxite, tar sands, manganese, and even industrial materials from quarries have been recovered in this manner, the most common deposits worked by strip mining are coal deposits, and for this reason this chapter will refer predominantly to coal.

The economic depth for strip mining is dependent on realizing a margin between its unit revenue value and the unit cost to recover, driven by depth and deposit complexity. The lesser-value bulk commodities will typically only be economic to shallower depths, whereas high-value coking coal, such as that found in Australia's Bowen Basin, and where the stripping ratio allows, shows some strip mines currently operating at up to 150 m of depth with potential to progress to more than 300 m. Strip mining is a bulk earth-moving operation making use of large-scale mechanized equipment.

An attractive geometry for strip mining is a tabular deposit with coal subcropping at economic depth and extending laterally, either flat or gently dipping, and constrained vertically within a number of seams. Deposit continuity relatively free of geological structure or intrusions is also attractive but not a prerequisite, although a heavily disturbed and/or intruded deposit and the resulting economies may limit the selection of equipment and operating method.

Strip mining is characterized by its method of waste material, or overburden, movement, which is placed almost entirely in-pit. An initial cut is made on the coal subcrop, called the boxcut, and the overburden is placed on a natural surface updip of the subcrop line. The exposed coal is mined out and successive cuts, or strips, are taken to progress the mining downdip with the overburden from each strip placed inside the previous mining void. Individual strip geometry is typically from 30 to 100 m wide and to the economically recoverable basal coal seam. A strip will extend along the strike until it is constrained by a mining property boundary, limiting surface feature, geological discontinuity, or economic limit with the maximum length of some strips being in the kilometers where conditions allow. A form of strip mining where the strips are aligned downdip instead of along the strike is applied in some instances but is rarer as economics generally favor strike mining. Each strip working area is denoted as a pit with a strip mine consisting of a number of pits.

Strip mining encompasses a number of different mining strategies, each a unique combination of pit configuration, equipment selection, and operating methodology. This chapter begins by outlining a generic strip mining process. It then goes on to discuss alternative pit configurations, equipment selections, and operating methodologies for each of four mining strategies. The strategies have been developed in some detail with a major case study example used to provide comparison and contrast. Additional examples are referred to for variants of note. The chapter closes with some discussion on what drives the selection of one strip mining strategy over another as well as a brief commentary on future trends in strip mining.

GENERIC STRIP MINING

The strip mining process from first workings to final closure can be described generically in six steps:

1. Clearing and topsoil removal
2. Fragmentation
3. Waste removal
4. Waste placement (including soil restoration and initial revegetation)
5. Coal mining
6. Mine restoration, maintenance, and eventual closure

Figure 10.8-1 provides a stylized schematic to show these processes. Each of these process steps will be discussed briefly as a prelude to discussing the four strip-mining strategies.

Some additional operational activities that support the mining process have been covered briefly due to their influence. These include rejects and tailings storage or disposal, river and creek diversions, and water management.

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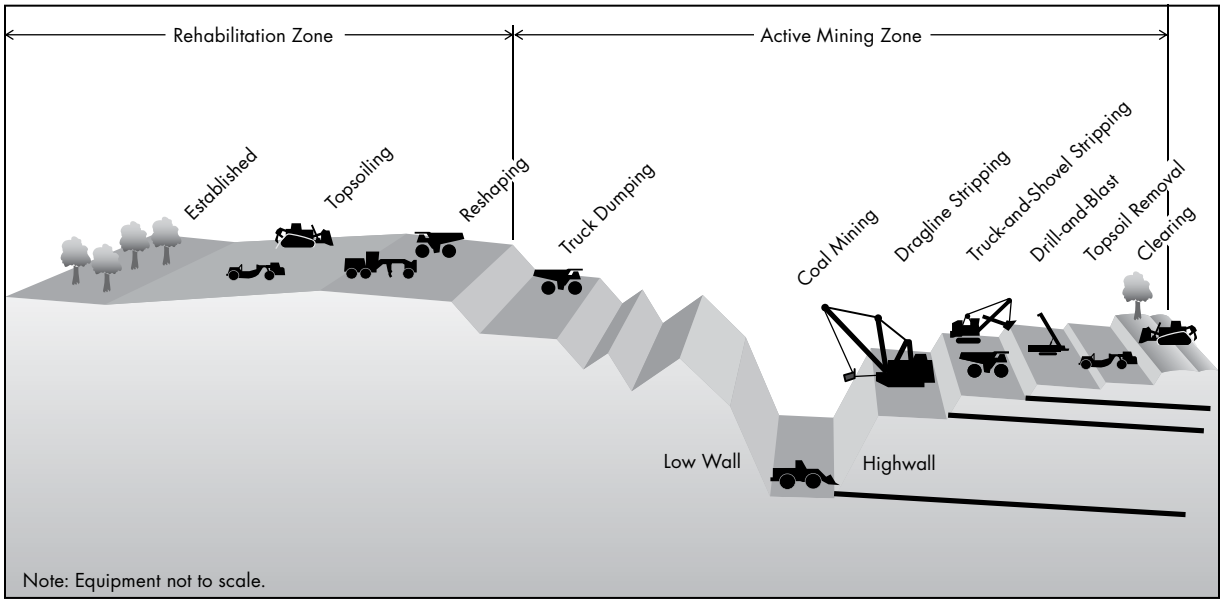


Figure 10.8-1 Strip mining process

Clearing and Topsoil Removal

Prior to first workings, a mining area is typically covered with vegetation and topsoil. The vegetation must be cleared, and the topsoil must be recovered and stockpiled for later use in postmining rehabilitation. Vegetation is cleared by track dozers and pushed into piles and disposed of by burning, shredding for mulch, burying, or a variety of methods dependent on local customs and laws. An examination is made as to the quantity and quality of topsoil, and a target thickness to recover is identified. High-quality soils may need to be separately identified and stored. Topsoil is removed with earthmoving equipment, typically scrapers. However, where thickness allows, dozers assisting excavators to load trucks may also be an efficient operating method. The topsoil is stockpiled in the vicinity of the future mining operation so that it may be recovered for postmining rehabilitation. For a mature mine with ongoing clearing and topsoil removal occurring in parallel to final rehabilitation, it may be possible, and is certainly desirable, to place topsoil directly in its final position without stockpiling. It is important that topsoil stockpiles are not compacted by machinery or excessive thickness as biological activity, and thus their value, will be adversely affected by the exclusion of oxygen and moisture from the topsoil mass.

Fragmentation

Following topsoil removal, there is a relatively small thickness of weathered and/or unconsolidated material overlying much thicker units of competent overburden. However, where there is a substantial thickness of weak material, it may be possible, or even necessary, to progress directly to a waste-removal process. This is the exception, and so fracturing, or fragmentation, of the overburden in advance of mining operations is required so that it may be handled safely and productively. Two main methods of achieving fragmentation are dozer ripping and drill-and-blast, with method selection dependent on the strength of the overburden, its thickness, and the volumetric demand of the mine.

Dozer Ripping

For relatively thin overburden of less than 4 m and where rock strength allows, a dozer-ripping process may be used for fragmentation. Use is made of the track dozer tines and the machine's mass to break the rock into a manageable size. Operational effectiveness is highly sensitive to rock strength, with high-strength rock leading to poor productivity, as well as maintenance downtime, due to adverse wear on the machine.

Drilling and Blasting

Drilling involves the creation of holes in the overburden within which explosives are placed. Drill holes are laid out in a regular pattern, either square or offset, the spacing of which is determined by the diameter of holes being drilled and the desired explosives density per volume of rock to be broken, or powder factor. Drill holes are drilled to either the roof of the target coal seam or to an operational depth to match the waste removal operating methodology. Where the valuable material (i.e., coal) is variable in depth, the driller may drill into the coal to ensure correct hole depth and then put a short length of stemming into the bottom of the hole to protect the relatively soft coal from damage when the overburden is blasted. Otherwise, holes are measured to ensure that the correct depth has been reached, with wet holes being marked and pumped out if the depth of water warrants it. Poor depth control will lead to uneven pit floors and inefficient waste removal. A presplit may also be used where a line of holes at a tighter spacing is drilled along the line of the next strip's highwall and is blasted separately from the main pattern. The intention is to deliver a smooth wall that is productive to excavate the waste back to and safe for personnel and equipment to work under during coal mining.

A blast design is prepared based on the overburden strength characteristics, the wetness of the holes, and the desired direction of movement of the blasted material controlled by placement and duration of hole delays. Bulk explosives are placed in the drill holes by mechanical means with ammonium nitrate and fuel oil (ANFO) being the most

common explosive used. Specialist explosives may be used for stronger overburden with water-resistant emulsions used in wet applications. As each hole is loaded with explosives, an explosive booster tied to a detonating cord is placed within the explosives near the bottom of the hole. The top 5 m or so of the hole is filled with stemming to prevent the blast energy from escaping at the top of the hole and instead direct the blast energy into the rock mass to fracture it (and in some applications to actually move it sideways). The cuttings at the top of the drill hole are often used as stemming material, but some mines import size-graded rock to deliver a superior blast-containment mechanism. After the area to be blasted has been loaded with explosives, the shot firer will connect each hole with detonating cord and time delays. A lead-in detonating cord is run out to a position of safety and, with guards in place and the area confirmed as safe to blast, the shot firer will initiate the blast. Electrically initiated detonators with millisecond delays may be preferred where sensitive vibration control is required. A good blast outcome is typified by an absence of flyrock, no excessive sound and/or vibration, no misfired holes, and good visible fragmentation with the rock mass traveling in the desired direction.

Strip mining is unique in that the drill-and-blast process itself can be employed as an overburden removal process. As the overburden is to be placed into the mined-out void immediately adjacent, certain pit configurations and operating methodologies lend themselves to cast blasting. Cast blasting is where a powder factor and delay design is selected to purposely cause the fractured rock mass to heave in the direction of the mined-out void with large quantities of overburden, up to 30%, resting in final position. It therefore requires no further handling by mining equipment. This is a particularly economical method of overburden removal.

Waste Removal

After blasting, the overburden may then be removed. Each end of the strip is called an endwall. The wall of the strip that is to progress successively downdip and is cut in in-situ material, or prime, is called the highwall. The in-pit overburden, or spoil side, wall is called the low wall because of its flatter slope compared to the highwall. (Refer to Figure 10.8-1.)

Cast–Doze Excavating

At shallow depths of <30 m, a combination of cast blasting and pushing with track dozers will provide the most cost-effective method of waste removal. Cast blasting can place up to 30% or more of the prime overburden into a final position. Track dozers can then push the remaining overburden into final position. If dozer pushes become excessively long or uphill, and therefore less productive, it may be more economical to supplement the operation with an excavator and a short-haul truck operation. The highwall will typically be cut to an angle of 65° with excavator assistance in cutting the batters back to hard material. The low wall will typically be placed at between 37° (the angle of natural repose for fragmented rock) and 45°. Effective rock fragmentation is critical to the success of this method, as blocky oversize overburden has a significant negative impact on the productivity of dozer operations. Poorly cemented overburden may be moved by scrapers.

Dragline Stripping

Where scale and deposit geometries allow (up to 85 m or so in overburden depth, depending on the class of machine being applied), a dragline can provide a more cost-effective method of waste removal. This requires sufficient annual demand for overburden stripping and mine life (in excess of 10 years) to pay back a high-capital, low-operating-cost piece of equipment. Simplistically, the dragline operates by moving overburden, or prime, from the current strip and placing it in the previous mined-out strip as spoil. Figure 10.8-2 shows a diagram of a dragline identifying its main physical features. The dragline needs to be placed at a level from which it can reach the top of the deposit to be exposed. This will either be its dig depth or operating level if less than the machine's maximum dig depth, although the dragline can handle overburden above its operating level via overhand methods. The total thickness of overburden to be allocated to the dragline therefore may consist of a number of passes made up of a combination of overhand and underhand operations and perhaps even repeated for multiple seams.

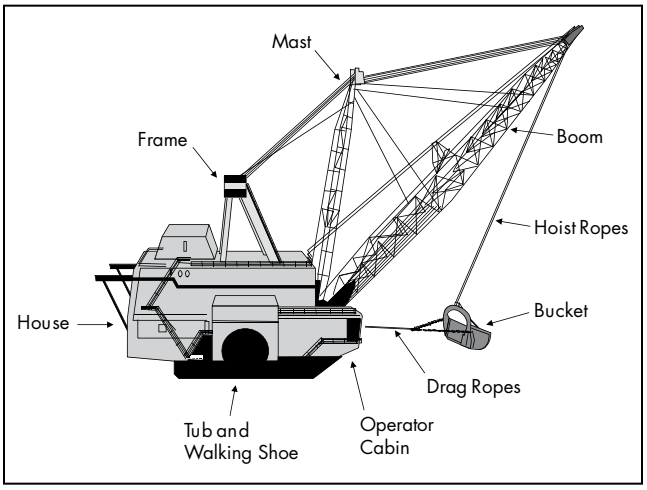


Figure 10.8-2 Dragline

Figure 10.8-3 shows a typical walking dragline operation. Highwall angles range from 45° to 65° and depend on the competency of the material to be excavated, as well as joint orientations. Offset benches in the highwall may be established for reasons of overall wall safety in terms of stability, to reduce risk to subsequent mining operations below the wall, or for operational reasons relating to the maximum reach of the overburden drills. Low-wall angles are typically cut at 45° up to the dragline operating level and then lay back at a 37° angle of repose to the peak of the dragline spoil.

The dragline will advance along the strip, moving by dragging itself on a revolving tub by 1–2-m "steps" delivered by two large feet on an eccentric cam, in regular blocks, typically 30 m in length, but they can be longer or shorter depending on the dragline's operating level and thus its ability to reach the toe of its excavated block. For a narrow strip width, the dragline is often placed directly into a final position, whereas for a wider strip, as is more commonly the case, the dragline builds temporary operating positions out of overburden from which it places waste from the previous block into final position. This is discussed in more detail later. Any overburden that the dragline has to handle a second time is termed *rehandle*. An efficient dragline operation will minimize rehandle, which is achieved by setting the dragline's operating level as low as possible. Unfortunately, the lower the dragline operating level, the lower its direct overall spoiling volume capacity,

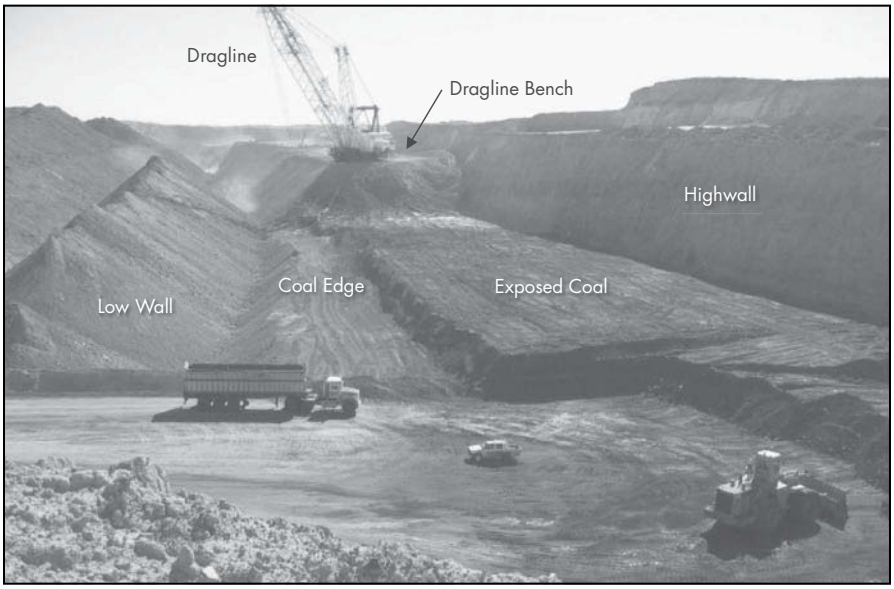


Figure 10.8-3 Typical dragline operation

which will then have to be compensated for, either by employing spoil side rehandle passes or removing material in advance by truck methods. Both of these are expensive, so rehandle is often an outcome of balancing desired coal exposure rates with installed dragline capacity and overburden removal economics.

Dragline operating geometries provide a relatively narrow operating envelope. For the larger class of draglines, their operating parameters are a dig depth of 65 m, a dump height of 55 m, and a dump radius of 95 m with an effective thickness of overburden allocated to the dragline system of around 85 m (Bucyrus 2008). Overburden outside of this envelope will need to be rehandled if the dragline is to deal with it; if it needs to be rehandled twice, it will generally be cheaper to remove by truck methods. Errors in design and unexpected geological impacts such as faults or discovered areas of geotechnical weakness may result in overburden being locally over-allocated to the dragline somewhere along the strip. This may result in the dragline becoming spoil-bound where it can no longer place all of the overburden it has been allocated to a final position using its planned operating methodology. This needs to be rectified either by ramping its bench higher to provide additional spoil room, or, if spoil fit is already maximized, by walking the dragline into the spoil and rehandling overburden even further away, via an elevated bench or pullback operation, to create room for the current strip's waste. This is an expensive exercise and can cause significant unplanned delays in coal exposure rates and so is to be avoided. Mobile, track-mounted draglines are still popular in variable terrain where overburden depths do not exceed the digging capacity of these smaller machines.

The annual operating capacity of a dragline is a product of its rostered time, mechanical and electrical availability, operating use, and productivity. As they are capital intensive (in excess of US$200 million per unit), draglines are typically rostered to operate 24 hours per day, 7 days per week. Their relative simplicity as a predominantly electrical machine leads to high availabilities, typically running in excess of 90%. Apart from geotechnical failures, excessive rainfall, or mine-planning failures (lack of blasted inventory), operating use is also high, running typically in excess of 90% of available time. Productivity will vary depending on the operation, with large swing angles from dig to dump, high spoiling, poor fragmentation, and rework of the excavated face or bench resulting in lower productivities. Total volumes moved per annum will range from 15 Mbm^3/yr (million bank cubic meters per year) for the most common smaller-sized machines to >30 Mbm^3/yr for the larger machines currently deployed. A dragline, depending on its size, will consume the equivalent electricity per annum to supply a small town of 3,000 to 5,000 people. Concentrating so much productive capacity in a single machine and in a pit configuration that requires vertical stripping, however, is a significant risk exposure, and there have been instances of catastrophic failure via machines "falling" into the pit because of bench failures, basic mechanical failure of key structural components (materials failure or operator error), or flooding. For mines relying on a single or a small number of draglines, this results in a significant business impact that is not quickly or inexpensively recovered from. Small efficiencies in dragline use lead to large cost savings, so most large draglines will be fitted with operating monitors. These assist the machine operator to select optimum swing angles, bucket-fill factors, and casting radii.

Dozer-Assisted Dragline Stripping

The dragline can be supplemented by track dozers pushing prime material into the void where, depending on strip width, it can be rehandled to its final position by the dragline. This operation is called production dozing and has the effect of increasing the effective prime waste movement rate, and thus coal exposure rate, of the dragline system, but at the expense of increasing dragline rehandles. Nonetheless, where there is sufficient spoil side room due to the dragline being under-allocated waste in terms of spoil fit for coal exposure rate reasons, it is a low-cost, low-risk way to increase the total thickness of overburden allocated to the dragline system.

Truck and Loader Stripping

For more-restrictive deposit geometries, shorter mine lives, and where a more-variable scale of operation is planned, the superior flexibility of truck-based waste removal methods may be suitable. Truck methods involve the use of trucks and loading equipment to dig overburden and dump it. Loading equipment is selected to match the trucks, whereas the truck and loader package is selected to match the task. Highwall batter angles are 60° for each bench in competent material, with flatter angles for poorer material, but often with 5-m berms between benches leading to much flatter overall highwall angles.

Large bulk thicknesses of overburden, typically >15 m, will be removed by electric rope shovel and ultra-class (>300-t payload) trucks as the most cost-effective operation, provided there is a sufficient annual requirement for waste stripping and mine life (>10 years) to pay back the higher-capital, lower-operating-cost position offered by larger electric shovels compared to smaller hydraulic excavators. For thicknesses >20 m and up to 25 m, overburden can be pushed down with dozers to create a safe working face height. Beyond 25 m of total thickness, multiple benches are often created. These might be separately blasted or through-blasted depending on material competency to allow them to be run over with the trucks. Where ready supplies of electric power are not available or at contract operations, this overburden task will alternatively be handled by large hydraulic tracked excavators, either in shovel or backhoe configuration. Electric rope shovels work most productively on a relatively flat bench, so where the dip of the coal seams exceeds the effective operating grade for the electric shovel, a secondary wedge operation using an excavator and trucks may be needed.

Truck-shovel operations can be used in an advanced-bench mode on upper layers of overburden to prepare a working bench for a large dragline. The truck-shovel operation removes variable topography, leaving a horizontal bench for the dragline. The dragline then is able to operate uniformly and efficiently at its optimum digging depth. Meanwhile, the truck haulage routes are arranged so that a natural-looking final topography is created without the need for spoil rehandle.

For overburden thicknesses <15 m, it will usually be more cost-effective to run with smaller hydraulic tracked excavators loading appropriately sized trucks. As overburden thicknesses reduce, smaller excavators may be selected. Below 2 m of thickness, it may become more effective to have a dozer ripping and pushing overburden up to the excavator, whereas sometimes a pure dozer method is employed where a void is available within a short push distance. Wheel loaders are rarely used to load waste, often due to their inferior economics in overburden operations.

A typical truck and loader fleet will consist of a loading tool, sufficient trucks to match the total cycle time (load, haul, dump, and return) to keep the loading tool running continuously, and various earthworks support gear including dozers, graders, and water carts. When used to remove overburden above the dragline, the truck operation is often termed a *pre-strip operation*. When used to remove waste between coal seams below the dragline operation, the truck operation will often be termed an *interburden operation*, whereas for thin waste bands it may be called a *parting operation*. Where the truck operation uncovers coal in its own right, either above a dragline or in a nondragline mine, it is more commonly termed a *truck and shovel operation*. The width of strip for the truck and loader operation will typically be set to match the dragline strips so that a regular release of dragline strips occurs. This can limit the effectiveness of the loading operation in narrower strips by reducing the operating room, increasing the influence of edge effects, and limiting the opportunity to deploy double-sided loading of the trucks, thereby reducing productivity. Wider strips, however, result in longer truck cycle times and increased work-in-progress in the pit that manifests as a larger invested working capital. Scheduling bottlenecks are also introduced by larger batch sizes via wider truck and loader strips. These effects need to be carefully evaluated depending on local conditions. Figure 10.8-4 shows a typical truck and electric rope shovel operation.



Courtesy of BHP Billiton.

Figure 10.8-4 Electric rope shovel and truck operation

The annual operating capacity of an electric rope shovel is a product of its rostered time, mechanical and electrical availability, operating use, and productivity. As they are capital intensive (in excess of US$20 million per unit), electric rope shovels are typically rostered to operate 24 hours per day, 7 days per week. Their relative simplicity as a predominantly electrical machine leads to high availabilities, typically running in excess of 90%. Apart from geotechnical failures, excessive rainfall, mine planning failures (lack of blasted inventory), or sufficient trucks in the cycle to match the haul duty required, operating use is also high, running typically in excess of 90% of available time. Productivity will generally vary less than a dragline because of the relatively generic nature of electric rope shovel operation most of the time. Productivity will be affected when digging less than optimal bench height, ramping in or out of an area, or when poor fragmentation and difficult materials such as clays result in carry back within the bucket leading to increased truck loading times.

Total volumes for electric rope shovels typically range from 15 Mbm^3/yr to 20 Mbm^3/yr for the larger machines currently deployed with the variability driven by the specific application. Three electric rope shovels will consume approximately the same electricity per annum as a large dragline. Electric rope shovels are inherently flexible but can present some constraints in terms of minimum required working areas, maximum operating cross-grade, and the requirement to manage a cable interacting with blasting operations and a fleet of trucks.

Continuous System Stripping

For pits that require large annual quantities of waste movement (>15 Mm^3/yr) to be placed via long haul cycles (>30-minute return), it may be cost-effective to deploy a continuous or conveyor waste system, either by itself or in conjunction with other waste removal methods. Relatively few of these operate globally, and their successful implementation is highly dependent on developing a specific and robust mine plan that marries overall pit configuration, equipment selection, and operating methodology to the fixed nature of these systems compared to other mobile overburden-moving equipment options. These systems comprise a crusher station at the dig end, a long conveyor (>5 km), and a spreader at the dump end. The crusher may be fed by trucks or directly by track dozer, excavator, shovel, bucket-wheel, or even dragline. These systems are capital intensive, being comparable in capital cost to a dragline with similar annual prime capacity but with an operating cost that sits between a dragline and truck and loader systems. For these reasons, only the longest-equivalent truck hauls provide an effective cost offset; economic payback can take a number of years and depends on achieving large annual volumes, which means minimizing the magnitude and frequency of partial- or full-system relocations. This is a developing mining strategy that will see more potential for economic application as pits deepen, but it is also under competition from emerging technology step-change improvements in truck and loader systems, such as partial or full automation. The environmental advantage of continuous system stripping is the ability to restore spoil in an approximation of the original strata profile.

Waste Placement

The movement of waste is a pure cost to the mining operation and has no direct economic benefit, so the placement of waste will generally be driven by a least-effort approach. For equipment other than trucks and conveyors, this dictates the overburden only be moved a short distance within the practical and economical operating envelope of the equipment in question. This will typically be <200 m in the horizontal from the point of origin with changes in elevation generally <50 m.

Cast–doze excavation, dragline stripping, and dozer-assisted dragline stripping methods generally place the overburden into the void from the previous strip either directly or with some rehandle.

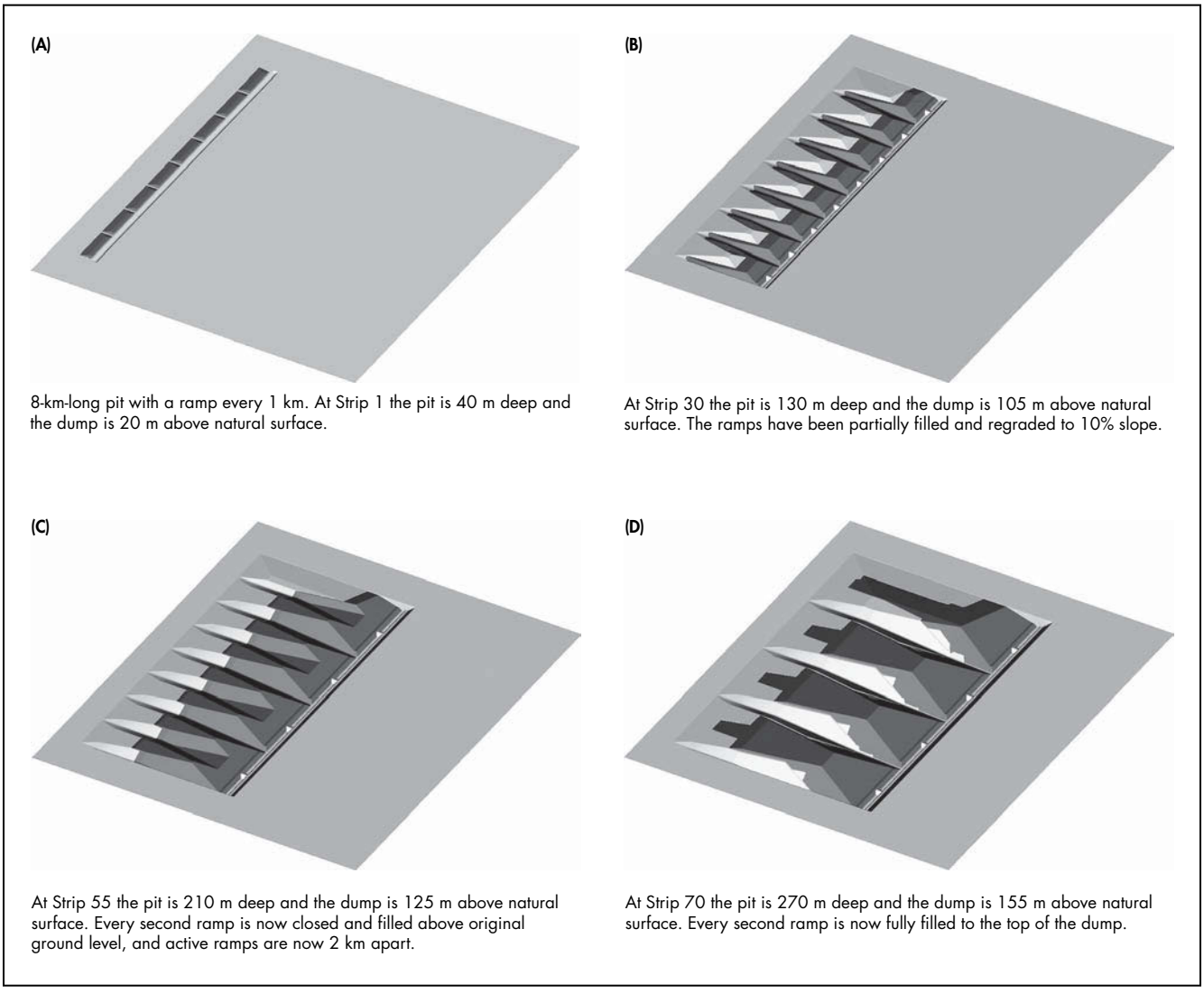
Minor truck stripping operations engaged in either parting or thin interburden operations, and generally at or near the bottom of the strip, will seek to minimize the haul. Overburden is thus placed inside the strip being mined or nearby, perhaps being used to regrade an access ramp that may be running at a flatter grade than that required by the coal mining trucks.

Bulk truck stripping operations, not supporting a dragline stripping method, also place the overburden in the previous void. Least-cost operation is generally achieved when material is hauled on grade, or as flat as possible, and it is desirable to establish the dumping strategy to sustain an average haul. This will ensure that short hauls do not idle spare trucks unduly and long hauls do not create instances where the loading equipment is undertrucked and production is lost. It is vital to have an established road system that allows access to multiple dump levels, and this road system will grow in complexity with pit depth and thus the number of dump lifts being developed.

For bulk truck stripping operations supporting a dragline stripping method, otherwise known as prestrip, the overburden is placed on top of the dragline spoil. The proximity of the truck dump to the active spoil area is determined by the geotechnical competence of the waste material itself and the strip floor; however, a good general rule is two spoil peaks, or prior strips, back. Dump geometry is dictated again by geotechnical competence but also the location of roads to access the dump and the desired intensity of dumping operations that need to be designed for. A typical overall operating angle of 19° for the active dump face is typical for truck dumping, and this profile is the result of a series of dump lifts that batter down at angle of repose interrupted by catch berms and roads. The economic cost of elevating overburden with trucks relative to horizontal haul is approximately 20:1, which is to say a dump will preferentially develop horizontally until economics dictate that an additional dump lift be added. For this reason, the dumps for mature pits will adopt a typical profile with the greatest height in proximity to the active dig area with a trailing back of the dump.

At this point it is worth considering the geometric drivers of dump development as well as economics. Figure 10.8-5 shows how the dumping strategy changes over time as a strip mine deepens. As the pit progresses downdip, batter effects demand greater and greater volumes of overburden to be moved, and this generally reports directly to the truck operation as the dragline is allocated a fixed thickness of overburden for each strip. On the dump side, the dump end batters play a similar role but have the inverse effect: as the dump gets higher, it has a correspondingly lower volumetric capacity due to less pit length available for dumping. The system dynamic is simplistically one of an inverted cone-shaped pit getting deeper, filling a conical stockpile that has a finite capacity for a fixed basal area.

After the dump reaches capacity, only three responses are available. First, any perpendicular ramps to the base of the strips that create large valleys reducing available dump room can be filled with alternative clearance of the mineral to be mined to occur by parallel low-wall ramps or via a high-wall ramp system. Second, the dump can be extended in the direction of the original commencement of mining. This can have grave consequences for any progressive rehabilitation that was undertaken without a final landform design and can lead to redisturbance of previously rehabilitated areas. The same issue might also apply to any infrastructure that was also placed without due consideration of long-term dump designs (e.g., coal haul roads, power lines, stockpile areas). Third, overburden can be placed out of pit or, in other words, beyond the limits of the excavated-to-date pit shell. In some regions, this out-of-pit material is called excess spoil. This can be expensive, involving long hauls or placing overburden on the highwall side of the pit and on top of future resource areas. This may be undesirable unless there are sterile areas due to geological disturbance, intrusions, or underground operations or other factors that may preclude strip mining in that area, or unless the subsequent rehandle of this waste at a future date is still economical. An additional downside to out-of-pit dumping is that it increases the total area of mining disturbance and creates an additional net area to be rehabilitated and maintained after mining ceases. Excess-spoil dumps have created environmental challenges whenever they intrude into established natural drainage channels. The placement of overburden for deep strip mines is an emerging problem and generally starts to manifest at depths of around 150 m. A further uncertainty and risk is the geotechnical stability of large in-pit dumps and is the subject of ongoing research.

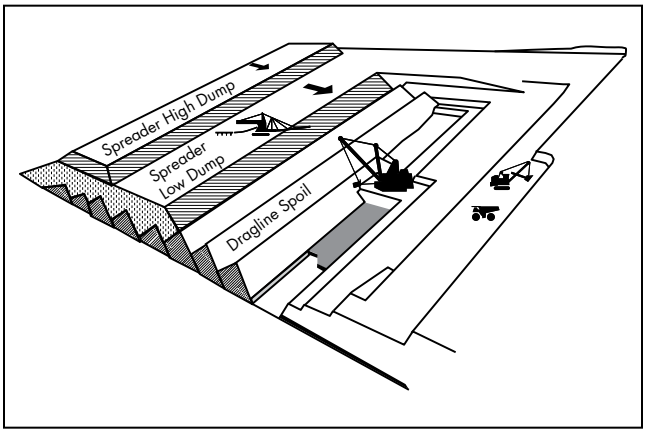


Courtesy of BHP Billiton.

Figure 10.8-5 Dumping strategy development with depth

For an integrated truck and dragline method, an additional operational complexity to be resolved is the interaction between the trucks and the dragline with both equipment types often having to operate in close proximity at the same time. The truck operation will seek to minimize its average cycle time by crossing the open strip when it is excavating near the middle of the strip to get to the dump, but the dragline operation seeks to operate long continuous strips for maximum efficiency. One compromise is the use of cross-pit bridges that the trucks travel across. Temporary bridges are constructed from blasted overburden and built to the height of the dragline operating level. When the dragline is a suitable distance away from the temporary bridge, the trucks use it to provide a short-haul access across the pit. They are "rolled" by the dragline on each strip to recover the coal that lies beneath them. The downside of temporary bridges is that the bridge is not always available for use by the trucks, or its use by the trucks becomes a constraint on the mine schedule, and the rolling of bridges has a productivity impact on the dragline. An alternative approach is the use of permanent bridges. These are constructed from in-situ material that is left in place in each successive strip. The coal beneath them is effectively sterilized as the successive mined-out strips are filled with overburden that either buttresses or buries the remnant mineral material. The bridges may be built to any height and may incorporate multiple cross-pit roadways at different levels to further optimize trucking operations. The downsides of permanent bridges are the sterilization of resources, which is magnified for higher bridges due to larger bases from the batter effects, as well as their role in limiting the length of strip the dragline operates with the corresponding productivity impact from end effects and more frequent machine relocations.

The rest of the trucked overburden either travels around the end of the pit or up a highwall ramp system to a natural surface and then around to the dump. A major advantage for endwall road systems is the provision of cost-effective on-grade highwalls, with a detraction being additional stripping demand from access roads cut into sterile ground, which can be substantial at depth, especially for short strips. A major advantage for highwall ramp systems is that they



Courtesy of BHP Billiton.
Figure 10.8-6 Spreader dump

are excavated in material that is part of the ongoing mining operation, and generally already exist if coal seams are present within the prestrip working horizon. A detraction is that productivity impacts the excavation operation from removing and re-creating ramps (if not already in place) as well as the potential need for loaded trucks to haul up to a natural surface and then haul downhill on the dump side, which is inefficient.

Placement of overburden by conveyor systems is most similar to truck operations. A spreader constructs the overburden dump. Depending on the equipment configuration, the overburden dump may be built in benches radially or along the strike. An upper stacked overburden bench of 15-m height may be built simultaneously with a lower filled overburden bench of up to 45-m thick before the whole system is advanced. Because of the immobile nature of continuous waste systems, the conveyor will either travel around the end of the strip or across a central permanent bridge. Figure 10.8-6 shows a diagram of a typical spreader dump.

Landform Strategy—All Placement Methods

The waste dump, or landform, strategy is dependent on a number of key factors including the long-term swell factor, operating pit and progressive and final landform geometries, any mining or dumping constraints, the total static volume balance, and the dynamics of optimal dump construction. The influence of the final pit limit and final pit treatment should also not be underestimated, with impacts for deep pits likely to affect the entire postmining landform constructed early in the mine life, but not manifested until the latter years of the mine life. Variations in the elevation of spoil dumps due to swell factor and the inherent displacement by draglines of contour highs and lows may lead to substantial drainage mismatch between disturbed and undisturbed lands. Surface contouring software can generate plans for final restoration that respect geomorphological principles.

Coal Mining

Although the overburden removal process accounts for the bulk of the operating expenditure for a strip mine, the coal itself is the reason for the stripping and provides the revenue to support overburden removal.

The overburden and coal seams are physically coincident at their boundary, so it is inevitable that imperfect mining processes will result in some unplanned removal of coal with the overburden, or loss, and some unplanned addition of overburden to the coal, or dilution. Typical losses and dilutions by mass can range from 5% to 10% with higher losses incurred for thinner seams and dependent on local practices and conditions.

Loss is directly incurred through blasting that can cast some of the coal into the final spoil position to be lost forever, but also blast-enabled through the disturbance of the overburden–coal interface. By using large-scale earthmoving equipment, driven by the paradigm and key performance indicator (KPI) of minimizing unit operating cost, removing the overburden near the coal can easily be biased toward coal loss. An effective remedy can be to set coal recovery as a KPI or deploy a more selective equipment configuration when approaching the overburden–coal interface in the last 1–2 m. This incurs a higher unit cost for overburden removal, but the affected volume is quite small and the total cost impact minimal. Similarly, the coal mining operation is often biased toward loss over dilution due to the high visibility of overburden within the mined coal.

For high-value coking coal where a processing plant is available to remove dilution and the cost of replacing lost coal is high, it will generally be economical to favor dilution over coal loss. For lower-value coal such as thermal coal, which generally has a lower cost to produce and is often crushed and sold without washing, it is usually more economical to favor coal loss over dilution. Some mines reduce dilution by use of a commercial road sweeper on the coal surface to windrow rocky material away from the coal-loading operation.

Coal mining is commonly by loader and truck systems. For thin seams or soft coal it is dug unblasted or may be ripped and pushed by dozers or even graders as part of the mining process. High-value, thin-seam coals have been excavated by tarmac recovery machines called fine graders or special continuous surface miners whose operation combines the fragmentation and mining process. For thick and/or hard seams a square drill-and-blast pattern is employed to fragment the coal prior to mining. Loading equipment used is either a front-end wheel loader, a diesel excavator in shovel or backhoe configuration, or, more rarely, a smaller electric rope shovel for some mines that have particularly thick coal seams. The choice of loading equipment will be driven by the required mining rate and the thickness of the coal seam and its continuity, with excavators in backhoe configuration being the digging tool of choice for highly faulted or banded coal seams, where selective mining is required. Trucks will either be rear dumps, electric or diesel, or belly dumps. Rear dumps have the advantages of being more maneuverable and faster up the coal access ramps compared to belly dumps, as well as being able to handle ramp grades of up to 10%, whereas they have the disadvantage of being slower on the flat with top speeds of 65 km/h compared to 75 km/h for belly dumps. For very long flat hauls (>20 km), and where ramp grades allow, highway-adapted road trains consisting of multiple self-tipping trailers with up to 300-t capacity are used. Figure 10.8-7 illustrates a typical coal mining operation.

Depending on blending requirements, coal may be direct-mined and fed to the wash plant continuously from a number of strips in different pits at once. Alternatively, it may be mined to stockpile for reclaiming later, either by a mechanical stacker reclaimer system or by manned equipment such as a batch-fed process.



Courtesy of BHP Billiton.
Figure 10.8-7 Coal mining operation

Mine Restoration, Maintenance, and Eventual Closure

The objective of mine closure is to return to the wider community a postmining or final landform that is safe, stable, and sustainable. Safety dictates that the final landform does not present a risk to humans or wildlife. Stability dictates that the final landform is both erosionally and geotechnically stable, with erosional stability more challenging to achieve. Sustainability dictates that the final landform reduces and prevents pollution, enhances biodiversity, and delivers a beneficial use with no ongoing liability to the company that owned and operated the mine.

Strip mining by definition disturbs large areas of land, so the greatest impact, and ultimately closure cost and potential trailing liability, relates to the creation and maintenance of the final landform from the open voids consisting of final strips and ramps. Other aspects of mine closure include the demolition and rehabilitation of mine and fixed plant infrastructure, but these are small contributors compared to the landform. In some jurisdictions the final landform is highly prescribed, whereas in others, legislation provides broad guidance within a set of principles. Final landforms can range from voids whose batters have been regraded to voids that have been fully backfilled to the original topographic levels.

In a strip mine it is worth noting that, by virtue of the operating methodology, mine closure in terms of the delivery of a final landform should be seen as a progressive activity that commences the day the mine opens and occurs more or less continuously until the site is fully rehabilitated. For this reason a strategic landform study should be undertaken as part of the initial development study, or soon after, so that the construction of progressive landforms can be undertaken in a cost-effective manner, maximizing the design of truck dumps to emulate as near as possible the final landform design. This will involve the bulk placement of material to progressively deliver the final landform fill, preferential placement of adverse material types such as acid-forming or highly erosive spoils within the spoil mass, preferential storage with progressive use of materials suitable for controlling erosion of external surfaces, and topsoil to aid in revegetation of the regraded spoil mass to deliver the final landform finish.

The other aspect of the final landform is the treatment of the final voids consisting of access ramps and the last mined strip. Significant costs and liabilities can be incurred if closure is not approached in a planned way. However, there are also significant operating benefits to be accessed from developing a comprehensive operating-to-close strategy and implementing it in the last 5 to 10 years of the mine's life. For deep pits it is likely that any perpendicular coal access ramps will have long been closed for dump room reasons, but for shallower pits approaching closure, and where still in use, they can represent a low-cost dump opportunity. Final strip voids become more valuable as a low-cost dumping opportunity for adjacent pits than as a source of marginally economic coal. A mitigated closure strategy will generally see all ramp voids, and most final strip voids, filled with waste from the remaining operating pits. In time these pits themselves will self-fill as a way to reduce unit operating costs and still yield a positive cash margin, albeit with ever-reducing total mine output. Where sufficient area exists and economics allow, some mines have wheeled their pits around in a large set of 90° offsets so that the last pit is adjacent to the first pit and thereby fills it.

Regulating authorities in some countries demand a final landform design and progressive delivery as a condition for operating a mine, whereas in other countries the approach is less prescriptive. Nonetheless, the final landform is an integral part of operating a strip mine, and it is critical that a strategic landform plan is developed early and followed throughout the mine life.

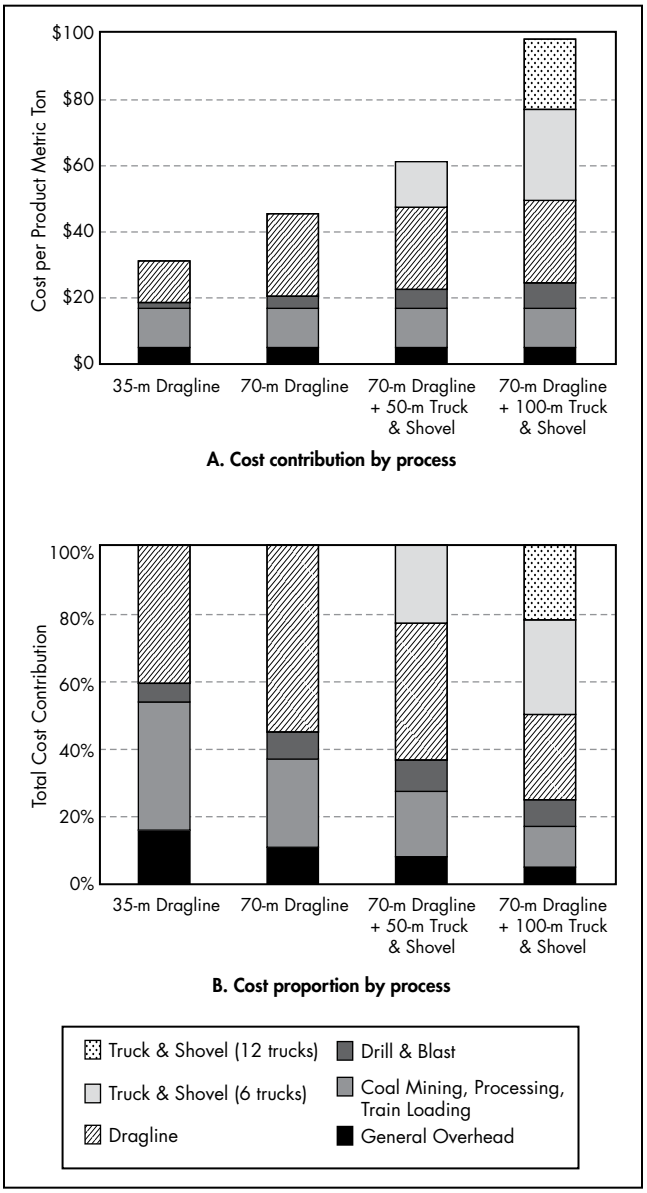


Figure 10.8-8 Relative cost by scenario

Mining companies are able to operate by virtue of the license to operate (LTO) they hold. A company's LTO is a product of its adherence to statutory requirements and the literal permits this provides in combination with figurative permissions provided by meeting the expectations placed on it by the wider community. At its broadest definition, this community can consist of local, regional, national, and international stakeholder groups comprised of individuals, institutions, and organizations. The strength of an open-cast mining company's LTO is directly related to how it manages progressive and final closure and specific mine planning and design aspects such as landform, treatment of final voids, water drainage and quality, and how they ultimately impact postmining land use.

Relative Economics by Business Process and Changes with Depth

Having discussed the key operational tasks that define the strip mining process, it is worthwhile contrasting the relative economics of each, how these relativities change with depth, and the operational implications of these movements. Four scenarios were developed based on a single 10-m coal seam, increasing waste depths, and differential waste allocation by waste-stripping process. Costs include both capital and operating components.

Figure 10.8-8 highlights the contribution waste removal makes to total mine gate cost for the four scenarios—dragline operating productively at shallow depth (35 m), dragline operating at its effective limit (70 m), 50 m of truck and shovel prestrip introduced with relatively short haul (6 trucks), and 100 m of truck and shovel prestrip introduced with relatively long haul (12 trucks). Some key points become clear. For shallow deposits (35 m), coal-related processes make up a significant proportion of total cost, and the operation will be oriented around minimizing coal mining cost. This suggests frequent coal access ramps with coal haul roads optimized for haulage to the processing plant with management focus on the coal mining process. As the seams progress deeper (70 m), drill-and-blast makes up a more significant proportion of cost, and focus will shift to optimizing drill-and-blast operating practices. Dragline operations are now the majority of total cost, and there will be extreme focus on productivity and equipment uptime. Beyond 70 m, as truck and shovel prestrip is introduced, it will work around the dragline process, and its role will be to ensure cost-effective dragline operation. At extreme depths (100 m of truck and shovel, and 70 m of dragline), waste stripping dominates the cost structure with truck and shovel operations now contributing as much cost as all of the other operations together. The coal mining and dragline operations will work around the truck and shovel operation to ensure its cost-effectiveness. The mine design will be modified to optimize truck and shovel at the expense of all else. The scenario also shows the additional cost loading due to task—the additional depth has a linear impact providing that the same haul cycles can be maintained, but simple dump geometry dictates that with depth, not only is there more waste, but it also has to be hauled farther and higher. The additional cost from this increased task is shown in Figure 10.8-8 as the 6-truck to 12-truck cost increment, with the result being an exponential cost increase. At this point, attention will be given to seeking out optimized dumping practices, new dump locations (potentially on top of high-cost coal or future underground areas), and innovative waste movement methods (conveyors being one example).

Consideration at this point should be given to the relative cost of underground operations as an alternative means of generating raw coal for processing. A 4-m-thick seam at around 70-m depth will be exposed by dragline for approximately the same operating cost per metric ton as a longwall operation and for around the same mining capital investment. The choice of approach around this approximate opencut stripping ratio of 12:1 prime to raw coal (9:1 for a pure truck and shovel operation) will be governed by practical considerations. Opencut mining is inherently lower risk and has a wider envelope of application given that underground operations can be severely impacted or even excluded by factors such as geological faulting, intrusions, weak immediate roof, seam gas, seam water, and so on.

Other Operational Aspects

Three other operational aspects are worth discussing in respect to their unique impact on strip mining. These are the storage or disposal of washery rejects and tailings, river and creek diversions, and water management.

Rejects and Tailings Storage or Disposal

Typical product yields range from 50% to 90% by mass for all material fed into the coal washery. Coal washery by-products, the remaining 10% to 50% of the mined coal mass, include coarse reject and wet tailings with the bulk by volume being the coarse reject material. Coarse reject is often stored in close proximity to the wash plant, either in a stockpile on a natural surface or, if available, a completed pit void, and it is placed there either by conveyors or trucks. Tailings are either placed in a conventional wet tailings facility constructed as a dam, or codisposed with reject in either a dam or completed pit void. Wet tailings facilities have large footprints, are high in capital to construct, are expensive to rehabilitate, and carry significant environmental risks both during their operation and post-rehabilitation. For this reason, a completed pit can often be seen as an ideal storage and disposal solution.

Deposit geometries, however, generally do not offer up completed pits early in the mine life, as a strip mine will work the entire strike on a cost equivalence basis for the entire mine life. Little economic delineation exists between pits that are often arbitrary subdivisions of what is usually a single deposit broken up by operational needs such as coal access ramps or creek corridors. More commonly available are final ramp voids left behind as the deep operation has moved to fill the mouth of a ramp as a short-haul dump option. Unfortunately, a competing alternative use for any final voids, either ramps or completed pits, is a potential low-cost dump, and given the cost ratio between horizontal and vertical haul, it is surprising how far away a truck system can reach economically when its alternative is to elevate overburden. For this reason, the best economic use of voids may not be to fill them with wet tailings, especially when water itself for many mines is becoming a scarce and valuable commodity. In these cases, a better alternative may be a partially or fully dewatered tailings paste or cake mixed with reject and disposed directly into the dump mass where it is contained and the long-term liability is minimized. No burial of washery wastes into the reclaimed overburden should occur without a thorough analysis of the wastes' potential for harm to the restored groundwater regime. This may be, among other things, an acid–base accounting or a column test for long-term solubilities.

River and Creek Diversions

Most strip mines are tens of kilometers long with few reasons to break a continuous strip into separate pits other than arbitrary subdivisions based on the needs of the mining operation; infrastructure corridors for road, rail, power, or water pipelines; and so on. It is highly likely that in this long strike there will be a number of water courses traversing planned mining areas. It then becomes an engineering and strategic analysis to weigh up the positive value of the coal that lies below a creek or river and the dump space created by removing a valley through the future maximum dump envelope against the negative value from the initial capital cost of a diversion, the ongoing operating cost to maintain the diversion, a final capital cost to reinstate the water course in or near its original course if applicable, and an estimate of the risk of exposure to a trailing liability to maintain the diversion into perpetuity. The nature of a strip mine means that it is likely that any permanent diversion will be quite long, whereas a temporary diversion can just shift the problem to another pit with the commensurate risk of reestablishing a water course through what is now effectively a postmining landform that is more susceptible to damage from water erosion.

Water Management

Because of their long strike and progressive development over time, mature strip mines can carry large internal water catchments. Environmental licensing conditions relating mainly to salt levels and turbidity can often mean that poor-quality mine water is not easily discharged off-site. For this reason, mine water is stored and either evaporated or partially consumed in the water-based coal processing operation. The large volumes that can accumulate over time or from single intense rain events often mean that a sacrificial pit is selected as the only economically viable water storage solution. The pit may be used for a number of years until it becomes the most attractive pit remaining on-site, at which point another pit is selected for water storage and the first pit is pumped out. After the water is removed, there is often a thick layer of mud in the bottom of the previously dormant pit that also needs to be removed. Mud removal is a slow and expensive process, and if not fully completed before stripping is recommenced in the affected pit, can lead to ongoing spoil stability issues. Massive spoil failure in a deep pit is prohibitively expensive to remediate. It is conceivable that for a very deep pit subjected to a massive failure, it would not be economical to recover the pit because of the large capital cost involved; thus, significant resource sterilization could occur if extraction by underground means is not deemed viable.

ALTERNATIVE STRIP MINING STRATEGIES

The four most common strip mining strategies will be examined as follows:

1. Cast, doze, excavate operation
2. Single-seam dragline operation with prestrip
3. Multiple-seam dragline operation with prestrip
4. Truck and shovel operation

A strip mining strategy may be typified by a unique combination of pit configuration, equipment selection, and operating methodology with the combination essentially a response to the deposit being worked. For this reason, a methodical approach making use of the following discussion framework will be applied to examine each of the four major strip mining strategies deployed in mines today:

- A brief generic discussion of the mining strategy
- A specific mine case study used to examine
  - Pit configuration—mine location and layout and the deposit's geological orientation,
  - Equipment selection—the physical and volumetric scale of extraction, and
  - Operating methodology—an operational description in terms of overburden movement and coal mining
- Reference made to variants of note as applied at other mining operations

Other variants within the presented strip mining strategies are deployed where the specific circumstances warrant it. The combination of final and interseam waste removal and coal mining in thin-seam mines may be achieved by the application of continuous surface miners or graders. Steep-dip mining, as in Colombian thermal coal mines or the United States anthracite district, gives rise to operational variations of the truck and shovel strategy, whereas mountain-top removal

in the United States' Appalachia region is unique and can be considered to be a multiseam modified area mining method.

Cast, Doze, Excavate Operation

"Cast, doze, excavate" refers to a strip mining technique whereby the overburden is cast blast, followed by significant removal by large bulldozers, and the remainder of the overburden is removed using an excavator and trucks, or sometimes by draglines. This method is most applicable to relatively shallow deposits.

In these operations, the overburden blast is designed to not only fragment the overburden to enable removal, but also for the blast to significantly move the overburden into the adjacent open void, reducing the amount to be removed with mining equipment. This style of overburden blasting is referred to as cast blasting. Because of the increased blasting vibrations and noise, the method should not be used near dwellings and residential districts without first determining potential dust, noise, and vibration levels.

Bulldozers are more productive and hence more economically attractive for combinations of relatively short push distances and fairly flat grades. Where bulldozers can be used in this manner they are usually lower total cost than truck and loader systems. In an ideal situation, the bulldozers are used to remove the overburden until the push distance and grade combination is no longer lower cost than the next alternative (either truck and loader systems or draglines). In some situations, cast doze is used to remove all of the overburden material.

Case Study: Groote Eylandt Mine

Groote Eylandt is located off the coast of east Arnhem Land in the Gulf of Carpentaria about 640 km from Darwin, Australia. The mining leases are located on the western plains and cover 84.5 km^2 or 3.74% of the island's area.

Pit configuration—Geological orientation. The Groote Eylandt ore body occurs as a subhorizontal sedimentary layer, gently undulating and dipping to the west. It is a continuous horizon, ranging in thickness from a few centimeters to 9 m, consisting of hard, high-grade, cemented pisolitic and massive manganese oxides. The overburden ranges in thickness from 0.5 to 35 m and consists of lateritic overlying clays and gravels.

The ore body is vertically zoned and can be mined as two distinct layers:

1. A middle mining horizon consisting of massive mangite, cemented mangan-pisolite, or loose mangan-pisolite
2. A bottom mining horizon consisting of siliceous mangite

The beneficiation plant is located in the middle of the mining leases, with 90% of the resource within 8 km of the processing plant.

Equipment selection—Physical and volumetric scale of extraction. The mine typically removes 18 Mbm^3 of overburden each year, comprising 0.4 Mbm^3 of topsoil, 17 Mbm^3 of dozer production, and 0.6 Mbm^3 truck and excavator operation. This total overburden removal exposes 6.9 million t of run-of-mine (ROM) ore to be delivered to the primary crushing station to yield 3.6 Mt of manganese product.

Current mining equipment is a mix of stripping dozers, excavators, and rear-dump trucks. The mining blocks or cuts are 40-m wide and 200-m long, with strips arranged to follow ore-body contours and to control groundwater.

Operating methodology. The Groote Eylandt Mining Company Pty Ltd. (Gemco) mine is a conventional, shallow, opencut strip mining operation involving the removal of manganese ore. The mining sequence is a continuous cycle; areas from which ore has been extracted are backfilled with overburden from the next strip to be mined before then being rehabilitated. This method results in the mining site moving across the ore body disturbing only a small section of land surface at any given time. To meet Gemco's customer product-quality requirements, ore can be mined from a variety of pit sources.

The first step is to clear trees on the planned mining blocks, using a bulldozer with a tree rake. The topsoil is then removed and returned to prepared backfill areas or stockpiled for future rehabilitation.

Overburden is primarily removed using a fleet of Global Positioning System–equipped track dozers that push the material into the adjacent mined-out cut (strip mining technique). The dozers are used to remove all overburden, exposing the ore. Typically, the dozers work in two separate teams, each team working a strike length of 250 m to 500 m and a pit width of 40 m. At times they are used as one fleet to provide peak capacity for short-term ore requirements. Historically, scrapers were also used to remove topsoil and overburden when the overburden was significantly thinner. To reduce costs and handle a wider range of material types, production dozing was introduced. A small amount of overburden is removed using a truck and excavator method, and this method is applied in thicker overburden areas and where digging from above with excavators is used to better handle wet conditions.

A rotary percussion rig is used to drill the ore horizon, and blasting is conducted using emulsions for both wet and dry holes. The blasted ore is removed using a hydraulic excavator. The excavator is also used for overburden stripping for initial or box cuts. Ore is trucked from the pits to the primary crushing stockpiles in 85-t-capacity rear-dump trucks. Some direct haulage also goes to the primary crusher, provided the ore type coincides with the ore treatment campaign through the plant.

Approximately 22 stockpiles are maintained near the primary crusher and categorized according to grade and lump-to-fines ratio. The haulage distance from the pit to the stockpiles varies from 2 to 10 km. In total there are about 30 km of haulage roads that are serviced by ancillary equipment.

Rehabilitation of mined areas started in 1970. Seeds from approximately 25 native forest species are collected from within the mine lease. This seed is stored until the wet season when it is broadcast over areas that have been backfilled and covered with topsoil. Backfill areas are designed to maintain the general premining slopes.

Single-Seam Dragline Operation with Prestrip

For single coal seams that occur deeper than 30 m and where economies of scale allow, a single-seam dragline method is often applied. Although multiple seams may be recovered, for the purposes of terminology where only one seam is directly exposed by dragline operations, it is called a single-seam dragline operation. Because of their specific geometric limitations, efficient dragline operations are dictated by the interaction of the particular deposit geometries and the machine's key working dimensions. A dragline moves overburden by a simple repetitive cycle involving filling the bucket through a combination of pulling in the drag rope and letting out the hoist rope, raising the bucket by the reverse rope movements,

swinging to face the spoiling area, emptying the bucket by releasing the tension on the drag rope, and swinging back to return the bucket to the dig face or bank.

Strip width is typically 50–70 m wide with the strip being placed into the adjacent mining void in discrete excavation blocks of 25–30 m in length, depending on the dragline operating level. A number of different dragline methods are available to place the overburden in the previous void. Selection of method is driven by a number of factors:

- **Deposit geology**—The number of seams to be uncovered by the dragline and the overburden and interburden thicknesses
- **Geotechnical stability**—Certain materials in the strata may be either unsuitable to place the machine on or unsuitable to place in certain locations within the spoil
- **Spoil fit**—The machine will need to sit low enough to reach the top of the coal, but it will also need to sit high enough to ensure it can place the spoil into its final position
- **Productivity**—Will be maximized with lower average swing angles, but swing angle may be less of an issue if there are a large number of hoist-dependent swings (where the swing speed is less than maximum due to the machine waiting on the bucket to be hoisted to height)
- **Scheduling**—At times it may be necessary to adopt a dragline method that may be less productive overall due to increased relocation times but that releases coal sooner and more consistently

Dragline methods are analyzed in two- and three-dimensional (2-D and 3-D) volumetric models as well as simulations. Two-dimensional analyses called range diagrams are used to check for theoretical spoil fit and calculate sectional rehandles. Figure 10.8-9 shows an example of a range diagram sequence. The section is perpendicular to the strip and shows where material is dug from (A), dumped to (A'), and the operational sequence (steps and fill type). Where topography is variable or the influence of certain design features are such that a 2-D analysis is unsuitable, a 3-D check for spoil fit and rehandle rates can be undertaken. In addition to the simple dig-to-dump balance, more sophisticated software models can be used to simulate the dragline machine itself to yield theoretical production rates in cubic meters per hour based on a dynamic analysis to determine the average swing angle and relative frequency of hoist and swing dependent cycles.

A brief overview of the main dragline methods are given in the following paragraphs. A completed range diagram is shown in Figure 10.8-10.

In the side cast method, the overburden is dug from in front of the machine to expose coal and dumped directly in the void from the previous strip (Figure 10.8-10A). It is most often used for shallow overburden thicknesses where the dip is relatively flat and there is no requirement for selective placement of overburden in the spoil. It is also used for multiseam operations for exposing the top coal seam as there is a large void to spoil the overburden into.

The key bridge method (Figure 10.8-10B) is suitable for deeper deposits up to the dragline's ideal operating envelope. Sequentially it involves excavating a block of overburden 30-m long to create the new highwall (the key) and using this overburden to build a bench out into the previous strip's void. The dragline then progressively repositions across the bench, excavating overburden and exposing coal as it goes

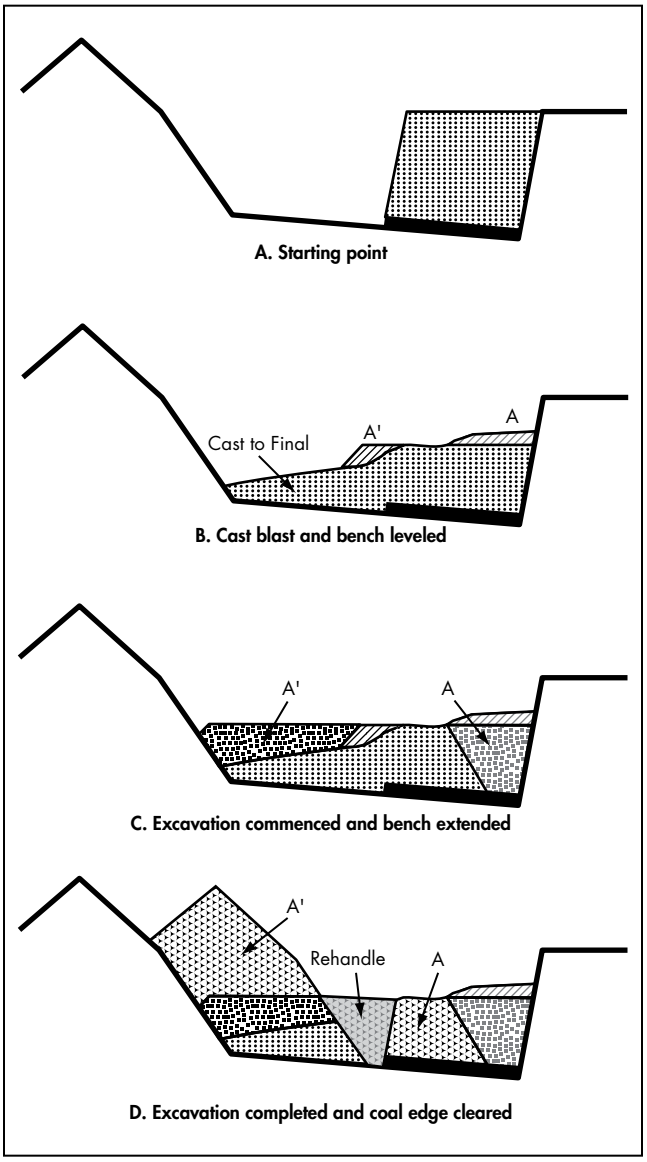


Figure 10.8-9 Range diagram sequence

(the block). After it has uncovered the full strip width of coal, it repositions diagonally down the strip into position to commence digging the next key. The method has higher rehandle but is productive because of the short swings, the previous strip floor can be cleaned up prior to overburden being placed in it, the highwall is dug in-line giving a safe wall, the coal edge is also excavated in-line, and it is a simple method to execute. A variant of this, called the extended key with in-pit bench method (Figure 10.8-10C), involves the key being excavated for 10 or 20 blocks and an in-pit bench built on the low-wall side. At the end of the extended key, the dragline then repositions to the in-pit bench and "pulls" the remaining blocks for the length of the extended key, exposing coal. An advantage of this method is lower rehandle than the key bridge, as the in-pit bench does not have to be filled in completely from the highwall to the low-wall side (no "bridge" required).

In the chop cut with in-pit bench method (Figure 10.8-10D), use is made of throw or cast blasting to move as much of the

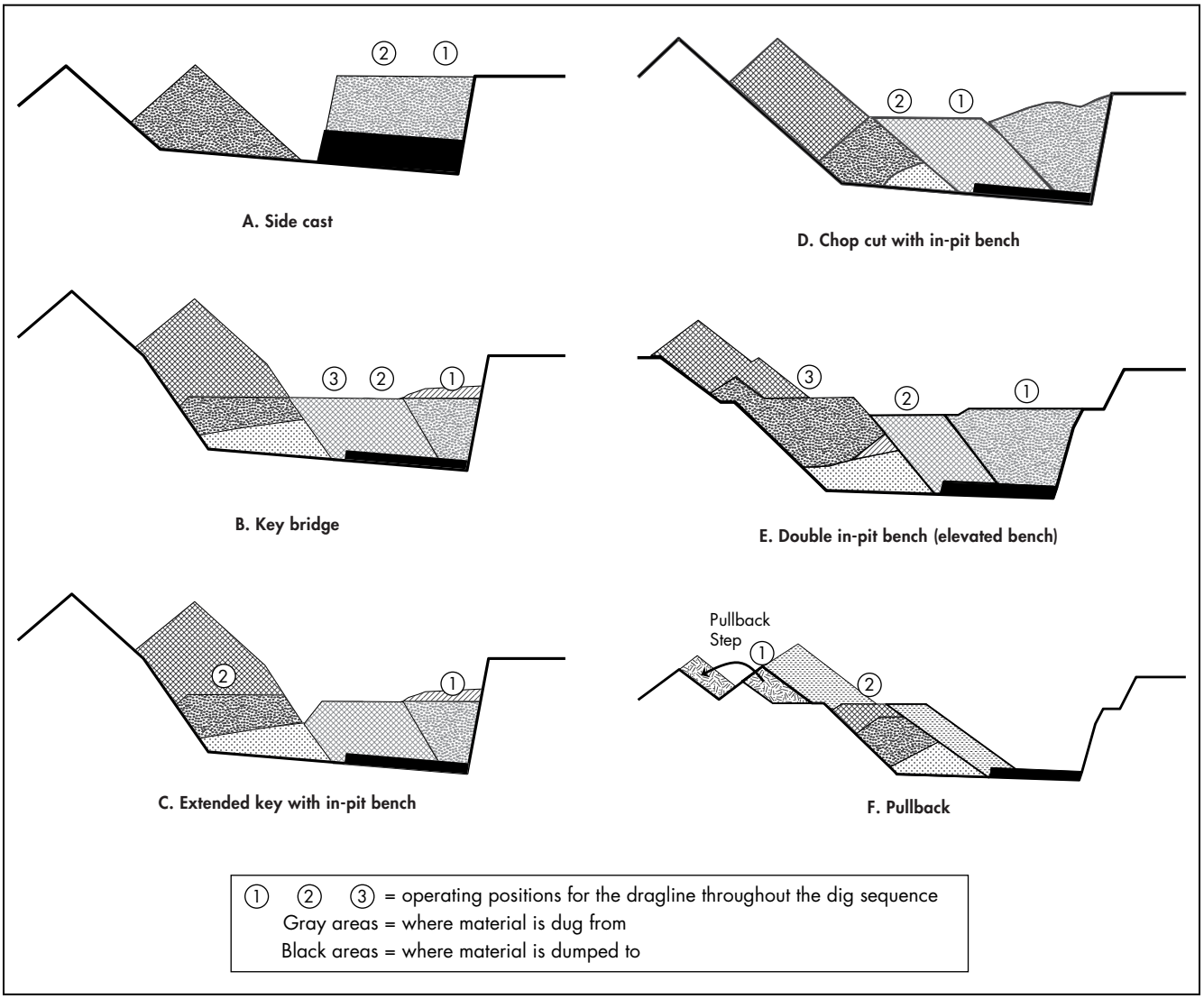


Figure 10.8-10 Main dragline methods

prime material across to the low wall as possible. The dragline excavates the key area along the strip by digging perpendicular to the new highwall and creates an in-pit bench toward the low-wall side of the strip. The dragline then retreats along the in-pit bench and excavates the block that it was previously sitting on and places it in final position. The method has low rehandle and high prime productivity but is a more complicated two-pass operation that gives poor control of highwall.

The double in-pit bench method (Figure 10.8-10E) is used when the depth of material is such that the dragline operating envelope is not sufficient to place all of the overburden into a final position from a single working level. For this reason it may also be known as an elevated bench method. An extended key or chop cut with in-pit bench method is used, but instead of the bench being fully excavated to final position, a partial excavation, or trim, is undertaken and the overburden placed to create another bench at a higher working level. When a suitable length of upper bench is created, the dragline moves onto the upper bench and excavates the remaining waste to expose the full strip width of coal. The higher second bench increases the volumetric capacity of the dragline and so allows the exposure of deeper coal. Walking delays increase, overall productivity is lower, the method is more complex, and coal exposure is more intermittent.

Although not a specific method, the pullback technique (Figure 10.8-10F) involves walking the dragline up into the spoil to pull existing overburden back prior to placing overburden from the current strip. It is time-consuming to prepare access for the dragline, and the pullback material does not expose coal so it is all rehandle, but productivity can be reasonable depending on the volume to be excavated. It is rarely used as part of an ongoing operation because of its high cost, but it can be used in areas where spoil room is particularly tight, such as around ramps or deep boxcuts or to recover from design errors or geotechnical failures. Operationally this method can be delivered by two separate draglines: one on spoil in the pullback mode and one on the bench to be mined.

As the seams progress to greater depths, typically at around 85 m, a truck and loader prestrip operation must be introduced to assist the dragline to continue to reach the top of the coal by creating a working level for the dragline that is lower than the natural topography. Prestrip may have

been introduced sooner than this, however, depending on the deposit and dragline geometry and general economics. Prestrip may be directed to relieve the dragline in tight spoiling areas such as when crossing ramps or endwalls or where the natural topography is variable, or prestrip may be applied more generally across the whole strip.

Case Study: Norwich Park Mine

The Norwich Park hard-coking coal opencut mine is located on the western fringe of the Bowen Basin coalfield in Central Queensland, Australia. The economic seams are contained in the Mid- to Late Permian German Creek Formation, which are overlain by up to 55 m of poorly consolidated cemented sediments consisting predominantly of sand and clay with irregular gravel beds and weathered basalt flows.

The depth of weathering varies from 15 to 25 m north of Leichhardt pit to 25–50 m in the Leichhardt pit and south. Four coal seam groups are present over the mining area with the dip of the sequence to the east and varying from 2° to 10° in some areas.

Equipment selection—Physical and volumetric scale of extraction. Initial mining operations commenced on the subcrop of the Dysart seam(s) in 1979 to uncover coal in strips oriented along the strike of seams. Mining has progressed along the strike and downdip over the intervening period.

Surface features are generally flat with a few ephemeral creek systems running across the deposit. The mining area has been divided into a number of pits with these features incorporated into the layout design. The mine processing plant and other facilities are located on the western side of the deposit in a generally southern location.

Currently, the mine employs a fleet of major mining equipment consisting of six electric walking draglines supported by excavators and rear-dump trucks for waste removal and front-end loaders and bottom-dump trucks for coal mining and haulage.

The coal haulers haul ROM coal to the crushing and processing plant along a haul road network. Total annual product metric tons are typically 6 Mt from a plant feed of 8.5 Mt running at an average yield of approximately 71%. Total annual prime overburden moved is typically 100 Mbm^3 giving a prime-to-product strip ratio of approximately 17:1. Allocation of this prime is mainly to dragline (60 Mbm^3/yr) followed by prestripping (27 Mbm^3/yr) and the remainder to production dozing and minor waste operations.

Operating methodology. Figure 10.8-11 provides a schematic of the mining process representing the typical dragline and truck and shovel activities. Truck and shovel stripping is used to complement and expedite dragline productivity with waste hauled around the dragline operation and dumped on top of dragline spoil. Typically, the truck dump is two spoil peaks behind the current dragline strip.

Single-seam dragline methods are used in the Price and Leichhardt pits, where successive seams are uncovered using trucks and excavators, as well as areas of Campbell and Roper where the upper seams are coked or weathered. The two methods used in the single-seam areas can be broadly categorized as the Jensen and Curtis off-line key bridge method (conventional or extended key), and the extended key and elevated bench method. The conventional Jensen and Curtis method progressively uncovers the full seam width, whereas the extended key does not. The extended key method is used where the height of the elevated bench above the in-pit bench is too great to use the conventional Jensen and Curtis method. This might be the result of an area that has a very low in-pit bench because of the blasting and dozing techniques, or an area where the elevated bench has to be built high to enable the dragline to exit the pit. The latter is often used from the endwall back and is known as a reverse key.

Little difference exists in the planning requirements in regard to range diagram analysis for spoil fit for these two methods. The main difference relates to the timing of the coal exposure. The conventional Jensen and Curtis method has continuous coal exposure whereas the extended key method has sporadic coal exposure due to the need to return to the front of the strip to pull blocks to expose the full seam width.

Jensen and Curtis off-line key bridge method—Leichhardt pit example. After blasting and any dragline prestrip, a dragline access road is formed to the ramp mouth. If time permits, dozers will be used to bulk push waste across the pit and lower the in-pit bench level. The dragline will then be walked to the ramp mouth and the dozer key will be set up to break away from the ramp mouth. This initial waste is spoiled behind the dragline to form the in-pit, or chop, bench. The dragline will then be positioned on the chop line to start working away from the ramp. The dragline digs key material and dumps it into the in-pit bench. The dozer will be cleaning coal and pushing up spoil on the low-wall side at this stage. The dragline will widen, or trim the key, and throw the material to final spoil. The dozer will continue preparing the in-pit bench. The dragline works off the in-pit bench and throws the remaining low-wall side material to final spoil. At this stage the dozer will be working against the highwall, cutting down the key material and cleaning the highwall. For the rest of the strip, dozer push will be used to excavate the highwall portion of the key and expose down to the top of the coal. The progress of the method in sectional view is identical to the example range diagram sequence provided in Figure 10.8-12. Operationally, the method makes extensive use of dozer operations to cost-effectively supplement the dragline operation.

Extended key and elevated bench method—Roper pit example. Using this technique, the floor of the previous strip is cleaned before blasting. Any mud and water is dammed up on the low-wall side. The area is not presplit before blasting. The overburden material is cast blast onto the pit floor with the blastholes drilled at 75°. After blasting, the dragline takes a highwall trim from the surface (Figure 10.8-12A) down to 20 m below the surface and forms a safety bench. The dragline or dozers then prepare an access into the pit and will prepare the chop bench. The dragline will walk into the pit and commence digging off-line extended keys from the ramp mouth (Figure 10.8-12B). The spoil will be used to build an elevated bench. When coal needs to be exposed to suit coal-mining requirements, the dragline will form a ramp up onto the elevated bench and will walk back to the front of the strip to pull blocks, exposing the full width of the Dysart seam (Figure 10.8-12C). Additional low-wall trim will be taken to ensure a stable low-wall angle and a defined low-wall edge offset.

Truck and shovel prestrip. Prestrip is usually conducted by contract. Planned prestrip is the difference between the waste removal requirement to uncover the annual coal production and the installed dragline capacity. Shovel/truck priorities include geotechnical recommendations, spoil relief, and post-strip activities. Free-dig prestrip areas comprise horizons that cannot be drilled because blastholes will not stay open (i.e., they are too sandy).

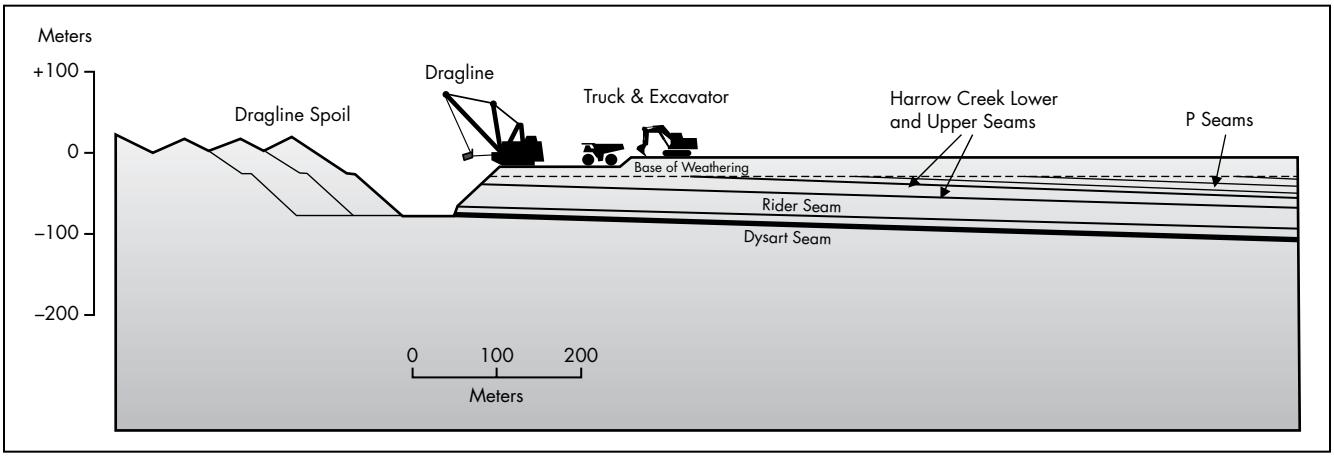


Figure 10.8-11 Cross section of Norwich Park mine

Variants at Other Mining Operations

A number of possible variants of single-seam dragline operations exist. Track dozers may be deployed to assist the dragline operation and speed up coal exposure rates as at the nearby Gregory mine. This mine has also used excavator and truck fleets to remove the dragline key material to accelerate coal exposure. The advantage is that this material is placed outside of the dragline operating area and does not contribute to rehandle, unlike production dozing. Shallow mines in the United States have employed a direct-cast or side-cast method. The operating strip width is narrower and the waste-to-coal ratio is such that the dragline is able to directly place all material into the final position without building working pads and thus incurring rehandle. For steeply dipping seams or highly faulted deposits, the dragline may expose a "false floor" and leave the remaining wedge of material overlying coal to be removed by smaller truck and excavator methods.

Multiple-Seam Dragline Operation with Prestrip

For multiple coal seams that occur deeper than 30 m, and where economies of scale allow, a multiple-seam pit configuration is selected. Within this broad configuration a combination of a number of the single-seam dragline methods is applied to uncover each successive seam. Strip-and-block geometry is similar to single-seam dragline methods, but the geometry dictates significant time and effort to move the dragline up and down between what can be quite different working horizons. A brief overview of the two broad categories of multiple-seam pit configurations—namely, stacked and offset—follows.

Stacked Configuration

In a typical stacked configuration, the dragline removes the overburden in a sequential operation working from the top of the sequence to the bottom. Multiple dragline passes are required with one coal seam being exposed on each pass. Generally, the first pass is a simple direct-cast operation into the void from the previous strip. Subsequent passes will generally involve key or chop operations and extended benching to expose the coal and place the spoil into its final position. The stacked configuration can result in low rehandle (25% or less) and high productivity, especially for narrow strips. One disadvantage is that the lower-pass burden blasts are buffered by upper-pass overburden, and so less cast blast is achieved and fragmentation can be poorer, leading to reduced digging productivity. A second disadvantage is that the material from the upper pass ends up on the floor of the whole spoil pile, may not be suitable to use as a spoil base, and can frequently remain in the base of the pit for extended time between passes where it undergoes further degradation. A third disadvantage is that if lower passes are relatively thin compared to the upper pass, high rehandles can result.

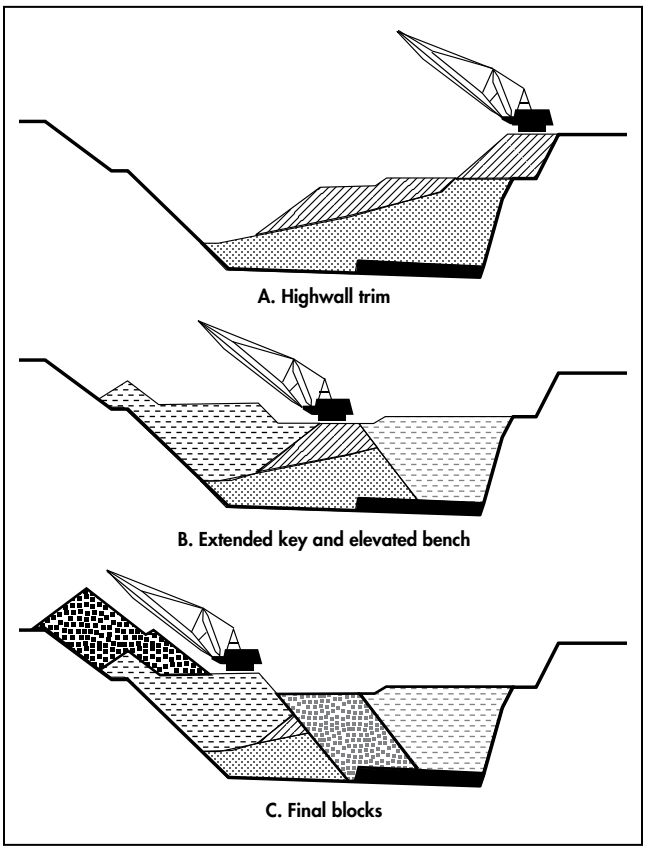


Figure 10.8-12 Extended key and elevated bench sequence

Offset Configuration

In a typical offset configuration, the upper seam is offset at least one complete pit width from the lower seam. This allows

the lower pass to be blasted first, followed by the upper pass, which is generally cast as much as possible to lower the dragline working level. The overall sequence is then very similar to an extended bench method where the extended bench covers the entire lower seam. It is important that the bench level is as low as possible as the bench is built entirely from rehandle. Advantages include the exposure of two coal seams simultaneously, which allows them to be mined at the same time enabling blending. Blast performance is improved for the lower pass as it is not buffered, and scheduling is simplified as the dragline does not require an alternative working area while a seam is mined out and the next pass of overburden is drilled and blasted. Disadvantages are that overall rehandle is generally higher and can increase dramatically with increased total waste allocated to the dragline, and in deeply prestripped mines the large open void requires a significant investment in working capital to establish and maintain as well as placing further stress on the dump volume balance.

The selection of one configuration over the other is dependent on an analysis of many factors:

- Seam dip
- Relative geometry of the upper- and lower-pass overburden and seam thicknesses
- Geotechnical characteristics of the overburden
- Scheduling and blending considerations

Case Study: Goonyella Riverside Mine

The Goonyella Riverside high-quality hard-coking coal opencut mine is located on the western fringe of the Bowen Basin coalfield in Central Queensland. The mine area has a strike length >30 km arranged in two groups of parallel pits representing the originally separate Riverside and Goonyella mines. In the west, pits target the single basal Goonyella lower seam (GLS); in the northeast, pits target all three major seams: the Goonyella middle seam (GMS), Goonyella upper seam (GUS), and the GLS; and in the southeast, pits target the GMS. An underground longwall operation is currently installed in one of these pits and is extracting GMS coal from panels oriented perpendicular to the strike.

Pit configuration—Geological orientation. The Goonyella deposit contains economic seams in the Late Permian Moranbah coal measures that are approximately 300-m thick. The Moranbah coal measures are Mid- to Late Permian age and are characterized by several laterally persistent, relatively thick coal seams interspersed with several thin minor seams.

The Permian coal measures dip east at 3°–6°. The mine area is covered by 0.5–30 m of poorly consolidated Cainozoic sediments consisting of lenses of river channel gravels and sands separated by sandy silts, sandy clays, and clays. The Tertiary silts and clays are densely compacted, hard, and generally dry.

The major seams commonly exhibit decreasing ash content and increasing vitrinite content toward their base and are recognized for their superior coking properties.

Equipment selection—Physical and volumetric scale of extraction. Mining operations commenced on the subcrop of the GMS in 1971 to uncover coal in strips oriented along the strike of the seam. In 1983, mining of the GLS commenced following the commissioning of the Riverside mine updip of the GMS operation. Also in 1983, the Goonyella mine commenced mining the GLS in addition to the GMS in a double-seam operation.

Surface features are generally flat with a few ephemeral creek systems and the Isaac River traversing the southern portion of the mining lease. The mining area has been divided into a number of pits with these features incorporated into the layout design.

Currently, the mine employs a major mining equipment fleet consisting of seven electric walking draglines supported by a number of electric rope shovels, hydraulic excavators, and rear-dump trucks to effect waste stripping. Coal mining is undertaken by front-end loaders and rear-dump trucks. The trucks haul ROM coal to one of two crushing and processing plants along a haul road network. Total annual product metric tons are typically 16 Mt from a plant feed of 20 Mt running at an average yield of approximately 80%. Total prime overburden moved is typically 125 Mbm^3/yr giving a prime-to-product strip ratio of approximately 10:1 for the opencut coal component. Allocation of this prime is mainly to truck and shovel (80 Mbm^3/yr) followed by dragline (45 Mbm^3/yr).

Operating methodology. Figure 10.8-13 provides a schematic of the mining process representing the typical dragline and truck and shovel activities. Truck and shovel waste is hauled around the dragline operation and dumped on top of dragline spoil. Typically, the truck dump is two spoil peaks behind the current dragline strip.

The four northern double-seam pits target the GMS and GLS and account for 70% of opencut production. The GUS is being exposed and mined in the prestrip areas of these pits. The double-seam pits in the north have the lowest strip ratio, and production is concentrated in this area. All double-seam pits have low-wall access ramps. The remaining single-seam pits target the GLS in the west and the GMS in the east and account for the remaining 30% of opencut production. As the single-seam GMS pits advance downdip, they are picking up the GUS in their prestrip operations.

The choice of dragline technique depends on a number of pit-specific conditions. Several dragline techniques are used that can be broadly categorized as

- Single-seam key bridge methods,
- Single-seam key bench methods,
- Double-seam offset methods, and
- Multiple-seam stacked methods.

Although the techniques are long-established for each pit, the basic selection criteria for single-seam methods is whether the low wall is geotechnically stable, with an extended key/bench method used where it is and a key bridge method used where it is not. In the dual-seam areas, the offset method is favored unless the interburden is considerably thinner than the overburden.

Double-seam offset method—Redhill pit example. The first step is to blast the interburden, which is cast blast followed by blasting of the overburden with material spilling over the interburden bench. The dozers then form a working pad on the shot overburden. Sitting on an off-line position, the dragline will then uncover the full width of the GMS. This essentially consists of a combination of off-line key and trim (widening) of that key to expose full-seam GMS. The spoil is cast beyond the GMS coal edge. Ideally, the entire length of GMS in the strip will be uncovered before the dragline has to move onto the in-pit bench. A combination of selective placement with the dragline and use of dozers creates an in-pit bench. The dragline forms and walks across a bridge onto the in-pit bench. This bridge is subsequently removed.

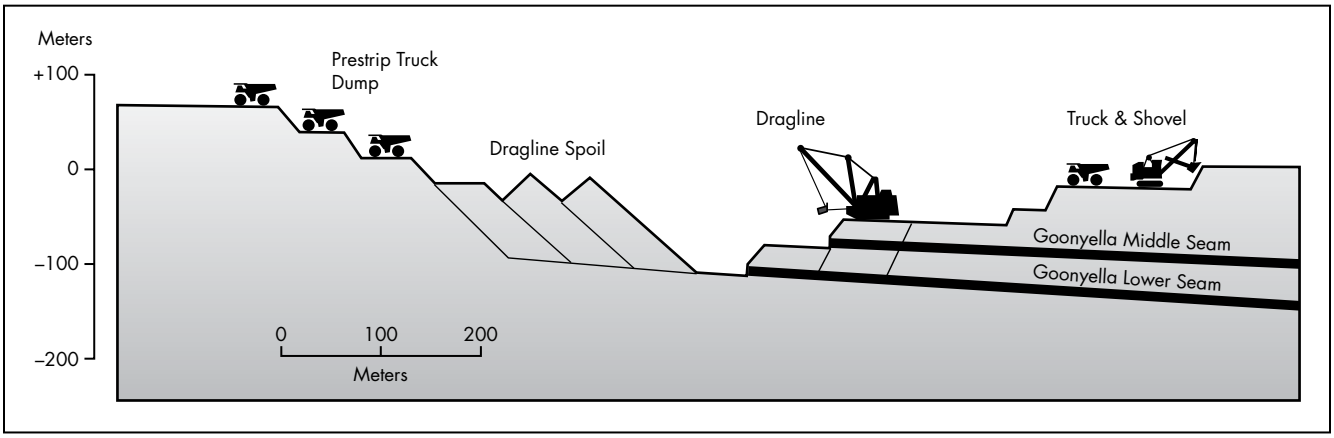


Figure 10.8-13 Cross section of Goonyella Riverside mine

The operation is now the same as a conventional off-line key/elevated bench method. From an off-line position, an extended GLS key and trim is taken, and the spoil is used to build an elevated bench. The dragline walks up onto the elevated bench and pulls blocks to uncover the full width of GLS coal. The GMS coal is accessed from the highwall whereas the GLS coal is accessed by a low-wall ramp at the southern end of the strip.

Multiple-seam stacked method—Thiess pit example. The upper split of the GLS (GLUS) overburden is blasted and a working pad is formed on top of the blasted ground. Full-width GLUS is uncovered by taking an on-line key, then trimming (widening) the key until the GLUS is exposed. This is essentially a side-casting operation. After the GLUS is exposed, it is accessed via a temporary coal ramp that is later removed by the dragline. Then the interburden is drilled and fired. The operation is now the same as a conventional off-line key/elevated bench method. From an off-line position, an extended GLS key and trim is taken, and the spoil is used to build an elevated bench. The GLS blocks are pulled from the elevated bench to expose the full strip width of the lower seam coal. The GLS is accessed by a conventional center ramp.

Truck and Shovel Operation

As previously discussed, truck and shovel strip mining is selected where flexibility beyond that offered by an efficient dragline method is required. Typically, these applications are in more complex multiseam and/or steeply dipping deposits. Truck and shovel is preferred in steeply dipping deposits where waste spoil room in the previous strip is insufficient within the efficient operating envelope of a dragline. Truck and shovel is also used for shorter life operations where there is insufficient mine life to provide a payback for the higher capital investment of draglines.

The two main types of truck and shovel mining operations are conventional along-strike operations, with the pit progressing downdip with each successive strip, and downdip mining or terrace mining, where the strips are excavated downdip usually to the final highwall and then progressed along the strike (effectively at right angles to a conventional along-strike method). The advantage of terrace mining is that the waste material can be short hauled either across the blasted waste or around the ends of the strips to the adjacent waste dump. Typical dimensions for terrace mining in Australian coal mines are 200 m downdip using 70-m strip widths. Terrace mining is very attractive for steeply dipping coal seams where the steep floor conditions significantly reduce the spoil room available for conventional along-strike methods. Examples of terrace mining include Australian coal mines such as Macarthur Coal's Moorvale mine, Peabody's Burton mine, Jellinbah Resources' Jellinbah East mine, and Cockatoo Coal's Baralaba mine.

Case Study: Mt. Arthur North Mine

The Mt. Arthur North mine is an example of a conventional along-strike truck and shovel operation. It is located in the Upper Hunter Valley of New South Wales, Australia. The mine site has a topography that is moderately undulating, steepening near the base of Mt. Arthur. Domestic product is transported via an overland conveyor connecting the mine to the Bayswater Power Station, and export product is shipped from Newcastle.

Pit configuration—Geological orientation. Coal seams within Mt. Arthur Coal's leases are contained within the Wittingham and Greta coal measures, which are separated by a thick noncoal-bearing unit known as the Maitland Group. Coal seams split and coalesce throughout the area, with 20 discrete seams splitting to some 80 unique coal plies.

In the northern part of the deposit, the seams dip from subcrop toward a major monocline at dips of 5° to 10°. Across the monocline, seam dip increases to 10º–25º before flattening again to dips of 2° to 4° in the broad Calool syncline. Indications are that the seams roll over a broad and less-well-defined Denman anticline west of the Calool syncline. To the west of the Denman anticline, seams are down-thrown by the Mt. Ogilvie fault zone.

The in-situ waste strip ratio (cubic meters per metric ton) generally increases to the southwest with current open-cut planning targeting areas of up to a 5:1 strip ratio, which extend downdip to the edge of the eastern monocline. Beyond the monocline, the dip steepens locally and the basal seam plunges to depths >400 m before the seams flatten out.

Equipment selection—Physical and volumetric scale of extraction. The mine is currently capable of operating at a nominal 14.5 Mt/yr, based on 73.5 Mm^3/yr waste prestripping capacity (including contractors) at a 5.1 m^3/t strip ratio.

The choice of mining equipment was dictated by equipment operating at the adjacent Bayswater No. 3 mine—electric rope shovels and hydraulic backhoes, which were progressively transferred to Mt. Arthur North during 2001–2002. Although the rope shovels provided a proven low-cost prestrip capability, their inability to operate on steep dips and their

limitations in shallower interburden resulted in the procurement of additional hydraulic backhoes to mine the "wedge" of overburden that remains beneath the rope shovel floor and the next coal seam as well as interburdens in the 2-to-10-m-thickness range. The rope shovels are generally limited to the "base-of-weathering" pass and the thicker prestrip passes. The current distribution of prestrip by operation type is 35% by rope shovel, 45% by large hydraulic excavator, 10% by smaller hydraulic excavator (coal/parting fleet), and 10% by mining contractor.

An 80-m strip width was chosen to maximize the use of two electric rope shovels while providing sufficient width to operate. Overburden is blasted prior to excavation. Coal is mined using smaller hydraulic excavators and a front-end loader. Coal is "free-dug" in situ. Very thin coal seams may be ripped by a bulldozer and pushed into piles for more efficient loading.

Operating methodology. The mining method is open-pit strip mining, using truck and shovel methodology, spread over multiple prestrip horizons, as shown in Figures 10.8-14 and 10.8-15. The advantages of this system are low technical risk, high resource recovery, and flexibility to change sequencing and production rates as required.

The pit design is based on a strip mining layout where the strips are oriented parallel to the strike. Having fully developed along a 6-km strike length, the mine progresses downdip in 80-m-wide strips. The coal seams dip at between 5° and 8° over the majority of the lease; however, the dip increases in the far southeastern corner to 15°–20° where a bench mining technique and through-seam blasting is used.

In the main pit there are large areas where the seams dip <7°, allowing hydraulic excavators and rear-dump trucks to work on grade. Thicker interburdens are dug by rope shovels and hydraulic excavators. The rope shovel is used to prestrip the majority of the interburden, leaving a remnant wedge for a backhoe to remove. Above 7° dip, it is necessary to reduce the apparent dip and work off temporary benches. In these steep dip areas, track dozers are required to prepare work area pads and/or push down and stockpile material ready for loading.

Material to be excavated is first prepared by blasting or ripping, or it is free-dug. Waste rock is hauled from the highwall to the low-wall side of the mining void to backfill mined-out areas. To cross the mining void, truck haulage is either along the mining strips and via the pit endwalls or via highwall ramps and low-level cross-pit bridges that connect the highwall to the low wall. Although generally a shorter distance, the cross-pit bridge system may necessitate significant de-elevation and re-elevation of material. Most mining blocks will have a choice of the two types of routes, and the one that offers the quicker cycle time is the one that will be selected. A high-level cross-pit bridge is installed between the main pit and the Ayredale pit and functions as an endwall.

ROM coal is mined in situ using hydraulic excavators in both backhoe and shovel configuration, according to the working section thickness. Coal is hauled to the ROM coal receiver hoppers for delivery to the coal handling and processing plant. ROM coal can also be stockpiled nearby to the receiver hoppers and rehandled later as required. Coarse reject from the coal washery is conveyed to an overhead truck loading bin, situated adjacent to the coal receiver hoppers and backhauled by trucks into the spoil dumps on the return haul to the coaling face.

Spoil dumping has migrated from ex-pit dumps during the start of the operation toward in-pit placement. This involves dumping on sloping floors up to 9° and generating a layback of the dump profile for stability. The layback is enhanced by a low-wall ramp and haulage system incorporated into the dump face. Nominal batter heights of 20 m and bench widths of 40 m are used for initial dump design and suitable for standard truck-tipping operations at the crest. This profile of 2:1 is able to be locally optimized to larger batter heights and smaller bench widths where floor conditions are deemed suitable and material composition is homogeneous and competent.

Floor preparation standards are enforced on the Ramrod Creek coal seam floor. This normally involves ripping and removal of silt, water, and remnant coal. Floor blasting has not been required but will be evaluated for future steeper dip areas. Placement of material in the dump is managed, and standards exist for the type of material to be placed on floors or high tip faces. Coal reject and base of weathering material



Figure 10.8-14 Mt. Arthur coal pit operations

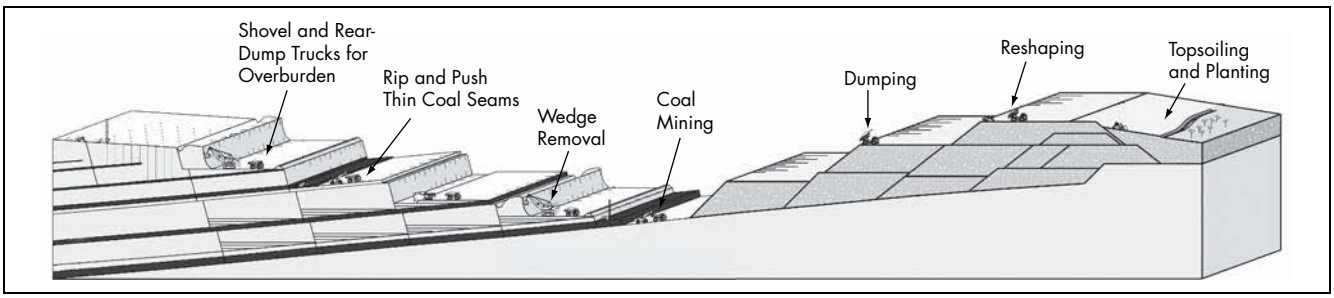


Figure 10.8-15 Mt. Arthur mining method

Table 10.8-1 Strip mining strategies compared by selection driver

Subcriteria	Driver	Strip Mining Strategies		
		Cast, Doze, Excavate	Dragline	Truck and Shovel
		Geology		
Faulting	Complex geology of ore–waste interface impacts, ore recovery, and equipment productivity. Selective mining practices are required.	Small scale of equipment units enables effective operation.	Ancillary equipment required for ore–waste interface. Large draglines can result in significant ore losses.	Flexibility of equipment enables resource recovery but with slower loading rates and some impact on costs.
Material strength	Instability due to low bearing capacity of soft waste material	Dozer production is only slightly impacted due to low equipment ground-bearing pressure. For extreme cases, install wider dozer tracks.	Dragline can sink into soft material. For localized weak materials, dragline can be used to dig off soft material and replace with competent. For more extensive occurrences, a buttress needs to be maintained to provide stable ground for the dragline. Equipment modifications include larger tub and wider shoes.	Shovel is also prone to sinking in unstable material. Equipment modifications include significantly wider tracks. For shallow soft waste, the shovel can be operated on the stable floor below the soft material, and if necessary, dozers can lower the material to a workable bench height.
Slope stability	Unstable material results in flatter highwall and low-wall batters and less-stable working faces.	Negligible impact due to the normal dozer operation having relatively flat highwall and low walls	Flatter working slopes require longer boom and smaller bucket dragline configurations to provide the required reach resulting in higher unit costs.	Fairly insensitive to working slopes with minor increase in haulage distances resulting in very minor increases in unit costs.
		Deposit		
Depth	Waste-stripping cost as a proportion of total mine costs	Optimal cost for shallow depths 40 m or less; deeper than this, dozers become less productive with long and uphill pushes.	Optimal cost for 40–85 m, may need some prestrip assistance; deeper than this requires a significant waste-stripping operation above the dragline.	Optimal cost for waste above 85 m.
Dip	Geotechnical stability and deposit geometry starts to favor certain equipment characteristics.	Suitable for flat to moderate dips; at steep dips, need to cast blast to a false floor and pick through rest of waste to expose coal with secondary truck and excavator operation.	Best for flat to moderate dips; at steep dips, low-wall stability can be an issue and capacity of dragline to reach top of coal and place the overburden in a final position is severely limited. Can expose a false floor and follow up with excavator and trucks.	Ultimate flexibility for all dips. For very steep dips, excavators will pick through waste and coal and may need to load trucks on the same level the excavator is sitting on; less productive.
Seam thickness	Impact of economics for thick seams and driver to selective mining	Ideal for thin seams where dozers and excavators can expose the coal with minimal loss; tool is selective and operator is close to the coal–waste interface.	Best for seams 1 m and greater; thin seams can be severely disrupted by large blasts, and bucket is a large digging tool trying to follow a waste; coal interface many meters away from the operator; excessive losses an outcome for thin seams. Thick seams will be profitable if mine is able to fund capital-intensive draglines.	A thick seam can subcrop economically at a depth beyond the reach of a dragline; leads to deep truck and shovel boxcut; potentially stick with truck and shovel due to impracticalities of getting dragline in and out of a very deep pit.
Number of seams	Waste geometry between seams drives selection of most appropriate method.	Overburden pushed into the low wall needs to end up in a final spoil position away from the coal to be mined; many seams will result in dozed spoil rilling back against the coal edge; will lead to loss or dilution.	Best operating on a few relatively thick waste sections; intensive set-up effort to get a dragline to ramp down into a pit and out again and will prefer to have a reasonable task when it is set up; prefer interburdens of minimum 10-m thick. Multiple seams at the bottom of the pit with thin interburdens mean the dragline will not be able to effectively clear the coal edge, leaving spoil up against it; creates a truck and excavator task to excavate the low wall back.	Best for many thin seams especially where waste between seams is thin or seams are split; is able to take the waste completely away from the coal, leaving it to be recovered cleanly.

(continues)

Table 10.8-1 Strip mining strategies compared by selection driver (continued)

		Strip Mining Strategies		
Subcriteria	**Driver**	**Cast, Doze, Excavate**	**Dragline**	**Truck and Shovel**
Stratigraphy development	Impact of upper seams subcropping at depth	This method is generally limited by depth; however, if upper seams subcrop, they interfere with the effectiveness of the cast operation, with thin overburdens difficult to cast and upper overburdens potentially buttressing lower interburdens and reducing overall cast effectiveness.	As more upper seams enter the sequence, the whole mine, driven by the downdip advance of the dragline, will actually generate more coal; one option is to expand output, but if this is not economical, the choice is either to lift draglines up in the sequence and abandon lower seams, raise dragline level to reduce advance (give more waste to the dragline), or, if it is a multiple dragline mine, to stand some draglines down as no longer required as the mine transitions to source more of its coal from the overlying truck and shovel pit.	Working multiple subcrops at once can lead to spoil from the downdip pit being placed on top of future potential resource as the updip pit catches up; truck and shovel systems provide the flexibility to place overburden more selectively to allow updip pits to mine through more effectively.
Size	Resource size will determine the ultimate scale of the mine to be constructed.	Optimal for small deposits that cannot justify large capital expenditures with long payback periods	Optimal for large long-life deposits seeking to generate large volumes for a long period of time where capital payback occurs. Comes in large lumps of capacity, so potentially need to supplement with truck and shovel stripping capacity. Ends up as either prestrip or a satellite truck and shovel pit only.	Highly scalable equipment that can be selected to match the deposit, both initially and also as it develops over time.
Topography	May influence choice of equipment and therefore mining method	Will affect balance between cast, doze, and excavate.	Highly variable topography is problematic for dragline; will generally lead to supplementary truck and shovel stripping.	This is the most flexible mining system, but truck grade limitations and variable topography can result in waste being brought forward in the schedule to access upper benches. Dumps can also lose capacity if topography is falling away on the dump side.
Geometry	Physical constraints to mining operations will influence selection of equipment and therefore mining method.	Highly flexible and well suited to variable deposit geometries	The preference is long, straight strips; short strips lead to lots of walk time and reduced productivity. Bends in strips can lead to inside corners that create spoil tight spots; this can be alleviated with truck and shovel assistance. It is very difficult to extend the strip after the mine has been developed to depth, essentially trying to dig a deep boxcut; this generally needs to be done with a truck and shovel. Vertical development to pick up deeper seams is a similar problem; a truck and shovel is needed to develop.	A coal seam that subcrops at the lease boundary means that the boxcut may need to be constructed by truck and shovel and the waste hauled elsewhere. Multiple valleys through low-wall dump mass means that for pits deeper than 150 m, total dump balance will come under stress; this may favor a move to hauling to highwall side and other out-of-pit locations; or for really long hauls, a conveyor waste system may be more economical.
Location	Lack of supporting infrastructure may influence choice of equipment and therefore mining method.	Smallest equipment and easiest to support from an infrastructure point of view	A lack of availability of electrical power, in total or reliable supply, may preclude use of draglines. It may be difficult to bring large equipment to location. A lack of specialist operating labor and technical support labor may also detract from draglines.	A lack of electrical power may preclude use of electric rope shovels and lean toward diesel over hydraulic excavators. A lack of specialist technical support labor may also drive toward smaller equipment.

(continues)

Table 10.8-1 Strip mining strategies compared by selection driver (continued)

		Strip Mining Strategies		
Subcriteria	**Driver**	**Cast, Doze, Excavate**	**Dragline**	**Truck and Shovel**
		Environment		
Noise	Noise limitations on operation due to proximity of third parties	Tracks and engine revolution are main sources, but activity is mostly in-pit, hence less off-site impact.	Significant noise from bucket and rigging. Dragline regularly dumping above natural surface and can carry off-site. At worst, may exclude draglines from sensitive areas close to third parties.	Significant noise from engine revolutions on acceleration and tipping, horn signals, and reversing alarms. Trucks are often dumping at height and locations remote from the active pit near the mine boundary. This can be reduced by technology ("whisper quiet" trucks). For sensitive areas, a limit on traffic intensity and hours of operation can be required.
Dust	Proximity to third parties	Minimal as mostly slower material speed and restricted to in-pit, therefore less impact	Significant due to dumping at height driven by dragline geometry	Significant with dust generated at the load point, haul roads, and final dump. Dust is usually controlled by adequate work area and haul road watering. Controlling dust generation from large elevated dumps is a challenge.
		Economics		
Capital	Capital cost per bank cubic meter	Lowest with shortest lead times due to factory line production of equipment	Highest and longest lead times due to single unit fabrication and on-site erection	Mid-range with variable lead times (1–3 years) depending on supply–demand balance
Labor	Total cost of labor and accommodation per bank cubic meter	Mid-range with typically 10 operators per operating shift for 15–20 Mbm^3/yr capacity	Lowest with typically 2–3 operators per operating shift for 15–20 Mbm^3/yr capacity	Highest with typically 15 operators per operating shift for 15–20 Mbm^3/yr capacity
Other costs	All other operating costs per bank cubic meter	Mid-range	Lowest with largest cost driver being electricity	Highest with largest cost drivers being fuel, tires, and maintenance
Expansion capability	Ability to increase production incrementally	Additional units add small capacity steps.	Minor step increases possible via motor upgrades; beyond this, additional units provide large-capacity steps.	Additional capacity by adding trucks up to loader; beyond this, additional units provide large-capacity steps.

Table 10.8-2 Forces for change and their impact on strip mining

Fundamental Force	Drivers for Change	Potential Outcomes
Health and Safety Focus • Regulators demand safer mines • Shareholders demand safer mines • Mine operators and mine workers demand safer mines	• Continuous reduction in injury rates demanded by all • More research reveals adverse long-term health outcomes from shift work and other mining-related activity • More risk of injury for aging work force	• "Smart" mining equipment and maintenance tools detect unsafe practices and require third-party risk assessment and lockout override before proceeding • "Smart" personal monitoring system detects unsafe behavior and raises central alarms and automatic notification to wearer and supervisor • Remotely operated, semi- or fully autonomous equipment with robots undertaking routine maintenance
Environmental Concern • Global warming • Climate change • Land degradation • Biodiversity	• Public perception of greenhouse gas–intensive companies and their products; remoteness of most mines does not deter environmental action groups from focusing public attention • Carbon taxes and trading • Water scarcity with users competing for allocations (domestic, industrial, and agricultural); mine discharge water quality becomes higher profile as background or natural flows reduce • Food security becomes topical with opencut mining targeted as an activity that degrades prime agricultural land	• Time-based Google Earth with enhanced image-processing tools means anyone on the Internet can monitor net land disturbance and rehabilitation deficits, leading to more focus on progressive rehabilitation • Electricity generated by nuclear power potentially tips economics toward electrically driven equipment, leading to emergence of widespread use of electric excavators and electric trucks using overhead cables; alternatively, more use of conveyors to move waste • Water collected from across the whole mine site at closure and directed into final voids that have been left open as large catchment dams, leading to treated water sold to local users • Underground mining becomes only form of mining allowed in areas of, or near, high-quality agricultural land • Carbon taxes tip economics heavily in favor of underground mining due to much-reduced energy expenditure
Social Forces • Urbanization and aging work force • Environmental values	• Increasing service gap between metropolitan and regional or remote areas; harder to attract highly skilled personnel to remote areas • Societal pressure for increased external control of licenses and leases (the social license)	• More-prevalent fly-in/fly-out as the standard model • Number of personnel on-site an absolute minimum • Real-time terrain acquisition means most geology, survey, engineering, as well as equipment performance and condition-monitoring tasks occur off-site in centralized technical centers • Region must be persuaded of sustainable development before it awards the social license
Technology • Automation • Increased equipment scale	• Need to eliminate safety incidents • High cost of labor and associated costs including accommodation • Increasing strip ratios for deeper strip mines • Shortage of suitable experienced personnel	• Less human exposure to accidents • Fully automated operations: truck–shovel fleets, draglines, and production dozing • Reduction in on-site labor requirement to maintenance and control functions • Significantly higher productivity • Lower maintenance costs as more predictable operating practices • Outcomes would have the largest impact on truck and shovel costs
Market Forces • Continued globalization • China/India industrialization	• Vertical integration for commodity security; commodity consumers buy and develop mines • Supply-side consolidation; companies continue to aggregate to pursue improved economics • Long-run prices reflect relative scarcity of energy and metallurgical coal in the face of sustained increasing demand but with short-run volume and price volatility	• Marginal undeveloped deposits become economical; thinner seams, so more likely to be truck-based mines • Existing deposits economical at greater depths • Huge variation in prices in short-term timeframes means there are no final highwalls; major mining companies exit mines and they are purchased by equity hedge funds trading on cash generated between care and maintenance and incremental marginal extraction cycle • China becomes a major manufacturer of draglines and earthmoving equipment; much reduced-capital pricing fundamentally shifts relative economics of equipment

must be incorporated into the dump tip faces and blended with stronger material to minimize their negative stability impact.

STRIP MINING STRATEGIES COMPARED BY SELECTION DRIVER

Having outlined the four major strip mining strategies, it is useful to explore the relative suitability of each strategy in a given mining application. Four significant decision criteria have been identified: geology, the deposit, environmental considerations, and economics. For each of these, a number of subcriteria and their respective drivers and influences on each strip mining strategy are summarized in Table 10.8-1.

FUTURE TRENDS

This chapter has outlined a range of pit configurations, equipment types, and operating methodologies that represent the breadth of current strip mining practices. Although the specifics of the future are uncertain, one thing is definite—change comes. In closing out this topic, Table 10.8-2 identifies some of the major forces at work today, comments on the drivers for change they introduce, and speculates on their potential outcomes.

The past 100 years in strip mining has seen the emergence and continuous development of powered earthmoving equipment. Today's trucks move >300 t/cycle and are filled by large hydraulic excavators or rope shovels in as few as three passes. Electric walking draglines have a dig-to-dump range >100 vertical meters, a similar operating radius, and move more than 200 t per 1-minute cycle. Advances have primarily come through scale (enabled by improved materials and engineering design), application of low-cost and mechanically reliable electric motors, and technology generally for improved mine planning, equipment operation, and condition monitoring. The result has been existing strip mines that remain economical at depth and previously uneconomical deposits now within reach. Recent rapid development in the areas of geospatial awareness, image processing, real-time communications, and general computing power suggest that the industry is on the brink of significant breakthroughs in remote and autonomous equipment and even whole-of-mine operation. A step change in the relative economics of conveyors over trucks could see conveyor waste systems deployed extensively in those mines where the deposit allows. These changes will fundamentally shift how strip mines are designed and operated with an accompanying continual improvement in mine safety and reduction in cost while, as always, improving underground operations will continue to offer a viable alternative.

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CHAPTER 10.9

Strip Mine Planning and Design

Ernest T. Shonts Jr. and Jerry M. Nettleton

INTRODUCTION

Modern strip mining began with the development of the steam shovel used in the excavation of the Panama Canal. However, underground mining continued to dominate the extraction industries until the early 1900s, when strip mining proved itself to be more economical for near-surface, flat-lying deposits.

Strip mining is a form of open-pit mining that uses strip cuts to mine generally shallow, flat-lying deposits. Strip mines are typically layered deposits—a classic example is surface coal mining. This chapter will concentrate on considerations in the mine planning and design process of strip mines. The primer for mining engineers is *Surface Mining* (Pfleider 1972); see Kennedy (1990) for the updated second edition of that text.

Mine Planning and Design

There are two general approaches to mine planning. The first is the development of a greenfield or new property. Capital investments have not been committed; the property may already have been acquired or is in the process of evaluation. The prefeasibility study establishes the value of the property and potential capital commitment. Prefeasibility may require a general mine plan to establish if mining is practical and may go as far as suggesting what equipment would be used. The feasibility study then details the capital commitment and life of the property, and includes a detailed economic evaluation. To develop economics in terms of cash flow, expected net present value (NPV), and internal rate of return (IRR), the feasibility study requires a detailed mine plan, equipment selection, schedules identifying the timing and amount of capital expenditures, and estimates of projected revenue and costs. Information on mining economics can be found in Stermole and Stermole (1987). Table 10.9-1 is a list of items to consider in a mine plan feasibility study.

An existing property or brownfield expansion considers an operation where equipment and facilities are already in place and future development is planned. Mine planning for existing operations may require a different approach to address market conditions and the need to expand, scale back, or maintain a steady-state production level.

In an existing property, where equipment and facilities are established, the two stages of mine planning are the short-term mine plan and the long-term or (life-of-mine) plan. These stages are significant in that mines often assign designated staff and instigate significant mine planning efforts to best meet the differing requirements and level of detail for these two functions. See Table 10.9-2 for a list of items to consider in the mine planning process.

Short-Term Mine Planning

The short-term mine plan (or operating plan) generally relates to a period of less than 5 years and is typically broken down into stages of monthly, quarterly, and annual operating plans. Each stage reflects a level of detail designed to guide the mining toward fulfilling annual production and economic budgets. Short-term mine plans consist of details and sequences for drilling and blasting, quality control, and material excavation and placement to be used by operations to direct equipment operations and meet production goals.

Long-Term Mine Planning

Long-term mine plans usually cover a period greater than a year beyond the current active operations, up to and including the point when the life-of-mine plan starts. Typically, these long-term mine plans are broken down into levels of detail that include quarterly plans for up to 5 years, annual plans out to a minimum of 10 years, and general plans out to the life-of-mine planning horizon. The defined planning periods can vary depending on the size and life of the reserves, along with the general purpose and objectives of the operation. The objectives of the long-term mine plan include identifying and scheduling labor levels, equipment production levels, and capital requirements (including major equipment purchases, infrastructure expansion, reserve expansion, acquisition of new reserves, and end-of-life reclamation). Other applications of long-term planning are to determine production sensitivity to revenues and costs (which would affect production rates),

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Table 10.9-1 Salient factors requiring consideration in mine planning and feasibility studies

Information About Deposit	General Project Information	Development and Extraction	Economic Analysis
Geology: overburden • Stratigraphy • Geologic structure • Physical properties (highwall arid spoil characteristics, degree of consolidation) • Thickness and variability • Overall depth • Topsoil parameters Geology: coal • Quality (rank and analysis) • Thickness and variability • Variability of chemical characteristics • Structure (particularly at contacts) • Physical characteristics Hydrology: overburden and coal • Permeability • Porosity • Transmissivity • Extent of aquifer(s) Geometry • Size • Shape • Attitude • Continuity Geography • Location • Topography • Altitude • Climate surface conditions (vegetation, stream diversion) • Drainage patterns • Political boundaries Exploration • Historical (area, property) • Current program • Sampling (types, procedures)	Market • Customers • Product specifications (tonnage, quality) • Locations • Contract agreements • Spot sale considerations • Preparation requirements Transportation • Property access • Coal transportation (methods, distance, cost) Utilities • Availability • Location • Right-of-way • Costs Land and mineral rights • Ownership (surface, mineral, acquisition) • Acreage requirements (on-site, off-site) • Location of oil and gas wells, cemeteries, etc. Water • Potable and preparation • Sources • Quantity • Quality • Costs Labor • Availability and type (skilled, unskilled) • Rates and trends • Degree of organization • Labor history Governmental considerations • Taxation (local, state, federal) • Royalties • Reclamation and operating requirements • Zoning • Proposed and pending mining legislation	Compilation of geologic and geographic data • Surface and coal contours • Isopach development (thickness of coal arid overburden, stripping ratio, quality, costs) Mine size determination • Market constraints • Optimum economics Reserves • Method(s) of determination • Economic stripping ratio • Mining and barrier losses • Burned, oxidized areas Mining method selection • Topography • Refer to previous geologic/ geographic factors • Production requirements • Environmental considerations Pit layout • Extent of available area • Pit dimensions and geometry • Pit orientation • Haulage, power, and drainage systems Equipment selection • Sizing, production estimates • Capital and operating cost estimates • Repeat for each unit operation Project cost estimation (capital and operating) • Mine • Mine support equipment • Office, shop, and other facilities • Auxiliary facilities • Labor requirements Development schedule • Additional exploration • Engineering and feasibility study • Permitting • Environmental approval • Equipment purchase and delivery • Site preparation and construction • Start-up	Short-range and long-range production plan Cash flow • Revenue • Capital • Labor • Operating costs • Royalties • Taxes Evaluation • Risk reward • Net present value • Internal rate of return

Source: Hrebar and Atkinson 1998.

capital projects, and the life of the reserve. Both short- and long-term mine plans are decision tools to help manage the risk and reward opportunities that confront mine management.

Risk and Reward

The successful implementation of a correctly formulated mine plan will yield the optimal recovery of the resource, as measured by optimizing the resulting revenues (reward) and effectively controlling costs (risk). Costs for a new property can vary substantially because of unknown conditions. It is the mining engineer's task to identify and characterize key conditions and factors, and to assign costs to them that realistically represent the risks involved in developing and safely mining the reserve. It is also the mining engineer's function to get the most value from the reserve by using proven methods to mine and produce the resource in a timely manner. Determining the time value of money—or the NPV of a project—is the most common method of measuring the economic balance between risk (costs) and reward (revenues) for a mine development project.

A base-case mine plan is typically developed to identify and quantify the costs and revenues associated with developing the property. Successive planning scenarios may then be developed to evaluate the cost/revenue effects of changing mine plan layout, sequencing, production schedules, equipment, or other parameters to minimize cost impacts and maximize the resource recovery and revenues to optimize NPV.

OUTLINING RESERVES

Mapping

Project mapping often begins with a general location map that shows geopolitical boundaries, towns, roads, general topographic features, the mine location, and any nearby residences or structures. Planning starts with more detailed mapping of topography, surface drainage features, surface and mineral

Table 10.9-2 Major steps in surface mine development preceding production buildup and full production*

Assembly of Mineable Coal Package	Market Development	Environmental and Related Studies	Preliminary Design, Machine Ordering	NEPA Process	Permits	Design and Construction	Mining Preparation
Lease acquisition	Market survey	Initial reconnaissance	Conceptual mining development	Identification of lead agency for EIS	State water well rights appropriation permits	Preliminary design and estimation	Stripping machine(s) erection
Mapping the area	Potential customer identification	Scope of work development	Economic size determination	Draft EIS	State special use permit, such as a reservoir	Material ordering and contracting	Loader erection
Drilling program	Letter of intent to develop and supply	Consultant selection	Mining system design, layout, and development	EIS review and comments	State mining permit	Water well development	Support equipment readying
Surface drilling rights acquisition	Contract negotiation	Implementation	Equipment selection	EIS hearing and record	State industrial siting permit	Access road and site preparation	Labor recruitment and training
Drilling, sampling, logging, analysis		Environmental impact report	Stripping machine ordering	Federal EIA review	Federal NPDES permit	Railroad construction	
Mineral evaluation (determination on commercial quantities present)		Environmental monitoring	Mine plan development	Council on Environmental Quality filing	U.S. Forest Service special land use permit	Power supply installation	
Drilling on closer centers (development drilling)				Mining and/or reclamation plan approval		Facilities and coal handling construction	
Sampling, logging analysis						Warehouse building and yards	
Surface acquisition						Coal preparation and loading facilities construction	
						Overland conveyor construction	

Source: Jones 1977.
*EIA = Environmental Impact Assessment; EIS = Environmental Impact Statement; NEPA = National Environmental Policy Act of 1969; NPDES = National Pollutant Discharge Elimination System.

ownership, existing facilities and infrastructure, and potential mine area and disturbance boundaries. Aerial flights provide an up-to-date surface contour of the property and help to identify existing structures, facilities, and other surface features. Ground surveys using Global Positioning System equipment provide additional detail and will become the basis for the ongoing process of mapping and documenting mining and related activities. Infrastructure includes the location of existing features—roads, rail lines, pipelines, power lines, communication lines, wells, buildings, and structures on and adjacent to the mining property—that may be used for or affected by mining activities. It is important to identify not only features related to the mining activities, but also features and resources that may require special environmental considerations, such as streams, forest land, wetlands, wildlife habitat, and other protected or environmentally sensitive areas. Initial geological maps should include surficial geology (outcrops, surface geologic exposures, structural features), and indications of strike and dip of structures. Maps prepared by governmental research organizations such as the U.S. Geological Survey are a typical starting point for new operations. Conversion of GPS surveys to a geographic information system format will allow the ongoing collection and integration of ownership, environmental, geologic, and mine and reclamation planning and operational information.

Since modern mapping is computerized, map information is developed and saved as individual layers that can then be selected and combined to produce a range of specialized maps. Although it is important to define and establish a common mapping datum for compatibility, determining a map scale is not critical because scale can be adjusted later to meet map presentation requirements. After a standard mapping datum is established, all mapping should be completed and input to that datum. The mapping datum and coordinate system should be compatible with the maximum area expected to be affected by mining and should accommodate both short- and long-term planning. It is important to document the basis for the mapping information (metadata) and to preserve original data in a digital format with appropriate backups.

Drill-Hole Coverage

Sufficient drill-hole information is critical for the development of geologic and mine planning models. If a company has a history of drilling in the area, then all known drilling information should be reviewed and evaluated for reliability and content. Information is then consolidated into a geologic computer model. In a new operation (or one that is expanding into a new reserve area), a drilling program will be required to collect enough information to adequately evaluate the property. A resource is typically defined by the level of confidence in the occurrence and extent of the mineable seams. In open-pit mining, the terms *measured*, *indicated*, and *inferred* convey both the extent of drilling and the level of confidence in the reserve characterization. In surface strip mines, the deposits

tend to have consistent thickness characteristics; therefore, drill spacing for exploration often involves a grid spacing of 0.4 to 0.8 km (0.25 to 0.50 mi). Areas that require more extensive drilling (or seismic studies) are seam boundaries, structural features, and surface disconformities such as valleys, surface displacements indicating faulting, and depositional unconformities. The greater the variation in reserve characteristics, the greater is the need for increased drilling density or supplemental data collection. Data quality can be evaluated and supplemented using geostatistics techniques, which take into consideration variances in the ore body when drill data are being correlated. In operating mines, drilling information can be supplemented by structure and quality information obtained from drilling and blasting activities in active operating areas. The engineer should become familiar with the reserve and should identify where structure, thickness, and quality issues may exist in planned mining areas.

Geology to Mine Planning

Using the drill-hole information and surface structural features, a geological model is developed. This model should include seam identification, structure, thickness, quality, and characteristics that may affect mining efficiency such as overburden/interburden thickness and characteristics, fracturing, variations in thickness, washouts, and water table. The engineer should be looking for inconsistencies in quality and thickness to determine the extent of mineable seams and the mining characteristics that would affect equipment selection and productivity. It is also important to understand overburden/interburden characteristics and structural constraints as indicators of potential stability problems. Information on highwall angle and spoil angle can be estimated by material type and stability characteristics. Thickness, strike/dip, strip ratio, and quality maps should be developed for all potentially mineable seams. This can either be a fairly straightforward process, or more complex and time-consuming if seams combine and split. Transitioning from a geologic model to a mining model requires the understanding of reserve characteristics, potential structure hazards, equipment capabilities, and strip mining methods. Geological cross sections, history and knowledge of the area, research on similar deposits, and knowledge of mining methods are all resources that the engineer can tap into in order to optimize mine planning.

The basis for a good mine plan is a good understanding of the topography and geology of the property. To facilitate this, an accurate topographical map must be developed with a minimum 6.1-m (20-ft) contour interval, and a grid base must be used that has a minimum spacing of 61 m (200 ft) and covers all of the projected mining area along with potential facilities and any significant surface structures. In addition, it is necessary to become familiar with the drilling information, including seam characteristics, structure, and the quality information for each mineable seam (referred to as target seams). Contour thickness maps (also known as isopach maps) should be generated for each of the targeted seams and should include information on overburden, interburden (waste material between targeted seams), and the undiluted thickness of the targeted seams. Quality maps should also be created for each targeted seam. Geologic structure maps need to be developed to identify any structures (e.g., faulting and offsets) that would affect the mining method or decrease recovery and increase dilution and waste.

Coal loss due to ribs, barriers, top/bottom of coal seam dilution, and overblasting can be as high as 10% in eastern operations (Anon. 1977).

Reserve Evaluation

Modern mine planning starts with a computer model of the resource. The extent, structure, quantity, and quality of the resource and the associated burden material should be well defined. These data will allow the engineer to develop a mining model. The mining model will incorporate and consolidate data on the resource into a model of mineable seams, waste, and soil materials (sometimes referred to as suitable plant growth material). For this purpose, a reserve is identified as a resource that is delineated by its economic strip ratio.

Strip Ratio

The economic stripping limit is usually the first factor to be determined in establishing the mine plan. The economic stripping ratio is defined as the cubic meters (yards) of waste material to be removed to uncover one metric ton (short ton) of product. For illustration purposes in this chapter, coal will be used as the resource. Developing maps that show the ratio of overburden thickness to a mineable coal thickness is a good place to begin. This ratio can be converted to a strip ratio map by mapping the thickness of the overburden and interburden converted to cubic meters and divided by the thickness of the coal converted to metric tons (short tons). This ratio is calculated using the cumulative thickness of both overburden/interburden and coal seams down to and including the lowest mineable seam.

For example, for a 61-m (200-ft) cumulative overburden/interburden thickness with a total coal thickness of 6.1 m (20 ft), and assuming the average density of bituminous coal at 1.28 t/m^3 (80 lb/ft^3), the stripping ratio would be calculated as follows:

$$\text{burden} = 200 \text{ ft} \times 3 \text{ ft} \times 3 \text{ ft} \div 27 \text{ ft/yd}^3 = 66.7 \text{ yd}^3$$
$$\text{coal resource} = 20 \text{ ft} \times 3 \text{ ft} \times 3 \text{ ft} \times 80 \text{ lb/ft}^3 \div 2{,}000 \text{ lb/st} = 7.2 \text{ st}$$
$$\text{stripping ratio} = 66.7 \text{ yd}^3 \div 7.2 \text{ t} = 9.26{:}1$$

or

$$\text{burden} = 61 \text{ m} \times 0.91 \text{ m} \times 0.91 \text{ m} = 51.0 \text{ m}^3$$
$$\text{coal resource} = 6.1 \text{ m} \times 0.91 \text{ m} \times 0.91 \text{ m} \times 1.281 \text{ t/m}^3 = 6.47 \text{ t}$$
$$\text{stripping ratio} = 51.0 \text{ m}^3 \div 6.47 \text{ t} = 7.88{:}1$$

These calculations can be used to define an economic limit:

- Cost for overburden/interburden mining is \$1.63/m^3 (\$1.25/yd^3)
- Cost of coal mining is \$2.76/t (\$2.50/st)
- Estimated revenue is \$13.23/t (\$12/st)
- Economic stripping ratio (ESR) is
  (\$13.23/t – \$2.76/t) ÷ \$1.63 = 6.41 or 6.41:1;
  (\$12.00/st – \$2.50/st) ÷ \$1.25 = 7.60 or 7.60:1
  = 6.41 m^3/t (7.60 yd^3/st) = 6.41:1 (7.60:1)

  Note: Ratios differ between metric (International System of Units) and English units because stripping ratios mix units: cubic meters to metric tons and cubic yards to short tons.

The ratio becomes the economic mining limit and is the first step in establishing the economically mineable reserve. The formula can become more complex as issues of quality affect the economic value of the coal. It is important for the engineer to understand what the economics are and what

contract restrictions, including penalties and bonus provisions, may affect revenues. Other adjustments to coal recovery (and actual mineable reserves) would be dilution and in-seam losses, which tend to downgrade the stripping ratio by increasing the cubic meters (cubic yards) of waste and decreasing the number of metric tons (short tons) of resource. Depending on mining conditions and on thickness and quality of the seam, losses can amount to several percent and have a significant impact on the reserve.

RESOURCE CONTROL

Land and Mineral Ownership

Ownership of land and minerals is a key consideration that should be determined and addressed early in the planning process, as it can affect development time frames, permitting requirements, development cost structure, and project profitability. Land ownership typically consists of two components: the surface estate and the mineral estate. Either or both may be held by private (fee estate) or public (public estate) entities. In some cases, the surface and mineral estates may be held separately (severed estate).

The rights to access and utilize the surface and to develop and produce the mineral resource can be secured through direct ownership (purchase of the surface and/or mineral estate), lease, agreement, or a combination of these legal vehicles. In many cases, the right to develop and produce the mineral resource (particularly where public-estate minerals are involved) carries with it certain rights of surface access and use.

With the exception of direct ownership, other access and development rights typically involve structured payments to secure and exercise these rights. These typically take the form of advance payments to secure the rights, flat-rate annual fees for use of the lands, royalty payments based on mineral production rates, fee payments keyed to specific activities, or combinations of these payment mechanisms.

Secure, well-defined access and development rights are an important element of a stable cost structure and are key to project cost control and profitability. It should also be emphasized that ownership rights for the project area and an adjacent buffer zone can be important in minimizing and successfully addressing potential conflicts with adjacent landowners and uses.

Other Resources

In Wyoming (United States), gas collected from coal seams (i.e., coal-bed methane), has become a significant consideration relative to permitting and the timing for development of coal reserves where both (independently recoverable) resources exist. Existing oil and gas (or other mineral) leases must be a consideration in the extent of and timing for development and production of reserves.

PERMITTING

Mine permitting is the process of preparing and submitting relevant project information for review and approval by jurisdictional government authorities to verify project plan compliance with applicable laws and regulations. In general, applicable laws and regulations as they relate to mining are designed to prevent, control, minimize, or effectively mitigate potential adverse mining-related impacts on the environment and on human health and welfare.

Mine permitting may involve submittal of individual permit applications for approval of specific mining-related activities (mining and reclamation, air emissions, water discharge, facility construction, and so on). Alternately, it may involve environmental analysis and plan approvals for the project as a whole. Or a combination of all of these approval mechanisms may be involved. An outline of typical mine permitting requirements is provided in Table 10.9-3.

Typically, the process involves review by national or state/provincial agencies that have approval authority over land uses or mining and reclamation plans. It may also involve review by agencies with authority over specific environmental resources such as air, water, wildlife, and other specific project aspects. Normally, some provision is provided in the permitting process for input by affected parties and nongovernmental organizations.

Environmental Baseline

At the start of the permitting process, it is important to effectively characterize environmental resources and values as they exist in the project area (baseline) prior to mining disturbance. Generally, baseline characterization involves field studies by qualified professionals of all resources and values that may be affected. The following resource values are typically characterized:

- Land use
- Cultural, archaeological, and paleontological resources
- Geology
- Meteorology and air quality
- Surface and groundwater hydrology
- Soils
- Vegetation and wetlands
- Fish, wildlife, and related habitat values
- Aesthetics and noise
- Socioeconomic conditions

Baseline characterization then forms the basis for development of specific measures to prevent, control, minimize, or mitigate the potential impacts and for evaluation of potential mining-related impacts with consideration of planned control and mitigation measures.

Mitigation Plans

At the point in a project at which permitting activities are initiated, mining and reclamation plans are generally well defined, including the locations, extent, and nature of surface-disturbing activities. Comparison of the extent and nature of mining activities with information from environmental baseline characterization provides the basis for development of project mitigation plans. In many cases, required mitigation measures are defined to some extent by specific regulatory requirements. A common example is the requirement for the collection and treatment of runoff from mine disturbance areas to comply with effluent standards before water is discharged to natural drainages. To address site conditions and constraints or enhance postmining land use, the mine operator may have an opportunity to develop site-specific mitigation plans or to modify mitigation plans within the limits of regulatory constraints.

Impact Analysis and Monitoring

Environmental impact analysis is an integral part of the permitting process, whether it is the evaluation of regulatory compliance as part of a permit review or of the significance of potential impacts.

Table 10.9-3 Typical mine permitting requirements

Mining and Reclamation Permit(s) (National/State or Provincial/Local Mining, Natural Resource, or Land-Use Authorities)	Water Discharge Permits (National/State or Provincial/Local Environmental or Natural Resource Authorities)	Air Emission Permits (National/State or Provincial/Local Environmental or Natural Resource Authorities)	Other Permits (National/State or Provincial/Local Environmental, Natural Resource, Land-Use, or Safety Authorities)
General information (mine overview and information on permit preparation) Exploration plans and compliance Legal information—owner/operator, surface and mineral ownership, right-of-entry Other permits and regulatory compliance information Environmental baseline information Land use • Cultural, archaeological, and paleontological resources and values • Topography and geology • Meteorology and air quality • Surface and groundwater hydrology • Soils • Vegetation and riparian/wetland resources • Fish, wildlife, and related habitat values • Aesthetics and noise • Socioeconomic conditions Mining and reclamation plans • Boundaries of mining, disturbed, and affected areas • Duration and sequencing of operations • Operating schedule(s) • Description of mine development and operations • Mine equipment and labor • Mine facilities and roads • Descriptions of specific mining and related activities – Drainage and sediment control – Soil material removal – Blasting – Overburden removal – Ground control – Material processing and handling – Mine and process waste handling – Material transportation • Environmental monitoring plans and reporting • Reclamation and land restoration/management – Facility conversion, salvage, or demolition – Backfilling and grading – Drainage reestablishment – Soil material replacement – Revegetation – Postreclamation management and monitoring • Reclamation bonding and bond release Environmental and socioeconomic protection and mitigation plans and measures	Source and location of discharge(s) Ambient flow and water quality conditions (receiving stream) Use(s) (receiving stream and downstream waters) Quantity and quality of discharge, and seasonal or temporal variations Treatment methods (if any) Water quality control/ protection measures (i.e., spill plans, retention structures, etc.) Monitoring plans and reporting requirements Discharge effluent limitations	Source and location of emission source(s) Ambient air quality and meteorological conditions Quantity and type(s) of emission(s), and seasonal or temporal variations Control methods and efficiencies (if any) Monitoring plans and reporting requirements Air emission limitations	Land use/zoning permits Safety permits/approvals (training, personal protection, noise, electrical/mechanical equipment, blasting, haulage, ground control, and impoundments) Riparian/wetland protection/ mitigation permit Water supply permit Wastewater control permit Solid and hazardous waste handling and disposal facility permit Radiation source permit (monitoring devices) Explosive material handling and storage permit Communication system permit

In the context of a permit review, impact analysis focuses on whether or not the proposed mining and related activities (and planned mitigation measure) meet specific regulatory requirements and performance standards.

For environmental factors, impact analysis typically includes the evaluation of direct, indirect, and cumulative impacts and assesses whether or not potential impacts meet an objective, or indeed the subjective measure of significance.

Prepared and evaluated as part of the permitting process, monitoring plans provide a mechanism for the direct measurement of impacts on specific environmental resources. If properly designed and administered, monitoring can identify significant changes in resource conditions. It can be used to assess the effectiveness of mitigation measures and the accuracy of impact assessments and to modify operating and mitigation practices.

Timing

It is critical to allow sufficient time in the project schedule for project permitting. It normally takes between 12 and 18 months to collect adequate information for environmental

baseline characterization. Depending on project complexity, permit preparation may require between 6 and 24 months. Required agency reviews and approval may extend the overall permitting schedule by another 6 to 24 months. Important factors in minimizing permitting time frames include ensuring adequate and timely baseline characterization, conducting effective ongoing communication with stakeholders, and coordinating closely with jurisdictional agencies.

INFRASTRUCTURE

Existing Infrastructure

Existing infrastructure (i.e., roads, water wells, gas pipeline, or power lines that run through the property) must be identified and mapped, and the landowners must be contacted and told the intent and potential impacts of mining. Where existing infrastructure may be affected by mining, mitigation action may be necessary, either in the form of compensation or relocation/replacement of the structures.

Mine Infrastructure

The necessary infrastructure to support mining and related operations will need to be planned, developed, and accounted for in the economic evaluation process. This infrastructure will include roads and utilities, office and changehouse facilities, warehouse and maintenance facilities, material handling, processing and product transportation facilities, drainage and sediment control systems, and so forth.

Infrastructure Mapping

Maps showing both existing and planned mine infrastructure are typically developed as part of the mine planning process. Features that will be mined around, or that are not within the mining area, are not shown on the reserve map. Some structures require economic analysis to determine whether mining around or compensation/relocation/replacement is the better economic approach. These areas may be included in the reserve, but until the status of the areas is resolved, they should be excluded in the base-case mine plan as nonmineable reserves.

PIT DESIGN

Mining Methods

For surface strip mines, the choice of mining method is dictated by the terrain, geology, and depth of the resource. The terms *contour mining* and *area mining* are used to describe mining methods that are suited to specific geologic and topographic conditions.

Where the terrain is variable and multiple seams are present, contour mining may be the best option. In the Appalachian mines of the United States, thin seams, undulating topography, and sometimes steeply dipping seams require equipment that is highly adaptive and mobile and can move material relatively long distances. These conditions favor contour mining using large-tracked dozers, rubber-tired scrapers, shovel-truck or loader-truck equipment fleets, and occasionally small draglines (23–35 m^3 [30–45 yd^3]). As the economic stripping ratio increases, contour mining may be coupled with follow-up to auger or with conventional underground methods.

Where the terrain is relatively uniform and the seam or seams are flat-lying, area mining is often the preferred approach. Relatively large flat-lying deposits such as the coal reserves of Wyoming's Powder River Basin (PRB), phosphate deposits in Florida (United States) and the Middle East, and the Canadian tar sands are well suited to area mining methods using large draglines (46–120 m^3 [60–160 yd^3]), fleets of large-scale shovel trucks (shovel capacity 46–76 m^3 [60–100 yd^3]; truck capacity 140–230 m^3 [180–300 yd^3]), or even bucket-wheel excavators (BWEs) and cross-pit conveyor systems. Using large stripping equipment and moving large quantities of material keeps unit operating costs to a minimum. The PRB mines were typically mined at a 1:1 stripping ratio in the 1980s; however, by the 1990s the stripping ratio had increased to more than 2:1. As mining progresses to the west, the stripping ratio increases as the coal seams dip between 1% and 2% and the overburden gets thicker. This increase in strip ratio is also reflected where split seams occur. Coal quality also increases from north to south, which can have a significant effect on the ESR. Changes in geology must be taken into consideration in pit design and equipment selection. Bise (1986) gives several examples of equipment selection. Bucyrus-Erie Company (1979) is a good source of information on shovel/truck selection and operation.

Large electric shovels are used in both contour and area mining. They offer good flexibility, reasonable mobility, and moderate to high loading capacity, and they can be used both for stripping overburden and for ore loading. However, electric shovels do require a truck fleet, along with associated haulage-road requirements, which can increase production costs.

Draglines are high-production machines used to strip and move overburden over short distances. Draglines have limited mobility and are generally not suited for loading product. It can take up to a year to construct a dragline, and its cost can range from $50 to $100 million. Nevertheless, draglines are dominant in the large surface coal mines in the United States and in other large flat-lying deposits throughout the world (Cassidy 1973).

In the lignite mines of Europe and the U.S. Gulf Coast, large BWEs and conveyor-belt systems are used to excavate and move large volumes of overburden and product economically. The BWE is a continuous-excavation machine capable of removing up to 240,000 m^3 (314,000 yd^3) of material per day. BWEs are found mostly in coal mining in Europe, Australia, and India. A BWE can cost more than $100 million and take 5 years to construct.

A compilation of equipment application and various mining scenarios as applied at operating properties can be found in Chironis (1978) and Kirk (1989). Support equipment including dozers, scrapers, loaders, graders, water trucks, and—depending on individual mine requirements—a multitude of other equipment required to complete the mining cycle is a significant capital and operating cost. Sources for accounting for this equipment include vendor publications such as the *Caterpillar Performance Handbook* (Caterpillar 1995) and vendor Web sites.

Cast blasting is a technique that is used in combination with dragline operations to increase overburden production capacity and reduce costs. Cast blasting utilizes the blast energy and gravity to cast overburden from the highwall side of the pit both into the pit and onto the spoils. The portion of the overburden cast into the pit is then moved onto the spoils using the dragline effectively, thereby reducing overall dragline production requirements and rehandling. However, effective control of blasting is important to prevent loss of the underlying resource, particularly in multiseam pits.

Highwall mining is a method of increasing coal recovery in a pit where the stripping ratio has reached its economic limit and surface operations are no longer cost-effective. Two types of highwall mining are commonly used: auger mining, in which a large-diameter auger bores parallel holes into the exposed coal seam; and conventional mining, in which a remote-controlled continuous miner, coupled with an extensible conveyor, extends parallel entries into the exposed coal seam. For highwall mining to be effective, the key criterion is that the coal and surrounding rock must be competent enough to be self-supporting when a portion of the coal seam is removed. Geotechnical calculations determine the size and allowable depth of the auger or miner entries. Recoveries above 50% are possible with augers as large as 2.4 m (8 ft) in diameter, and conventional methods can result in slightly higher recoveries. Experience has shown that highwall mining is not well adapted to steeply dipping seams.

Pit Geometry

After the boundary of the mineable reserves is established, the geometry and structure of the reserve dictate pit orientation and configuration. When the pit orientation is laid out, the mapped dip and strike of the seams and any geological structures such as faulting or discontinuities should be taken into consideration. Geotechnical evaluations provide additional valuable information for determining highwall and spoil slope angles, slope angles for waste and soil stockpiles, pit-end and inter-bench slope angles, and the overall highwall angle for multiple-seam operations.

For both planning and operations tracking/reporting, a standard naming convention should be established for designating and referring to specific mining areas, pits within each mining area, and cut sequencing within each pit. This naming convention should designate and explain mining units to be used in the scheduling process. The width and length of the pits are typically constrained by the pit geometry, physical limitations of the equipment (e.g., dragline reach), and the targeted production rate required. Pits are broken down into mining cuts based on both production scheduling considerations and the accuracy required for reporting volumes. Short-term planning often requires smaller cuts, depending on drill spacing and quality variations, but it provides increased accuracy relative to production volumes and quality. Long-term planning does not require the same accuracy, so larger cuts or mining blocks may be acceptable, thereby reducing data requirements and the amount of time required to input and analyze the data for mine scheduling.

Points to consider when developing the design of mine pits include the following:

- Orientation of the pit with respect to the strike and dip of the seam and site topography. Pits with inside curves result in insufficient spoil room and additional rehandle. Outside curves create additional spoil room and provide for spoil-side access to the coal seam. Similarly, mining updip creates more spoil room and reduces the potential for spoil failure. If the terrain is relatively flat, straight pits usually give better control over the pit and tend to result in better productivity.
- Prominent structural features, such as faulting, along with intersecting drainages and groundwater. Dewatering may be required, which would have an impact on highwall stability.
- Length and width of the pit. Generally, a maximum pit length of 1.6 km (1 mi) is a good rule of thumb where ramp access to the pit floor is sufficient. Most ramps are spoil side, but there are occasions when highwall ramps and drilling and blasting ramps are required. Minimum pit width is driven by the depth of the pit and equipment clearances needed in the pit bottom. There generally needs to be sufficient width in the bottom of the pit to operate coal-loading equipment with truck and drill access. Keeping the pit width to a minimum is necessary to minimize rehandling.
- Haul access. This consideration of pit design should provide sufficient room for drilling, blasting, and coal extraction. Intermediate spoil ramps, usually at least one per pit, are required to access the pit bottom from either side of the main haulage ramp. Pit-end ramps are also a consideration, so that coal extraction, drilling, and blasting are not cut off by overburden operations. Haulage access design becomes more complex for multiple-seam operations.
- Highwall and pit-end ramps. These are used when spoil material is unstable. The disadvantage of highwall ramps is that the ramp must be reestablished after each cut, and the main haul road requires relocation when mining advance encroaches on the existing haul road.

Rehandle

Rehandle is defined as material that is moved more than once to uncover the same amount of resource. A normal amount of rehandle for a dragline is 10%. Some circumstances require increased rehandle, such as when overburden depth is increasing above the average design thickness for the equipment being used. In such cases, rehandling material may be more cost-effective than purchasing additional equipment or using equipment that is more costly on a unit production basis. When rehandle costs and volumes for major production equipment increase beyond normal limits, it reduces the availability and effective productivity of the equipment for stripping. Utilizing shovel-truck prestripping on a dragline pit may decrease rehandle significantly, resulting in additional stripping capacity. Managing rehandle becomes a trade-off between production capacity and lowest-cost utilization of the equipment.

Unnecessary rehandle can occur when stockpiles are placed too close to the active pit or when production scheduling requires rehandle of stockpiled material. Spoil and highwall failures, which can also result in unnecessary rehandle, may result from poor maintenance of the pit bottom, stacking spoils too high, or highwall/spoil angles that exceed design parameters. An elevated water table, water in the pit, and poor surface-water drainage all can greatly influence highwall and spoil stability. Unplanned rehandle is an unnecessary cost that can have a direct impact on production cost.

Mining Sequence

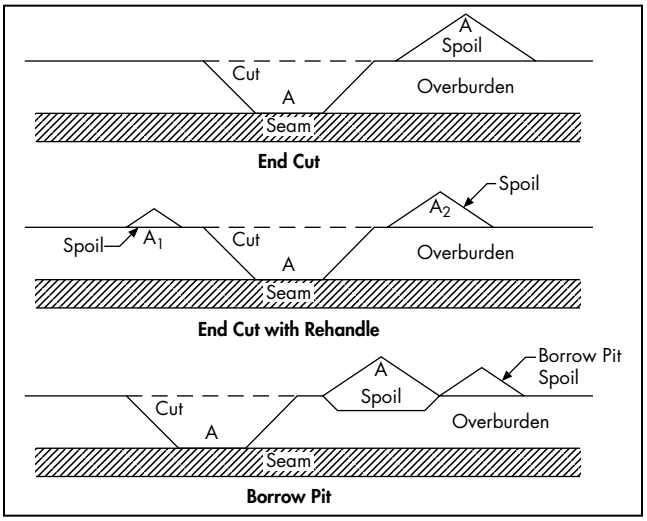
For a new property, the mine planning process comprises setting a target production, selecting the equipment necessary to meet that production rate, and running a projected production schedule to determine the economics of the plan. Production sensitivities are then run to determine if the plan can be improved by changing the production level, equipment, production sequence, or other preselected parameters. It is not uncommon to run several iterations to generate an economically optimized mine plan.

For an operating mine, production targets and equipment fleet may be preset. In such situations, meeting the target production rate becomes the focus of planning runs to best fit adjustments to pit configuration and sequencing to the equipment for a given production rate. In this case, planning options may include supplementing, replacing, or idling existing equipment to meet the required production rate. The mining sequence is considered in the pit design phase.

A typical example of mine sequencing is a dragline operation, which is initiated with a box cut. In a dragline operation, the box cut is typically the initial excavation, which creates sufficient spoil room for the first dragline pit. Figure 10.9-1 illustrates three box-cut scenarios.

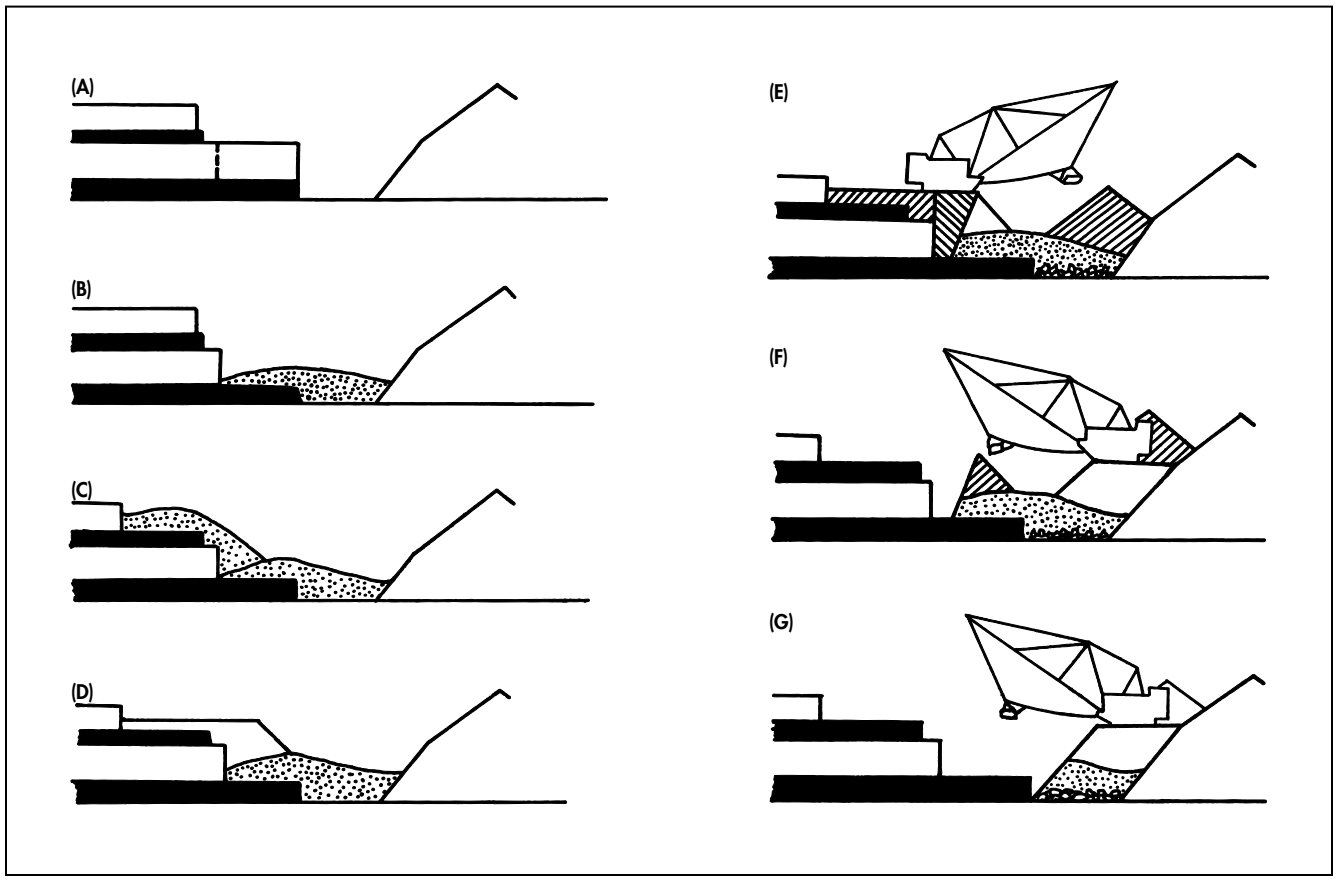
A typical mining sequence would involve topsoil removal, overburden removal, coal removal, backfilling of the resulting pit with overburden spoils, and grading of the spoils to the final reclamation configuration, followed by reclamation grading, replacement of soil materials, and revegetation. For most regulated operations, unless there is a variance in the permit, soil materials and, in some cases, underlying weathered materials are stripped and stockpiled for future replacement on regraded spoil materials. In some cases, these materials can be directly replaced on existing regraded spoil areas, avoiding the need for stockpiling and often resulting in better reclamation.

Figures 10.9-2A–G illustrate the steps in a typical dragline sequence. Figure 10.9-2A shows the pit as it would look at the end of the previous cycle. The coal is extracted and the pit is ready for the next cut, which is shown by the vertical dashed line. Figure 10.9-2B and Figure 10.9-2C illustrate the results of cast blasting on both the upper and lower seams. Figure 10.9-2D shows the upper bench being graded in preparation for the dragline bench. Figure 10.9-2E illustrates the several phases of stripping that a dragline would perform; the first is extending the bench to allow the dragline to walk out



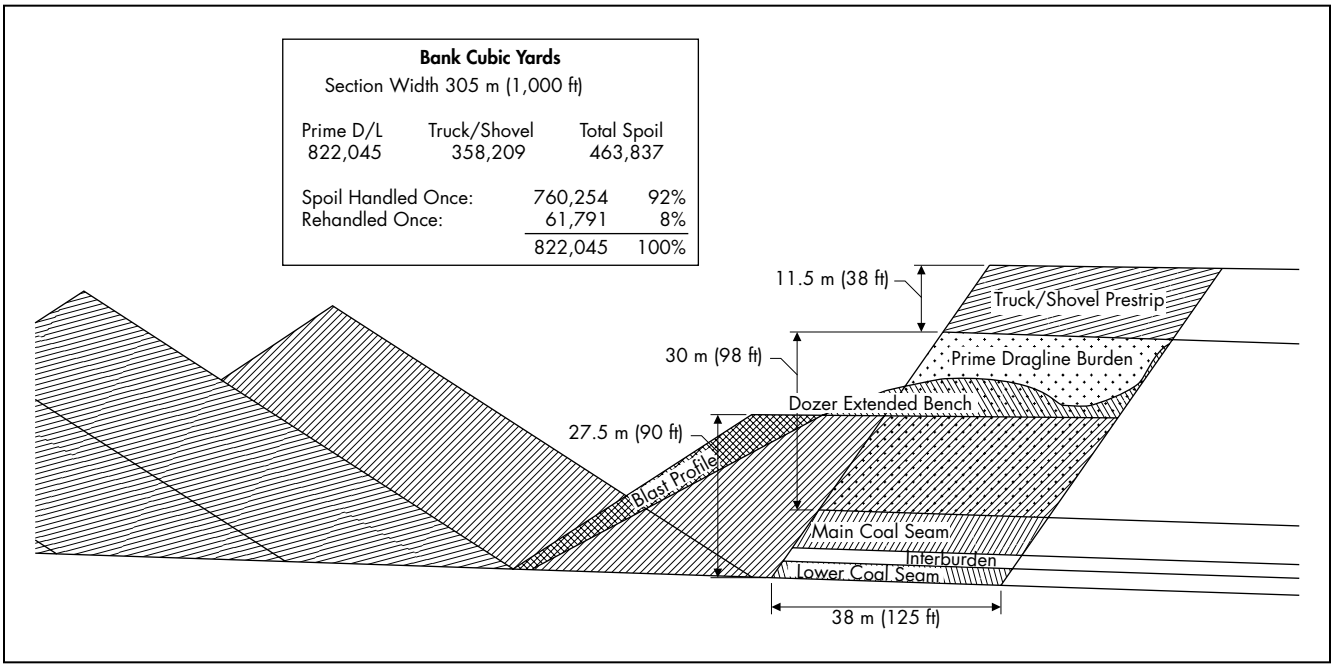
Source: Skelly and Loy 1975.

Figure 10.9-1 Box cut methods



Source: Hrebar and Atkinson 1998.

Figure 10.9-2 Dragline pit sequence cross section



Courtesy of Mincom, Inc., Denver, Colorado.
Figure 10.9-3 Typical pit section

far enough to cast the next cut of spoil to build a spoil-side bench for the dragline to sit on. In Figure 10.9-2F, the dragline is moved to the spoil side, where the bench has been graded to an elevation that allows the dragline to uncover the coal seam. Finally, Figure 10.9-2G shows the section immediately before removing the coal, which is the last step in the cycle. When the coal has been extracted, the cycle is repeated.

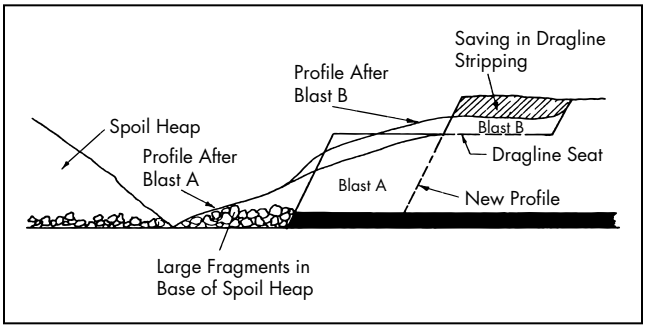
The purpose of generating range diagrams is to determine the amount of material moved by the production equipment in each sequential step. These volumes are then used to determine the productivity of the equipment for that specific pit geometry. Overburden thickness and changes in the coal seam geometry can affect the production levels, and a different configuration may be required.

The trade-off with the cast-blasting technique is the loss of coal and dilution due to fracturing of the coal seam at the exposed highwall of the pit. Other configurations that could be compared to cast blasting would be a pre-bench shovel-truck or scraper fleet, depending on the depth of the material above the first seam.

Computer programs that simulate dragline scenarios are used to generate the equipment volumes needed for scheduling. By adjusting the equipment configuration for different geometries, a database of scheduling volumes can be generated by equipment type, with rehandle quantities and expected coal production volumes being taken into account. Each schedule is run through an economic model, and scenarios are compared to evaluate the impacts of each on NPV. This process is used in both proposed and ongoing mine projects to provide justification for equipment purchases and modifications to the mine plan.

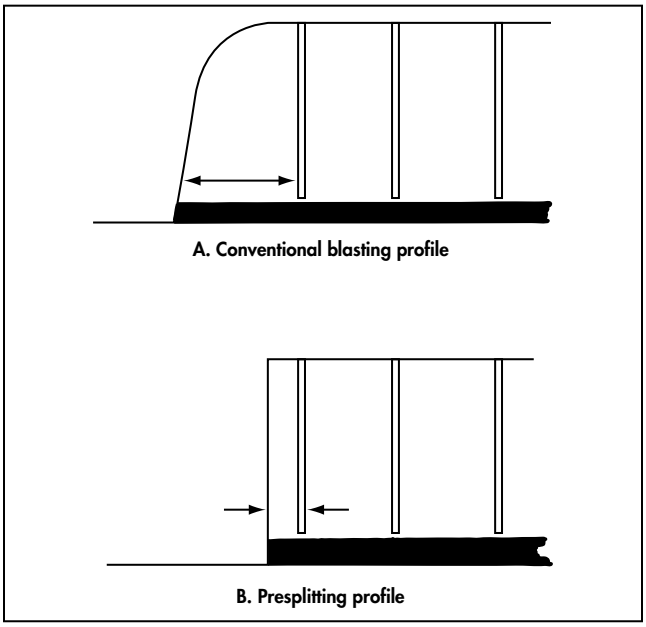
Specialized Blasting Techniques

For strip (open-cast) mining, overburden must often be fragmented by blasting so it can be efficiently and economically



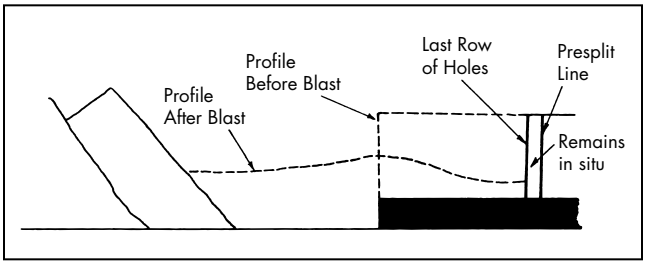
Source: Hrebar and Atkinson 1998.
Figure 10.9-4 Blast cast profile

excavated. For many years, mining engineers have considered cast blasting (explosives casting) of overburden both to take advantage of the explosive energy used to fragment the overburden and to reduce material excavation and handling costs (Brealey and Atkinson 1968; McDonald et al. 1982). The low cost and high gas pressures of ammonium nitrate and fuel oil (ANFO) explosives make cast blasting more attractive as a production tool. The advantages of cast blasting become increasingly significant for thick overburden with resistant layers requiring high powder factors (kilograms of explosives per cubic meter [pounds per cubic yard] of overburden blasted)—0.65 kg/m^3 (1.1 lb/yd^3) is typical for some very strong sandstones, for example, in South Africa and Australia. Applications have shown that in certain circumstances, cast blasting in deep overburden can be more economic than conventional stripping. This method, which is based on reducing the primary overburden cast by the dragline (Figure 10.9-3), has the added advantage of promoting gravity segregation of the cast material. Large rock fragments come to rest near the pit floor to form the base of the spoils, thereby improving spoil



Source: Hrebar and Atkinson 1998.

Figure 10.9-5 Blasting profile



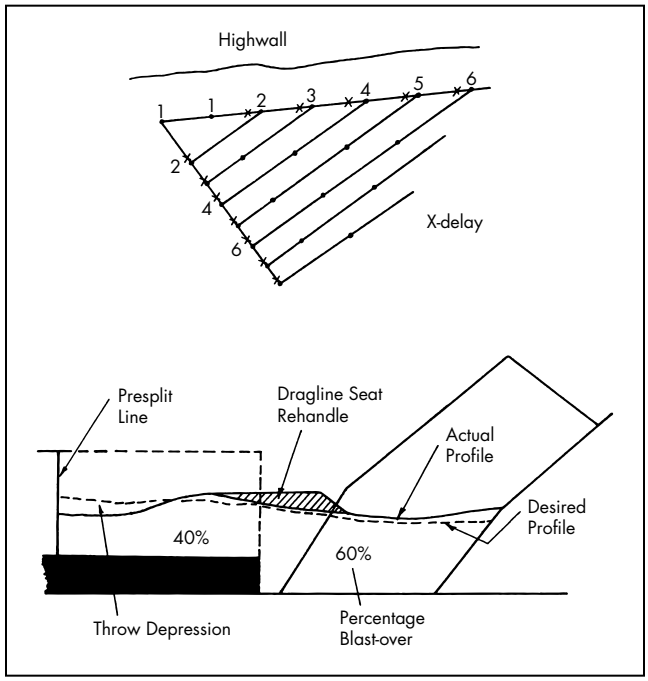
Source: Hrebar and Atkinson 1998.

Figure 10.9-6 Blasting with parallel tie-up

drainage and stability. Figure 10.9-4 illustrates a typical blast cast profile.

Presplit blasting may be used in conjunction with cast blasting to dewater permeable overburden. Additionally, in strong ground, a vertical face can be created in conjunction with cast blasting, rather than the irregular sloping face produced by conventional blasting (Figure 10.9-5). It is obvious from Figure 10.9-5B that the vertical face, with greatly reduced distance from the front row of blastholes to the toe of the highwall, will result in far more efficient cast blasting. Nonvertical highwalls with an inclined presplit line can be incorporated into the blast design along with modification of the blasthole pattern. Where clay-rich and similar materials are absent, low-cost bulk-placed ANFO has been used in place of more expensive water-resistant explosives (e.g., water gels, heavy ANFO).

The geometry of the spoil after cast blasting must be controlled to provide a section suitable for dragline operations while moving as much spoil as possible across the pit. Conventional blasting patterns in strip mining are usually square or rectangular; rows of holes perpendicular to the highwall are detonated sequentially so that the spoil is thrown parallel to the highwall toward the last open cut. In cast blasting, rows of blastholes parallel to the highwall are detonated sequentially, initiated from the highwall progressively from



Source: Hrebar and Atkinson 1998.

Figure 10.9-7 Blasting method

front to back, resulting in maximum spoil cast across the pit. Experience shows that marginally more spoil is cast using this design, but a section of overburden between the presplit line and the last row of blastholes may remain in place (Figure 10.9-6).

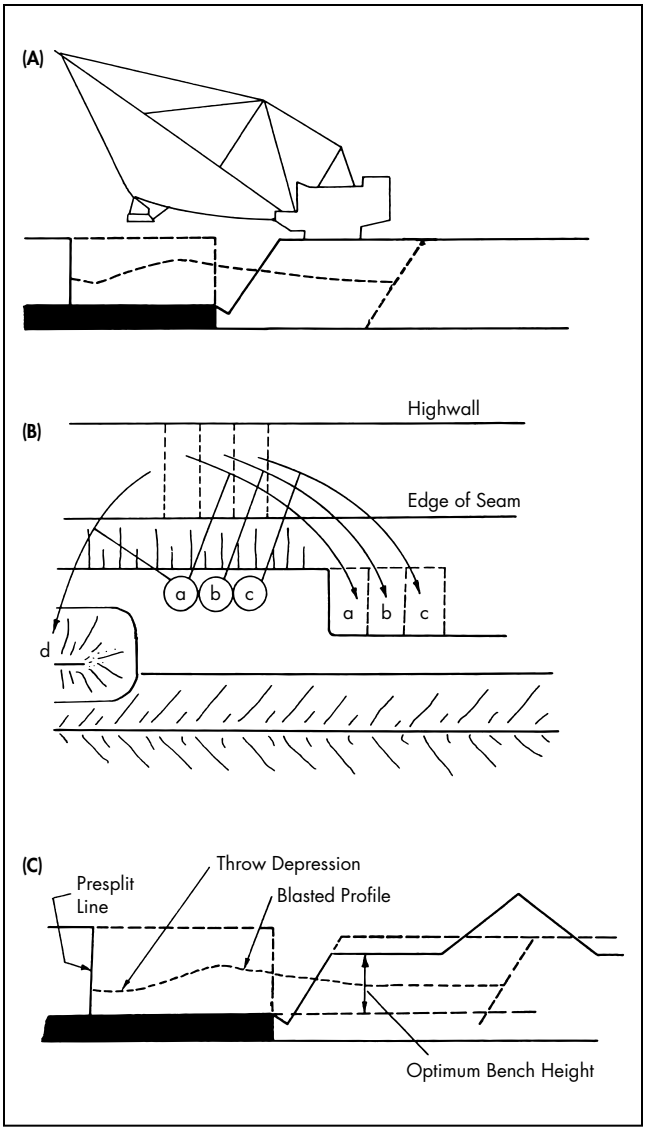
The best results appear to occur with about a 30° offset using a staggered V1 pattern with relatively long, inter-row delays. Figure 10.9-7 illustrates a typical section and the desired section. The throw depression can be greater than desirable, and some minor rehandle of the thrown spoil is necessary to establish a bench for dragline operations. The spacing/burden ratio of the blasting pattern should be determined using site-specific factors and requires detailed consideration to control the trajectory of the thrown spoil.

A means of evaluating the feasibility and effectiveness of cast blasting is the use of the depth/width (d/w) ratio. This is a ratio varying from 0.4 to 0.9, where d is the pit depth from the highwall crest to the bottom of the lowermost seam and w is the width of the exposed pit bottom (toe of highwall to toe of spoils).

Blasthole diameters of 254 to 311 mm (10 to 12.5 in.) are typical. It is doubtful whether larger-diameter holes will provide additional advantages in strong ground, unless explosive density/energy can be increased in the blasthole. Closer spacing rather than larger blasthole diameters may be preferable.

Cast blasting (Walton and Atkinson 1978) can facilitate spoil-side dragline operations, which offer several operating advantages:

- Access ramp roads can be located on the highwall side of the pit, eliminating problems caused by lack of spoil room at the junction of the active pit with the access ramps.
- The stripping sequence is uninterrupted, since access ramps do not have to be reestablished on the spoil side, resulting in increased dragline productivity.



Source: Hrebar and Atkinson 1998.

Figure 10.9-8 Cross-pit, chop-down with single-seam method of operation

- Equipment scheduling is simplified.
- Access to individual seams is simplified in multiseam operations.
- Surface reclamation is simplified since spoil grading does not have to address reclamation of access ramps.

Disadvantages of spoil-side operations are limited, but include the following:

- The dragline may operate in the less efficient chop-down mode.
- Separate power distribution systems are required on both sides of the pit—on the spoil side for the dragline and on the highwall side for blasthole drills.

Single-Seam, Cross-Pit, Chop-Down Operation

A dragline used in the chop-down mode is about 60% efficient compared with the conventional drag mode. Bucket maintenance costs are also higher (although this is considerably alleviated by the better fragmentation achieved in cast blasting). The depth/width ratio of the pit should exceed 0.4 for cast blasting to be considered (i.e., this method is best suited to deeper pits).

Figure 10.9-8 shows the method of operation. The dragline bench height in Figure 10.9-8C can be fixed so that the spoil crests are essentially level, virtually eliminating the need to grade the spoil peaks during reclamation. This advantage can only be fully realized, however, if the height of the dragline bench above the top of the seam does not exceed the optimum digging depth. Where this height exceeds the optimum digging depth, the bucket must be cast and dragged prior to the swing cycle, thereby increasing cycle times. In these circumstances, it is usually more economic to level the spoil peaks with either the dragline (rehandle) or conventional mobile equipment.

The single-seam, chop-down method may be used to strip seam partings in multiseam operations. This operation results in a reduction of dragline productivity of up to 50%, but if the ratio of parting to lower seam thickness is low, this approach can be an economical option.

Two-Seam Method

The steps utilizing cast blasting are illustrated in the following sequence (refer to Figure 10.9-2A–C, and as described previously in the "Mining Sequence" section):

1. The pit section prior to blasting (Figure 10.9-2A).
2. Interburden blasted into the void left after extracting the mineral from the previous strip. Large rocks will form the base of the spoil heap and are not handled by the dragline (Figure 10.9-2B).
3. Blasted overburden. The throw depression of the interburden blast is filled by overburden spoil (Figure 10.9-2C).

This method results in an increase in dragline productivity above that of conventional dragline stripping, but insufficient experience is available in comparable conditions to quantify this increase. There are, however, other marked advantages:

- Both seams are exposed, allowing simultaneous seam recovery. The upper seam can be dozed over the side of the bench (or by other suitable means) onto the lower seam, thereby concentrating and simplifying mineral loading operations and improving blending. The volume of in-pit exposed reserves is increased, allowing greater flexibility in operation.
- Spoil stability is enhanced by gravity segregation of the cast-blast materials, with larger durable rock forming a base for the spoils.
- With both seams exposed simultaneously, additional dragline moves and dead-heading can be eliminated or reduced.
- Reduced need to handle durable rock materials with the dragline can result in reduced bucket maintenance costs.

This method can also be used for single-seam, thick-overburden applications to eliminate chop-down operations for upper overburden benches. Where weathered material exists near surface, the upper bench can be presplit with more closely spaced holes than for the lower, more competent overburden. Provided that weathering is not too deep, the upper bench slope can be increased, up to a maximum of 90° (vertical), since presplitting will limit blasting effects and reduce drainage concerns.

Stripping Previously Worked Deposits

In some situations, thick coal seams that have been previously partially extracted by underground methods (e.g., room-and-pillar mining) can be economically recovered by strip mining. In these situations, stability and the safe operation of the dragline can be in question where conventional dragline stripping is proposed. Cast blasting provides an alternative stripping method for suspect areas where the dragline is not located over the pillared coal, thereby minimizing stability and dragline safety concerns.

Equipment Selection

The following criteria are among those that should be considered when selecting equipment for a new or expanding operation:

- The life of the project. Certain capital and operating cost considerations should be evaluated. Smaller equipment (e.g., scrapers) has lower capital costs but much higher operating costs, depending on the haul distance and the unit volume moved. In comparison, draglines have much higher capital costs and much lower unit operating costs. For example, a scraper may have a capital purchase cost of $2 million, with a recurring replacement cost every 5 years, a unit capacity of 23 m^3 (30 yd^3), a life of 35,000 hours, and an operating cost of $0.98/$m^3$ ($0.75/$yd^3$) varying with the haulage cycle. A 115-m^3 (150-yd^3) dragline might have a capital purchase price of $50 million, an operating cost of $0.20/$m^3$ ($0.15/$yd^3$), and a life of 150,000 hours. It typically will last for the life of the project, which could be 30 years, with a single major rebuild costing $10 million.
- Depth and thickness of the seam(s) overburden and coal, with total depth to last mineable seam. Typically, pits up to 150 m (500 ft) in depth can be surface mined.
- Strike and dip of the property. Is the deposit type more suited to area mining or contour mining?
- Comparison with similar operations. Consider similar operations, but do not restrict the decision by existing convention. In the 1980s, there were only truck-shovel operations in the PRB. The first dragline was introduced in 1982, and since then, draglines have dominated United States surface mines in Wyoming, Montana, Colorado, Texas, and North Dakota.
- Combinations of mining methods.
- Equipment type and size. For draglines, evaluate the equipment type and size by generating range diagrams from cross sections across the length of the reserve. Figure 10.9-3 illustrates a typical range diagram and the volume distribution. Choose typical cross sections that represent variations in the reserve. For shovel-truck operations, bench height and passes per cycle help to determine equipment matching. Three to five bucket passes by a shovel to fill a truck is considered a good match. It will be important to select equipment that can handle changes in pit geometry.
- Dragline capacity. Multiple-seam operations require handling of intermittent coal and interburden, which will impact the productivity of a dragline and require loader truck equipment to handle the smaller seams. Dragline capacity must take into account such production interruptions.
- Haulage scenarios. Evaluate haulage scenarios for truck-shovel and scraper operations.
- Cast blasting. Look at the possibility of augmenting production with cast blasting.
- Used draglines. Evaluate the market for used draglines that are currently idle. Finding a dragline that can be relocated for half the cost of a new dragline may fit the mine plan needs.

All options should be evaluated for the highest NPV and IRR.

Ground Control and Pit Slope Stability

Effective ground control and planning for pit slope stability comprise a two-step process. First, geological structures and groundwater conditions that may affect stability need to be identified and characterized. The planning process then focuses on the proper layout and design of the pit geometry and mine structures to address site-specific structural and groundwater conditions.

In addition to adequate exploration drilling, sampling, and logging to characterize geology, lithology, general structure, water levels, and reserve characteristics, seismic studies can prove invaluable in delineating structural features and characteristics. Groundwater studies can also provide supplemental information on groundwater occurrence and flows.

Proper pit layout minimizes exposure of structural unconformities in the pit highwall, provides for effective drainage of both surface water and groundwater, and avoids geometries that tend to adversely concentrate ground forces (i.e., noses or notches). Similar considerations are appropriate for mine structures, including mine spoils, overburden and waste piles, tailings facilities, water impoundments, and mine buildings.

For pit slopes and spoil piles, as well as engineered structures such as tailings facilities and water impoundments, geotechnical analysis by qualified professionals is appropriate. These analyses assess both static and dynamic (seismic) stability and are often required by applicable regulations. In the case of engineered structures, foundation testing is often a component of the geotechnical analysis, and specific recommendations for foundation preparation and drainage measures are incorporated into project construction plans.

MINE SAFETY

As a critical element of modern mining operations, safety must be a key consideration in planning all aspects of mine operations: layout and design of pit geometry and mine structures and facilities, road design, equipment selection, and operations planning and scheduling.

Applicable regulatory requirements often factor in safety as well as environmental considerations. However, effective protection of worker safety and health requires thought and effort well beyond the scope of regulations. For this reason, the planning phase should address both engineered safety considerations (i.e., pit design and equipment safety features) and safety systems (i.e., worker training, safety awareness, equipment inspection and maintenance, and health monitoring).

LAND REHABILITATION

On completion of active mining operations, whether for an individual mining area or for the mine as a whole, mine disturbance areas should be returned to a stable condition. The potential for any short- or long-term adverse environmental

effects or hazards to human health and safety should be minimized, and productive postmining land use(s) should be supported. Generally, this involves backfilling and grading mine pit areas to blend with the surrounding terrain and promote effective drainage, replacing stockpiled soil materials, and reseeding with compatible native vegetation species.

As part of the planning process, it is sometimes possible to enhance the natural conditions and land uses that existed prior to mining or to make land modifications that may facilitate higher and better land uses.

Effective land rehabilitation can minimize the potential for any short- or long-term liabilities and maintain or increase the value of mined lands. Land rehabilitation is important in promoting a positive perception of the operator and the mining industry as a whole as being environmentally aware and responsible.

ACKNOWLEDGMENT

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CHAPTER 10.10

Highwall Mining

Eric N. Berkhimer

INTRODUCTION

The concept of highwall mining refers to recovering coal by boring openings beyond the highwall limits produced by strip mining after the economic limit is reached due to increasingly high strip ratios. In general, the term encompasses traditional auger mining as well as the more recent advances using a relatively new class of equipment under the heading of "highwall miners." Auger mining is, in essence, the practice of using a large drill, turned horizontally to bore into the coal seam, and is generally limited to thicknesses of 0.6 to 4.8 m (2 to 16 ft). It began in West Virginia (United States) in the mid-1940s, but the machines at that time were limited in the penetration they could achieve. Advancements in technology were made intermittently over the ensuing decades and peaked in the 1970s and 1980s with higher horsepower units that could achieve a deeper penetration into the seam. Highwall miners, developed primarily in the 1990s, use continuous miners to advance into the coal seam and are able to achieve greater penetration depths and increased coal recovery.

Highwall mining is most commonly practiced when contour strip mining has been exhausted, such as in the Appalachia region of the United States (mainly Kentucky and West Virginia). It has also found some application in other parts of the United States as well as a number of coal regions around the world. According to Newman and Zipf (2005), the concept of highwall mining has been used effectively to mine

- Abandoned pre–reclamation law highwalls,
- Points or ridges that are considered uneconomic to mine by traditional underground or other surface methods,
- Outcrop barriers left at the entrance to underground mines,
- Previously augered areas containing otherwise inaccessible reserves, and
- Multiple coal seams.

In another paper, Zipf (2005) further reports that highwall mining may account for approximately 4% of total U.S. coal production or upward of some 60 Mt (65 million tons) of raw coal annually. There are reported to be approximately 60 highwall mining systems and as many as 150 auger systems in use across the United States.

SITE REQUIREMENTS AND CONDITIONS

In order for a site to be considered for highwall mining, a number of site conditions must be taken into account. Important details include the overburden thickness (including maximum expected thickness in mountainous areas), the pit floor condition, the possibility of intersecting abandoned or current underground mine workings, any intersection of previously drilled auger holes, and any fractures or jointing in the highwall. Highwall stability is a major ground control–related safety concern because of the proximity of the highwall mining equipment to the highwall. In fact, some states have certain regulations for reclaiming areas mined by highwall mining as well as for working below a highwall (including the requirement for benching of the highwall). For example, Ohio and Kentucky (United States) mining regulations include special requirements for reclaiming mining pits that have been auger mined (McCarter and Smolnikar 1992). These include special sealing of the auger holes and time requirements for backfilling the area when the augering has been completed.

Other geologic condition requirements include continuity of the coal seam, a relatively uniform seam thickness, and a near horizontal orientation (less than 10° pitch). These are considered essential because of the potential adverse effects encountered, particularly related to contamination/dilution of the coal.

Any highwall mining plan should consider the hole width (or diameter), the web pillar width (coal left in place between holes), barrier pillar width (a wider block of coal left between two series of holes), and the number of holes between barrier pillars. Additional care must be taken regarding the design if operations are recovering multiple seams in close proximity to one another.

The most widely accepted empirical formula for relating pillar strength to coal strength, pillar height, and pillar width in the United States is the Mark–Bieniawski pillar design formula, which, because the pillar length is much greater than either the pillar height or width in highwall mining, can be simplified to

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Table 10.10-1 Pillar width calculation based on mining height and depth of cover

Design Depth of Cover, m	Mining Height, m						
	0.9	1.2	1.5	1.8	2.1	2.4	2.7
30.5	21						
38.1	25						
45.7	29	32					
53.3	33	36	39	41	43		
70.0	36	40	43	46	48	50	53
60.9	40	44	48	51	54	56	58
76.2	43	49	53	56	60	62	65
83.8	47	53	57	61	65	68	71
91.4	50	56	62	66	71	74	78
99.1	53	60	66	71	76	80	84
106.7	57	64	71	76	82	86	90
114.3	60	68	75	81	87	92	97
121.9	63	72	79	86	92	98	103

Source: Adapted from Vandergrift et al. 2004.

$$S_p = S_c\,[0.64 + (0.54 \times W/h)] \qquad (10.10\text{-}1)$$

where S_p is the pillar strength, S_c is the in-situ coal strength, W is the pillar width, and h is the pillar height.

According to Vandergrift et al. (2004), a standard figure of 6.2 MPa (899 psi) for coal strength can be applied for an initial calculation.

After pillar strength is calculated, pillar loading is required to calculate a safety factor. Pillar loading can be estimated using the tributary area method:

$$L_p = S_v\,(W + W_E)/W \qquad (10.10\text{-}2)$$

where L_p is the average vertical load on the pillar, S_v is the in-situ vertical stress, and W_E is the entry width.

Finally, an appropriate safety factor must be determined. Although this is somewhat dependent on specific site conditions, a web pillar safety factor of 1.5 and a barrier pillar safety factor of 1.0 are adequate for many sites.

Table 10.10-1 shows the results of pillar width calculations for the case study detailed later in this chapter based on varying seam heights and overburden cover depths (affecting the vertical load on each pillar). Numbers have been omitted where the W/h ratio is less than 1.0.

The major equipment manufacturers often offer their experience and expertise to users to design the web and barrier pillars and for hole pattern planning. In addition, several modeling programs are available to assist with highwall mine design. LAMODEL is a nonlinear boundary-element method used to examine in-seam pillar behavior. UDEC is a distinct-element code used to examine the stability and interaction of the floor, seam, and roof. Additional details can be found in the paper by Vandergrift et al. (2004).

Preparation in advance of using a highwall miner is relatively straightforward. The highwall should be groomed to remove any loose materials that could fall. The pit floor needs to be leveled and cleaned, and a roadway parallel to the highwall should be constructed for access to the equipment and for beltways and/or truck transportation to move the mined coal. The area should be at least 15 to 25 m (50 to 70 ft) in width, although some design engineering is under way to make the equipment more compact for working on narrower benches.

ADVANTAGES AND LIMITATIONS

Highwall mining methods are generally a low-cost, high-production application. Operation is generally considered to be safer than traditional surface or underground mining, but equipment size is somewhat limited. It can often result in less ash (from dilution by the surrounding rock) than surface mining. Maximum recovery is achieved with straight highwalls, while the inside and outside of curves require fanning of the holes and loss of the reserves located between holes. Because of this, careful mine planning is required.

Figure 10.10-1 shows a typical range of costs per metric ton of coal recovered by highwall mining. As shown, the primary factors influencing costs are seam height, and roof and floor conditions. Seam height is defined as low (<1 m [3.2 ft]), medium (1–2 m [3.2–6.6 ft]), and high (>2 m [6.6 ft]). Roof and floor conditions include flatness, smoothness, and hardness of the floor material as well as competency of the roof material. For example, a floor that has less than a 1° incline and is free of undulations will have a significant cost advantage over an area where the floor is at a higher angle or has undulations that are cut with the coal causing contamination of the coal or a roof that has material that will fall once the coal is removed. These have been classified as easy, average, and difficult.

Augers are also a highly productive method of coal recovery with relatively low capital and labor requirements. A three- to four-person crew can average 90 to 2,200 t (100 to 2,500 tons) per shift. However, because a web pillar generally of 0.3 m (1 ft) for every 0.6 m (2 ft) in hole diameter is left in place, recovery averages only 40% to 60%. Additionally, productivity decreases as the depth increases because of the greater torque requirements from the power unit. After the auger is in the seam, the operator cannot see the moving machinery, so operating an auger unit is a highly specialized skill requiring an experienced operator. Further, guidance is difficult as holes tend to drift downward and in the direction of rotation. Most auger mining is limited to a penetration distance into the coal seam of 90 to 150 m (300 to 500 ft) or less.

Because of these limitations, more advanced highwall mining systems were developed in the 1990s. These units also allow operation with three- to four-person crews, but they can produce at a considerably higher rate than traditional auger miners—up to 3,600 t (4,000 tons) per shift. The coal recovery is also improved, up to 70% of the coal being recovered. Penetration into the coal seam can be up to nearly 500 m (1,640 ft). Technological advances also allow better ability for the operator to keep the unit operating in a straight line and in the coal.

EQUIPMENT AND OPERATION

Augers are essentially large drills turned and used in a horizontal direction. They can range from 0.6 to 2.4 m (2 to 8 ft) in diameter and 18 to 61 m (60 to 200 ft) in length. A typical auger consists of the cutterhead, auger flights to add increased penetration depth and move the coal to the surface, and a prime mover to provide the power to drive the auger. Augers achieve production by exploiting the low tensile strength of the coal rather than overcoming its high compressive strength. As such, the key parameters in achieving production are the cutterhead (auger) diameter, the available power, the penetration depth, and the coal type and hardness. Auger performance is directly related to the machine power and the cutterhead diameter.

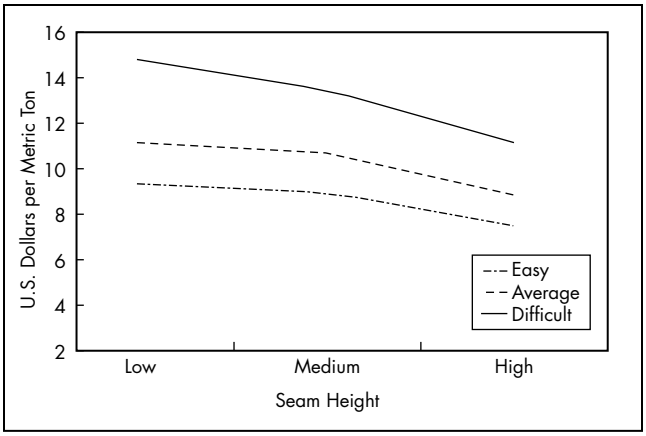


Figure 10.10-1 Comparison of mining costs based on seam height and floor conditions

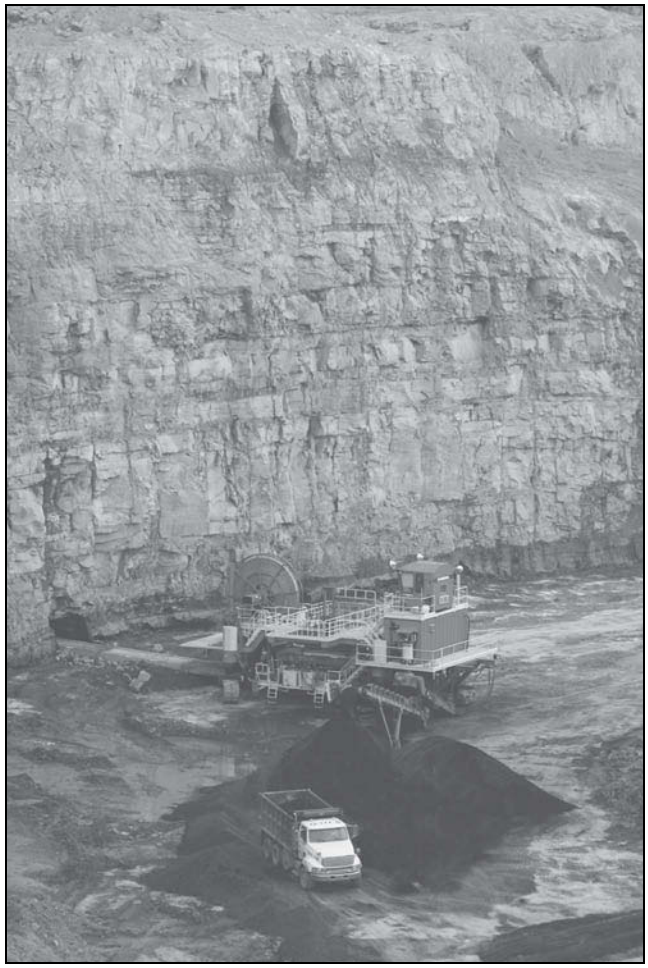
Highwall mining systems were developed in the 1990s to combat the deficiencies experienced with traditional auger mining. Dominated principally by two original equipment manufacturers (OEMs), these systems consist of four basic parts: (1) the cutterhead module consisting of a continuous miner head and gathering arms; (2) the powerhead assembly, which pushes the cutterhead forward; (3) some type of loading and conveying system; and (4) the base unit, which contains the electrical and hydraulic systems. These units are used to mine parallel entries rectangular in cross section into the coal. The cutterhead modules range from 0.75 to 5 m (2.5 to 16 ft) in diameter. A highwall miner is shown in Figure 10.10-2.

RECENT DEVELOPMENTS AND FUTURE ADVANCES

In the mid-1980s, the U.S. Bureau of Mines (Mayton and Volkwein 1989) conducted research in developing a thin-seam continuous miner with transportation to the surface using a multiple-unit continuous haulage system (as used in underground room-and-pillar mining). The key feature and development of this system was an onboard ventilation system that kept the air around the cutterhead clear to permit an operator, functioning by remote control, to see the face using video cameras. Although some features of this were put to use by the later highwall mining systems described previously, this system is not being used commercially.

As a system, recent advances in highwall mining (and particularly with the highwall mining systems described) have centered on greatly improved productivity, better clean coal recovery, and deeper penetration. OEMs have also labored to develop narrow-bench machines that allow operation on previously mined highwalls (highwalls that were mined before reclamation laws came into existence and are still open). In addition, a more global reach has been developed, resulting in highwall mining moving out of Appalachia to locations including, among others, the western United States, Russia, Australia, and South Africa. Finally, there has been a more engineered approach to web and barrier pillar design to improve overall highwall stability and safety for the mining crews, as evidenced by several studies and design formulae detailed previously and OEM involvement.

Future developments in the field of highwall mining are expected to center on achieving deeper penetration, resulting in higher productivity and the ability to work on steeper sloped seams (greater than 10° pitch). This will allow more applications to be used and better coal recoveries from a multitude of applications currently not conducive to highwall mining.



Courtesy of Terex SHM.

Figure 10.10-2 Highwall mining system

A BRIEF CASE STUDY

Vandergrift et al. presented a case study in 2004 detailing a project to perform highwall mining at Bridger Coal's Jim Bridger mine in southwestern Wyoming (United States). The mine was using the ADDCAR highwall mining system supplied by Mining Technologies, Inc., of Ashland, Kentucky. Potential target areas were identified in four areas of the mine with mining to occur on up to five seams.

Because of the large number of areas identified and the mining of multiple seams in each area, a number of geotechnical issues were first identified. In order to work through these issues, Bridger Coal drilled a series of five geotechnical core holes. Inspection, testing, and analyses of these cores provided the basic input parameters for later calculations.

Roof competency and stability was analyzed using the CSIR Rock Mass Rating (RMR) method. Pillar design calculations were then made using the modified Mark–Bieniawski formula (Equation 10.10-1), and pillar loading was calculated using tributary area theory (Equation 10.10-2). Table 10.10-1 shows the results of these calculations at different mining heights and cover depths.

The empirical results were then confirmed based on previous on-site experiences. Numerical modeling using LAMODEL and UDEC were then applied to confirm pillar stability, explore the effects of seam interaction, test designs against cascading pillar failure (when failure in one pillar results in stress transfer to adjacent pillars that, in turn, fail), and examine roof and floor stability. LAMODEL and UDEC modeling essentially confirmed the validity of other calculations and the design plan formulated.

When the design plan was completed, Bridger Coal filed the necessary applications and revisions with the Wyoming Department of Environmental Quality–Land Quality Division and the U.S. Mine Safety and Health Administration for mining permit revisions and ground control plan approvals. Final approvals were received in January 2003.

The mine then scheduled highwall mining and laid out the panels consisting of 20 entries each using the web and barrier pillar design tables developed previously. Subsidence monitoring (as a condition of the permit approvals) was established over the area to be affected. Mining was initiated in the D41 seam of the Northern area of the mine in April 2003 with the first hole completed on April 10, 2003. Average penetration of the first 18 holes was 458 m (1,503 ft) or 94% of the planned penetration. A previously undetected fault was then encountered in holes 19 through 40, which caused average penetration to drop to 71% of planned penetration—or 344 m (1,129 ft). The plan was then altered to mine only one additional panel at a planned penetration of 305 m (1,000 ft). This panel achieved an average penetration of 298 m (978 ft) or 98% of the target.

Despite the geologic problems encountered, the implementation of the ADDCAR system at the Jim Bridger mine is viewed as a success. Penetration depths as high as 488 m (1,600 ft) have been achieved, and the web pillar design has provided a stable highwall.

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CHAPTER 10.11

Quarrying

Ian Brown

INTRODUCTION

Throughout the developed and developing world, vast quantities of aggregates are quarried every year, estimated at around 25 billion t (metric tons). Production in Western Europe and the United States is broadly static, whereas output in Eastern Europe, India, China, and Southeast Asia is increasing rapidly as these countries develop their infrastructures.

Quarried aggregates are essential materials for building the modern world, and these products are used every day in houses, roads, schools, hospitals, factories, shops, offices, airports, and railways. For these projects, aggregates are often used in the form of concrete and asphalt, but they are also used for industrial manufacturing purposes—for example, glassmaking, cement manufacture, paper, fillers, flue gas desulfurization (which results in artificial gypsum), and steelmaking.

The quarrying of rock for aggregates is similar to surface mining for various minerals (Figure 10.11-1), but there are a number of significant differences. The mines are often on a much larger scale, and there is usually a greater depth of overburden, which is still economic to move because the value of the minerals, such as coal, copper, or gold, is many times greater than the value of construction aggregates. The yield of minerals from the ore can be very low (<5%), which means that most of the material extracted from the mine remains on the site in either waste tips or as backfill. In comparison, the product yield from a typical aggregate quarry is commonly >85% and is sold off-site.

GEOLOGY AND LOCATION

Quarrying is the extraction of rock from the ground, usually through open-pit methods. As such, the geology of a country or region determines where a quarry is located. The type of rock also affects the end use of the aggregates produced; for example, granite makes good rail ballast, limestone is essential for cement, and graywacke is excellent for road-surfacing products.

The other key factor when considering geology is the distance from the marketplace, because aggregates are often a high-bulk, low-value product that is expensive to transport. Many construction materials can be produced from local materials, such as sand and gravel, which are abundant in many countries. The high value of cement makes it more worthwhile to transport this product farther.

Aggregates can be naturally occurring and extracted from a quarry, recycled from old construction materials, products of other processes, or artificially made. Some rock types are suitable for many uses; others are restricted to specific needs. The main general categories of rock are igneous, sedimentary, and metamorphic.

Igneous

The most common igneous rocks are granite or basalt, usually formed by the cooling and solidification of molten rock or magma. They can be extrusive from volcanic lava or intrusive as a result of slow crystallization in the earth's crust. Igneous rocks are high in silica, which make good construction materials, particularly rail ballast and aggregates for concrete and asphalt. However, they are more expensive to produce because of high rates of wear on processing equipment.

Sedimentary

The most common sedimentary rocks are limestone, sandstone, or graywacke, which are formed by the consolidation of sediment. Over millions of years, fragments of rock, sand, and soil are deposited by water action, wind, or ice and then compressed by increasing layers of material. When laid down in large quantities, material organically deposited from the shells of marine creatures can form limestone, a widely used rock quarried for cement, industrial powders, lime, flue gas desulfurization, and construction aggregates. Consolidation of chemical precipitates in multiple layers can also form sedimentary rocks.

Metamorphic

The most common metamorphic rocks are slate, quartzite, or schist, which originate from igneous or sedimentary rocks but have undergone a change due to exposure to extremely high temperatures and/or pressures. Quartzite is used in construction, and slate is extracted for specialist roof tiles and roofing felt.

Quarries can be split into two main types: rock operations or sand/gravel sites, each consisting of any of the main rock

Ian Brown, National Production Manager, Lafarge Aggregates, Mountsorrel Quarry, Loughborough, Leicestershire, UK



Courtesy of Lafarge.

Figure 10.11-1 Mountsorrel (United Kingdom) granite quarry showing multibench operation

types. Rock quarries extract material from a solid mass, from the side of hills, or from the surface working down into a pit. Sand/gravel is usually the result of weathering and consists of unconsolidated material that has been broken down and deposited some distance from the source rock—either as glacial moraine, in river floodplains, or in a marine environment. Sands and gravels usually exist together as a blend of different particle sizes and proportions. In addition, these materials will also be either rounded in shape or angular, depending on the extent of weathering by water, wind, or ice.

The location of any potential quarry should be considered vis-à-vis the distance to the market and numerous local factors when undertaking site investigations for a new quarry (Table 10.11-1).

Many countries have detailed geological records that have been built up over many years and should give an indication of the type of material and geological structure in any given area. Several methods are available to give a better indication of the type of material in a given piece of land. For rock deposits, a large drill rig is required, and the objective is to extract a solid core to allow further physical and chemical tests to be carried out. In sand and gravel deposits, samples of the unconsolidated material are brought up from much shallower boreholes by an auger. Also, geophysical surveys can be carried out, consisting of seismic testing, ground radar, and magnetic analysis to determine the size and extent of the mineral deposit.

When the geologist is satisfied that a particular area of land contains a large enough source of rock suitable for use as a construction aggregate or industrial mineral, the mineral surveyor either secures the purchase of the land or agrees to a lease with the landowner. Such negotiations usually include an upfront payment and a royalty paid on each ton of aggregate extracted. This usually time-consuming process can take many months or even years to achieve a satisfactory outcome.

The next stage is to obtain a permit or planning consent from the local regulatory authority to start operating the quarry. This process will vary from country to country.

PERMIT TO OPERATE

In most countries a permit is required before any organization can open a quarry. In many instances during the last 50 years, this process has evolved significantly and is now one of the most expensive and time-consuming aspects of setting up a quarry: US$3 million and a 5-to-10-year process span is not unusual.

Table 10.11-1 Factors affecting location of quarry

Geological Factor	Actions to Consider
Depth of overburden	A ratio of more than 2:1 is usually uneconomic.
Size of reserve	Reserve of 1 Mt (million metric tons) is the minimum required for a new sand and gravel quarry. A reserve of at least 10 Mt would be needed to justify a rock quarry.
Water table	If this is within the reserve, dewatering will usually be necessary.
Streams and rivers	An appropriate distance, usually at least 10 m, must be left between the site and water course. Sometimes a river or stream is diverted.
Rock type	Common rocks for basic products must be close to the market. Less abundant materials (for specialist uses) can be shipped some distance.
Visual impact on local residents	Landscaped amenity banks must be constructed and/or large numbers of trees planted.
Roads and railways	Reserves are sometimes lost because of the proximity of these structures, although they are also essential for product distribution. Roads or railways should be close to the plant area but not too close to the mineral reserves.
Distance to the market	Because of high transport costs, this is very important.

Local residents have a right not to be adversely affected by the quarry, and most governments have established a process to control all aspects of land development. Quarries are no exception, and permitting legislation helps to determine suitable locations and satisfy a number of conditions to ensure that the site is operated in an environmentally acceptable manner; in addition, a site restoration plan may be needed to implement when quarrying has been completed. To meet the demand of construction and other needs, the local regulatory authorities usually have a duty to grant sufficient permits to extract aggregates.

In the United Kingdom, the planning application is usually a set of documents and drawings submitted to the local mineral planning authority. The application is reviewed by professional officers and numerous third parties (such as the Environment Agency), and a decision is made by elected representatives. For example, the study work and drafting required to obtain the appropriate permits for the Lafarge Aggregates Brooksby Sand and Gravel Quarry, established in Leicestershire in 2006, consumed an estimated three full-time equivalent years. The final submission consisted of 30 separate documents, which came to approximately 750 pages and 25 detailed drawings or maps.

These documents and drawings contain a report on all potential areas of concern, with a detailed description of how the quarry will be set up and operated. The landscaping is shown, as are the site entrance, roads, weighbridge, offices, wheel cleaners, and so forth. Plans detail how the overburden will be stripped and stored, as well as how the mineral will be extracted. Details and drawings of the proposed processing plant and associated infrastructure are also included. The rehabilitation design is a key section of any application.

The planning consent or permit will usually include up to 100 separate conditions that must be complied with. Failure to

do so by the operator can result in fines or, in extreme circumstances, cancellation of the permit (Smith and Watkins 2007).

The following are examples of the type of conditions issued with a permit to open a new quarry or to extend an existing operation:

- An environmental management system is established to monitor compliance with the permit conditions and implement remedial action if necessary.
- Visual impacts and landscaping requirements include details of the type of soils mounds required around the perimeter of the site, a tree planting plan, and the site entrance design.
- Vegetation management requirements ensure that landscaped features are maintained in good condition by watering, weeding, and mowing. Trees or hedges should be replaced if they die.
- Biodiversity and ecological management may apply to animal habitats in and around the operation. For example, badger sets must be left undisturbed or relocated; ponds or lakes must be protected from pollution; and, where possible, quarry faces must be left for bird nesting.
- Fuel and chemical storage must be stored in well-maintained double-skinned tanks to prevent the risk of leakage.
- Water management plans must clearly show the location of water bodies on the site and how water is to be used on-site for mineral washing or dust suppression. Dewatering, abstraction, and discharge must be licensed and managed to ensure there is no risk of pollution by suspended solids or hydrocarbons.
- Solid waste management includes surplus materials from maintenance activities, such as scrap metal, conveyor belting, worn crusher parts, and office waste, that must be stored in segregated areas and disposed at a licensed facility, not just buried on-site.
- Dust from plant and machinery is not acceptable. It is detrimental to the health of the quarry work force, is a visual hazard to passing traffic, and can cause damage to nearby properties. Dust emissions must be minimized by the use of sprayed water or approved chemical additives during periods of dry weather. Enclosed crusher buildings can also help solve this problem.
- Noise from plant and machinery must not exceed statutory limits. This condition is imposed to protect employees' hearing and prevent operations from being a nuisance to local residents. This can be achieved by fitting suitable and well-maintained silencers to mobile equipment, putting crushers and screens inside insulated buildings, installing rubber-lined chutes, and fitting synthetic screen media.
- Vibration from blasting must be controlled. Most local authorities apply a vibration limit well below the level that may damage nearby properties. The operator is expected to design quarry blasts to achieve vibration levels within these limits. Monitoring the vibration level of each blast and reporting the results to the local authority is often a requirement of the permit.
- To minimize the effect of heavy trucks used for the distribution of aggregates on the local road network, various measures will be necessary. These include sheeting the loads, driving through a wheel-cleaning system, ensuring trucks are not overloaded, maintaining legal speed limits, and ensuring that trucks are well maintained. In some circumstances, trucks will be prevented from using certain roads based on weight limits.
- Archaeology must be investigated during soil stripping, and time must be allowed to ensure that historical remains can be investigated, recorded, and removed off-site, if necessary. This work is usually at the expense of the operator.
- Emergency procedures to protect the environment must be in place to be implemented if any installed system fails—for example, if a fuel tank leaks. It is also essential to train employees in this activity, including rehearsals/exercises.
- Soil-handling requirements during stripping and restoration are necessary. Soil can only be handled when the moisture content is low enough not to damage its structure and must be stored in separate mounds and retained on-site for use in rehabilitation.
- Time limits and hours of operation may be required. If operations are located a long distance from residential property, it may be acceptable to operate 24 hours a day. However, if not, it is common to restrict running the site between 7 AM and 5 PM, five days a week.
- A rehabilitation plan is required. When quarries were exhausted of mineral reserves in the past, they were often simply left. However, all modern operations have a rehabilitation plan built into the conditions of the permit. Options for rehabilitation include low-level agriculture or nature conservation (created by placing soils on the quarry floor), a lake formed by letting the void fill with water, or sometimes filling the site with domestic or industrial nonhazardous waste and returning it to agriculture. If the plan is established in advance, the mineral operator can plan the material handling to ensure that the final design is achieved economically and to the required standard.

These mineral extraction permit conditions have often evolved over time. Some will be the result of national guidelines or good practices applied to all mineral operators. Others will be based on the needs of local residents or are the result of negotiation between the mineral operator and the local authority.

When the conditions for the mineral permit have been agreed, it is not unusual for them to be enforced by a legal contract. Inevitably, the whole process takes months or even years to complete. The mineral operator must be prepared for a planning application for a quarry to take a long time and to make a submission well before on-site work needs to start.

QUARRY INFRASTRUCTURE

When commencing quarry work at a greenfield site, the location will have been chosen by the operator, based on the quality and quantity of the mineral and its location relative to the market and competitors.

After the necessary permits are in place, the first task will be to secure the site with fencing, gates, and tree planting around the boundary and by the main entrance from the nearest highway. At many sites, obstacles will need to be removed or diverted, such as public rights of way, overhead power lines, trees and hedges (to be cut down only if absolutely necessary), water courses, and pipelines.

If the product processing system includes a washing plant, then a water-handling system will need to be constructed. This usually consists of the licensed abstraction point from groundwater or from a nearby water course and the necessary pumps and pipelines. Water for cleaning aggregates will also need to be stored in a large-capacity pond. The residue from any washing operation is usually fine silt, which must be separated from the water before the water is reused or discharged off-site. The most economic method of handling silt is gravity settlement through a series of lagoons. Thickeners and mechanical presses are more complex and can cost up to 10 times more than settling ponds, but in certain circumstances they may be the only option. Some quarries extract from below the water table, in which case dewatering pumps and—if the waste is not then reused for mineral processing—settlement and discharge facilities will be needed.

The next task is to strip the soils from the first phase of the area to be quarried (most quarries are worked in a series of phases) and the land to be used for plant, roads, and buildings. Soils are often up to 1-m thick and can consist of topsoil and subsoil. This material is often used to rehabilitate the site in the future and should be stored carefully, including being handled only when it is dry and friable. Soil storage mounds are often placed around the boundary of the site to act as a noise barrier and minimize the visual impact for local residents.

During the planning/permitting process, a walkover and/or desktop study into the potential for finding archaeological remains on the site is conducted. The subsequent permit will instruct the quarry operator how to proceed and whom to inform if a discovery is made and what steps will likely follow. During soil stripping, an archaeologist usually observes the work in person; if anything of interest is located, the area will be cordoned off to allow further investigation, often with hand tools. This work will usually only lead to minor delays, but sometimes projects can be held up for months if the find is deemed to be archaeologically significant.

Overburden stripping is then carried out. This involves the removal of the material below the soils and above the mineral to be extracted. Overburden can consist of poor-quality rock, clay, sand, and peat, and can vary in depth from 1 to 20 m or more, depending on the local geology. The initial cut is often placed in storage for rehabilitation use at a later date. However, most of the remaining overburden should, wherever possible, be directly placed for rehabilitation in order to avoid the extra cost of multiple handling of these materials.

The most common equipment used for stripping soils and/or overburden is a combination of hydraulic backhoe excavators, articulated or rigid haul trucks, and tracked bulldozers. In the past, motor scrapers were commonly used for this work, but they are slowed by wet conditions and can cause excessive compaction of sensitive soils. However, they may still have a place in drier climates.

As the mineral is being exposed, it is usual to construct roads and foundations, install the electrical supply, erect the processing plant, construct the necessary administration buildings (such as an office, scale house, employee welfare facilities), and build the maintenance shop.

In the past, quarries have tended to extract the most easily accessible material, often with little regard for a proper plan and geotechnical design. This resulted in many accidents due to high faces collapsing or rockfall. Also, it was common for some reserves to be left in the ground because the haul road was "quarried away" prematurely and there was no access when needed.

Every quarry should have a design that includes the following:

- Local geological details
- Hydrogeology
- Direction of working or phasing and timetable
- Safe access roads
- Bench heights and widths
- Stability of the geology when material is removed
- Details of chemistry for cement or milling plant quarries
- Geotechnical surveys

The level of detail and complexity of the quarry design will reflect the scale of the operation. For example, the plan for a 100,000-t/a sand and gravel operation will contain significantly less detail than one for a 5-Mt/a blasted limestone quarry.

Although some governments have legislated in favor of strict geotechnical management of quarries, many quarries around the world still lack appropriate quarry working plans and likely operate at higher levels of risk and greater inefficiency than is necessary or indeed acceptable. The purpose of a managed geotechnical plan is to ensure the safety of the site employees and local residents and to secure the long-term economic viability of the site.

Quarry faces must be designed to ensure they are safe and the risks of rockfall are minimized. The geotechnical design must also include safe access roads, the overall slope stability of the operation, and a method to economically extract the mineral being quarried, without sterilizing any reserves. A risk assessment should be carried out to help determine how high the rock face should be; factors include the geology, results of blasting, and the size and type of the machine working the face. Historically, face heights were often limited only by drilling capacity, and 30 to 40 m was quite common. As a result of rockfall incidents, many jurisdictions have placed limits on the height of quarry faces; for example, in the United Kingdom, it is typically 12 to 15 m.

The face should be inspected daily, and loose rocks should be removed by scaling with a hydraulic excavator. All edges at the top of quarry benches must have edge protection strong enough to prevent a haul truck from going over the quarry face; this is usually an embankment of quarried material placed to a height of 1.5 to 2 m along the open edge of the haul road. Unworked faces should have rock traps at the base to ensure that falling rocks do not roll into a position of danger. Haul roads should be designed with a slope of no more than one in ten and should be at least three times the width of the haul trucks (Darlow 2007).

Stockpiles of soils, overburden, and quarry products must be designed to ensure they are stable. The underlying ground must be assessed to ensure it is able to take the weight of the stockpile, which may be more than 100,000 t, and taking into account the geology and the groundwater that may be present. The stockpile should be constructed by building up from the base and compacted in 1-m-thick layers. To ensure it is stable, the type of material being stocked will often determine the angle of the slopes of the stockpile's perimeter. The maximum safe height of the stockpile will be determined by the surface area available and the design of the slope. The location of the stocked material will also affect the factor of safety to be used in the calculations; if it is close to the site boundary, the processing plant, or water storage lagoons, the risk of damage would be greater if there was a failure. A drainage system should be

included in the design to ensure that heavy rain does not erode the surface of the stockpile or cause a major slip of the slope.

Geotechnical design should also be applied to water storage and silt lagoons, which must be constructed on a suitable base with the necessary drainage systems. The sides must be built using compacted material that will form an impermeable seal to prevent leakage from the lagoon. Sufficient freeboard, 1 to 2 m, should be included to protect against wave action, and an overflow system should be in place to cope with heavy rainfall.

All geotechnical structures must be designed by a competent engineer and constructed to that design. When in operation, these structures must be regularly inspected and maintained to ensure they remain stable and fit for purpose.

DRILLING AND BLASTING

Drilling and blasting is an essential part of the quarrying activity at a rock quarry and enables the operator to break the solid rock into suitable sizes for loading, hauling, and primary crushing. Five important factors should be considered:

1. **Safety.** There should be no premature initiation or fly rock (Darlow 2007).
2. **Ground vibration and air overpressure.** These must be minimized to avoid disturbing local residents.
3. **Fragmentation.** The rock is broken to a size suitable for further processing.
4. **Efficiency.** The operation should not take too long or be expensive.
5. **Optimization.** It is often cheaper to break rock chemically with explosives than mechanically with crushers. The blasted rock must be neither too small nor too large for the available processing equipment.

Since the 1950s, when technology rapidly developed, large-capacity drilling rigs have become the equipment of choice for creating blastholes in quarries. These machines are usually mounted on tracks or sometimes on wheels and are self-propelled with their own onboard diesel engines and compressors. Compressed air provides the power to operate the hammer drill and flush the holes clean of the rock fines generated during drilling operations.

The drill bits are often made from a wear-resistant material called tungsten carbide. However, different parts of the world prefer driving the bit in different ways. In Scandinavia, where many of these machines are manufactured, a hydraulically powered vibrating hammer is used to drill at the top of the drill rig mast. In the United Kingdom, the compressed-air-driven down-the-hole hammer is used to provide the vibrating power. In softer material, it may be possible to drill by using a hydraulic drive to rotate the drill tubes, which, in turn, rotate a tungsten carbide bit.

Whichever type of drill rig is employed, the objective is always the same: to create 75-mm- to 150-mm-diameter holes at the quarry face ready to receive explosives. Most quarry blasting is carried out using one or more rows of holes located parallel to the open quarry face (typically 12 to 15 m high). The spacing between holes and the burden (distance from the open face) is related to a number of factors and varies for almost every quarry. The blast design should start with a laser survey of the quarry face to be blasted, which assists the competent blaster in establishing the position of the face bottom, so that the burdens of the holes are within acceptable parameters. If the burden is too large, the blasted rock will contain oversize, or if too small, the risk of fly rock increases. The detailed design is usually calculated using proprietary computer software and taking account of local conditions relating to that particular face and the experience and knowledge of the blaster.

The cheapest and most commonly used explosive is ANFO, which is a mixture of approximately 95% prilled ammonium nitrate and 5% fuel oil (or diesel). This material can be mixed by hand on a small scale or more often by a truck-mounted mechanical mixer. However, ANFO needs to be initiated by a detonator-sensitive explosive and is not suitable in wet holes, as it dissolves in water.

Although nitroglycerine explosives were commonly used in quarries up to about 1990, in many parts of the world their use has been replaced by packaged slurry explosives or bulk emulsion, which are mixed from various ingredients, mostly ammonium nitrate, and are pumped into the blasthole from a truck. Initiation of these explosives was usually achieved with detonating cord, but this method was phased out and replaced by electric delay detonators in the 1970s. The microseconds delay between each hole being fired minimized the accumulated ground vibration without affecting fragmentation.

A decade later, nonelectric shock tube (known as nonel) became the most common means of initiating quarry explosives. This was due to the reduced risk of premature initiation from radios, thunderstorms, or other interference. Nonel systems were also cheaper to manufacture. In recent years, the world of electronics has been applied to quarry blast initiation. It is reliable, has almost infinite settings for accurate delays, and is safer to use. Although vibration control and/or fragmentation is often improved, the cost of programmable initiation is currently too high for its common usage in aggregate quarries.

FACE LOADING

Sand and gravel quarries are often to be found in valley floors or floodplains—that is, under the water table. The most effective way to deal with this is to dewater the working area and excavate dry, because it is often safer and cheaper. If pumping is uneconomic and working underwater is the only option, this can be done with a dragline or alternatively (due to significant improvements in technology) with a long-reach hydraulic backhoe excavator. Alternatively, when working sand/gravel from underwater, a dredger can be used. These machines are built on pontoons or attached to ships and can operate as suction pumps, grabs, or bucket elevators. Their operation is not dissimilar to dredgers used in mining of placer deposits. At the top end of the scale, these dredgers can operate in large freshwater lakes or at sea.

Because sand and gravel is usually unconsolidated, its extraction from the ground is achieved without the need for in-situ breaking, although some deposits may need some additional ripping with a tooth attached to the back of a large bulldozer or a heavy-duty excavator.

Since the post–World War II reconstruction boom of the 1950s and 1960s, diesel-driven draglines were commonly used in sand and gravel quarries to excavate and load haul trucks or field conveyors. Draglines were able to sit on top of the seam of sand and gravel and reach depths of up to 10 m, and it was not necessary to construct good quarry floors. They were also used on some sites to strip and cast the overburden.

Currently, many quarries use wheel loaders to excavate and load sand and gravel, as long as the floor is competent



Courtesy of Lafarge.

Figure 10.11-2 Typical method of excavation of blasted rock by a backhoe machine

Table 10.11-2 Advantages and disadvantages of tracked and wheeled loading equipment

Backhoe Excavators	Wheel Loaders
Easy to operate	Easy to operate
Minimal whole-body vibration	Significant whole-body vibration
High rate of productivity	Variable productivity
Good fuel consumption	More fuel per ton consumption
Long-lasting tracks	Tire life dependent on ground
High breakout force	Lower breakout force
Slow mobility	Good mobility
Able to drop load into haul trucks	Loads on same level as haul trucks
Longer reach for scaling loose rock	Limited in scaling ability
Useful for secondary breakage	Load-and-carry capability

enough to carry heavy-wheeled traffic in all types of weather. This method is commonly utilized in North America, where in many areas these machines work safely and efficiently, as long as a number of core factors are present: competent operator, suitable sized machine, dry level working area, and a working face height that is less than the maximum reach of the machine. In many parts of the world, sand and gravel quarries do not have the right geological or weather conditions to operate using wheel loaders, so the machine of choice is the hydraulic backhoe excavator. Set up correctly, such machines will work the sand and gravel face in a safe and cost-effective manner.

Loading at the face in a rock quarry has closer links to open-pit hard-rock mining for metal ore than to a sand and gravel operation, but the type of machines available are similar: wheel loaders, face shovels, and hydraulic backhoe excavators. However, the equipment in quarries and mines is usually larger than that employed with sand and gravel.

In North America, the blasted rock is usually loaded into haul trucks by wheel loaders. The trend on other continents is to excavate and load shot rock using hydraulic backhoes or face shovels (Figure 10.11-2). Because both methods have advantages and disadvantages, each quarry operator should consider its own application carefully before deciding on which method (and equipment) to employ (Table 10.11-2).

HAULING TO THE PLANT

After the sand and gravel or blasted rock has been excavated, it is often transported to a fixed processing plant, which is usually on the same site and within 1 km of the working area. Since World War II, the diesel-driven haul truck (in ever increasing sizes) has monopolized this work. The haul truck is simple to operate, flexible because it can drive anywhere within the quarry, reliable, and effective.

Two main types of haul truck operate in modern quarries. The rigid haul truck (RHT) is a two-axle machine with rear wheel drive and easily recognizable by the large body canopy extending over an offset cab. Its capacity ranges from 30 to 300 t, and it is more commonly found in a rock quarry where its higher payload provides economies of scale. Some large sand and gravel quarries in North America also use rigid haul trucks where the geology and weather conditions combine to allow good haul roads to be utilized.

The other category of haul truck is the articulated haul truck (AHT). This is a smaller machine, capacity 20 to 50 t, which has three axles and an articulated hitch between the centrally mounted cab and the material-carrying tipping body. Usually a six-wheel-drive machine, it is designed to cope with wet, sticky conditions and is commonly found in European sand and gravel quarries. AHTs are often used for hauling stripped overburden to stockpiles and tips, usually because all-wheel-drive traction is required for difficult ground conditions.

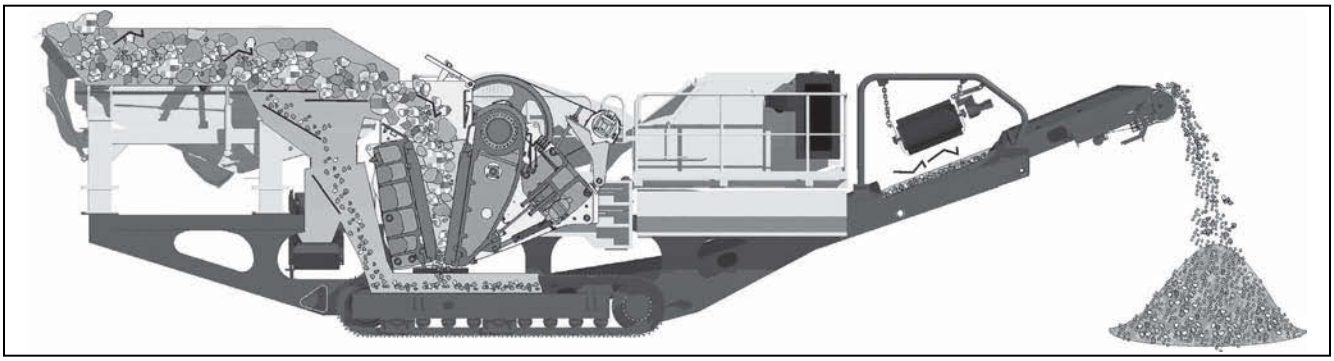
Several factors are important in optimizing the safe and efficient use of haul trucks, both RHTs and AHTs: a good haul road, the right-sized machine, and the size of the haul truck fleet.

Good Haul Road

The road should have as hard a surface as possible, an incline no steeper than 10%, and a camber to allow drainage into ditches or runoff areas. The road should also be at least three haul-truck widths wide and have edge protection made from quarried granular material at least 1.5 times as high as the largest haul truck tire using the route. The haul road must be maintained to ensure it remains safe and effective to use. For this work, the best machine is a road grader. In dry conditions a water bowser will also be required for dust suppression.

Right-Sized Machine

A truck that is too big to feed the processing plant or too small for the quarry excavator has few benefits to recommend it. The truck should be loaded in four to five passes of the excavator or wheel loader. To enable it to tip into the primary crusher, the capacity needs to be of an appropriate size. The tipping body should be suitable for the material being quarried. If the rock being carried by the haul truck is abrasive, additional protection will be carried to minimize the cost of excessive wear. Hardened steel or rubber liners can be effective. In particularly difficult conditions, a suspended rubber body could be fitted to the haul truck, similar to a steel haul truck body, except the floor is made of industrial rubber and suspended from the steel frame by special ropes. In cold conditions it may be necessary to heat the tipping body to prevent wet material from sticking. Heating is achieved by directing the engine's exhaust through channeling in the tipping body.



Courtesy of Metso Minerals.
Figure 10.11-3 Metso track-mounted mobile jaw crusher (including a magnet for removing tramp metal)

Size of Haul Truck Fleet

The factors that determine the size of the haul truck fleet are the size of the individual trucks, the tons per hour required to feed the plant, the distance and gradient from the loading point to the hopper, expected truck availability, and productivity. Having too many trucks will be expensive and lead to delays in the haul cycle. Too few trucks will produce inefficiency and leave the excavator and/or the plant running empty. In addition, maintenance of a single machine should not affect the production schedule.

For some quarries, particularly in sand and gravel, the most cost-effective method of material haulage is a field conveyor that runs from the excavator directly to a surge pile at the processing plant. This type of system is usually feasible only where it is unnecessary to blend material from different benches and where the gradient is shallow. Usually the conveyor system is made up of a number of straight sections, changing direction at transfer points. These transfer points often cause material spillage. During the last few years, some quarry operators have installed shallow radius curved conveyors, which can be a cost-effective way of avoiding expensive conveyor transfer points. As quarries are extended, often the most cost-effective means of connecting the new extraction area with an existing plant is by a long field conveyor from a hopper at the edge of this new area, which is fed by a smaller team of haul trucks.

A number of disadvantages are associated with conveyors: they are not very flexible, (i.e., they cannot be moved from day to day), the installation cost is often high, and difficulties often exist in providing power supplies. However, they do have many advantages. They require little labor; are reliable, productive, and quiet; reduce on-site heavy vehicle movements; and only need small amounts of dust suppression at transfer points. They do not require large haul roads and can be a safe and low-cost method of transferring material.

IN-PIT PROCESSING

The traditional quarry consists of the working face, a haul road, and a fixed processing plant. However, in the second half of the 20th century, during the huge surge in demand for construction aggregates, many North American operators set up quarries with portable equipment so they could supply projects in remote areas, such as dams, bridges, and power stations, and also optimize the use of their equipment during seasonal construction campaigns. Equipment suppliers rose to the challenge by designing and manufacturing portable quarry equipment, which has been further developed and improved over the years. A typical crushing *spread* consists of a number of wheel-mounted chassis for crushers and screens, with wheeled hoppers and stacking conveyors for stockpiling material. The machines are linked together and usually powered by a diesel generator.

In Europe since about 1990, there has been a revolution in the track-mounted aggregate processing plant. Scandinavian manufacturers, such as Metso (Eloranta 2009) and Sandvik, were the pioneers of this technology and are still the global leaders. The machines consist of a set of Caterpillar-type tracks, used on backhoes and dozers, and a strong chassis. Fitted at one end is a feed hopper and grizzly screen. In the middle are the crusher (jaw, impactor, or cone) and the diesel/hydraulic power source. At the other end is a discharge conveyor (Figure 10.11-3). Over the years, these primary units have been modified to include a sizing screen with more discharge conveyors and a closed-circuit return belt back to the crusher. This type of unit is now available from numerous manufacturers in a variety of configurations, including the installation of different types of screen, secondary and tertiary crushers, hoppers, stacking conveyors, and other attachments.

This type of track-mounted processing plant is usually located at the quarry face and can consist of one unit producing a crusher run material or can be linked by conveyors to a fixed secondary processing plant. Alternatively, it can be linked to two or three other track-mounted machines to form a complete plant on the quarry floor.

In-pit processing has an advantage in the following situations:

- If the quarry face has moved a long distance from the processing plant, expensive haulage with off-highway trucks can be eliminated by crushing at the face. The material can be transferred to the fixed secondary plant by a conveyor system.
- Small reserves of aggregate that do not justify the cost of a fixed plant can be operated using tracked mobile machines, which then move to the next reserve on completion.
- Seasonal campaign processing is common in parts of North America. A single portable crushing plant can travel between groups of quarries.
- In long-established quarries, the processing plant is commonly located on good aggregate reserves. As the original quarry becomes exhausted, it is often cost-effective to

Table 10.11-3 Advantages and disadvantages of belt conveyors

Advantages	Disadvantages
Continuous operation	High capital costs
Can move large volumes of material	Unreliable if not maintained properly
Simple to maintain	Not flexible, after it is installed
Quiet in operation	
Low operating cost	



Courtesy of Lafarge.
Figure 10.11-4 Excavator and conveyor

dismantle the old plant and extract the reserves beneath it with a tracked mobile unit.

MATERIALS HANDLING

Aggregates manufactured from rock (both blasted, and sand and gravel) consist of graded particles usually less than 100 mm in dimension, which are heavy and often abrasive for quarry plant and equipment. The crushing and screening of aggregates is a continuous process, and the handling of material between the plant's major components is as important as the crushing and screening operations themselves. A modern quarry plant will consist of hoppers, feeders, conveyors, storage bins, surge piles, chutes, and transfer points.

The most commonly used material-handling equipment is the belt conveyor. It has numerous applications and, if constructed and maintained in the correct manner, will be reliable and effective (Table 10.11-3).

Numerous types of conveyor are available for almost every conceivable application in a quarry.

- Field conveyor: long-distance material transport (Figure 10.11-4)
- Curved conveyor: wide radius curves to eliminate transfer points
- High-angle conveyor: very steep, or vertical, sandwich belt
- Plant conveyor: typical structural steel conveyor within a plant
- Truss conveyor: for heavy-duty applications bridging large gaps
- Stacking conveyor: inclined to discharge on a stockpile
- Radial conveyor: stacking conveyor that moves sideways in an arc
- Shuttle conveyor: moves horizontally to feed hoppers or storage bins
- Tripper conveyor: horizontal with a moving discharge point
- Pipe conveyor: where the belt wraps around to enclose the material

Conveyors move material around the quarry, but the material needs to be fed between conveyors or to/from processing equipment by feeders. There are several types:

- Belt feeders. Belt feeders are usually very short conveyors (1 to 3 m).
- Vibrating pan feeders. Usually electromechanical, the vibrating motion is provided by conversion of the rotary motion of an electric motor and an eccentric weight into an inclined stroke.
- Apron feeders. These feeders are heavy-duty chain-driven steel conveyors.
- Grizzly feeder. Often used to feed a primary crusher, its vibrating bars have a dual purpose of removing undersize material and feeding the oversize.

One of the most frustrating problems experienced when operating conveyors is potential material spillage around some transfer points and in particular underneath the conveyor from the return rollers. Many types of belt scraper are available for minimizing this problem. One method that can almost eliminate spillage under the conveyor is the belt turner, a device that turns the empty belt over at the conveyor's tail. This allows the belt to return right side up, with the clean side in contact with the rollers. As a result, any material that remains in the belt stays on it rather than being knocked off. When the belt reaches the head of the conveyor, another belt turner turns the belt over again to carry its normal load.

CRUSHING

The use of quarry explosives is the first application of energy to the virgin rock in its journey to being converted into a marketable product. Blasting is an important and effective form of crushing. A well-executed blast transforms a solid rock formation into fragments small enough to be accepted by a processing plant. It is generally cheaper to effect size reduction chemically (explosives) than mechanically (crushers), albeit with far less control (Rothery and Mellor 2007).

The manufacture of aggregates requires larger rocks to be broken down into smaller rocks; for example, blasted rock may be up to 1 m across, whereas rail ballast is a 50-mm-diameter product, and concrete aggregate is usually 20 mm in size. Various stages of mechanical crushing are necessary in the production of aggregates, with the following key objectives:

- Maximize product yield
- Minimize wastage
- Optimize the particle shape
- Remove deleterious material
- Minimize energy consumption
- Optimize crusher wear characteristics

A typical rock quarry will usually have three or four stages of crushing:

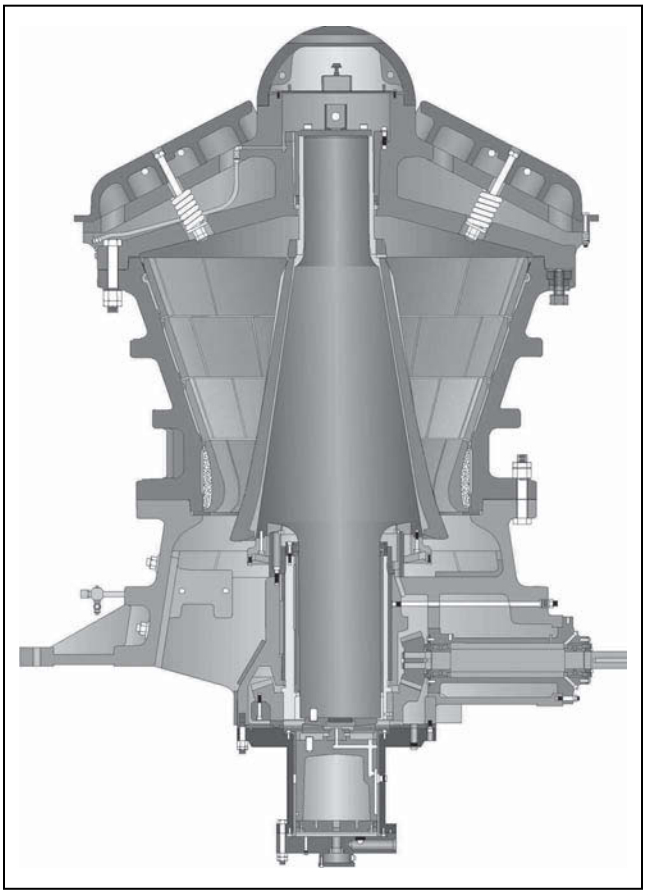
1. Primary crushing: Blasted rock is reduced from up to 1-m diameter to less than 300 mm.

2. Secondary crushing: The product from the primary crusher is further reduced in size to 50–60 mm, which may form some of the final products.
3. Tertiary crushing: Taking the secondary product produces the final aggregate sizes, usually –20 mm.
4. Quaternary crushing: This may be used to produce manufactured sand or recrush surplus oversize.

Several types of crushers are used in numerous applications: gyratory, jaw, cone, impact, and roll.

Gyratory Crusher

A gyratory crusher (Eloranta 2009) consists of a long spindle carrying a hard steel conical grinding element, known as the head, seated in an eccentric sleeve. The spindle is suspended from a *spider* and, as it rotates (about 80 to 150 rpm), it sweeps a conical path within a fixed crushing chamber. Maximum movement of the head occurs near the discharge at the bottom. During the crushing process, large rocks are compressed between the rotating head and the top shell segments of the crushing chamber, becoming smaller as the material drops through the machine. Gyratory crushers are large-capacity machines (up to 10,000 t/h) and are only used in primary applications. A gyratory crusher is suitable for most rock types (Figure 10.11-5).



Courtesy of Metso Minerals.

Figure 10.11-5 Cross section of a gyratory crusher

Jaw Crusher

A jaw crusher consists of two steeply inclined, heavy-duty metal plates (known as *jaws*) within a cast steel frame. A wide opening at the top receives blasted rock, and the product is discharged through a smaller opening at the bottom. One jaw is fixed, while the other is powered to swing back and forth.

Jaw crushers can be used as primary crushers for many different rock types unless very large capacity is required. They are usually designed according to the size of their feed opening. The first dimension is the width of the jaw, and the second is the gape (distance between the fixed and the swing jaws at the entry to the crushing chamber). *Scalping-off* (or removal) of –50-mm-diameter material usually takes place before the crusher and feeder, as this is essential for efficient operation. All jaw crushers have a large flywheel attached to the drive, which stores energy on the idling half of the stroke and delivers it on the crushing half. There are two main types of jaw crusher:

1. Double-toggle jaw crusher: The swing jaw is suspended from a shaft, and its crushing cycle comes from the rotation of the eccentric shaft, causing the pitman to rise and fall, which, in turn, rises and lowers the twin toggle plates, moving the jaw stock in and out. Although double-toggle jaw crushers are not commonly used, they are suitable for hard, abrasive rock.
2. Single-toggle jaw crusher: A driven eccentric shaft passes through the top of the swing jaw, creating an elliptical motion at the bottom of the jaw. This is a simpler design than the double-toggle machine, with fewer moving parts, and is suitable for many primary crushing applications.

Some older quarry plants may use small jaw crushers as secondary or tertiary machines.

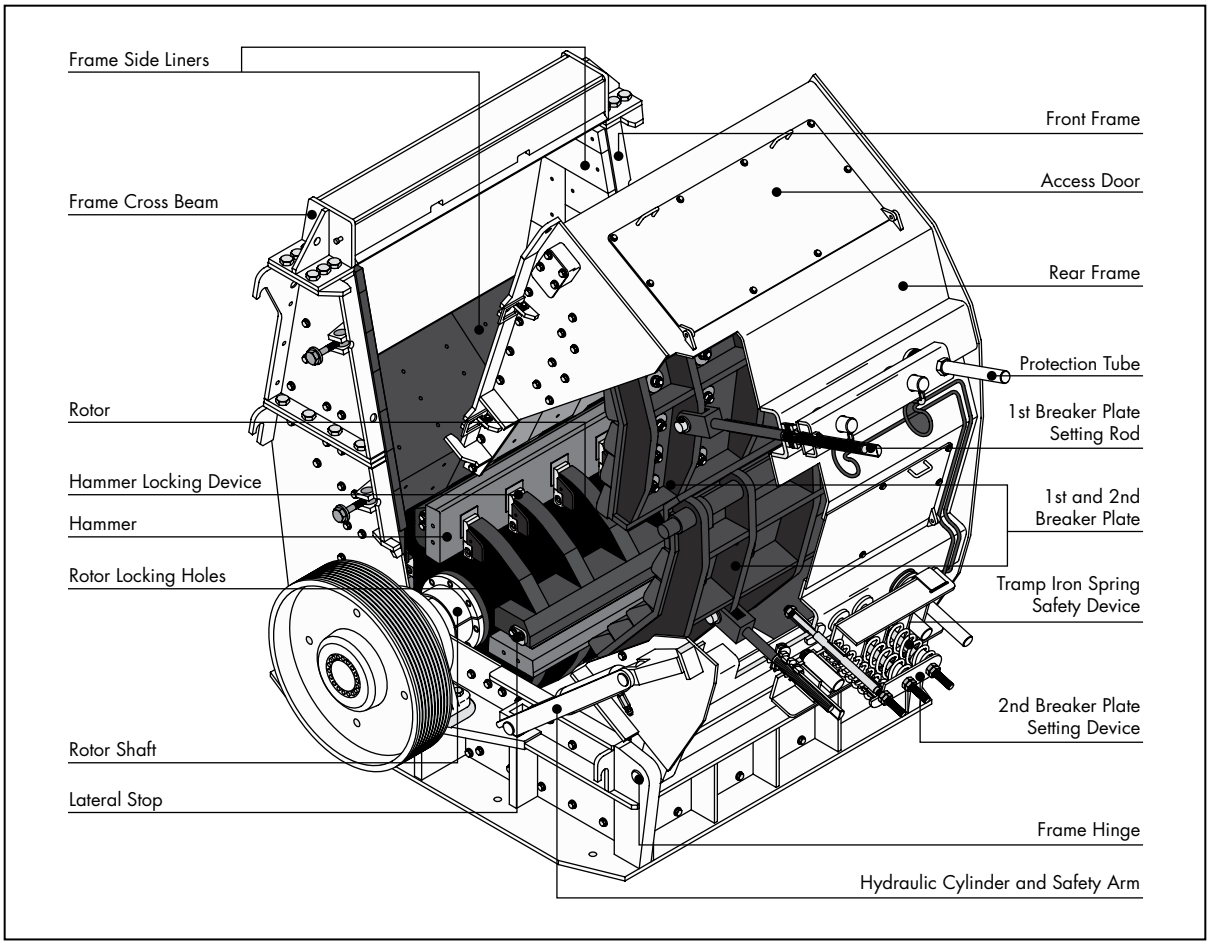
Cone Crusher

A cone crusher is a smaller and modified version of a gyratory crusher. The shorter spindle of the cone crusher is suspended from a spider in some models but not in others. The capacity of a cone crusher is directly proportional to the diameter of the head, which can be quite high, given sufficient motor power. Also, the steeper the head angle, the larger the capacity. The *throw* of a cone crusher is many times greater than a gyratory crusher, and the cone crusher operates at higher speeds.

Older machines were designed to operate at a slower fixed speed, whereas newer models are faster and adjustable within a speed range. Cone crusher speed is inversely proportional to capacity but directly proportional to size reduction; for example, as the speed of a cone crusher increases, the capacity decreases, and the product becomes finer.

The wide displacement of the head at each stroke is at a speed that allows each piece of rock to fall under gravity and be caught farther down by the rising head (on its return stroke). In this way, the material passing through the crusher is subject to a series of hammer-like blows rather than being gradually compressed, as is the case in a gyratory crusher. At the bottom of the crushing chamber, the faces of the upper and lower mantles are parallel so that all rock will be crushed and at least one dimension is equal to or less than this closed side setting.

The cone crusher is a well-established machine in the production of aggregates and is suitable for many applications (except primary crushing) and most rock types. However, older cone crushers have been criticized for sometimes producing material with a poor product shape. The newer



Courtesy of Metso Minerals.

Figure 10.11-6 Cross section of horizontal impact crusher

generation of high-speed cone crushers virtually eliminates this characteristic.

General principles for good cone crusher performance follow:

- Ensure that the motor is big enough for the duty, and consider the benefits of variable speed inverters and direct drive from the motor to the crusher.
- A controlled and continuous unsegregated feed, distributed evenly to the full periphery of the crushing cavity, is essential for good performance.
- Avoid feeding sticky, wet, or contaminated rock to a cone crusher because this will reduce its throughput capacity.
- The closed side setting (i.e., the smallest gap between the upper and lower mantle) should be set in accordance with the products required and once set should be monitored regularly to ensure it does not deviate.
- Magnets and metal detectors should be fitted within crushing circuits to pick off the numerous pieces of metal that wear away from the plant. Such devices protect the crushers from damage and unnecessary downtime.

Impact Crusher

An impact crusher (Eloranta 2009) consists of a fabricated steel chamber with breaker plates mounted at certain points. In the middle of the chamber sits a horizontal rotor, which, electrically driven at high speed, carries fixed blow bars or swing hammers. Large rocks are fed in at the top, and, after a series of high-speed collisions between the rock and the rotor, as well as between the rock and the chamber linings or even between rock particles, much smaller rocks are discharged from the bottom of the machine. Although most impact crushers have a horizontal shaft (Figure 10.11-6), some specialist applications, such as shaping aggregate or manufacturing sand, employ a vertical shaft impact crusher.

Impact crushers can be used in fixed or portable applications and at primary, secondary, or tertiary stages but only with less abrasive rock. The impact crushing process causes immediate fracturing of the rock, which can be important in the production of high-quality construction aggregates. Impact crushers are also well known for their high reduction ratio, good aggregate shape, and the generation of significant quantities of fines. When considering using impact crushers, the desired product type needs to be evaluated.

Methods of adjustment–product grading can be changed by altering the following:

- Rotor speed
- Feed rate
- Position of breaker bars
- Metallurgical content of blow bars
- Spacing of grid bars

Another type of impact crusher is the hammer mill, where high-strength metal hammers are pivoted on the rotor rather than as fixed blow bars. The hammers can pivot out of the way of oversize or tramp metal and are usually best suited for a smaller throughput and for softer material; otherwise the *way* rate would be too high. (Note: The way rate is that percentage of material that passes uncrushed through the mill.) The base of the crusher chamber is perforated so that only products of a certain size can pass through; any oversize passes through the crushing process again. The hammer mill will achieve much of its crushing by stone-on-stone attrition in the crushing chamber, which makes it difficult to control particle size. Although the product will contain a large proportion of fines, the aggregate products will have a good cubical shape.

Roll Crusher

Roll crushers are still used in some quarries, although in many cases they have been replaced with modern cone or impact crushers. However, they have a use for crushing friable, frozen, or sticky materials such as chalk, gypsum, and soft limestone.

The most common type of roll crusher consists of two horizontally mounted steel cylinders, which are free to revolve toward each other. One of the rollers usually rotates around a fixed-point axle, while the other has its axle set at a variable distance away. This mechanism provides the adjustment for different sizes of feed material and products. Diameters and speeds of the rollers can be varied to produce different sized products. Some roll crushers have a single cylinder rotating to a fixed plate, whereas others have multiple rollers, but these machines are less common.

The feed to a roll crusher must be evenly spread across the full width of the roller; otherwise wear rates will be differential. The rollers are often smooth and lined with abrasion-resistant manganese steel to make them last longer. For some particularly friable or sticky materials, the rolls are made with protruding, intermeshing teeth that dig into the material through a process of ripping and compression, which helps to pull the material into the machine for crushing.

SCREENS

Screening is the separation of aggregate particles into various size ranges and can vary from the separation of 50-mm scalpings from blasted rock to the separation of coarse dust from fine dust. A typical screen consists of a rectangular steel frame mounted on springs, to which the screen media is attached, and an exciter unit to vibrate the frame and create the screening action. The exciter unit usually consists of out-of-balance weights mounted on a shaft driven by an electric motor, which may be attached to the screen or mounted on the support structure and connected through V-belts. The weights can be adjusted to give the required amplitude and stroke of the screen, which, in turn, determines its performance in screening different types of materials into a wide variety of products.

The following screen types are used:

- Dry screens for blasted rock processing
- Wet screens for sand and gravel washing
- Inclined screens for general sizing
- Horizontal screens for final product sizing
- Dewatering screens for removing surplus moisture from sand
- Multideck screens for making more than one product
- High-energy screens for separating sticky material

The following are guidelines for good screen performance:

- Screens should be sized at least to 133% of the required capacity.
- Springs must be regularly maintained.
- Material bed depth should not exceed four times the aperture of the screen media.
- Material should be fed evenly across the full width of the screen.
- Discharge chutes should be large enough to avoid material flow restrictions.

Screen Media

Three main types of material are used: steel, rubber, and polyurethane, with numerous designs for each of these media. The selection of a particular medium depends on many factors and should be analyzed carefully. For example, polyurethane lasts much longer than steel but is also more expensive. Rubber will also last longer than steel but does not have as much open area. Typical screen media used in quarries are

- Punched steel plate for scalping blasted rock,
- Heavy-duty woven wire for medium-sized limestone products,
- Fine piano wires for removing oversize from dry sand,
- Tensioned rubber for wet screening and abrasive materials,
- Tensioned polyurethane for sticky or fine materials, and
- Modular polyurethane for dewatering screens and abrasive material.

SAND AND GRAVEL WASHING

The majority of sand and gravel produced around the world needs to be washed to remove deleterious material such as clay and silt. Sand and gravel deposits contain these materials because they have been laid down in relatively recent geological times by rivers or glaciers.

Most sand and gravel quarries produce less than 500,000 t/a. Therefore, the processing plant (Figure 10.11-7) is relatively small compared with many hard-rock quarries. Sand and gravel, brought from the quarry face by haul trucks or field conveyors, is usually fed to the plant from a hopper or a surge pile onto a washing screen fitted with numerous spray bars (delivering 20 to 40 L of water per minute per ton of feed). This screen simply removes any large lumps of clay or other oversized material, which is retained on the top deck. Gravel retained on the bottom deck is sent for further processing, and fine material passing the bottom deck, with most of the water, is sent for separate processing and dewatering.

Some quarries where the sand and gravel is contaminated with lumps of clay have fitted ingenious devices to remove this material. Part way along the conveyor to the first screen, the troughing rollers are sometimes replaced with flat rollers for a couple of meters to enable a rotating spiked shaft to be employed, which knocks large lumps from the belt. This is simple but effective.

Although the gravel from the washing screen may be clean enough to be screened into its product sizes, stockpiled, and sold, further washing is often required. Washing trommels or scrubbing barrels are large horizontally mounted cylinders, perhaps 2 m in diameter and up to 10 m long. Traditionally, a trommel was driven by a chain on a toothed sprocket encircling the unit, and the whole unit rests on two or more rings on

solid rubber or polyurethane trunnions. Today, a trommel is usually driven through pneumatic tires rotating on a shaft.

Trommels are almost autogenous (i.e., they rely on gravel, sand, and clay tumbling in slurry, and rubbing and rolling against each other to loosen and bring into suspension any silt or clay). The transport of material through these machines usually relies on the discharge aperture being greater than the feed aperture, as the fall between the two is usually only 5°. The dwell time for materials in the trommel is around 2 minutes, which is usually sufficient to clean the material to the required standard; for particularly dirty material a longer dwell time would be set. Trommels are often lined with wear-resistant rubber liners with rubber lifter bars installed longitudinally to assist in cascading the material in order to improve the cleaning action. The peripheral speed ranges from 16 m/min in washers to twice this speed in scrubbers. (Note: Scrubbers operate at about 60% of critical speed—that is, the speed at which the contents would cling to the sides by centrifugal force). Trommels use approximately 25 L of water per minute per ton.

Log washers are used for particularly dirty material. They consist of two counter-rotating steel shafts (logs), fitted with many steel blades, which agitate and scrub the material, pushing the coarse fraction up the inclined tank, with the fines overflowing the weir at the back. The two shafts rotate inward toward each other so the material is forced into scrubbing contact. Blade mills work in a similar manner to log washers except they contain an Archimedean screw instead of the bladed logs.

The discharge from these machines is usually fed to another screen, which gives an opportunity (by using further spray bars) to rinse any remaining fines from the cleaned gravel. The clay, sand, silt, and water are screened out to a collecting sump from where they join the fines-handling process. Cleaned gravel is then fed by conveyor to a product-sizing screen or crusher if necessary.

During sand production, clay and high levels of silt in sand and aggregates used for concrete cause weakness and therefore have to be removed, usually by washing. Particle size definitions used to describe sand production are as follows:

- Sand: +75 μm to 4 mm
- Silt: +2 μm to 75 μm
- Clay: –2 μm

Methods of Processing Sand

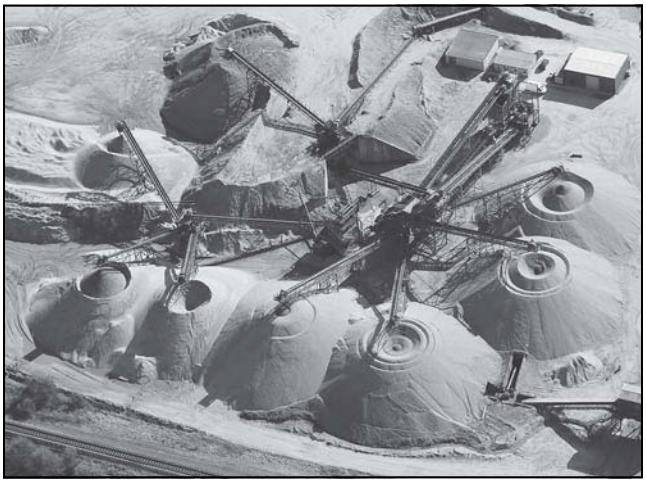
From the collecting sump, a number of methods are available for separating the sand from the silt and clay and at the same time classifying the sand into different grades based on particle size (Littler and Millburn 2007).

Free Settling Classifiers (30% to 70% efficient)

Free settling classifiers operate on the principle that sand settles out of slurry into a sump by gravity and is then mechanically separated and partly dewatered by rakes, bucket wheels, or screws. From there, it is often carried by a radial stacking conveyor for further dewatering by gravity. The silt and surplus water overflows a weir adjacent to the bucket wheel or screw and is then pumped to settling lagoons or mechanical thickeners.

Horizontal Current Classifiers (<50% efficient)

Horizontal current classifier machines are commonplace in North American sand and gravel quarries but rarely seen in Europe and other parts of the world. These classifiers provide consistently graded products from a varying feed and allow enough flexibility to achieve the required specification. The long horizontal tank typically consists of 11 compartments, known as stations. Slurry enters at the feed end, where the coarser (i.e., larger) particles begin to settle out. Settling continues along the length of the tank, with the particles that settle out in each station being smaller than from the previous. Because horizontal classifiers rely on free settling, a large amount of fines, unfortunately, are entrained in the early coarse sand stations. This can be overcome by introducing (through slatted plates) a rising current of water, which pushes particles farther down the tank. The required grade of sand can be obtained by blending the material from a number of different stations, each of which has three outlet pipes discharging at 60% to 70% solids by weight into three separate flumes. Each flume runs the length of the tank, picking up material from all stations and transferring it to a dewatering system, which often involves a fine-material screw classifier.



Courtesy of Lafarge.

Figure 10.11-7 Alrewas sand and gravel washing plant

Hindered Settling Classifiers (80% to 90% efficient)

Hindered settling occurs when the settling rate of a particle (within the sand slurry) is increased. The transition from free settling to hindered settling occurs as the concentration of the solids in the slurry increases. This reduces the distance between particles sufficiently that the drag force created by the settling particles affects the movement of nearby particles. However, if the concentration of solids in the slurry is too high, then entrapment and misplacement of particles will dominate. One of the most common hindered settling machines currently in use is the hydrosizer, which works in the following way:

1. The slurry is fed through the top opening of the hydrosizer with additional feedwater (if required).
2. The upward current of water is created by introducing more water through the teeter plate, at the bottom.
3. The upward flow of water counters the settling velocities of the solids.
4. The remaining solids build a high-density fluidized bed until a desired height (or bed thickness) is achieved.
5. After the desired bed thickness is reached, an automatic controller is used to discharge the product at the bottom.
6. Surplus water and silt are discharged from a weir at the top.

Elutriators (<50% efficient)

An example of this group of equipment is the S-type classifier, which is designed to take a continuous feed of up to 25% solids by weight. Feed is passed down a central tube until it meets a deflector plate and is turned radially outward. At this point the flow is split, with fine sand and silt tending to move upward to the overflow, while the coarser particles move downward to the underflow. Which particles move in which direction is determined by the velocity of the upward current, with inevitably some fine particles becoming entrained with the coarser particles and passing downward. A second classifier is often required to further separate the overflow into fine sand and silt.

Hydrocyclones or Hydroclones (60% to 80% efficient)

The hydrocyclone is a simple machine with few, if any, moving parts. It consists of an upper cylindrical section over a lower downward-tapering conical section, the bottom of which is known as the spigot and is 20% of the upper section's diameter. A hollow pipe, the vortex finder, extends through the top of the cyclone and is usually around 30% of the upper section's diameter. Hydrocyclones are usually constructed in sections for ease of maintenance and are lined with polyurethane or rubber to minimize wear. The feed of slurry, sand, and water enters the unit tangentially via a pipe and onto an involuted feed well, which, in effect, wraps around the top of the cyclone. This process pressurizes the feed, which is forced onto the cyclone walls under centrifugal force. The relatively density of sand is 2.5 times that of water and as such forces its way to the walls of the cyclone, pressing out the less dense water, which moves toward the central axis of the cyclone. The partially dewatered sand spirals down the inside of the cyclone and discharges through the spigot. Dirty water exits through the vortex finder into a lower pressure zone. The purpose of the vortex finder is to stop the sand in the feed shortcutting to the overflow.

After sand exits any of these machines, it should be dewatered prior to stockpiling. The most efficient machine for this task is the dewatering screen, which usually accepts sand at 60% to 80% solids by weight and discharges it at greater than 90% solids. Dewatering screens are often horizontal or inclined slightly up at the discharge end. The decks can be stainless-steel wedge wire or slotted natural rubber. However, by far the most common screen media is modular polyurethane. The underflow water is usually fed back to the sand plant in order to recover any remaining fines.

Management and Disposal of Silt

The waste from sand and gravel washing is a suspension of silt and clay in water. By far the most economic way of dealing with this material is to allow it to flow by gravity to settling ponds, which are usually excavated in the floor of the quarry where the sand and gravel has been extracted. The ponds should be geotechnically engineered to ensure they are stable and do not leak. The silt and clay settle out by gravity usually in the first of a series of ponds, the water in each pond becoming successively cleaner. This cleaned water is often pumped back to the washing plant for further use. Ideally, the silt ponds remain where they are and (when they are full) form part of the quarry restoration.

However, in a number of cases, settling ponds are not practical because of, for example, lack of space, severe shortage of water, no planning consent, high silt content, environmental issues or proximity to local amenities, or the airspace/quarry void has been allocated for landfill. If any or a combination of these factors apply, then the silt-laden water has to be mechanically and chemically (with flocculent) thickened prior to discharging to the silt pond. This will provide recycled water for immediate reuse in washing, thereby reducing the size of the silt pond. If no space exists for a silt pond, the thickened silt has to be made into a semidry cake using a filter press.

The conventional silt thickener consists of a large tank containing a slowly revolving rake mechanism. The dirty water is fed in at the top of the tank where a measured amount of flocculent is added. This causes the silt particles to coagulate together, which makes them heavier, and they sink toward the bottom of the tank. The clean water overflows at the top of the tank, usually to another adjacent tank for immediate use in the washing process. The thickened silt is drawn to the center of the tank by the revolving rake, from where it is pumped to a silt pond or filter press.

The filter belt press is one of a number of systems available that will create a semidry filter cake. The thickened silt is treated with more flocculent and is forced between two filter cloths that are pressed together with mechanically driven rollers to remove the surplus water. The water is reused, and the semidry cake is usually stockpiled by the conveyor, to be disposed of by haul trucks.

A more efficient method of filtering is the plate press, made up of a number of recessed plates forming chambers into which the thickened silt is pumped. The plates are supported in a fabricated steel frame and are pushed together by the force of a hydraulic ram. Each plate is covered by a filter cloth, which retains the solid particles but allows the water to pass through. The solid particles gradually build up on the surface of the filter cloth to form a solid cake. After a predetermined period of time, the pressure is released, the filter plates separate, and the cake falls into a stocking bay to be disposed of later.

Removing Lignite from Sand

Some sand and gravel deposits contain small amounts of organic matter and/or lignite, which will cause damage in concrete if not removed. Several ways to separate these deleterious materials rely on the principle that these materials are less dense than sand. One of the more common methods of lignite removal uses an upward current of water in a tall cylindrical tank, which collects the much lighter lignite particles that overflow from a weir at the top of the tank. This mixture of lignite, fine sand, and water often flows over a sieve bend, or curved steel fixed screen. The water and fine sand return to the product, while the lignite is removed by a separate conveyor or chute. If lignite is still present in the sand, the process is repeated in a second adjacent set of equipment.

Dry Screening

In many cases natural sand can simply be dry screened straight from the quarry face without the need for complicated and expensive washing equipment. This is typically done with a mobile screen fed from the face by a wheel loader into a hopper. Sand then passes up an inclined conveyor to a double-deck screen, the top deck of which, having 20-mm apertures, is used simply to protect the bottom deck, which is often a series of longitudinal piano wires. The oversize and the product are usually stockpiled separately with conveyors. The productivity of a dry screen will depend on the feed size

and moisture and clay content, and in some cases has to be processed twice.

Manufactured Sand

In certain parts of the world, no local deposits of sand are available, or, if present, they are unsuitable for use in construction materials. In such areas, the only solution is to manufacture the sand.

From a normal crushing and screening process, making sand for producing concrete is quite difficult. Often, too many coarse or gritty particles or too many fines exist, and sometimes particle shape is a problem, especially if the fines have been scalped without crushing.

Manufactured sand can be made from surplus 20-mm or 10-mm aggregate by reprocessing it through a vertical shaft impactor. With a high proportion of stone-on-stone crushing in these machines and with high rotor speeds, well-graded sand with good particle shape is achievable. However, as always, this is often dependent on the geology of the feed material. Further fine screening may be needed to fine-tune the particle size grading of the manufactured sand. The Mogensen sizer is a multideck dry screen that is capable of making a cut between different sizes of fines. If manufactured sand is made dry, it should be stored in a building to avoid being windblown, causing an environmental problem.

An air classifier can be used to remove surplus fines from sand made from crushed rock; this operates in a similar way to the cyclones used, except that pressurized air is used instead of water.

SPECIALIST QUARRYING OPERATIONS

Super Quarries

Typically, aggregate quarries are relatively small, especially when compared to mines. This is primarily due to the relative abundance of suitable geology for the raw materials and the need for the operation to be close to market because of high transport costs, as aggregates are heavy and of relatively low value. On average, sand and gravel quarries are smaller than rock quarries, as the deposits are fairly shallow (i.e., between 3 and 10 m below the ground level) and consume huge areas of land.

Most quarries in Europe produce less than 1 Mt/a, and many generate much less than half this amount. However, a number of large super quarries (>3 Mt/a) have significant reserves, large processing plants, automated plant control systems, and fewer but larger items of mobile equipment, and can take advantage of economies of scale when distributing by rail or water. The United Kingdom, for example, has eight super quarries (out of a total of about 1,500). Taking advantage of these economies of scale, they produce approximately 15% of the nation's requirements for primary aggregates.

The typical quarry in North America is usually larger than those found in other parts of the world, because land is relatively cheap, aggregate consumption per head is higher, and fuel costs are lower. A number of super quarries in the United States and Canada have been established for various reasons over the years:

- Remote location with massive reserves
- Special properties for cement or lime production
- Close proximity to the rail network for aggregate distribution to rapidly growing urban areas
- Innovative plant designs using large-scale equipment borrowed from the mining industry
- Large ships (30,000 t or more) that have proved cost-effective for transporting aggregates on the Great Lakes and along the West Coast

Underground Quarries

The overwhelming majority of quarries operate as open-pit mines. However, sometimes a rare combination of circumstances makes it cost-effective for aggregates to be mined from underground. These include

- Favorable geology;
- High-strength rock structure to allow room-and-pillar operation;
- Relatively scarce, high-specification material;
- Extension to an existing quarry so a quarry face to tunnel into and other infrastructure are in place;
- Significant environmental constraints that prevent further surface quarrying; and
- Large depths of overburden.

Marine Aggregates

The European continental shelf is relatively shallow and contains similar sand and gravel deposits to those found on land. The reserves are considerable, but a complex system of permitting is required to ensure that the rules governing a country's territorial waters and international waters are respected. Additionally, possible damage to the marine habitat, which could affect fish stocks, and the potential of an increased risk of coastal erosion are concerns, although no evidence currently exists to substantiate either of these claims.

Marine sand and gravel is dredged from the seabed by large, specialist ships, which operate 24 hours per day, 7 days per week. The dredging areas are clearly defined, and the ship's efficiency and compliance with the permit are monitored closely by sophisticated Global Positioning Systems. Suction pumps attached to steel pipes are lowered from the ship during dredging. Up to 10,000 t of sand and gravel may be extracted in one sailing. Most ships have the facilities to screen either surplus sand or gravel from the as-dredged cargo.

When the ship is full, it returns to port (i.e., Amsterdam, London, or Antwerp), where the cargo is discharged at a marine aggregate wharf, which is just like any other wharf except it has a sand and gravel processing plant, usually with a high capacity, 250 to 500 t/h. There is a ship-to-shore conveyor and a stocking area capable of holding at least two ships' worth of material. The sand and gravel is washed (in fresh water) and screened in much the same way as land-won material. The main differences are as follows:

- The salt content will corrode the plant quickly unless galvanized and well maintained.
- Fresh water must be used in the washing process.
- The aggregate products have to undergo extra testing to ensure no deleterious effect from the extra seashell content or chlorides.
- The risk of dredging unexploded munitions left from World War II exists, and suitable precautions need to be in place.
- Marine-dredged aggregate has a low silt content, which minimizes waste. However, a wharf has no place to put traditional settling lagoons, and therefore silt thickeners and filter presses will be necessary to handle this difficult material.

Dimension Stone

Prior to the invention of concrete, many buildings were made of natural stone quarried for that purpose. Many castles, mosques, stately homes, banks, and government buildings are built with dimension or building stone. Historically, every town needed its local building-stone quarry, which were often small and labor intensive.

Since concrete has become one of the world's most heavily consumed materials, dimension stone production has fallen dramatically and is used now only for the cladding of some prestige buildings. Quarries supplying these materials are few and, as such, support a niche market.

As ever, the local geology is critical. A dimension stone quarry must have massive, unfractured rock strata that are strong enough for a building but not too strong to extract in blocks. Overburden will be stripped by conventional methods, but a variety of different systems will be used to extract the blocks of stone:

- Slow-burn explosives in small-diameter holes
- Diamond-toothed saws
- Ultra-high-pressure water cutters
- Traditional wedge and feather system
- Hydraulic version of wedge and feather
- Large hydraulic backhoe excavator with a ripping tooth and/or bucket
- Wire ropes (impregnated with sand or industrial diamonds), which are looped through drilled holes and cut the rock as they travel at high speed against the rock

When the large blocks have been removed from the quarry face, they are moved to a plant for further processing by various combinations of wheeled loaders, cranes, and haul trucks. Processing of the blocks into paving slabs or walling stone is carried out either by automated diamond-toothed saws or workers with hammers and chisels.

REHABILITATION

In old quarries, all the effort was put into the task of winning material from the ground, converting it to a product, and selling it. No plans were made to rehabilitate the land, either during the quarrying process or at the end when the reserves were exhausted. Nevertheless, nature has done a reasonable job in rehabilitating some of these old quarries over many years, and indeed some are now protected areas for flora and fauna (Smith and Watkins 2007).

Modern quarries in many parts of the world will not receive an operating permit without a rehabilitation plan; this must be included as part of the mine plan. In some circumstances, the rehabilitation result is designed first, and the quarry is planned to achieve it. Many quarry rehabilitation projects have been carried out in the United Kingdom during the last 20 years or so, and it is difficult to identify where the quarry was located because of the continuously improving techniques of the quarry industry, vis-à-vis postextraction planning.

Sand and gravel quarries are usually easier to rehabilitate than rock quarries because they are shallower and often operate quickly over a large area of land, in which case the rehabilitation can be progressive. The overburden stripped from one part of the quarry is directly placed as backfill onto another part of the quarry that has already been worked. This is good rehabilitation practice and economic, because the material is only excavated and transported once.

Rock quarries are much more difficult to rehabilitate progressively, as they are often worked to a greater depth than sand and gravel sites. It is not uncommon for rock quarries to be exhausted before they undergo any rehabilitation process. Also, the number of options is limited in designing rehabilitation schemes for rock quarries, especially if they are deep.

Examples of types of rehabilitation for worked-out quarries are discussed in the following sections.

Lakes

To create a lake, the margin of the worked-out quarry is graded to a shallow slope, trees and aquatic plants are planted, and the water table is allowed to rise. The lake could be used as a nature reserve with shallows for wading birds or reed beds for other species and blinds for bird watchers. There are many spectacular examples of former quarries that have been converted into some excellent wildlife habitats (Goodquarry 2009a).

Lakes can also be used for fishing, sailing, rowing, diving, and other water sports. These activities will usually require a means to enable public access, such as roads and parking lots, as well as buildings for cafes and washrooms or the storing of equipment. The rowing lakes in Sydney (for the 2000 Olympic Games) and in London (for 2012) were both constructed following sand and gravel quarrying.

Agriculture

Many quarries are rehabilitated back to agriculture, which is often what the land was used for prior to quarrying. In some operations it is a simple matter of leveling the quarry's base and replacing the soils at a low level, usually with a pump-operated drainage system (Goodquarry 2009b).

In other circumstances the quarry void is backfilled with construction and demolition waste to the original level and the soils put back on top. Some quarry voids are backfilled with household waste, although this requires the construction of an impermeable seal to prevent the leaking of unwanted pollutants into the water table. This type of quarry landfill usually needs a large dome of material to allow for settlement, but after this process, the land can be converted back to agriculture.

Other Uses

Quarries, either rock or sand and gravel, can be valuable as landfill sites. If the waste is from households, it will emit methane as the waste degrades, which can be harnessed through pumps and pipelines and used to generate electricity. Some quarries are converted back to open land or forestry for wildlife and later the harvesting of timber. Other rehabilitation projects include residential housing, industrial and commercial premises, golf courses, and shooting ranges.

MARKETING

The production of quarried aggregates can be a complex, time-consuming, and expensive business. The sales and marketing of quarry products is a specialist field but essential to ensure a good return for the operator. Unlike metals and coal, no international market exists for such commodities, nor are aggregates (in the normal course of events) worth extracting and storing until they can be sold and used.

A quarry's location is determined primarily by the local geological conditions and the numerous environmental conditions necessary to secure a permit. The distance to the market is important because the cost of transport, whether by road, rail, barge, or ship, ranges from high to very high in

comparison to the value of the product. Most quarries usually serve a market within a radius of approximately 50 km.

Many quarries are operated by vertically integrated companies which sell the aggregates internally for the production of ready-mixed concrete, and asphalt and precast concrete products, such as blocks, roof tiles, curb stones, and paving blocks. For many years, these products were seen as basic with little or nothing to choose between them. However, the 21st century has seen changes in concrete and asphalt technology. The major producers now heavily promote their brand of self-leveling, colored, or crack-resistant concrete. Similarly, asphalt producers focus their marketing on skid resistance, low traffic noise, and high-durability materials.

The most significant user of aggregates and aggregate products is the construction industry. Most of the materials used in civil engineering are seen as commodities. Because price is usually the most important factor, the nearest quarry to the construction project has an advantage in terms of transport costs. Sometimes other factors will give an operator that is farther away from the site the opportunity to supply a project with superior or more consistent quality materials or a better service by delivering the correct quantity of material on time, every time.

Because the market for aggregates can be quite variable, operators have to be able to react to significant changes in sales volumes. In the northern hemisphere, for example, extreme winter weather can reduce the production season to 8 or 9 months a year. If material has to be washed, only a 5-month window of operation may be available. Quarries adapt to these climatic conditions by operating a night shift in the summer and building up stocks for the winter. Concrete production can continue at low temperatures by using hot water and/or special additives, thereby giving the operator a competitive advantage.

Quarries located close to major railway lines, navigable rivers, or the sea enables them to compete in markets a considerable distance from the site by using low-cost bulk transport by rail or water. Local distribution is then carried out by short-haul road deliveries. Quarries with rail access are also in a good position to supply ballast for maintaining the railway infrastructure.

A large construction project in a particular area can also affect the sales and marketing strategy of local quarrying companies. New roads, railways, airport runways, dams, power stations, or ports consume huge quantities of materials, usually over a 2- to 3-year time period. Sometimes luck will give a quarry an advantage simply by being the closest to the project.

The structure of the quarrying industry in a country or a region can also influence how aggregates are sold and distributed to the construction sites. In the United Kingdom, 80% of the quarries are operated by five global companies. Peaks in demand give a national operator the ability to back up material supplies from other sites. In Italy most quarries are operated by different companies competing in different local markets.

As in every other aspect of quarrying, geology plays a part in marketing as well. Some quarries produce aggregates with special qualities simply due to the rock type. This means the product has a higher value, often due to scarcity, and can be distributed effectively over a much larger area. For example,

- The ideal geology for the manufacture of cement is a deposit that is near the market and contains limestone and shale.
- High-purity calcium carbonate is required for flue gas desulfurization in large volumes, and distribution by rail is essential.
- The aggregate for high-friction road surfacing can be quite rare and often located long distances from the market, so the costs will be much higher.

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PART 11

Hydraulic and Pipeline Mining

CHAPTER 11.1

Hydraulic Mining

Mark Richards

The use of water plus energy to mine unconsolidated material has a long history. In recent years, however, the most popular techniques have been based on either water-jet cutting or dredging. Water-jet cutting is used for the mining of unconsolidated materials, alluvial deposits, freshly blasted ores, and for the recovery (or remining) of dewatered tailings dams. Dredging is used for the recovery of unconsolidated materials in rivers, lakes, in other extremely wet environments, and for the recovery (remining) of fluid tailings dams, see Chapter 11.2.

HISTORY

There are many claims for the invention of the hydraulic method. With the basic principles so obvious, however, it is thought that there was concurrent development of similar mining operations around the world in the 18th and 19th centuries. Certainly its use was heavily championed by the gold miners in North America; by gold, diamond, and possibly rutile miners in South Africa; and by china clay miners in Cornwall, United Kingdom. (Numerous sources regarding the history of hydraulic mining are available through the Internet.)

In North America, the heavy industrialization and scaling up of this mining method led to millions of tons of waste materials flowing down major rivers. The uncontrolled release of this debris led to one of the western world's first environmental action laws as the authorities forced the miners to capture the waste on the mine site and reuse the water.

Uncontrolled hydraulic mining and the use of rivers for high-volume waste removal continued in many countries well into the early 20th century and indeed still occurs today in some South American and Far Eastern countries. Now, in a more environmentally conscious age, this method is largely restricted to operations that can successfully contain the waste and reuse the water at an acceptable cost, such as the current china clay industry (where rheological parameters apply) and some tailings dam recovery operations.

The initial development of the technology concentrated on *monitors*—high-volume and relatively low-pressure output water jets. These were very effective but quite inefficient in energy terms. The monitors of the time had nozzles that ranged up to 200 mm (8 in.) in diameter, with feed pipe work being much bigger. This led to a fairly static operation, which was fine for the denudation of hillsides and thick alluvial operations. The technical considerations of balancing the resultant thrust vectors and steering the monitor were solved by "brute-force" applications using big counterweights, thrust vanes, and so forth.

At the end of the 19th century, the "easy pickings" were beginning to disappear in both the gold and china clay operations. The need to wash blasted material and less easily sorted material resulted in the need for a higher-pressure, more robust, and more transportable monitor. The technology for this was freely available in the parallel development of fire-suppression systems. The early 20th century saw the introduction of higher-pressure and lower-volume systems, loosely based on fire-industry designs. From that moment on, design diverged to match the mining method.

HYDRAULIC MINING TODAY

Today, after the factual features (i.e., grade, size, structure, depth, and location) of a deposit are taken into account, the energy costs associated with hydraulic mining are probably the most important factor influencing the viability of this mining method. To minimize the water pumping requirement, the liberation of ore from country rock using water must be carried out at the highest fluid density possible.

Low fluid density liberation means that a great deal of energy is required for

- Pumping the water to the production face,
- Pumping the resulting ore streams away from the production face,
- Recycling the water and separating the ore, and
- Setting aside a great deal of space for large settling ponds or investing in high-cost dewatering systems.

Mark Richards, Mines Manager, Imerys Minerals Limited, Cornwall, UK



Figure 11.1-1 Hydraulic monitor

Water-Jet Cutting

Water-jet cutting is normally achieved by using a water canon or hydraulic monitor (Figure 11.1-1).

Water-jet cutting technology has been developed regionally to match the specific mining requirements of the ore bodies. Alluvial gold mining, for instance, and china clay mining use quite different designs. The basic principles remain consistent, however, and there is starting to be a convergence of design based on significant step changes introduced by the fire-suppression industry.

Design Parameters

Conditions that will maximize the efficiency of hydraulic mining include the following:

- A reliable source of water, high above the washing faces, that minimizes pumping energy
- Pumps that are energy efficient and in the correct location
- A ready supply of large-diameter pipe to feed water to the monitors
- Monitors that do not need to be moved very often to maintain their optimum position at the mine face
- A mine face drifting into a hill with a gravity runoff that will direct the ore stream toward the processing plant
- No need to recycle the water

In today's operations, however, the mine designer may have to cope with many suboptimal parameters, probably the most crucial of which will be the requirement to recycle the water. This reflects on the energy use most of all and has been the reason that many hydraulic mining efforts have been replaced with mechanical make-down systems, such as washer barrels and water-sprayed vibrating screens.

Whatever these parameters may be, the task is exactly the same: to maximize the efficient use of the energy available.

SYSTEM DESIGN

Water-jet mining projects require a pumped water supply. It is essential that the pump and pipe-work system be designed to make the most efficient use of the electricity supply. More energy efficiency gains are possible from pump and pipe-work optimization than from monitor optimization.

Pumps

In an ideal project, a single pump would be preferred; however, most pressure pumps have a fairly narrow efficiency curve. If there is a requirement to move monitors between benches (thus varying the head requirement of the pump) or if there is a need to run varying numbers of monitors, then a multiple-pump setup should be considered. A single pump can be used for varying loads if the pump has a very flat efficiency curve and if it can be driven by an inverter-controlled motor. Large inverters (above 200kW) are also very expensive, and in certain circumstances they can cause unwanted harmonic effects back to the grid. For this reason, some electricity utilities place limits on their use. A complete cost analysis should be carried out, therefore, to determine which system is more appropriate.

The ideal pump arrangement is based on one pump per monitor, where the head and volume requirements are matched. If the water is free of suspended solids and has a neutral pH, consideration should be given to hydrophobic pump coatings which will definitely improve pumping efficiency, sometimes by up to 5%. If suspended solids are present, they should be minimized by the use of a large settling pond, which can also be used as a water storage lagoon. The presence of even small amounts of suspended solids or adverse pH conditions will significantly alter the choices regarding pump construction materials. In these cases, it is recommended that duplex stainless steel be used.

Modern pressure pumps need a net positive suction head in order to run efficiently, and therefore the pumps should be gravity fed with a minimum of least 1 m (3.2 ft) of head. Consideration needs to be given to stone traps, isolation valves, servicing room, crane access, and most importantly pipe friction; it is always good practice to oversize the pump pipe work on the suction side.

On start-up, a lot of damage to the pumps can occur through cavitation, especially if they are required to pump into an empty and unpressurized pipe-work system. It is therefore advisable that the pressure-side isolation valve be designed so that it can be partially closed during the start-up sequence, allowing the pump to be "throttled" during the start-up phase. Many installations have a timer sequence built into this valve for automatic throttling during start-up and shutdown.

It is advisable to have some facility for pump health checks that does not necessitate pump dismantling. Many modern diagnostic systems need pressure tappings before and after the pump, as well as on the pump casing, and also a facility to measure the bearing temperatures.

On high-power installations, it is always advisable, even with flexible couplings, to use laser alignment techniques between the motor and pump. Misalignment will mean heat buildup and bearing damage in the long term.

Finally, pressure pumps can have very high starting currents. It is therefore prudent to consider this when designing the power supply and also when designing the control panels. It is wise to have these control panels linked in some way to prevent operators from turning pumps on simultaneously.

Pipelines

Multiple-pump systems should be fed via a well-designed manifold for efficient distribution to the pipe-work system. If the manifold has not been well designed and properly aligned, a great deal of energy can be needlessly wasted at this stage. Ideally there should be a swept bend off each pump into the manifold, and the manifold itself should be at least large enough and long enough to reduce the turbulence of the water before it is fed into the pipe-work system. Manifolds without



Figure 11.1-2 A typical multipump arrangement in the Cornish clay industry

swept bends can be used if they are considerably oversized. The manifold will need drain and inspection points.

Figure 11.1-2 shows a four-pump (duplex stainless steel) arrangement with 355-kW motors. Of particular note are the oversized suction and delivery manifolds, the suction isolation valves, and the motorized delivery valves for throttling the start-up. The stone traps are out of view to the right.

It is all too common to build a system based on a "Christmas tree" approach to pipeline design, where sections are added over time with little thought for either isolation or friction losses. Where possible, the entire pipeline system should be designed to have a pipe friction of less than 1%. Many free on-line calculators for pipe friction and other pumping parameters are available on the Internet.

Because collisions between mobile plant and the pipeline network are inevitable, having the ability to isolate a section without losing production is a necessity. The use of *ring mains* with multiple pathways and isolation points will allow the friction losses to be kept to a minimum. Multiple pathways enable smaller-diameter pipes to be used, thus saving money and making pipe handling, maintenance, and movement considerably easier. The location and inclusion of both drain points and air bleed-off points should be considered during the design process.

If there are a variety of pipe sizes in the system, it is obviously wise to put the biggest diameters where the highest flows occur. Any large pipe, however, wherever it is located in the main feed system (before a branch occurs), will always reduce the friction losses.

It is normal to use steel pipes with rubber-lined steel wraps for pipe jointing; this allows a slight give for ease of fitting, movement under varying pressure, and some flexibility in alignment. Some industries have experimented successfully with high-density polyethylene pipes, which can be constructed in convenient lengths for dragging around the mine (typically 100 m [328 ft]). These longer lengths of pipe suffer proportionately less from the friction associated with the pipe joints and short bends.

MONITOR DESIGN

Historically, the major centers of hydraulic mining have tended to develop their own style of monitors with little concern about what was happening in other industries or areas. This has resulted in specialized designs that have given rise to an "evolutionary fit" for the hydraulic mining industry.

The basic principles are universal, and the overriding principle of jet cutting is to maximize the energy per unit area exerted by the water jet impacting the face. Therefore, the water jet should hold its shape and coherence for as long as possible, which means that the main efficiency driver for a monitor is the nozzle. To perform efficiently, the nozzle must be provided with nonturbulent water.

To steer the water jet in two dimensions, it is necessary to have two axes of maneuver and therefore at least two sets of bearings in the monitor. To balance the forces involved, a loop arrangement is needed where the bearings cross over. Figure 11.1-3 shows a monitor in operation in the Cornish clay industry. The thrust is in excess of half a metric ton (5,000 N), and therefore the forces must be balanced. Other approaches to this balanced-forces issue are shown in Figure 11.1-4.

A monitor is built of the following components:

- A base to counteract any overturning moment
- A stone trap
- A loop for balancing the forces as much as possible
- A multiple bearing assembly for two-dimensional movement
- Hydraulic rams, hydraulic ring motors, or electrical actuators for steering the jet
- A barrel that contains a streamformer, a stilling zone, and a nozzle

The nozzle needs to have a quick-change facility (normally using steel pipe wraps) for varying the volume and pressure in order to keep the pressure pump(s) on the peak efficiency point.

All of these elements must work together to minimize any turbulence before the water hits the nozzle. To facilitate this,

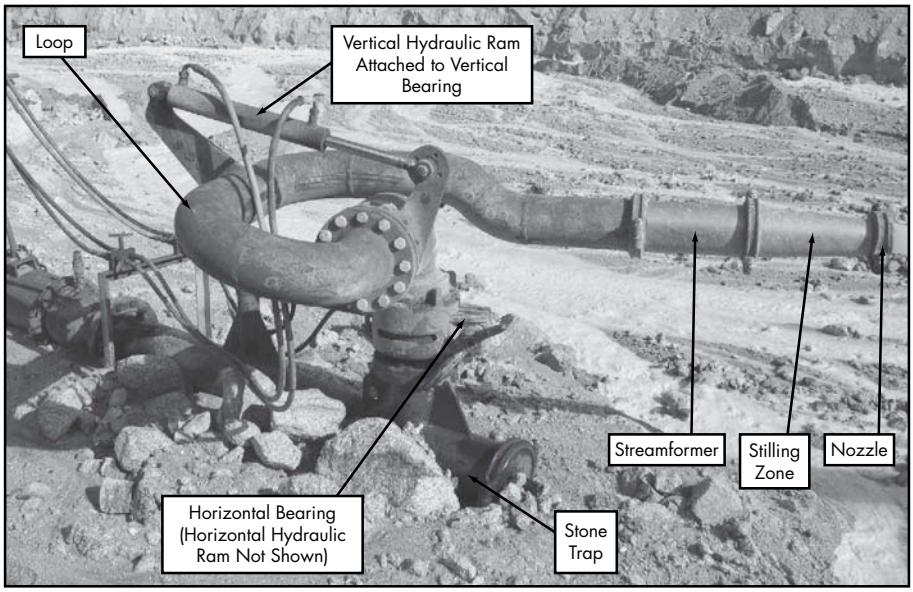
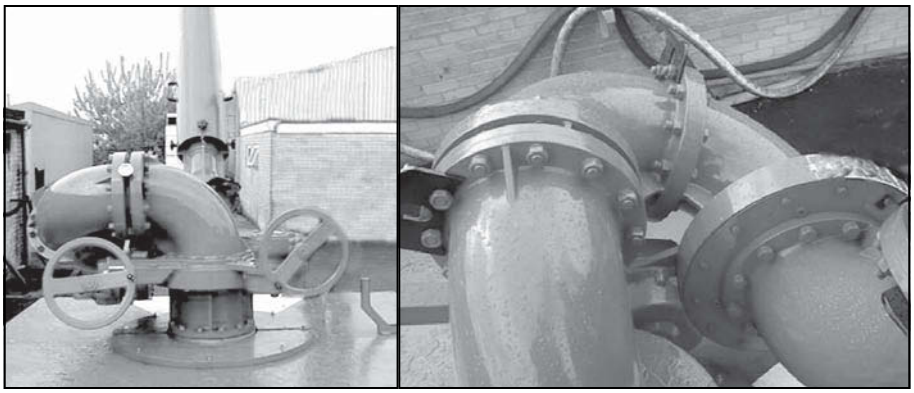


Figure 11.1-3 Close-up, showing bearing arrangement and hydraulics



Note: The long barrel is for foam generation, which would not be used for hydraulic mining.

Figure 11.1-4 Latest designs from the fire-suppression industry

there must be minimum friction all the way through the unit, which calls for large-diameter pipes, swept bends, constant diameter, and a low-friction streamformer. There is a trade-off among all these elements as, for instance, large-diameter pipes will require much tighter bends, while an efficient streamformer will cause friction. Current design cues coming from the fire-suppression industry, however, indicate that large diameters are more important than tight bends. As shown in Figure 11.1-4, the latest thinking suggests that the bends be approximately two pipe diameters with the pipe indexed downward off the top flange to keep the turning moment as low as possible.

The streamformer is to some extent dependent on the operating conditions. Since even the most efficient streamformers can be blocked or damaged by foreign bodies in the water, a degree of pragmatism has to be used in this case. The Cornish china clay industry, for instance, uses a streamformer that has no center at all (Figure 11.1-5); the fins are mounted on the circumference of the barrel and point inward. The fire industry uses a much more delicate honeycomb design because their water tends to be much cleaner.

The nozzle construction material is dependent on the water quality and ranges from aluminium-backed polyurethane to brass and gunmetal (Figure 11.1-6). The industry is converging, however, on a fire-hose-type design of a simple cone with some variations in angle. Many other designs have been tried, but ultimately the original fire-hose type has tended to be the most efficient.

Of major importance is the surface roughness of the nozzle. It is essential that the nozzle be free of nicks and gouges. As a general rule, no roughness should be felt when a fingernail is passed over the surface. The water emerging from a good nozzle setup should appear as a straight bar for at least 50 nozzle orifice diameters and should degrade slowly but steadily on its path to the mining face. Any sign of instability, expansion, or pulsing in this critical area should be seen as a sign of excessive turbulence within the monitor.

Actuation

In most monitor designs, the actuation, or steering, is accomplished by the use of hydraulic rams, although some designs use hydraulic ring motors or electric actuators. In smaller

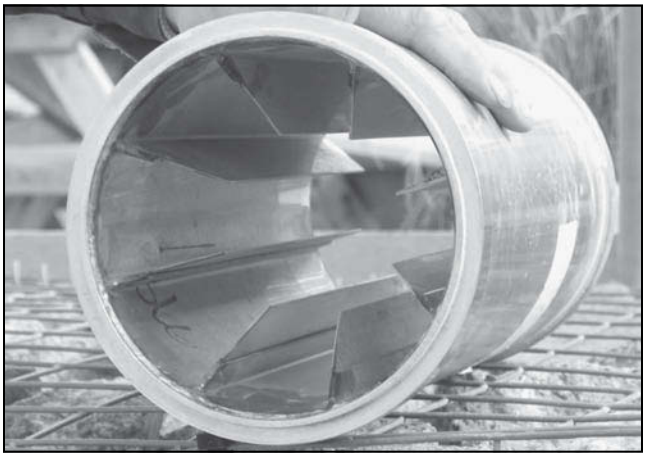


Figure 11.1-5 Self-cleaning streamformer



Figure 11.1-6 Polyurethane nozzle with aluminum backer

operations, or where the risk level is low, monitor steering may be a manual task by means of a yoke on the back of the monitor. In cases like this, it is normal to have a small tray on each axis where weights can be added to fine-tune the balance.

Respectively, motive power has to be supplied in the form of a hydraulic power pack or a generator. In all of these cases, it is common for a Pelton wheel to be used to supply the raw energy. The Pelton wheel is essentially a high-pressure water wheel that takes a small amount of water from the pressure line prior to the monitor to provide the motive force (Figure 11.1-7). Operation of the actuators is normally via pilot hydraulics from levers operated from the control hut (Figure 11.1-8).

In some instances, a secondary Pelton wheel has been used to supply lighting, although this is an inefficient use of pumping energy. In most cases, diesel-powered mobile lighting units or fixed lighting are preferred.

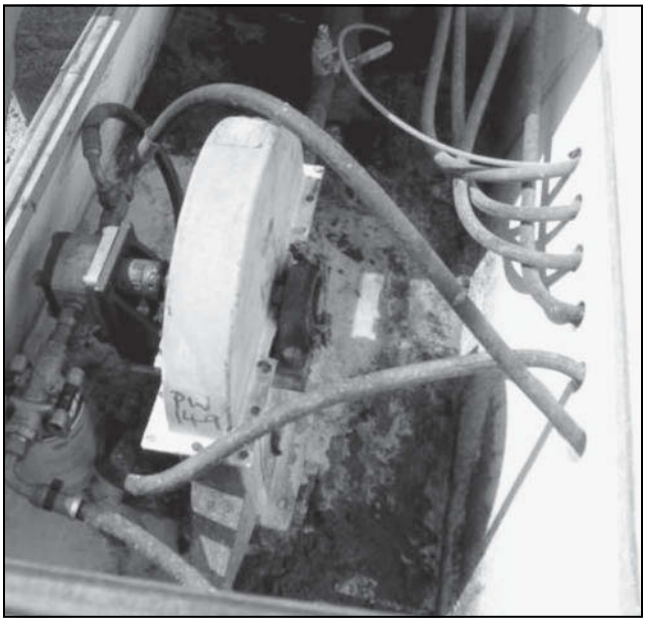


Figure 11.1-7 Pelton wheel

Control Hut Design

A demonstrable and significant increase in productivity, and therefore efficiency, can be achieved if the worker is able to maintain concentration on the production task, with eyes focused on the work face. Holding this attention span is difficult, however, if the worker is uncomfortable or having to work in a badly designed environment. Ergonomics, therefore, is the main driver for control hut design.

Obviously the prevailing climatic conditions will have a bearing on the control hut design; however the main design elements can be split into two categories: operator comfort and safety.

Operator Comfort

It is essential to treat the operator's seating position as if it were a workstation (similar to that for someone who uses a computer routinely) and to allow for the posture associated with operation of the levers. In other words, the controls should be designed to prevent repetitive strain injuries, and their positioning should prevent musculoskeletal stress. Other important considerations for operator comfort include the following:

- Soundproofing
- A draft-free operating environment
- A door design that can be kept open, shut, or anywhere in between, without being caught by the wind



Figure 11.1-8 Monitor control hut

- An ability to control glare if working at night or when the sun is low on the horizon
- A means of preventing windscreen dirt or, failing that, an efficient windscreen cleaning system
- The need for the mobile plant drivers that move the huts (normally small wheeled loaders) to be able to see through the hut while carrying it

Safety Considerations

Stones can be flung a long way by a working monitor, and it is therefore necessary to have a basic understanding of this process:

- Stones get ejected when the monitor jet hits a reflector, such as a flat stone or a hole, where the majority of the jet is reflected back toward the monitor.
- The vertical angle of the monitor jet is crucial; when pointing down, it is much more likely to reflect.
- Stones falling through the reflected jet pick up acceleration only while they fall through the jet.
- The accelerated stones are self-limiting on the distance they can be thrown. The small ones are accelerated quite rapidly but slow down rapidly because of air resistance; the larger ones are difficult to accelerate and thus do not travel as far. Overall, the small stones travel the farthest but do the least damage.

It is wise, therefore, to have a parking rule and a distance rule for avoiding ejecta where possible and to have regard for the safety of others where it is not possible. Recommended precautions include the following:

- The monitor should be "parked" when the wheeled loader is operating in front of the monitor position.
- A minimum distance should be established (the Cornish clay industry recommends a minimum distance of 80 m [262 ft]) requiring that
  - People not involved in the act of jet washing be kept out;
  - All individuals use a safety helmet and eye protection within the zone;
  - Vehicles without safety guard glass (i.e., service vehicles) be advised to park outside the zone until they have the attention of the monitor operator, who will then carefully park the monitor to allow safe entry.
- Because it is obviously difficult to physically put a cordon around the monitor in an ever-changing mining environment, education and warning signs are vital.

If the host rock consists largely of stones, then some armoring of the glass in the hut may well be necessary to prevent breakage. This can be achieved through metal mesh guards or with a ballistic film (a polymer sheet coating similar to laminated glass). In the case of a hand-operated monitor, the energies involved are so low that ejected stones should not be a problem.

OPERATION

The following series of steps provide an overview of the water-jet mining process:

1. The monitor jet washes the material to separate the ore and fine contaminants from the host rock.
2. The resulting ore stream should be tightly channelled to ensure that there is no dropout en route to the processing area or to the gravel pumps which then transport the stream to the processing area.
3. A wheeled loader is normally employed to remove the washed host rock from the mining area.
4. Subsequent removal of the waste host rock can be by haul trucks being loading directly, or the waste can be stockpiled for later removal. Economies of scale may favor stockpiling.

Monitor and Control Hut Placement

The optimum distance from the monitor to the washing face is commonly considered to be 100 nozzle orifice diameters. This will maximize the energy per unit area of the water jet impacting the face. However, from an operational (health and safety) point of view, this is very close and, depending on the scale of the operation, may well compromise the reach in terms of the overall washing area. What is not in doubt is that the monitor should be as close as is safe and practical to the face, and that the control hut should be situated out of harm's way. This distance obviously has to be tempered by the fact that the operator needs a clear view of the jetting and washing process.

It is very easy to bury a monitor with debris runoff from the washing face. Although the operator should be able to prevent this by periodically cleaning the area in front with the monitor jet, it is sensible to place both the monitor and the control hut on small platforms made from local debris. This removes the equipment from the ore stream and makes it much more visible to any mobile plant working in the area.

Rewashing of consolidated tailings and other materials may result in a collapse of the face when liquefied. The operation must be risk assessed according to local conditions. Nevertheless, certain points are common to these operations:

- The control position must be substantially distant from the monitor.
- The control position should be placed on a high bank or mound of material that is not likely to be washed away.
- Washing face heights should be much lower than normal.
- Frequent checks of the washing face should be carried out by a suitably qualified person.

Monitor Operation

With due regard for safety, the operator should use the water jet in a horizontal action at the base of the ore pile in order to undercut the pile. Gravity will then bring unwashed material into the washing area. There is little need to raise the water jet into the higher reaches of the pile unless there is a need to dislodge a hanging rock. Washing from the higher reaches will mean that washed debris comes down to cover unwashed ore near the bottom, making the process inefficient. In addition, this practice tends to reduce the overall slope angle of the ore pile, which means that the optimum monitor position is quickly compromised as the ore pile is washed away.

To dislodge a hanging rock, a technique known as *doubling the stream* can be employed. The operator will direct the water jet to the top of the face above the rock for long enough that the face is covered with a falling cascade of water. The operator will then lower the water jet to the bottom of the hanging rock and take advantage of the weight of the cascade.

With the monitor centrally placed in front of the ore pile, the operator can wash one side until the density begins to drop off and there is a buildup of cleaned host rock or debris. At this point, the operator can move to the other side of the target

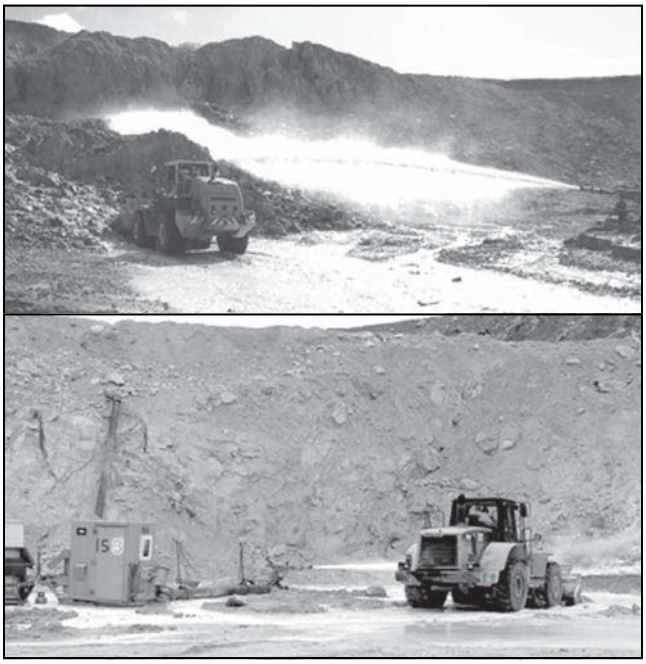


Figure 11.1-9 Wheeled loader removing washed host rock

area, thus allowing the wheeled loader access to load out the waste (Figure 11.1-9). When the wheel loader is forward of the monitor position, the jet should be parked so that flying stones are minimized.

It is at this point that good communication between the monitor operator and the wheeled loader driver is essential. At the beginning of their period together, they should agree on a set of signals or actions so that each can clearly understand the other's actions and intentions. It is normal, therefore, for each side of the ore pile to be worked in turn, allowing the wheel loader to be sufficiently available to remove the debris on demand and thereby keeping the density of the ore stream as high as possible.

Mobile Plant Operation

In most operations, the wheeled loader is sized to allow one machine to be fully occupied by one monitor. Additionally it is normal practice to stockpile the debris so that a fleet of trucks dedicated to waste removal can take advantage of the economies of scale.

The efficient interaction of the wheeled loader driver and the monitor operator is paramount to maximizing productivity and minimizing accidents. That the two must work as a team at all times cannot be stressed enough.

Ore Stream Pumping Versus Trucking

The method of mining used depends to a great extent on the debris ratio and the method of its removal. Across all industries the debris percentage can range from 90% down to 20%. Thus, it is wise to consider the cost of waste removal and the cost of ore stream removal as a whole.

If the debris ratio is very high, it may be appropriate to dispense with ore stream removal costs and carry the raw ore to a central washing point with a gravity run into the processing system. Conversely, if the debris ratio is very low, then electrically pumped ore stream removal is the most efficient. In addition, some ores degrade quite rapidly with too much energy input or shear. Thus, a high degree of pumping may do damage, and trucking may again be more appropriate.

In recent years, there has been a trend toward centralized hydraulic mining, and in some cases its abandonment, in favor of wet mechanical separation. In every new case, however, a comprehensive total costing of the ore stream *and* the waste stream will indicate the preferred process.

ACKNOWLEDGMENTS

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CHAPTER 11.2

Placer Mining and Dredging

Richard L. Bullock, Michael J. Cruickshank, and Ian Sherrell

PLACER MINING METHODS OTHER THAN BY DREDGING

The Spanish word *placer* means sand or gravel bank or shoal where gold can be obtained by washing. As it is defined for this chapter, a placer is any area of unconsolidated rock fragments or particles that may also contain minerals potentially of economic value. Placer mining involves the extraction of these unconsolidated materials without the use of explosives or any other significant means of rock-breaking force. This chapter contains a classification of placer deposits and describes the mining and processing of these deposits, which requires the application of mining and metallurgical engineering.

Many placers of economic value can be and have been mined by hand; indeed, the famous gold rushes of the American West and Klondike were in this category. Although it is fascinating history, such mining does not require the engineering methods described in this handbook. For individual prospecting and hand mining, as well as processing, see Wells (1973) and Peele (1945).

Classification of Placers

The many types of placer deposits described in this chapter all have four things in common:

1. A source of the original mineral existed.
2. Some mechanism released the mineral from its original location.
3. The valuable mineral must be resilient enough to survive transport and remain commercially recoverable.
4. A reason existed for the placer being located and concentrated where it was found.

Breeding (1985) describes the steps involved in the formation of a commercial placer as follows:

> The formation of a commercial placer requires a source of valuable minerals. Above primary deposits, eluvial deposits may exist that were formed by the erosion of gangue minerals and the concentration "in-situ" of valuable minerals. Down slope from these deposits are the hillside or colluvial deposits, and below them are the alluvial deposits of redeposited material. Most of the great placer fields of the world are the result of several generations of erosions and deposition. Well-known examples are in California and Colombia.
>
> Gold is a very resistant and malleable matter, and gold placers may extend for 40 to 50 mi (64 to 80 km) along a river system. Platinum is less malleable, but is very resistant to disintegration. Diamonds are extremely hard, and (especially gem diamonds) may be found over great lengths of a river system. Cassiterite (i.e., tin crystals) are less resistant to disintegration, and such placers seldom extend more than 2 mi (3.3 km) without resupply from an additional source or sources of mineralization. Tungsten minerals are generally more friable, and within a few hundred yards (meters) of the source disintegrate to the point that they are uneconomical to recover. Rutile, ilmenite and zircon placers generally result from the weathering of massive deposits and may be encountered over extensive areas; most are fine grained and durable.

Source of Mineral

Sometimes the source material is a mineralized vein (lode) containing high-grade mineralization that is attacked by the release mechanism. Sometimes the source material is a more massive mineralized rock containing disseminated bits of low-grade mineralization. In both cases, the matrix rock and the mineral are broken down into smaller particles that may or may not then be transported by the release mechanism. Portions of the original vein or host rock may remain, or it may all have been removed by the release mechanism.

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Release Mechanism

Physical release. The more common types of placers are those whose release mechanism is by flowing water. It may be normal stream movement, very rapid seasonal stream movement, flooding rivers or streams, or intermittent streams. When the water is moving fast enough to pick up the small particles, it will carry them to the point where a loss in velocity causes the stream to drop its load. Another release mechanism is the movement of ice as a glacier. This can be followed by streams flowing from the glacier, which then transport the material to a point where it will drop its load. Finally, wind erosion can also wear the rock particles down and separate them from the mineral particles.

Hydrothermal alteration and weathering. Both vein and massive types of mineral deposits can be broken down into smaller rock particles by hydrothermal alteration or by weathering. In these cases, the mineral usually stays in place and may have little chance of physical displacement other than by rolling down an adjacent slope, so there may be little chance for concentrating the mineral.

Deposit Type Formation

Several placer-building mechanisms can take place over long periods of geologic time and form the various types of deposits discussed in this chapter. The mechanisms are

- In-situ residual building,
- Eluvial and gravitational creep movement,
- Stream transport, and
- Steam transport and the wave action of a sea or ocean.

Residual Placers

Residual placers form when the remaining eluvials, consisting of mineralized rock, decompose in situ principally by chemical weathering or in some cases by hydrothermal alteration, often with little or no change in volume (Daily 1973).

Eluvial Placers

Eluvial placers, also termed *colluvial* or *hillside placers*, are transitional between disintegrated rock and stream gravel. They are formed on slopes where smaller particles are removed by rilling, wind, and dissolution (Daily 1973).

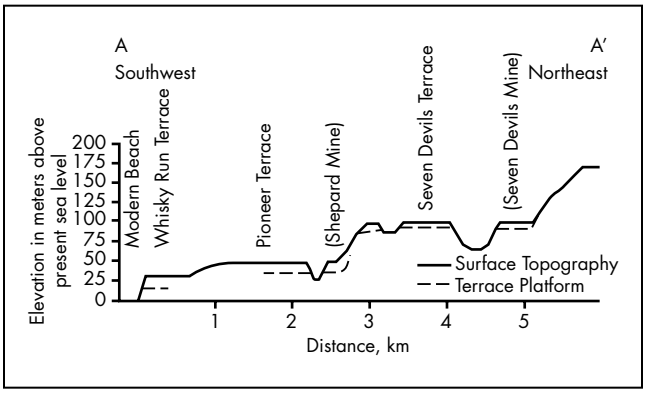
Various Stream Placers

Stream transport is a mechanism for forming placers.

Gulch placer. Creek placers, river placers, and gravel plain deposits (Wells 1973) are forms of the gulch placer. The differences are the volume, water force, and areas of deposit attributable to the water's action. McCulloch et al. (2003) classifies these deposits as colluvial, debris flow, pulse placer, and alluvial fans.

Debris flow, pulse placers. These deposits form when the original lode deposit has been sufficiently weathered so as to be weakened. Then, because of above-normal precipitation, the strong rush or pulse of water carries the debris from its original location down and across the valley. Coarse and fine material will flow with the pulse; fine material could be transported farther by normal precipitation.

Alluvial fan placers. When the normal stream flow carries its load down the narrow mountain valley to meet a wider valley or plain, the water fans out, whereupon it drops its load. It fills the current channel to the point that the stream is forced to find a new channel. In this manner a fan deposit is gradually created.



Source: Drew and Lessard 2009.

Figure 11.2-1 Schematic section depicting the relationship between terraces

Eolian placers. Eolian placers are concentrations resulting from the removal of lighter material by wind action (Daily 1973).

Bench Placers

When streams cut downward through a previously deposited alluvial deposit, a bench placer is developed. The economic bench placers are characteristically larger in volume than stream placers. These types of placers were typically mined in 1850 in California (United States) by hydraulicking.

Beach Placers

Where mineral-bearing streams reach the sea or ocean before they drop the transported mineral load, the wave action of the sea tends to concentrate the heavier minerals and forms bands or layers of the concentrated mineral or gem. These types of deposits are extremely important around the world for concentrating heavy minerals such as ilmenite, chromite, rutile, zircon, garnet, diamonds, and so on. It is important to understand that frequently such "beaches" are no longer by the seaside and are at elevations much higher than sea level. They may now be miles from the sea—for example, the Oregon chromite sands (Figure 11.2-1) or the deeply buried deposits of ilmenite/rutile sand east of Nashville, Tennessee (United States).

Desert Placers

Desert placers are found in arid regions where erosions and transportation of debris depend largely on fast-rising streams that rush down gullies and dry washes following summer cloudbursts that fall at higher elevations. These are typical in the southwestern United States and in other desert regions of the world. In these areas, the deposits can be quite large in extent, but erratic in location and mineral grade.

Glacial Placers

Glacial placers are mineral deposits that have been associated with a moving glacier that scraped off loose rock debris, soil, and sometimes minerals. The mineral is later concentrated either in glacial moraines or the streams forming from these glaciers. The gold deposits that have been worked along the foothills of the Sierra Nevada (United States) and the placer

deposits of Fairplay and Breckenridge, Colorado (United States) are examples of glacier-related placer deposits (Wells 1973).

Marine Placers

Any of the placers just described may be found submerged offshore on the continental shelf, which extends at its outer edge to a water depth of about 150 m (500 ft) and as much as several hundred kilometers (nautical miles) from the current shoreline. All coastlines have been subjected to frequent changes in sea level over geological time and during periods when existing placers were subjected to high-energy surf or ocean currents. As the shoreline advanced or retreated, they may have been dispersed, reconcentrated, or buried. Occurrences have been reported of placer gold in Alaska (United States) carried off the shelf by turbidity currents in submarine canyons to depths of approximately 4,000 m (13,000 ft).

Placer Exploration and Sampling Techniques

Before an exploration program is undertaken, it is important to understand what type of placer deposit is being prospected for. The type of deposit will have to be addressed to judge how the mineral concentration took place, how far the mineral may have been transported, where the larger particles of the mineral are likely to be, and where the finer particles will probably be. The normal range of thickness of the deposit will also have to be judged.

In performing exploration and sampling of a potential placer deposit, it is necessary to be aware of the mining and recovery method likely to be used to extract and recover the valuable mineral. To perform exploration and sampling effectively, it is necessary to understand the limits of the mining and recovery methods. This same philosophy also applies to assaying: If the mineral cannot be recovered by the concentrating methods being applied in the mining and processing, then fire assaying of the total mineral content of the material sampled does not make much sense, since recovery may not be possible.

As to the sampling technique, sampling tools that are normally used for hard-rock minerals or coal are simply not adaptable to most placer deposits. What is needed is a sampling tool that extracts a sample large enough to be processed to determine what is recoverable and to assay that which is recoverable.

The many methods of extracting samples from a placer deposit vary from being highly prone to error to fairly accurate. They are

- Hand grab samples gathered from existing terrain,
- Hand-dug pits or trenches,
- Backhoe trenches,
- Bulldozer trenches,
- Churn drill holes,
- Auger drill holes,
- Reverse-circulation drilling,
- Vibratory drilling,
- Other types of drills that case and capture the entire sample within the hole, and
- Large-area bulk sampling.

One method of capturing a large sample for bulk sampling and still cover a wide area of a deposit is with a drill. Between 1979 and 1984, a subsidiary of St. Joe Minerals Corporation (Missouri, United States) combined a measuring hopper, a portable jig plant, and the drilling of large bulk samples for sample testing. The jig concentrate was panned and amalgamated to recover the gold. This system was used to prospect 28 holes in the tertiary gravels of the San Juan gold property of the North Columbia area of California. The drill first sank a 915-mm (36-in.) casing and used a cactus-grab bucket to muck out the sample. Where the gravels were cemented, they had to be broken with a chopping bit. The advantage of the total system was that it minimized the possibility of unintentional salting, and it only sampled and assayed that which could be recovered by the pilot processing.

If the gravels are re-cemented, it is also possible that a diamond drill can be used. One such case was reported (Stone et al. 1988) in which a 100 $\times$ 140 mm (4 $\times$ 5.5 in.) diamond drill bit was used very successfully on the cemented gravels described previously.

The problem with the majority of the drilling methods is that a typical alluvial placer deposit contains boulders that are too large to be broken down on the surface by a drill bit or auger flights. This is why for most placer deposits, a bulk sample extracted with mining tools that can handle the boulders is more accurate. However, for a desert placer deposit that is likely to have only a few boulders, an auger drill is the best option. Another problem with drilling holes for the sample is that unless the hole is cased, or unless the deposit is a cemented conglomerate, caving of the hole will occur. If the sampling horizon passed a high-grade lens, the samples below this lens are likely to be unintentionally salted. The resulting sample values will be elevated and thus inaccurate.

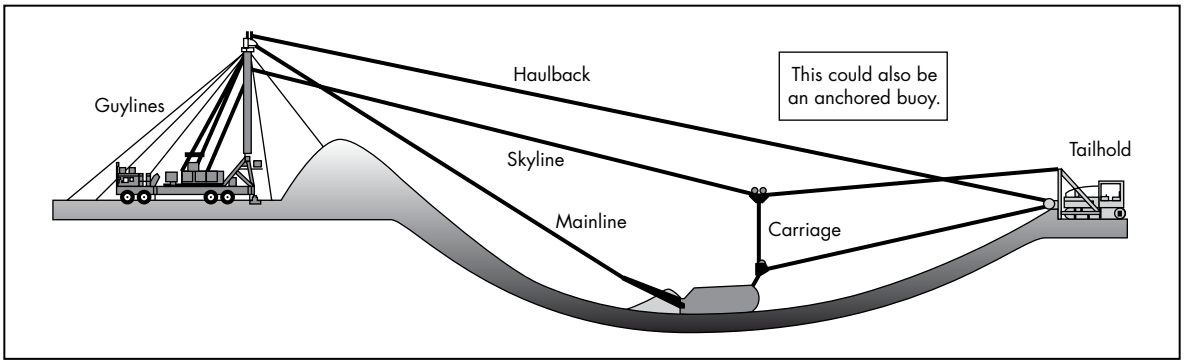
Probably the biggest problem is trying to sample the highly erratic nature of the precious-metal placer deposits. It will take many holes over a large area to statistically even out the nugget effect, which often occurs. In fact, with enough drilling and sampling, the outliers should be discarded from the sample collection.

Bulk Sampling Advantages and Disadvantages

The advantages of bulk sampling include a good view of the gravel in place and knowledge of the amount of force required to excavate it. Some of the same digging machines used in bulk sampling are used in production. The large samples obtained minimize the nugget effect and errors. On the other hand, a serious disadvantage is the fact that pits or trenches may not reach bedrock in all cases. Other disadvantages include the following. Curves often make it difficult to accurately measure a pit if the excavation is by a backhoe, because of sloughing from the sides. If groundwater is encountered, sample integrity is lost. Larger samples require larger transport and processing equipment, and the situation becomes more complicated. As unit costs of each sample are high, the exploration cost to delineate reserves is very expensive (Cope and Rice 1992).

Drill Sampling Advantages and Disadvantages

Historically, drilling has been the sample method of choice and has been proven by operation in most cases. Drilling gives greater coverage than bulk sampling in developing gravel grade and reserves at less unit cost, the profile of bedrock, and indications of enriched and barren areas. A disadvantage of drilling is that it gives a small sample, which aggravates the nugget effect. Other disadvantages are that at times boulders



Courtesy of Ramsey Company, Inc.

Figure 11.2-2 Sand or gravel being mined by Ramsey scraper bucket

will necessitate abandonment of uncompleted drill holes, vibration from driving casing or from a reverse-circulation hammer drill may cause gold to migrate downward into the gravel, and drill rigs require at least minimal access roads and drill pads (Cope and Rice 1992).

Other Mining Methods

Although the most common method of large-scale mining of placers is by dredging, several other methods of placer mining have been used over the years, and they are discussed in this section.

Drift Mining

Frequently, the precious metals of a placer deposit are located in relatively thin lenses on or very close to bedrock or some other impervious material, and they may be covered by many feet of worthless gravels and debris. The economic value does not warrant the removal of all of the overburden; in such cases, if the mineral is above the valley floor, adits may be driven into the hillside and the high-grade placer mineral extracted by drifting wherever the deposit exists. Usually, these adits or drifts must be timbered to maintain stability. Occasionally, the rock has re-cemented itself into a conglomerate and drifting could take place without timbering. Typical deposits that have been mined extensively for both gold and platinum are remnants of ancient channels now above groundwater in California, British Columbia (Canada), and Alaska; frozen stream-valley and bench-type placers in the Yukon (Canada) and Alaska; and deep lead deposits in gutters in the bedrock below the groundwater table in southwestern Australia and British Columbia (Daily 1973).

Conventional Equipment for Dry Unconsolidated Placers

The term *conventional equipment* refers to the same type of equipment that is used in various types of standard earth moving in surface or open-pit mines. It includes

- Front-end loaders and trucks;
- Tread-mounted power shovels or hydraulic excavators and trucks;
- Bucket-wheel excavators and trucks;
- Draglines combined with some other materials-moving system;
- Diesel-powered scrapers, which are excellent for large-area and desert placer deposits; and
- Bulldozers and a truck-loading chinaman chute (a colloquial term for a loading ramp that carries material to a truck parked below it).

Also included are various types of wire rope scrapers (tuggers), which move loose material by bucket to a truck- or hopper-loading system. These systems having curved-type buckets are used for mining diamond placers off the coast of West Africa. Another example of a scraper that can be used in deep water is the Ramsey bucket, shown in Figure 11.2-2.

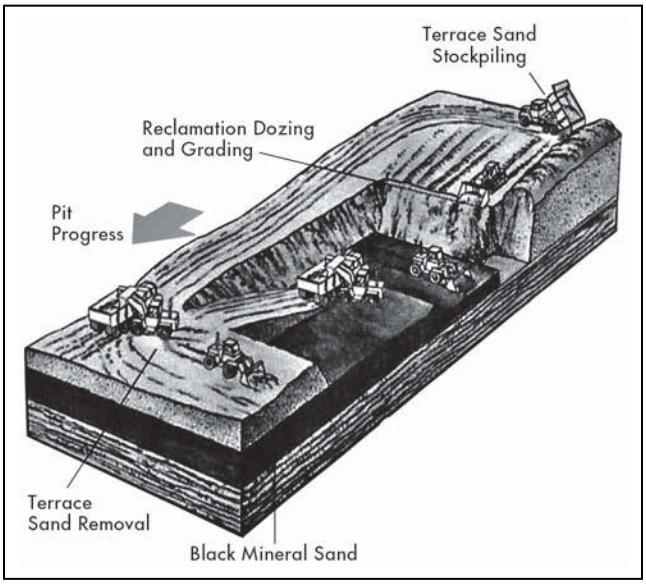
It is unnecessary to describe the many combinations of mining sequence for each of these pieces of equipment, since their use is current practice in hundreds of sand-and-gravel, quarry, and open-pit operations around the world. However, how to mine the placer and reclaim it at the same time is worth at least one example.

At the Oregon Resources Corporation Coos Bay mine, the material to be mined is a series of deposits of dry black sands containing commercial chromite, zircon, and garnets located on ancient beach terraces. In some ways, the mining will be more like a sand-and-gravel operation than an open-pit mine. Two mining systems were proposed (Figures 11.2-3 and 11.2-4), both ending in reclaiming the strip as shown in Figure 11.2-5.

After the small amount of waste overburden (averaging only 6 m [20 ft]) is removed, the mining method is simple front-end-loader/dozer, truck-operation strip mining. The process is similar to coal strip mining in the Midwestern United States, but on a much smaller scale. That is, they immediately mine and reclaim a series of parallel pits across the deposit. After the stripping has taken place for that pit for a length of 91 to 122 m (300 to 400 ft), the mining can commence. Most of the pits are not level, and they will be worked from the lower elevation upgrade.

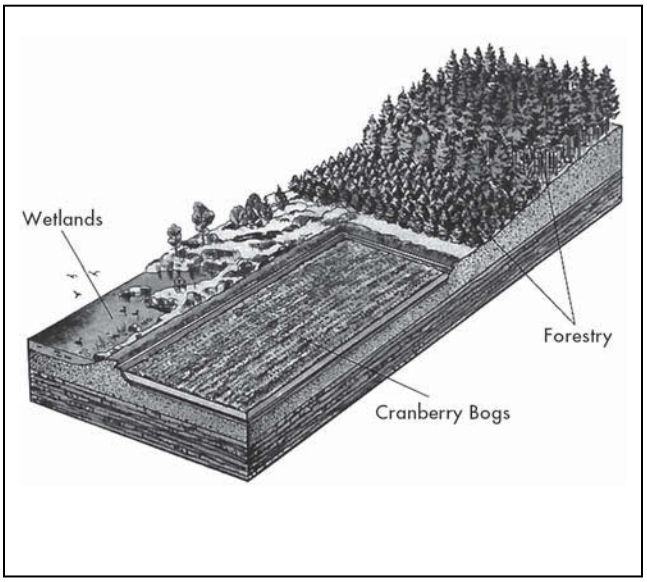
The mining is a simple process of bulldozing the black sands down a slope to the pit bottom, where a front-end loader loads the material to a portable scalping-screening plant hopper, which removes the deleterious materials and transfers the heavy mineral sands to a stockpile. From the stockpile, the material is loaded into bottom-dump trucks and hauled to the plant in Coos Bay, approximately 33 km (20 mi) away. When developed, the mine is expected to produce an annual average of 977,810 t/yr (metric tons per year) (1,077,840 stpy [short tons per year]) for the first 8 years and an average of 850,000 t/yr (936,935 stpy) for at least the next 12 years.

The trucks returning from the plant carry cleaned and dewatered sand from which the ore has been extracted. This



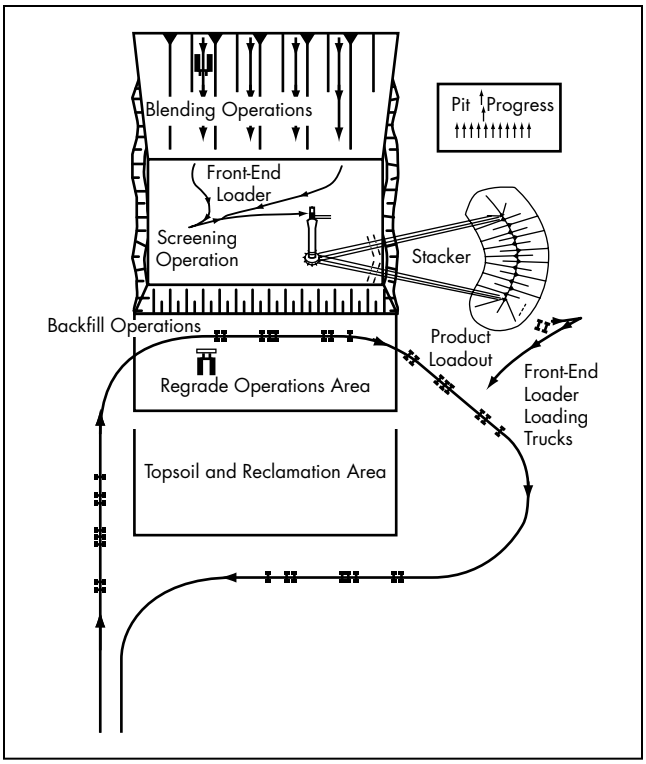
Source: Smith 2008.

Figure 11.2-3 Beach placer being mined and reclaimed within the same strip of mining



Source: Smith 2008.

Figure 11.2-5 Results of the mining and reclamation process shown in Figures 11.2-3 and 11.2-4



Source: Smith 2008.

Figure 11.2-4 Mining with dozers and front-end wheel loaders with reclamation being accomplished concurrently

is dumped at the edge of the pit, and a dozer pushes the material into the pit and fills it. The backfill pace keeps up with the mining extraction, so that no more than about 30 m (100 ft) of pit bottom is ever exposed. The disturbed swell of the material being brought back will fill the pit within about 81% of the original elevation.

The dozed backfill is then graded to the approximate contour of the original ground or is restored to a contour requested by the landowner to be reclaimed or to satisfy environmental regulations.

Frozen Placers

Mining the frozen placers of the earth's Arctic regions poses special problems. The following summarizes the problems connected with the mining of frozen placers (McLean et al. 1992):

> Practically all the auriferous gravel deposit north of the Alaska Range in Yukon and Alaska were frozen and had to be thawed for mining by all methods other than hydraulicking (Daily 1973). Gravel deposits to about 15 ft (5 m) in depth often could be thawed by solar action after removal of all vegetation and muck. The early drift miners used wood fires to sink shafts, and steam was delivered to pipe points driven into the face of drifts. During earlier years, ground was thawed for dredging by the use of steam introduced through pipes driven into the ground, which was inefficient and expensive. The fact that frozen gravel could be thawed best with cold water was discovered in 1918–1920, and that method was developed to a high level of efficiency as a result of research of the United States Smelting, Refining and Mining Co. The method consist of delivering water at ambient temperature, under low natural or pump pressure, through special pipe points either driven into the ground to bedrock or set in predrilled holes. Water temperature was seldom over 50°F (10°C). Flow was continued until the ground was thawed, which required six weeks [for] two or more seasons, depending on the depth of ground and character of deposit.
>
> The total volume of ground thawed with cold water at the three major properties in Alaska and

> Yukon from 1924 until termination in 1964 was more than 360 million yd^3 (330 million m^3). Average water duty was approximately 11 yd^3 (10 m^3) per MID (miner's-inch-day) of water (1 MID = 1.5 cfm per 24 hours). Average rate of thaw in stream gravel to depths of about 45 ft (14 m), using ¾-in.- (19-mm-) diameter pipe points driven in an equilateral triangle spacing of 16 ft (4.9 m), was 1½ days of water under pressure.

In the mid-1970s, the University of Alaska performed a research project demonstrating that drifts could be driven through frozen ground using drilling and blasting techniques. Likewise, frozen fractured rock is often drilled and blasted in mine sites like Polaris on Little Cornwallis Island near the North Pole.

Hydraulic Mining of Placers

The term *hydraulicking* as applied to placer mining describes the excavation of gravel banks by streams of water under pressure from nozzles. Thrush (1968) defines it as follows:

> The process by which a bank of gold-bearing earth and rock is excavated by a jet of water, discharged through the converging nozzle of a pipe under a great pressure, the earth of debris being carried away by the same water, through sluices, and discharged on lower levels into the natural streams and watercourses below.

This practice was at one time (the late 1870s through 1984) a very efficient manner by which to mine the auriferous (gold-bearing) gravels in northern California, Alaska, and the Yukon. These early mining techniques brought about the first antimining environmental laws in the United States (the 1884 Sawyer decision), which prohibited any further discharge of waste material into the Sacramento and San Joaquin rivers. The mining method that was applied—well documented in Peele (1945)—has few applications in "enlightened" countries with environmentally restrictive regulations. However, hydraulic mining is also applied to nonplacer mining, namely coal and tailing reprocessing, neither of which is covered in this chapter.

Subgrade Hydraulic Mining

Places in Thailand and Malaysia for many years were mined by subgrade hydraulic mining (Daily 1973). The following conditions favor this type of mining:

- Many deposits of limited areal extent exist that are unsuitable for mining with dredges.
- An abundance of clay is pulped satisfactorily by water action.
- Extremely irregular limestone bedrock has deep channels in many places, which become exposed in the bottom of the pit.
- High-grade concentrations of mineral are often present in recesses in the top of the bedrock, which can be cleaned effectively only by handwork.
- Water is available in sufficient quantity during all or most of the year.
- Electric power is available at a reasonable rate.

Table 11.2-1 Estimated cost for dryland placer mining (2009 U.S. dollars)*

	Stripping Ratio			
Daily Production	**1:1**	**2:1**	**4:1**	**6:1**
Per metric ton of ore				
500	6.40	7.70	10.55	13.25
1,000	6.00	6.65	7.20	7.85
2,000	5.60	6.10	6.75	7.30
Per short ton of ore				
500	5.80	7.00	9.55	12.00
1,000	5.45	5.95	6.55	7.10
2,000	5.10	5.55	6.15	6.60

*Estimate includes wages/benefits, operation of equipment, fuel, power, and supplies. It covers stripping, mining, waste replacement, and minimum reclamation. Long haulage is not included. It does not include processing except for a scalping screen on-site. Assumes 1,672 kg/m^3 (104 lb/yd^3) material.

- Space is available for slime-settling ponds sufficient to permit compliance with regulations.
- An adequate supply of labor at acceptable cost is available; many operations are conducted by a type of cooperative or family organization.
- Security can be provided. Security is a constant problem, especially protection from roving individuals or groups at night.

Estimated Placer Mining Cost

The average cost of a timbered drift in consolidated ground is estimated to range from approximately \$219 to \$390/m^3 (\$6 to \$11/ft^3), depending on the size of the drift.

Table 11.2-1 shows the range of mining costs using wheeled front-end loaders or hydraulic shovels, bulldozers, and trucks to mine a dry placer deposit above the water table in mild terrain and in a location where the weather allows work year-round. The cost also includes in-pit scalping or screening but not mineral concentrating. The accuracy is no better than ±30%–40%.

PLACER MINING BY USE OF DREDGES

Placer mining is the removal of alluvial material of economic value. Traditional placers were gold deposits found in active or paleo and buried river courses. More recently, the term has come to include alluvial deposits containing economic values of gold, platinum, or silver (rare); heavy minerals of tin, titanium, rare earths, or iron; diamonds; and commercial aggregates. These deposits may be concentrated beneath an overburden of alluvial material some hundreds of meters thick, or they may be disseminated throughout the material or exposed on or near the surface. They may be in small concentrations or cover very large areas, and they may be deep beneath the oceans. The mining of unconsolidated materials such as manganese nodules and metalliferous muds from the deep oceans will be touched on only briefly.

Dredging is the recovery of mined materials from under water. The two basic types of dredge, mechanical and hydraulic, are distinguished by the method used to lift the material to the surface. Many variations and combinations of these types are available. The final selection of a dredge for a mineral deposit, however, is based on many factors

including, but not limited to, the natural and social environments in which the deposit is located, the characteristics and current value of the deposit, the depth of the water, transportation distance, the efficiency of recovery, and the cost of the dredging system.

The decision to dredge or use other means of excavation depends on factors relating to the presence or availability of water and the characteristics and tenor of the deposit. With rare exceptions the dredge is used only in alluvial material, and the type of dredge chosen may be strongly influenced by the presence or absence of coarse materials, boulders, or other obstructions. Mining dredges for gold and heavy minerals are generally self-contained; the concentration plant is an integral part of the dredge or floats in close proximity. When there is a considerable overburden and little water, dry excavation employing large bulldozers, scrapers, or draglines may initially be used to remove the overburden and expose the ore, much like strip mining for coal. Offshore the depth of water may be paramount, but the nature of the deposit may call for specially designed systems for the platform and for excavation and recovery of the minerals.

Preparations and Planning the Dredging Operations

Dredging preparation begins with accurate evaluation of the area by sampling on an appropriate grid and preparing a three-dimensional representation of the deposit. Reverse-circulation or churn drills suitable for sampling are normally used. More modern methods of downhole sensors are sometimes employed. In lacustrine or coastal areas, continuous surveys with seismic or acoustic penetration may be preferentially used to develop the dimensions of the deposit in place and to select appropriate sites for drilling. The grade of the ore and characteristics of the deposit determined from the sampling data are used to select a dredge of the appropriate type and capacity.

The initial placement of large mineral dredges on land may typically involve the excavation of a pit in which the dredge is constructed. On completion, the pit is flooded and the dredging proceeds according to plan. In offshore or coastal areas, dredges may be towed directly to the site, but many more modern dredges have ship-type hulls and are self-propelled.

Crew sizes will vary considerably depending on the size and type of dredge and the social environment in which the deposit is located. The offshore tin recovery bucket ladder dredges in Indonesia typically had crews of 80 or more. Modern dredges are more automated but each shift needs a dredgemaster, an engineer, surface crews to maintain the control lines, and mechanical crews to conduct constant maintenance on all moving parts. Including shore, office, and transportation crew, the operation usually employs about 70 during the dredging season and 30–50 during the winter months. Fully automated systems require fewer personnel on-site (Garnett 1999a).

Dredge Location and Grade Control

Mining and grade control are determined from the sampling data. The sampling data will be treated in the same way in lakes as on land, although it may be necessary to adapt the sampling equipment to the environment. In northern latitudes, drilling may be carried out from an ice cover during the winter. Control in ocean beach environments is difficult, particularly where heavy surf or high tides are prevalent, and draglines may be used to mine in bulk without regard to whether the recovered material is high or low. In other situations, notably the diamond operations in southwestern Africa, the ocean may be held back by specially constructed massive berms or dikes so deposits can be worked in the dry.

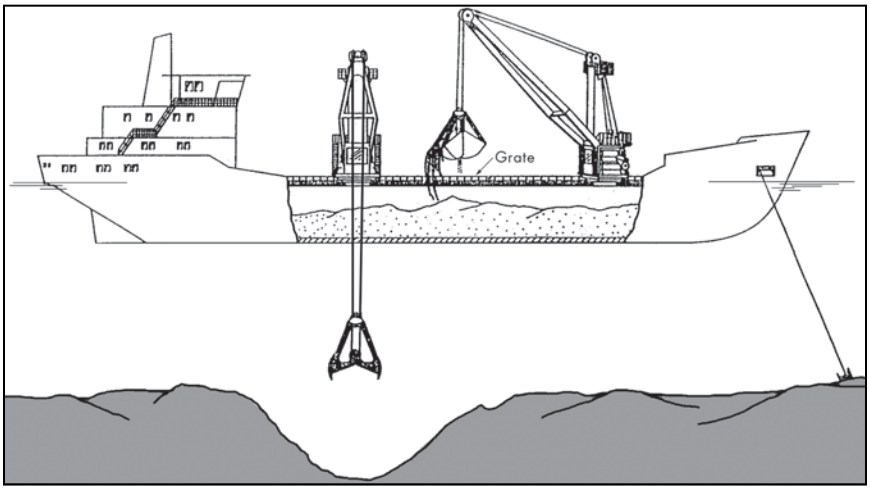
Sampling equipment varies considerably in its methodology and cost. The more valuable the economic material, the more accurate and difficult the sampling will be. Sampling of beach sand may require only a determination of the depth of the sand to bedrock. Sampling of alluvial gold deposits requires extreme accuracy because of the value of the grains, and a "rich" deposit may contain only a few parts per million of gold. Because the specific gravity of gold is approximately 19, the strong tendency of gold grains is to migrate to or away from the sampling device when it penetrates and disturbs the ground. This effect can create significant sampling errors. An alluvial diamond deposit may contain only one part per billion of diamonds; these areas are generally sampled by very large bulk samplers that may be prototypes of the proposed mining device (Garnett 1999b).

An important aspect of determining the efficiency of the dredging operation is the R/E factor, a measure of actual recovery divided by the recovery estimated from the sampling data. This number may vary between 0.5 and 2.5 depending on the accuracy of reserve estimation and the recovery of the gold by the dredge and in the beneficiation process. A factor of 1.04 was recorded by the Alaska Gold Company, averaged over a period of 4 years (Garnett 1991).

Natural Hazards, Waste Disposal, and Environmental Controls

Natural hazards in minerals dredging may include severe winter conditions in Arctic environments, the inclusion of large boulders in river gravels, buried trees in tropical areas, or deeply gullied bedrock in other areas. The overcoming of these problems may be expensive and must be factored in to the unit costs of the operations. In the Arctic, it may be possible to keep a dredge pond open by sawing the ice and removing it from around the dredge, but more often the dredge may be put on a maintenance basis during the winter months. If large boulders are present, the dredgemaster may be required to mine around them, thus losing time and pay dirt, or it may be possible to lift them out of the way. Some large bucket-ladder dredges have such a capability. Buried trees, commonly found in the tin fields of Southeast Asia, may be circumvented in similar fashion. The problem with gullied bedrock can be quite severe if the rock is too hard to disintegrate with the dredge. In the diamond placers on the southwest African coast, massive berms have been built to keep out the sea, and the overburden and surficial waters are completely removed. The gullies where many of the diamonds are retained are swept by hand with small brushes. Offshore, new dredge heads have been developed that have the capacity to disintegrate the bedrock incorporating the gullies.

Discharge of tailings from a dredge may be the result of overburden removal, or the separation of the pay dirt from the gangue material. In either case the waste materials may be a significant part of the dredge output, and they must be disposed of in a manner that does not interfere with the recovery of the pay dirt. This may be done by hydraulic or mechanical diversion of the waste to an area normally located behind the dredge. If the mined ground must be restored, the soil cover may be removed separately and stockpiled away from the dredged area for later replacement over the mined area by earthmoving equipment.



Source: MMS 1993.

Figure 11.2-6 Hopper dredge using clamshell for both loading and unloading

Environmental controls may preclude the use of certain areas for tailings disposal because of the runoff from the waste. However, most primary treatment at the dredge site involves only sizing and separation by physical properties such as specific gravity, color, or magnetic susceptibility. Chemicals are rarely added and the environmental problems may not be paramount. However, the issuance of permits is generally a political matter highly influenced by local and subjective feelings and can result in much frustration and delay. Waste-disposal permits must be secured at a very early stage in the planned development process. It should be recognized that all forms of placer mining are extremely cost sensitive. As such, dredging operations cannot withstand imposed expenditures on generic environmental research programs that are geographically located well outside the zone of influence of the dredging operation (MMS 1993).

Silting of watercourses can be prevented or lessened by containing and recycling the wastewater using cyclones or other dewatering devices. Containment or settling ponds may also be installed in appropriate areas. Offshore, some hydraulic dredges have used an antiturbidity system that allows the major solids to settle in a holding tank before discharge.

Types of Dredges

Mineral dredges may be generally classified as mechanical or hydraulic, depending on the means of transporting the dredged material from the dredge head to the dredge platform (Duane 1976).

Mechanical Dredges

Mechanical dredges transport the material from the deposit via moving buckets or moving belts. Major types of dredge buckets include the dragline, grab bucket, fixed arm, bucket ladder, bucket wheel, and continuous line.

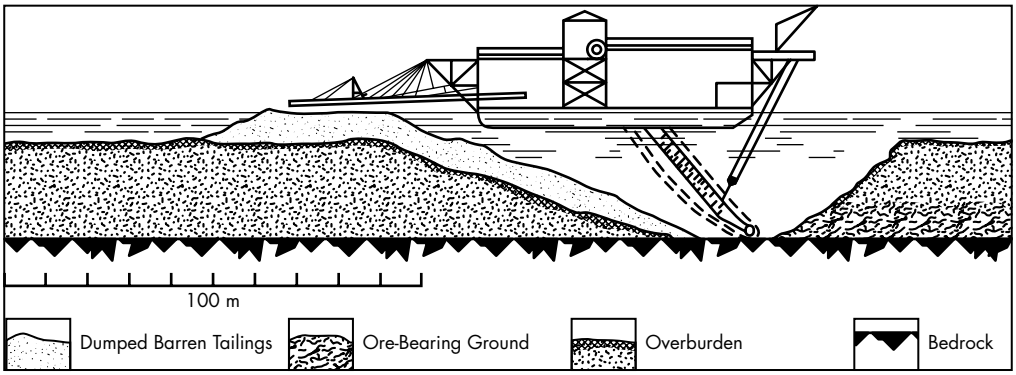
Dragline. The dragline is a simple excavating bucket attached to a cable on a cranelike ladder structure. The bucket may be drawn back and forth until the required depth is achieved. Excavated materials are swung aside on the crane for dumping. Normally used in the mining of nonmetallics such as phosphates, aggregates, coal, or overburden, the dragline has the advantage of simplicity and comparatively low cost. Bucket size may vary from 0.76 m^3 (1 yd^3) or less to more than 38 m^3 (50 yd^3) for some mega-draglines.

Grab bucket. The grab bucket is commonly used for the mining of aggregates or overburden. The bucket or clamshell is manipulated by a crane-type ladder to dig vertically into the material. The open bucket is dropped onto the material and its two halves automatically close on the load when the bucket is raised and swung to the point of discharge. Normally limited in capacity and depth of excavation because of multiple cables, the grab bucket is, however, frequently used in coastal marine environments for reef coral sands (see Figure 11.2-6.)

Fixed arm. The fixed-arm bucket is a more rigid machine that can dig into a deposit using an upward and forward force applied to a toothed open bucket on the digging arm. It is normally used for dry excavation but has been adapted for underwater use on submersible platforms with the addition of a bucket wheel or a rotating cutter feeding in to a suction line.

Bucket ladder. The bucket ladder, a heavy and sometimes ponderous dredge, has been traditionally and widely used for the mining of gold and tin placers throughout the world. Bucket ladders are generally capable of handling heavy ground containing coarse materials and boulders. On land they may be constructed on a barge in a dry dredge pond that is flooded after construction is completed. Bucket ladders use a continuous rotating loop of heavy steel buckets on the ladder, cutting into the forward face of the pond and discharging into a washing and concentrating plant on the dredge. Waste materials are discharged well behind the dredge from chutes or belt conveyors as the bucket ladder moves forward. The concentrates are gathered onboard and shipped by local means to the smelters on shore.

There are two basic types of bucket-ladder dredge. The New Zealand type is moved forward on a headline anchored several hundred meters ahead of the barge and controlled laterally by anchored lines on each side. The more common California or spud type is propelled forward by a walking effect using two widely separated spuds at the stern of the barge and alternately raising and setting them as the ladder is swung from one side of the cut to the other, also using lateral anchored lines (see Figure 11.2-7). Bucket ladders are limited in the depth of cut by the length of the ladder. When



Source: MMS 1993.

Figure 11.2-7 Bucket ladder dredge with head line and anchors for estuarine or offshore operations

the bucket-ladder dredge is used offshore, the barge hull is modified to oceangoing standards, and headlines and multiple anchors are used for control. The dredge may be towed directly to the site, but many of the more modern ones have ship-type hulls and are self-propelled.

Bucket wheel. The bucket-wheel system consists of a rotating wheel with digging buckets on the periphery that discharge into a ladder-mounted belt conveyor or a hydraulic pipeline. The wheels may vary in diameter from a few meters to 10 m (30 ft). The bucket wheel is very adaptable, and with the major advantage of a powerful digging torque, it is now commonly preferred over the heavier and less mobile bucket ladders.

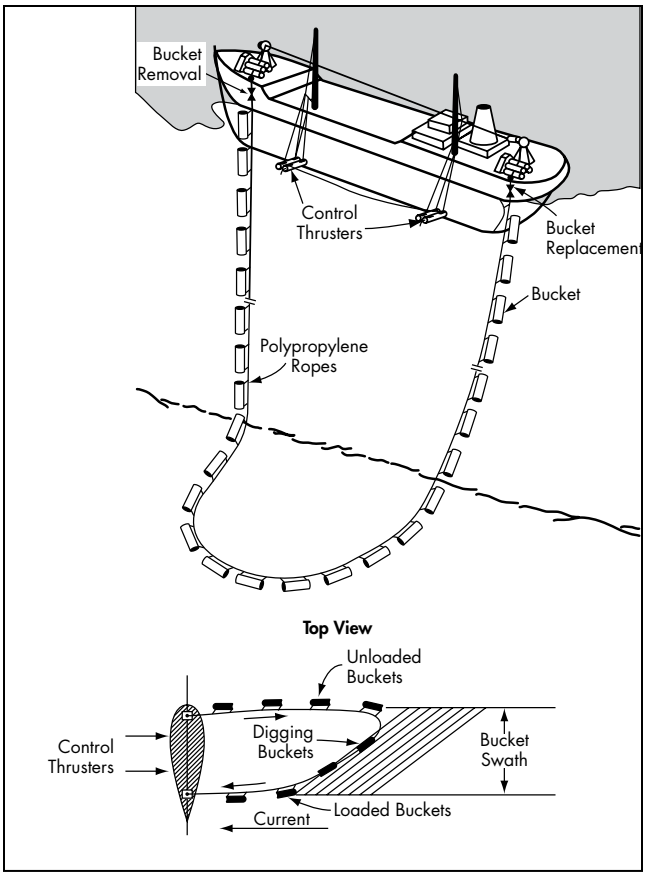
Continuous-line bucket. The continuous-line bucket is based on a system for unloading bulk granular materials from stockpiles or ship cargo holds (Figure 11.2-8). The bucket may be bell shaped or flat and attached to a continuously rotating flexible line. Its advantage when excavating loose materials such as sand is its capability to adapt to water depths. The system has been modified and tested for undersea mining of surficial deposits such as manganese nodules in water depths of 4,000 m (12,000 ft).

Hydraulic Dredges

Hydraulic dredges transport the dredged material from the dredge head to the surface in a slurry through pipes that may form, or be contained in, the dredge ladder. The dredge pump, usually a specially designed solids-handling centrifugal pump, may be on the dredge or submerged on the ladder. Submergence of the pump or addition of a submerged booster pump eliminates cavitation in the suction and may be necessary when the dredging depth exceeds cavitation depth (Figure 11.2-9). In special cases, air lifts or Venturi suction may be used in place of centrifugal pumps. The pump may discharge directly to a hopper, a treatment plant, or a sidecasting device on the dredge or it may be piped to a remote treatment plant or directly (in the case of a beach reclamation) to a fill area.

Variation in Dredge Heads

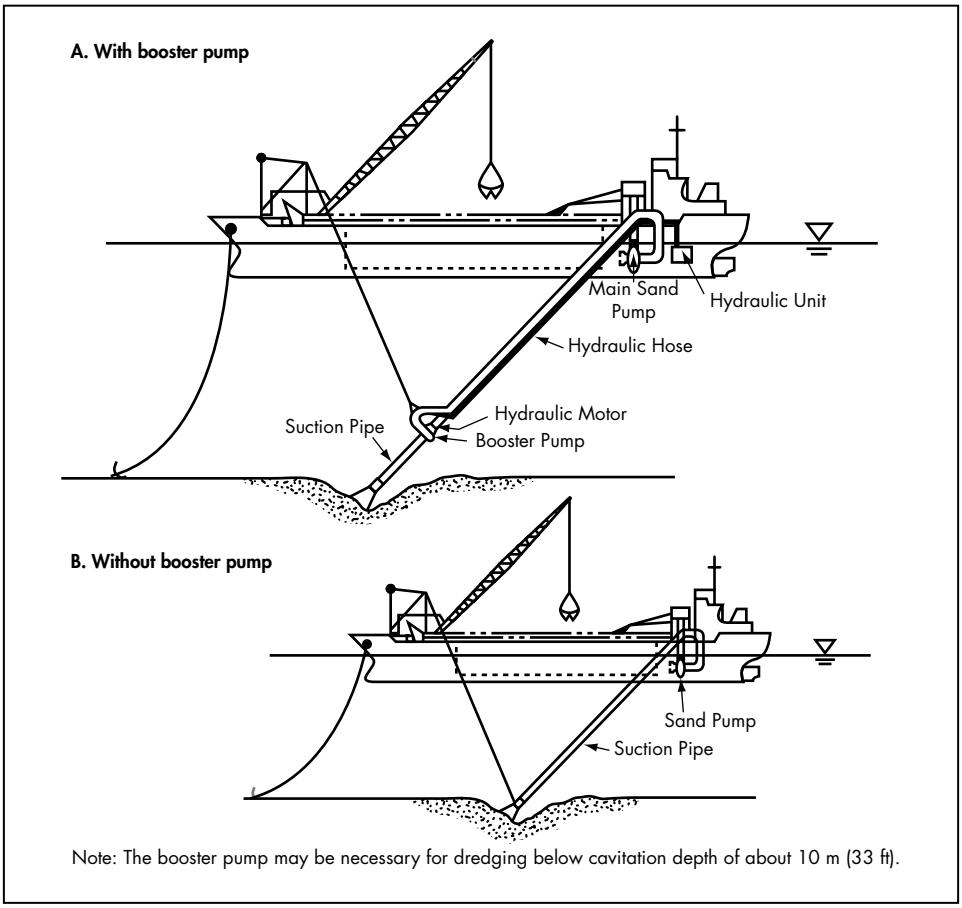
Straight suction heads are commonly used for removing sands or mud in harbor maintenance or for back-of-the-reef collection of carbonate sands for beach maintenance or cement manufacture. Small diver-operated suction devices are sometimes used for prospecting for river alluvials or in archaeological work for the recovery of artifacts.



Source: MMS 1993.

Figure 11.2-8 Continuous-line bucket dredge tested in 4,000-m (12,000-ft) water depth. Note the bucket swath width controlled by the separation of the lines and the transverse movement of the vessel.

Trailing drag heads are commonly used from seagoing hopper dredges to mine commercial aggregates or for the clearance of river channels for commercial shipping. They are also adaptable for the mining of some seabed surficial mineral deposits such as heavy mineral sands or manganese nodules (Figure 11.2-10). They have a shallow but strong digging capability because of the momentum of the moving vessel.



Source: MMS 1993.

Figure 11.2-9 Suction hopper dredges

Rotary-cutterhead dredges are commonly used in heavy ground where considerable energy is required to penetrate the material being mined. The material at the working face may be extensively disturbed, and use of these heads for the recovery of heavy minerals may result in significant losses of pay dirt, particularly if the underlying bedrock is very hard, rough, or gullied (Figure 11.2-11).

Horizontal auger heads are used in soft material that may be passed from a wide cut into the suction pipe opening. They are more often used for maintenance work in harbors than for mining. However, their use has been proposed for the recovery of surficial unconsolidated mineral deposits such as manganese nodules on deep seabeds.

Bucket-wheel cutterheads are a relatively recent innovation that has proven to be ideal for mining dredges. Digging buckets similar to those on a bucket-ladder dredge, but generally smaller, are mounted on the periphery of a rotating wheel attached to the digging end of the ladder. The buckets have tremendous torque and are capable of removing weathered bedrock. They can be designed to discharge directly into a suction pipe or conveyor without spillage of materials. For the mining of precious metals and heavy minerals, the bucket-wheel suction dredge has capabilities similar to those of the bucket-ladder dredge, and they generally cost less to operate. They have also been used in the form of remotely operated vehicles for sampling of gold placers and deepwater marine metalliferous sulfides.

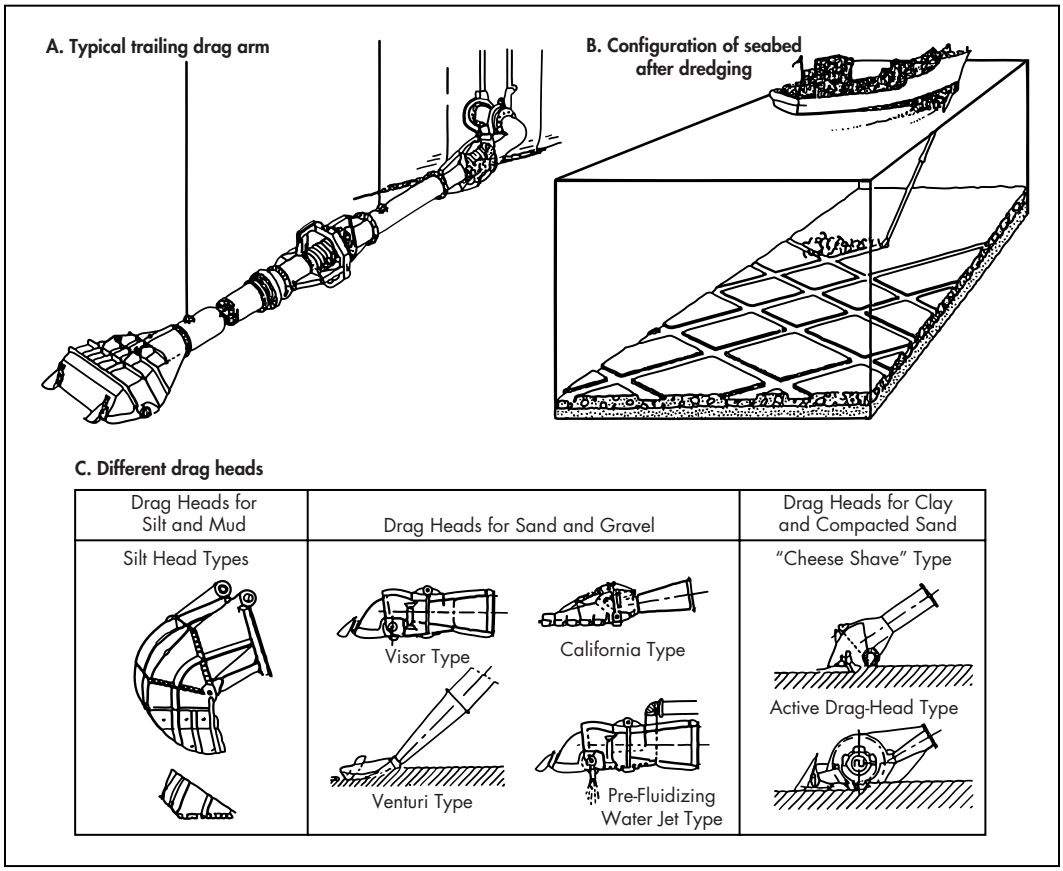
Dredge Manufacturers

Mining dredges of all kinds are manufactured throughout the globe. *Dredgers of the World* (Clarkson Research Services 2009) lists 233 backhoe, dipper, grab, and clamshell dredges; 18 barge-unloading dredges; 53 bucket-ladder dredges; 412 cutterhead suction dredges (including bucket-wheel dredges); 32 types of special equipment including agitation, water-injection, and wormwheel dredges; and 39 suction and 481 trailing suction hopper dredges. The reader should refer to appropriate Web sites such as www.westerndredging.org or www.worlddredging.com for additional information. Manufacturers of pumps are listed on the Internet at www.dredgepumps.com.

Dredging Costs

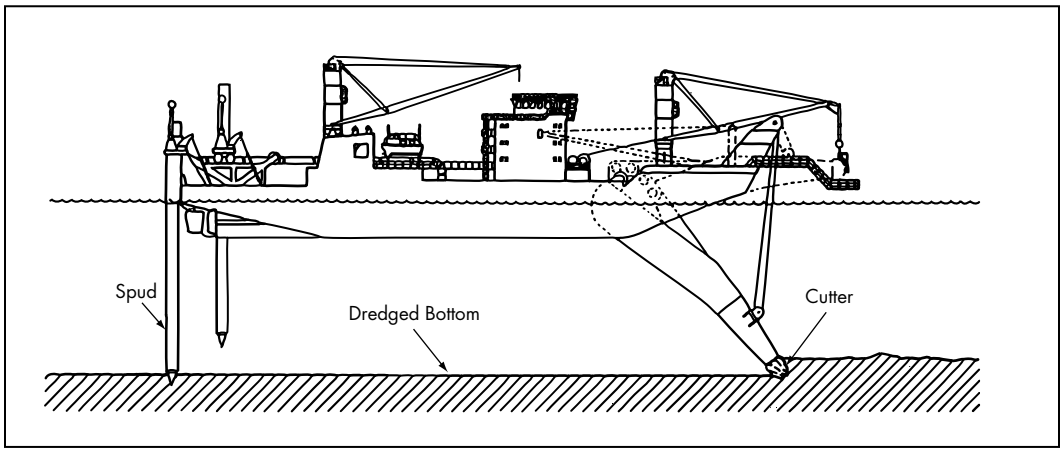
Costs vary widely depending on many factors, as outlined previously. Economies of scale may provide a significant cost advantage for dredging over many other forms of earth moving and transportation (Correia 2005).

Other data have been presented by Garnett (1999a, 1999b) for gold placers and diamonds. These include costs of $\$1.10/m^3$ ($\$0.87/yd^3$) for Indonesian bucket-ladder tin dredges offshore and $\$7.00/m^3$ ($\$5.51/yd^3$) for gold dredging offshore in Nome, Alaska. The latter cost was similar for an experimental remote-controlled underwater miner bucket-wheel excavator with a higher R/E factor. Offshore diamond recovery in Southern Africa was reported in the same sources to be



Source: MMS 1993.

Figure 11.2-10 Trailing suction hopper dredge and various configurations of drag head



Source: MMS 1993.

Figure 11.2-11 Cutterhead suction dredge with rigid boom and spuds (cutterhead could be replaced by a bucket-wheel cutter)

about \$1/m^3 (\$0.79/yd^3) in the ore stratum and \$1.25–\$1.50/m^3 (\$0.98–\$1.18/yd^3) in the overburden. In the same location, costs for the underwater miner (Namsol) were reported to be \$10–\$20/m^3 (\$7.87–\$15.75/yd^3).

Recent costs for beach replenishment sands from immediately offshore at Waikiki Beach in Hawaii (United States) in 2007 were stated to be about 59/m^3 (47/yd^3) using barge-mounted flexible-line submersible pumps piped directly to shore.

The average costs for all dredging carried out by the U.S. Army Corps of Engineers for 2008 ranged from \$3.00 to \$17.85/m^3 (\$2.37 to \$14.06/yd^3), more than doubling their reported costs since 2000.

Obviously, dredging costs are highly variable, and extreme care should be taken in preparing cost estimates for any future operations.

Future of the Dredging Industry

The technology for placer mining on land will continue to improve as new materials and automation are applied to earth moving in general. This will bring down costs and improve the capability to work sustainably within the natural and social environments involved. The worldwide demand for heavy minerals is increasing, and operations are prevalent in newly industrializing countries such as India, People's Republic of China, and Mongolia (Yernberg 2006).

An even brighter future is likely to be associated with the deeper waters of the world's oceans, where applicable technology from other industries is already being applied. Placer diamond mining off the coast of southwestern Africa was instigated in the early 1960s (Cruickshank et al. 1968) and has since been developed into a significant industry involving several companies and countries. See also www.diamondfields.com/s/Namibia.asp and www.theartofdredging.com/peaceinafrica.htm.

In July 2008, Neptune Minerals (www.neptuneminerals.com), a leading explorer and developer of seafloor massive sulfide (SMS) deposits, lodged its first mining license application over the Rumble II West seamount within the exclusive economic zone (EEZ) of New Zealand. The application represented a key milestone in that company's progress toward commercialization and pilot mining by 2010. Neptune also successfully received prequalification proposals for its first seafloor mining contract. Leading international dredging and subsea engineering companies were invited to undertake a build-own-operate contract mining system for the commercial development of the deposits.

Nautilus Minerals, Inc. (www.nautilusminerals.com), has reported that the company was following the lead of the oil and gas industry to tap into the vast marine resources in deep waters offshore. Nautilus was the first company to commercially explore for deposits of SMSs and potential sources of high-grade copper, gold, zinc, and silver in water depths of approximately 2,000 m (6,000 ft). Its initial project, Solwara 1, under development in the archipelagic waters of Papua New Guinea (PNG), utilizes technologies from the offshore oil and gas, mining, and dredging industries, as well as current technologies from marine scientific exploration and positioning. In December 2009, the company reported eight new discoveries in the area, secure port capacity for its operations at Rabaul, and the grant of an environmental permit from PNG authorities. Farther afield, high-grade SMS systems were discovered during a Tongan exploration cruise. Long-term plans include an increase in tenement licenses and exploration applications in the EEZs and territorial waters of PNG, Fiji, Tonga, the Solomon Islands, and New Zealand.

Commercial dredging for deep-seabed nodules, crusts, or other mineral deposits has not yet been carried out in the United Nations area beyond the EEZs. Significant research has been conducted, however, on the potential technology available for their mining at water depths from 3,000 to 5,000 m (9,843 to 16,404 ft) and for the environmental sustainability of proposed operations. Extensive documentation may be accessed at the headquarters of the International Seabed Authority in Jamaica at www.isa.org.jm/en/home.

The future of placer mining by dredging is obviously changing and matching the global needs of the 21st century. Many other useful data may be accessed at www.OneMine.org.

PLACER MINING PROCESSING

When placer ores are mined, they must be upgraded to produce salable product(s) by either a wet or dry process. This section describes the most common upgrading processes and equipment. A heavy mineral sands example shows how processes can be combined.

Placer Ore Beneficiation Methods

Wet Versus Dry Beneficiation

Beneficiation of placer ores generally involves wet processing, with further downstream dry processing necessary for certain mineral assemblages and markets. A majority of the deposits (i.e., tin, phosphate, precious metals) can be upgraded purely by wet processes, but ilmenite, rutile, zircon, and other minerals found in heavy mineral sand deposits almost always require some sort of dry processing.

When compared with dry processing methods, wet processing offers the benefit of easy ore movement (pumping versus dry conveyance) and typically cheaper operating/maintenance costs (per ton treated or transported). For these reasons, combined with the common placer mining method of dredging, wet processing is used at the beginning of placer ore beneficiation flow sheets. The wet separation equipment may not always achieve the product grade and recoveries that are needed. In these instances, dry processing can be used.

In dry systems, the circuit always includes a dryer, which can require an equivalent of 8+ L/t (2+ gal/st) of diesel fuel per metric ton dried. The cost of drying, almost entirely made up of fuel costs, only allows further downstream equipment to achieve the desired separation. Drying does not increase the value of the ore itself and should always be minimized. When dry processing is required, a cost reduction may be obtained if it is preceded by a partial upgrade of the ore by wet processes. The partial upgrading will reduce the amount of material that must be dried and can drastically reduce drying costs. This process is decidedly different than minerals processing that requires flotation, in which the drying that takes place is primarily to meet customer specifications or reduce shipping weight of the concentrate.

Even when wet processing equipment obtains acceptable grades and recoveries, dry processing equipment may be able to achieve higher grades and recoveries. To determine if dry processing is required, lab testing should be performed. Each upgrading process has advantages and disadvantages, and specific mineral assemblage separations can only be determined by testing. The benefits and drawbacks of each type of processing must be weighed in full before a processing flow sheet can be finalized. The performance of the equipment and the overall capital and operating costs of the circuit should also be taken into consideration.

Sizing

Unlike most mineral processing, in which comminution is required to achieve a specific liberation size, particles generally have already been liberated by the time placer deposits are being processed. However, a wide range of particle sizes, from rocks (centimeters/inches in diameter) to clay (microns in diameter), must be properly handled. Trommel screens are used to remove the large waste rock and organic matter and are a last place to break up or remove large clay conglomerates. Cyclones are commonly used to deslime (remove clays and ultrafines) and remove the fine fraction of the ore that cannot be economically upgraded.

The nature of placer deposits may make it necessary to remove clay coatings on the minerals to allow proper separation and improve grade. These coatings are not typically seen in hard-rock operations in which comminution has been used to reach the liberation size. To clean the particle surface, attrition scrubbers are used. These scrubbers mix the slurry at high percent solids, which results in multiple particle–particle collisions. These collisions remove the softer surface coating of clay, which can then be easily washed away.

Vibrating screens are also used to either remove uneconomic size ranges of the ore or to separate the feed into multiple size fractions that can be upgraded using separate circuits. The resulting tighter size range will increase the separation efficiency of physical separation equipment since the size effect of the particles does not come into play as much. Generally wet screens are used on the higher-tonnage initial feed portions of the circuit, whereas dry screens can be found in later portions of the flow sheet or on final product sizing.

Hydraulic classifiers can also be used to size material. They can achieve excellent size separation given a mono-density feed, and they offer the benefits of high throughput, low operating cost, and very low maintenance cost when compared with wet vibratory screens.

Gravity Separation

In general, the greatest upgrading in placer ore beneficiation occurs with spirals. They are typically the first separation devices that the ore encounters after the initial sizing operations. The ore must be sized prior to spiral feeding, with oversize and slimes removed. Particles outside of a separable size range (that varies from mineral to mineral) do not separate well on spirals and can even hinder the separation of particles of more acceptable size. Initial wet concentrator spiral plants are generally installed on a barge that trails the mining dredge. Having the concentrator in the mining area reduces transportation cost of the raw ore and tailings.

In addition to their use as sizing devices, hydraulic classifiers can also be used as density separators. Denser particles report to the underflow and lighter particles report to the overflow. Because hydraulic classifiers combine density and sizing separations, coarser light particles may be misplaced to the underflow while finer heavy particles report to the overflow. To remedy the problem that this poses, both the underflow and overflow can report to separate spiral circuits. The misplaced coarse lights and fine heavies, the result of processing by the hydraulic classifier, must be separated from their respective streams, and feed that has undergone the sizing process lends itself to easier spiral separation than a raw mixed feed would allow (Benson 2005). Because of this phenomenon, newer flow sheets are incorporating hydraulic classifiers into spiral plants at the beginning and middle of a spiral circuit, whereas they traditionally were incorporated at the end (Elder et al. 2001).

Centrifugal jigs are another common gravity separator for placer ore. These enhanced gravity separators use a ragging material that can be varied to change the apparent specific gravity of the separating medium. Both product and tailings streams are screened to remove the ragging material, which can then be recycled. The adjustability of the apparent specific gravity of the separating medium and the centrifugal action of the jig allow for efficient separations. The drawbacks to these separators include the cost of the ragging material and the high maintenance required for the enhanced-gravity portion of the separator compared with other common placer ore gravity separators. Maintenance can include daily (or more frequent) shutdown and cleaning, and yearly rebuilds that can be costly and incur extensive downtime. Typically, centrifugal jigs are used in difficult specific gravity separations to recover higher-valued minerals. For example, they can be used in a scavenger stage of zircon wet gravity circuits.

Wet shaking tables can still be found in placer ore beneficiation, generally where near-specific-gravity separations are required at low throughputs. They provide excellent gravity separations but do have big drawbacks such as low feed rates and large size. Because of these drawbacks, wet tables are typically only used for higher-priced products (e.g., gold and zircon) that have already been at least partially concentrated. Their low feed rate (0.5–2 t/h [0.55–2.2 stph] per industrial-size table) and capital and operating expense must be economically justified before introduction into a flow sheet.

Magnetic Separation

Magnetic processing can be done both wet or dry, and so it lends itself to all aspects of a placer deposit flow sheet. In wet magnetic processing, a low-intensity magnetic separator (LIMS) is used to remove magnetite or other ferromagnetic minerals. In most placer deposits, the magnetite is usually a tailings stream, but in the case of iron sands, it can be a product. In general, medium-intensity wet magnets are not used for placer deposits. However, wet high-intensity magnetic separators (WHIMSs) are widely used. They offer the advantage of high throughput and wet processing (coinciding with the common wet mining methods), and they excel when used at the beginning of circuits where product grades are easy to achieve or extra tailings can be removed prior to drying, resulting in cost savings. As an example, in a typical ilmenite/rutile/zircon deposit, a WHIMS can be used to remove ilmenite that does not meet grade (e.g., low percentage of TiO_2).

Dry magnets have ranges of field intensities similar to those of wet magnets, and they are used to remove similar minerals. Some common dry magnets used in placer beneficiation include the rare earth drum (RED), induced-roll magnet (IRM), and rare earth roll (RER). REDs are medium-intensity dry magnetic separators and are commonly used to remove, as product or tails, more highly paramagnetic minerals such as ilmenite. Higher-strength dry magnets are available as both permanent rare-earth magnets, which are found within RERs, and electromagnets, which are used for IRMs; each has its advantages and disadvantages (Dobbins and Sherrell 2009). The higher-strength magnets are typically used to remove the middle to weakly paramagnetic minerals such as staurolite and leucoxene or to reduce the final $\%Fe_2O_3$ grade of a non-magnetic product (e.g., zircon).

In general, wet magnets are a little less selective than dry because the liquid medium adds a restrictive drag force. On the other hand, dry magnets are not as tolerant of fine material and generally can only effectively treat down to 75 μm, whereas wet magnets routinely treat material finer than –45 μm (although that fine size is not very common in placer ore deposits).

In placer deposits where iron-bearing minerals are present, magnetite can be found. If enough ferromagnetic material is fed to higher-strength magnets, the ferromags cannot be effectively released, and wearing of shells and belts or plugging of matrices become issues. In these instances, low-intensity magnets can be used as scalpers for both wet and dry processes.

Electrostatic Separation

The typical separation equipment that coincides with the dry magnetic separators are the electrostatic separators. These can be found before (standard for heavy mineral sands) or after the magnets in placer ore beneficiation plants. They are mostly found in heavy mineral sand separation plants where the added physical property of conductivity is used to separate and upgrade the various products. Because the surface conductivity is so important for this type of separation, mineral surfaces must be clean (see information about attrition scrubbers previously in the "Sizing" section). All electrostatic separators either require minerals to be heated for a separation to occur or separate more effectively when minerals are heated. Some minerals, such as titanium-bearing sands, change conductivity at higher temperatures. The heat (+100°C [+212°F]) also removes the surface moisture that may be present, which can mask the true conductivity of a mineral.

The two common types of electrostatic separators are high-tension rolls (HTRs) and electrostatic plates (ESPs). The HTR has a higher capacity per unit length than the ESP (e.g., for heavy mineral sands 2 t/h/m of roll length versus 1 t/h/m [2.2 stph/m versus 1.1 stph/m]) and are generally the first stage of electrostatic separators. Because of the nature of HTRs, coarser nonconductive particles can report with the conductive material, and finer conductive particles can report with the nonconductive product stream. The opposite is true for ESPs. The misplacements associated with these machines make it possible to place ESPs within or after an HTR circuit to achieve effective separation of all particle sizes.

A much less common third type of electrostatic separator is the triboelectric separator. These can separate two nonconductive minerals based on the pre-charge that the minerals receive before they are introduced to the separation chamber. These may be used in place of flotation where water or environmental concerns are an issue. The greatest potential for these within placer ore processing plants is upgrading of phosphate by removal of silica sand and fine zircon cleaning by removal of undesirable nonconductors such as sillimanite and kyanite.

Other Processing

Flotation is not that prevalent in most placer ore upgrading but is frequently found in phosphate operations to separate silica from phosphate. The phosphate flotation circuits are generally large, with feed rates in excess of 1,000 t/h (1,100 stph). The flotation circuits are generally located after hydraulic classifiers that split the feed into tighter size ranges to allow for better separation efficiencies of the flotation cells. Circuits can be set up as rougher-cleaners, where the rougher flotation uses fatty acids as collectors for the phosphate. Because the phosphate floats very quickly, silica is usually entrained within the phosphate concentrate. To clean up the phosphate further, a cleaner flotation is used with amines as collectors to float the silica.

A placer ore beneficiation process generally limited to zircon sand is hot acid leaching. This process removes iron staining on sand grains with the addition of a high-concentration sulfuric or hydrochloric acid. This would only be incorporated where low $\%Fe_2O_3$ grades are needed for final product specifications that cannot be accomplished by other means and where the value of the treated mineral can pay for the addition of this type of circuit. A hot acid leach circuit would include a dryer for heating, reactor vessels, rinsing equipment, acid storage and distribution systems, and an effluent treatment plant. Effluent treatment plants are generally simple and include addition of a base to neutralize the acidic effluent and a thickener to remove precipitated solids from the effluent system. The effluent water would be recycled back to the acid leaching circuit so that no off-site discharge would have to occur.

Tailings Handling

Tailings from the floating concentrators are immediately pumped off the barge to backfill the pond or cut (in the case of offshore tin mining). In pond-dredging operations, clays can be pumped into the operating pond when they quickly drop out of suspension or are not much of an issue. If they build up in concentration and hinder separation, clays can be pumped to settling ponds where they dewater and are eventually mixed with the top sand layer during reclamation.

With dry mining there is no working pond, so all tailings from the concentrator must be deposited into settling ponds. These ponds allow reclamation of water as well as settling of clays. In some instances, mixing the clay and sand (which come from different tailings streams within the concentrator) speeds overall dewatering rates.

When mineral separation plant tailings streams are not pumped directly back to a mining pit, they are generally sent to a stacker. The stacker includes a dewatering cyclone with some type of distribution mechanism for the underflow of high-percentage solids. The distribution mechanism (e.g., swinging chute or movable cyclone) allows the underflow tailings to stack because of their high percentage of solids and to become partially dewatered (5%–10% moisture) over time. The dewatered solids can then be trucked back to the mine to backfill mining areas or to create berms for dewatering ponds. Clays report back to process water ponds or thickeners, where they are allowed to settle.

General Heavy Mineral Sands Flow Sheet

Heavy mineral sands operations use nearly all of the processes mentioned previously. For this reason, an example of a typical heavy mineral sands processing flow sheet is given to illustrate how the previously mentioned equipment would be incorporated into one flow sheet (Figure 11.2-12). The overall flow sheet is broken up into a wet concentrator section that produces a heavy mineral concentrate and a mineral separation plant that produces the final products. The wet concentrators are mostly found on floating barges located behind the mining dredge. Mineral separation plants (MSPs), on the other hand, are always land based and are located away from the mine site so concentrate can be easily transported from any location within the mine plan and so that multiple mine sites can feed a single MSP. The central-location concept can move the MSP far from the mine site(s); 50+ km (31+ mi) can still be economical.

When feed is introduced to a wet concentrator, it is first sized by trommel to remove oversize and then treated with hydrocyclones to remove clays and fines. The feed is then sent to a gravity circuit that includes spirals (Reichert cones can be found but are rare) and may include hydraulic classifiers. Typically, the classifiers would be placed at the end of the spirals circuit to help remove any misplaced silica and any fine material that would not be recoverable within the MSP. An attritioning step would be included between the spiral circuit and classifier to clean the sand surface and allow washing of clays if they are present. Spiral circuits have multiple stages; the final product usually passes through four spiral stages

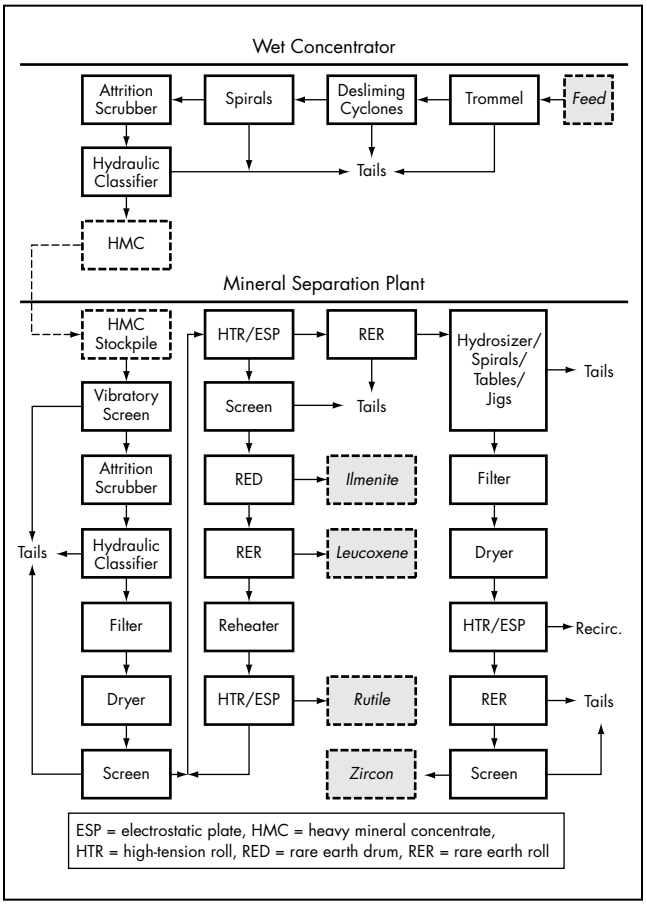


Figure 11.2-12 Generic heavy mineral sands flow sheet

including at least one scavenger step. The final product from the wet concentrator is referred to as heavy mineral concentrate (HMC).

After the HMC is produced, it is transported to the MSP. Blending may occur from multiple mine sites to achieve consistent product grades and mineral assemblages in the feed. Consistent mineral assemblages can reduce bottlenecks in the MSP that may occur at low feed rates if too much of one mineral is present in the feed.

The first circuit of an MSP would generally be a feed-preparation step that may include wet screens to remove oversize as well as attrition scrubbers and hydraulic classifiers to guarantee clean grain surfaces and minimal silica. The feed would then be filtered and dried. High-tension separators are used after the drier to utilize the inherent heat from that process to ensure efficient separation of conductors from nonconductors. Conductor minerals would then be run through various magnets (RED and RER magnetic separators) to remove ilmenite and leucoxene as products. The resulting conductor/nonmagnetic minerals would be reheated and passed through more electrostatic separators to produce a rutile product.

The nonconductor stream from the first high-tension separator circuit is run through a separate magnet circuit to remove nonconductive/magnetic minerals such as monazite and staurolite. Depending on market conditions, monazite and staurolite may be marketable products. The resulting nonconductor/nonmagnetic minerals have a high percentage

Table 11.2-2 Heavy mineral beneficiation cost example

Area	Ore, US$/t	Heavy Mineral Concentrate, US$/t
Mining (dredge)	0.60	5.10
Wet concentrator	1.40	13.45
Mineral separation plant	0.60	6.50
Product handling and delivery	0.95	10.95
General and administrative	0.35	3.75
Concurrent reclamation	0.10	1.05
Total operating cost	4.00	40.80

Courtesy of Outotec (USA), Inc.

of zircon but also an upgraded silica content. To remove the lighter silica, this stream is rewetted and run through wet gravity separators that include a combination of spirals, hydraulic classifiers, wet tables, and/or centrifugal jigs. The heavy concentrate from the MSP wet gravity circuit has a high zircon content but generally does not meet product specifications on $\%TiO_2$, $\%Fe_2O_3$, and potentially $\%Al_2O_3$. To clean up this zircon stream, it is dried and passed through high-tension separators and RER magnetic separators. Based on customers' product specifications, a final screening stage may be needed to remove any oversize or to size the zircon appropriately.

Cost for Mineral Processing of Placer Deposits

Table 11.2-2 shows typical costs for beneficiation of a heavy mineral sands placer deposit. The data assume a 1,000-t/h (1,100-stph) feed rate to the wet concentrator and ore containing an average total heavy mineral (HM) grade of 12.5%. The heavy mineral concentrate produced by the wet concentrator contains 95% heavy minerals; the valuable heavy minerals are ilmenite (65% of HM), rutile (5% of HM), and zircon (5% of HM). Recoveries in the wet concentrator are on the order of 95% for ilmenite, 90% for rutile, and 95% for zircon. Recoveries in the mineral separation plant are 98% for ilmenite, 90% for rutile, and 80% for zircon.

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CHAPTER 11.3

Heap Leaching

Travis J. Manning and Daniel W. Kappes

HISTORY

The method of heap leaching has a long history in the mining business. In the mid-1500s, mines in Hungary recycled solutions bearing copper through waste rock for additional copper recovery. Around that same time, Georgius Agricola wrote about piling rocks in heaps and sprinkling them with water for alum recovery in his book *De Re Metallica* (Agricola 1556). At the original Rio Tinto mine in Spain, acid solutions were added to large heaps of copper oxide ores circa 1750. In the 1950s, heap leaching was used for uranium extraction from ore. It was not until 1967 that heap leaching was developed for precious metals by the U.S. Bureau of Mines; the first large-scale gold heap leach project was at Cortez, Nevada. Fewer than 20 years after heap leaching of gold was first developed, it accounted for 30% of all gold production in the United States. In parallel with gold heap leaching, the last 40 years have seen a tremendous expansion of copper dump and heap leaching (Scheffel 2002). Uranium ores have been heap leached with both acid and alkaline solutions since the 1950s. A list of current and historical uranium heap leach operations can be found on the Web site for the World Information Service on Energy: Uranium Project (WISE 2007). Heap leaching of nickel laterites is still in its infancy with only one project near production, the Çaldag project in Turkey.

HEAP LEACHING EXPLAINED

In a simplistic sense, heap leaching involves stacking of metal-bearing ore into a "heap" on an impermeable pad, irrigating the ore for an extended period of time (weeks, months, or years) with a chemical solution to dissolve the sought-after metals, and collecting the leachant ("pregnant solution") as it percolates out from the base of the heap. Figure 11.3-1 is an aerial photograph showing the typical elements of a heap leach operation: an open-pit mine is shown on the left; on the right is a 2-Mt heap of crushed, conveyor-stacked ore on a plastic pad. Pregnant and barren solution storage ponds are located downslope from the heap. Buildings include a solution process facility for recovering metals from the pregnant solution, a laboratory, a maintenance shop, and administration offices. For a small operation such as the one illustrated here, very limited infrastructure is required.

In a more complex sense, heap leaching should be considered as a form of milling. It requires a nontrivial expenditure of capital, and a selection of operating methods that trade cost against marginal recovery. Success is measured by the degree to which target levels and rates of recovery are achieved. This distinguishes heap leaching from dump leaching. In dump leaching, ores are stacked and leached in the most economical way possible, and success is measured by any level of net positive cash flow.

WHY HEAP LEACHING?

Options for recovering metals from ore are many, including agitation leaching, gravity separation, magnetic separation, flotation, and vat leaching. If the ore can be heap leached (which requires good metals recovery by leaching and a rock type that allows the construction of permeable heaps), this technique offers some significant economic benefits. Recent large-scale, very simple heap leaches (40,000+ t/d) have been constructed for a total project capital cost as low as $3,000 per daily metric ton of ore processed, whereas the total capital costs for an agitation leaching plant are often $15,000–$35,000 per daily metric ton. The capital costs of a flotation plant are $10,000–$25,000 per daily metric ton. Another advantage of heap leaching is that it is a chemical process and the product of the leach operation is usually a metal. For gold and silver ores, the primary product at the mine for a leaching operation is an impure gold/silver (doré) bar that can be sold for 95% or more of the quoted price for the metals recovered from the ore. For copper ores, the primary product is usually copper metal that can also be sold for a high percentage of its value. In comparison, flotation plants, which are the primary competitor to the chemical leaching processes, produce a concentrate of the main value mineral(s), which often carries lower-value components and must be sold to a smelter so that the realizable value of gold, silver, and copper is often only 60% of the quoted metal price of the recovered metals.

A number of factors will determine whether heap leaching is the best fit for a project:

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Source: Kappes 2002.
Figure 11.3-1 Heap leach installation at Mineral Ridge, Nevada

- **Risk.** An increasing number of mining projects are being developed in exceedingly remote locations or places with political and social upheaval. It can be a wise choice to select heap leaching in these situations where a mill, with higher capital costs and more complex equipment systems, would be harder to maintain (or stop and restart) as social and political changes occur.
- **Lack of sufficient reserves.** In many underground and some open-pit mining situations, it is difficult (occasionally even impossible) to develop a large ore resource early in the project. Heap leaching offers a very quick, low-cost avenue for treating such ores. As an example, the Sterling mine in Beatty, Nevada (Imperial Metals Corporation) began life as an underground mine with a reserve of 100,000 t (metric tons) of ore at a grade of 11 g Au/t but eventually heap leached nearly 1 Mt.
- **Differential recovery is not sufficient to justify the added investment.** The operating cost plus debt service for a medium-sized mill (3,000 t/d) will typically be $34/t of ore treated, whereas the comparable cost for the heap is $12/t (year 2009 cost basis). Since the heap leach typically gets 25% less recovery than the mill, the gross value of the ore would typically have to be greater than $85/t in order to justify the mill.
- **Capital is very difficult or expensive to raise.** Heap leaching can give a small company a project that can catapult it into the "big league" from a relatively small investment. Often early-stage capital can be raised only by excessive equity dilution or acceptance of onerous bank covenants. The heap leach also gives the management of a small company more time to build up an experienced operational staff before it tackles more complex processes.

HEAP LEACHING CONFIGURATIONS

Configurations of heap leaching available include dynamic heaps, permanent single-lift and multilift heaps, and valley-fill heaps.

Dynamic Heaps

Dynamic heaps reuse the same lined area by loading the ore onto the pad, leaching, washing, and then removing the ore from the pads. They are also referred to as "on/off heaps." The advantage of an on-off heap is that a large amount of ore can be leached in a limited area in relatively thin layers. For ores where the permeability of thicker layers cannot be maintained, which is typical of oxidized copper ores, dynamic heaps often offer the only option. One factor often overlooked is that the leached ore is still an environmental liability and must be moved to another lined area. However, the water balance is easily manageable because of the limited area of the pads. The small area also makes it possible to cover the heap if necessary. Dynamic heaps are applicable to low-grade, precious metal ores containing sulfides that will start to oxidize to form acid within 1 to 3 years. The benefit of using a dynamic heap for this type of material is that the values can be recovered during the short leach time associated with the dynamic heap and the generation of acid would occur in the waste dump, not affecting the leach process.

The pad below a dynamic heap needs to be of sturdier construction than that for a permanent heap because there is more traffic at elevations close to the pad. Pads for dynamic heaps are frequently made of concrete or asphalt for this reason, though with careful ore removal a geomembrane (plastic liner) could be used. Slowly leaching ore can be re-treated by simply restacking it along with the new ore, but this will reduce capacity. Typically the maximum recovery is not reached with dynamic heaps because of the limited time frame for leaching. Before the ore can be removed from the pad, it is usually washed to recover any dissolved values and detoxified if necessary.

Permanent Heaps

In permanent heaps, the ore is stacked on a low-permeability surface and never removed. While this definition also applies to valley-fill heaps, the term *permanent heap* is usually reserved for a heap that is stacked on a relatively tabular ground surface where solution can exit at multiple points across the face of the heap. To expand the available volume for stacking ore, the pad is either expanded or additional lifts are added on top of the ore already stacked. The primary advantages of permanent heaps are that operations are less expensive than dynamic heap operations, and they allow for much longer leach times (up to several years).

Multilift heaps have been built as high as 200 m with successive lifts stacked on top of previously leached lifts. Single-lift heaps are appropriate for some ore types or leaching situations where, for example, the ore is high grade and not very permeable. For single- or multilift permanent heaps, the initial capital cost is relatively low because the liner does not need to be as robust as for a dynamic heap, and a small area can be lined at first with expansion in subsequent years. As multilift heaps grow taller, there are added operating costs for pumping solutions and transporting ore to higher elevations. Permanent heaps require a large area with gently changing topography. The large area required often leads to issues with the water balance in high-rainfall environments, but permanent heaps can be effectively used where rainfall exceeds 2.5 m/yr.

Valley-Fill Heaps

A valley-fill heap leach is just as it sounds: the ore is dumped at the bottom of a valley and is built up, "filling" the valley. Valley-fill heaps are built in areas that do not have enough level terrain to build an expanding permanent heap. Valley-fill

heaps can accommodate long leach times well. A valley-fill expands upward and outward as the valley widens near the top. The necessity for multilifts means that the ore should be strong and maintain high permeability under high loads. The pregnant solution is often stored inside the bottom of the heap and a small pond is used to collect the solution for recovery. This has advantages of less cost in liner, less precipitation collection, and it can keep the solution from freezing in cold climates. If solution is stored within the heap, a better liner needs to be in place because of hydraulic head on the liner. The front slope of the heap needs to be thoroughly designed and very stable to prevent any catastrophic failure. Frequently a retaining structure, or dam, is used to support the toe of the heap.

GEOLOGY OF GOLD DEPOSITS

Carlin-Type Sedimentary Ores

These ores consist of shales and "dirty" limestones containing very fine (submicroscopic) gold. Oxidized ores leach very well, with low reagent consumption and production recovery of 80% or better being achieved. Ores are typically coarse-crushed (75 mm) but may show recovery of 70% or better at run-of-mine (ROM) sizes. The largest of northern Nevada's heap leaches (Carlin, Goldstrike, and Twin Creeks) treat this type of ore. Unoxidized ore contains gold locked in sulfides (typically 1%–3% pyrite) and also contains organic (carbonaceous) components, which absorb the gold from solution. This ore shows heap leach recovery of only 10%–15% and is not suitable for heap leaching. Because of the different ore types, the northern Nevada operations (for instance, Barrick's Goldstrike mine) may employ roasters, autoclaves, agitated leach plants, and heap leaches at the same mine site. Crushing is usually done in conventional systems (jaw and cone crushers) and ores are stacked by trucks.

Low-Sulfide Acid Volcanics or Intrusives

Typical operations treating this type of ore are Round Mountain (Nevada) and Wharf mine (South Dakota, United States). Original sulfide content is typically 2%–3% pyrite, and the gold is often enclosed in pyrite. Oxidized ores yield 65%–85% recovery but may have to be crushed to below 12 mm. Usually the trade-off between crush size and percent recovery is a significant factor in process design. Unoxidized ores yield 45%–55% gold recovery and nearly always need crushing. At Round Mountain, approximately 150,000 t/d of low-grade oxide ore is treated in truck-stacked ROM heaps, 30,000 t/d of high-grade oxide ore is treated in crushed (12 mm) conveyor-stacked heaps, and 12,000 t/d of unoxidized ore is treated in a processing plant (gravity separation followed by leaching in stirred tanks). Primary and secondary crushing is done using jaw and cone crushers; finely crushed ore contains enough fines that conveyor stacking is preferred over truck stacking.

Oxidized Massive Sulfides

The oxide zone of massive sulfide ore deposits may contain gold and silver in iron oxides. Typically, these are very soft and permeable, so crushing below 75 mm often does not increase heap leach recovery. The Filon Sur ore body at Tharsis, Spain (Lion Mining Company) and the Hassai mine in Sudan (Ariab Mining Company) are examples of successful heap leaches with this type of ore. Because the ore is fine and soft, the ore is agglomerated using cement (Hassai uses 8 kg cement/t), and stacking of the heaps is done using conveyor transport systems.

Saprolites/Laterites

Volcanic- and intrusive-hosted ore bodies in tropical climates typically have undergone intense weathering. The surface cap is usually a thin layer of laterite (hard iron oxide nodules). For several meters below the laterite, the ore is converted to saprolite, a very soft water-saturated clay, sometimes containing gold in quartz veinlets. Silver is usually absent. These ores show the highest and most predictable recovery of all ore types, typically 92%–95% gold recovery in laboratory tests, 85% or greater in field production heaps. Ores are processed at ROM size (which is often 50% minus 2 mm or 10 mesh) or with light crushing. Ores must be agglomerated and may require up to 40 kg of cement/t to make stable, permeable agglomerates. Many of the western African and central American heap leaches process this type of ore; good examples are Ity in the Ivory Coast and Yatela in Mali. When crushing is required, one or two stages of toothed roll crushers (Stamler-type feeder-breaker or a mineral sizer) are usually employed. Conveyor systems are almost always justified; ore can be stacked with trucks if operations are controlled very carefully.

Clay-Rich Deposits

In some Carlin-type deposits and volcanic-hosted deposits, clay deposition or alteration occurs along with gold deposition. These ores are processed using the same techniques as for saprolites, except that crushing is often necessary. Because of the mixture of soft, wet clay and hard rock, a typical crushing circuit design for this type of ore is a single-stage impact crusher. Truck stacking almost always results in some loss of recovery. Agglomeration with cement may not be necessary, but conveyor stacking is usually employed.

Barneys Canyon (Utah, United States) uses belt agglomeration (mixing and consolidation of fines as it drops from conveyor belts) followed by conveyor stacking. The La Quinua operation at Yanachocha, Peru, employs belt agglomeration followed by truck stacking.

Silver-Rich Deposits

Nevada deposits contain varying amounts of silver, and the resulting bullion may assay anywhere from 95% gold and 5% silver to 99% silver and 1% gold. Mexico also has multiple silver-rich deposits, like the Dolores and Ocampo projects. Silver leaches and behaves chemically in the same way as gold, although usually the percentage of silver recovery is significantly less than that of gold. Examples of nearly pure silver heap leaches are Coeur Rochester and Candelaria mines in Nevada, and Comco in Bolivia.

GEOLOGY OF COPPER DEPOSITS

Copper occurs in three basic assemblages: oxides, secondary sulfides, and primary sulfides. Porphyry deposits produce the bulk of the world's copper, and if these outcrop, they usually contain all three mineral assemblages (Bartlett 1998). The secondary sulfides are often the highest grade because of the oxidized cap leaching naturally and depositing copper in the secondary sulfide zone. If acidic conditions are not sufficient, the copper is not transported and an oxide deposit is formed.

Normally, there is no distinct zoning and there is frequently a transition zone from oxides to sulfides. The varying chemistry of copper makes the chemistry of copper heap leaching more difficult and complex than that of gold or silver. Adding to the chemical issues, feldspar minerals in copper ores often break down over several weeks of leaching to form clays, which reduce the permeability of the heap. Dump leaches—where uncrushed low-grade mine waste is leached with dilute acid—are fairly common at copper mines because extra copper can be recovered at a very low cost.

Oxide Copper Ores

There is a wide range of copper oxide minerals. With the range of minerals comes a range of metallurgical responses. Most copper oxide minerals leach relatively quickly although some, especially the silicates (like chrysocolla), may be only partially soluble because of the formation of impermeable coatings.

Sulfide Copper Ores

There are primary sulfides, like enargite and chalcopyrite, and secondary sulfides, like chalcocite, bornite and covellite. The secondary sulfides usually leach faster and more completely than the primary sulfides. All sulfides need an oxidant present for dissolution, and the slow introduction of oxygen or oxidizing chemicals can result in a significant lengthening of leach times over what was predicted by laboratory tests. Within the heap, oxygen is usually converted by bacteria into oxidized (ferric) iron, and it is this ferric iron that reacts to oxidize the sulfides, so the heaps must be built in such a way as to encourage the growth of bacteria.

GEOLOGY OF URANIUM DEPOSITS

Sedimentary Uranium Deposits

Oxidized uranium is readily soluble and may be transported from granites or tuffs by groundwater to another location where it is concentrated. The uranium may precipitate in an area for a number of reasons, including lower temperatures and pressures, reduction, ion exchange, neutralization, and chemical replacement. Generally, for ore bodies, the uranium is deposited by reduction from the soluble, hexavalent uranium dioxide to an insoluble tetravalent state. The classic uranium deposits of the western United States are "roll front" deposits in which uranium is dispersed from sandstone and reprecipitated in a continually moving zone.

Granitic Uranium Deposits

Uranium may also be concentrated as large bodies of granite solidify. The initially low concentrations of uranium are continually forced into the remaining solution in the granite until small concentrated amounts of uranium-bearing solutions are left. The fluid can get into fissures in the surrounding rock and form veins of ore.

GEOLOGY OF NICKEL DEPOSITS

Nickel laterite deposits are created when a parent rock is weathered by high rainfall and elevated temperature, typically in a tropical or subtropical climate. The minerals that dissolve easily are removed by the rainfall, leaving behind higher concentrations of the less-soluble elements like iron and aluminum. Nickel is somewhat mobile and concentrates near the base of the leached zone.

CHEMISTRY OF GOLD DEPOSITS

The chemistry of leaching gold and silver from their ores is essentially the same for both metals. A dilute alkaline solution of sodium cyanide (NaCN) dissolves these metals without dissolving many other ore components (copper, zinc, mercury, and iron are the most common soluble impurities). Solution is maintained at an alkaline pH of 9.5–11. Below a pH of 9.5, cyanide consumption is high. Above a pH of 11, metal recovery decreases. A pH above 11 can also result in dissolving silica, which can cause problems with scale control and blinding of carbon.

Many heap-leachable ores contain both gold and silver. Deposits in western Africa and Australia tend to be very low in silver, while those in Nevada and Mexico are highly variable, ranging from pure gold to pure silver. Silver is usually not as reactive with cyanide as gold. This is because gold almost always occurs as metal, whereas silver may be present in the ore in many different chemical forms, some of which are not cyanide soluble. The consequence is that heap recoveries for gold typically range from 50% to 90% while silver typically ranges from 25% to 60%.

Recovery of gold from leach solutions is usually accomplished by adsorption of the gold cyanide complex onto granular activated carbon by pumping the solution through columns filled with the carbon. The leach solutions are recycled to the heap. Gold-rich solution from the heap usually contains only a few parts per million gold, whereas the carbon loads up to about 5,000 ppm. Gold is removed from the carbon by contacting it with a hot chemical solution that is then sent to an electrolytic cell for production of gold metal. This is usually melted on-site to produce a doré bar for sale to refineries (Marsden and House 2006).

The recovery of silver, and the recovery of both gold and silver from solutions high in silver, is usually accomplished using the Merrill–Crowe process. In this process the metal-rich solution from the heap is prepared by filtration and then vacuum treated to remove oxygen from solution. Zinc dust is added to precipitate the metals, and the precipitate is collected on filters. This precipitate is melted on-site to produce a gold/silver doré bar.

The level of cyanide in the heap leach on-flow solution typically ranges from 100 to 600 ppm NaCN, although some ores may require more than 1,000 ppm. Cyanide consumption via complexation, volatilization, natural oxidation, or oxidation by ore components typically ranges from 0.1 to 1.0 kg/t of ore. Cement and/or lime consumption ranges from 0.5 to 70 kg/t of ore. Several operations use cement for alkalinity control (instead of or in addition to lime) as well as for agglomeration.

Other leaching agents—thiosulfate, thiourea, hypochlorite, bromine—have been experimented with as an alternative to cyanide, but cyanide is by far the most effective and the most environmentally friendly leaching agent.

CHEMISTRY OF COPPER DEPOSITS

Copper ores are nearly always leached with a dilute sulfuric acid solution. Copper in heap pregnant solution is typically several grams per liter, whereas for gold/silver ores the solutions contain only a few parts per million of metals. With copper oxides it is not uncommon for the solution availability to the metals in the ore to limit the leach rate of copper. If a slow leaching rate is simply due to the avpplication rate, this can be remedied, but it is often due to the capillary action in the ore

particles, and this is difficult to speed up. Wetting agents have been used to overcome this, with limited success.

Depending on the ore, the natural production of sulfuric acid can be sufficient for the leaching process. The pH of the leach solution needs to be in the range of 2.0 to 2.8 or lower to prevent hydrolysis of the ferric ions. If other oxidizing sulfides (such as pyrite) are present, the acidity can drop to the point where gangue minerals react quickly to form clays, which plug the heap.

Recovery of copper from leach solutions is usually accomplished by processing the copper-rich solution through a solvent extraction plant where copper is extracted into an organic liquid and then back-extracted into concentrated acid for concentration and purification of the copper solution. The barren solution is recycled to the heap. The concentrated solution is sent to electrolytic cells for production of metallic copper. Copper cathodes produced from this process are normally of marketable purity and are sold directly to copper end users (Jergensen 1999).

CHEMISTRY OF URANIUM DEPOSITS

Uranium ores are typically leached with dilute sulfuric acid, often naturally generated, or ammonium carbonate. The tetravalent uranium in oxide minerals needs to be oxidized to the hexavalent state. This is normally done in an acid heap by natural bacterial oxidation, but use of chemical oxidants may be employed. For an alkaline heap, hydrogen peroxide, H_2O_2, is usually employed as an oxidant. The roll of the ferric ion in the oxidation in an acid uranium heap is the same as that in copper heaps. As a result of similar chemical processes, acidic uranium and copper heaps have the same issues to overcome (Merritt 1971).

If lime is present in the host rock, this can cause permeability issues as the leach solution dissolves and reprecipitates the calcium as gypsum. It is common to begin the leach process at a low sulfuric acid concentration and increase it after the gypsum has had a chance to reprecipitate throughout the heap instead of in a single layer, as would occur if higher initial acid concentrations were used. If the lime in the host rock causes excessive acid consumption, alkaline ammonium carbonate $[(NH_4)_2CO_3]$ leaching can be used.

Ammonium carbonate leaching is selective to uranium and as a result the uranium bearing minerals need to be exposed to wetting, possibly requiring finer crushing.

Recovery of uranium from both acid and alkaline leaching is typically accomplished by upgrading solutions with solvent extraction or ion exchange resins followed by a caustic treatment to precipitate "yellow cake" uranium, which can be refined to uranium dioxide (UO_2).

CHEMISTRY OF NICKEL DEPOSITS

Heap leaching of nickel has been proposed but rarely applied. Either hydrochloric (HCl) or sulfuric (H_2SO_4) acid may be used as lixiviants. Recovery of nickel from recycled heap solution requires expensive neutralization of the acid, and this is a barrier to application of the process.

LABORATORY WORK

As with any processing method, it is very important to base the design on the results of a comprehensive program of laboratory testing. For a proper heap leach laboratory program to be developed, it helps to know early on, preferably in the exploration stage, that heap leaching may be an option. To conduct representative column tests, the samples need to be coarse rocks that cannot be produced from reverse circulation drilling. Either large-diameter core diamond drilling (100–200 mm) or bulk sampling (tons) is required. It is common practice to drive a drift into the ore body to take a sample, but as the resource gets larger, this sample becomes less representative. The initial ore samples are seldom representative of the entire ore body; therefore, laboratory tests, including column leach tests, should be continued on a regular basis during mining.

After heap leaching is selected, there is a range of variables that need to be tested. These include crush size, heap stability, permeability versus heap height, solution application rate, reagent strength and consumption, the need for agglomeration, and the type (usually portland cement for gold/silver heaps) and the amount of agglomerating agent required, leach time, and percent recovery.

Heap leaching has inherent risks that can be largely eliminated if the design and operating practices follow the results of the initial and ongoing laboratory testing. The risks result from the nature of the operation. The results of the process are usually not known for several weeks or even months after the ore is stacked, and at this point it is not economical to reprocess the ore. Mistakes made in the initial plant design or incorrect operating practices (e.g., not crushing finely enough or not agglomerating or stacking properly) can result in cash-flow issues that might persist for up to a year after the problem is solved.

An on-site laboratory is an important part of the infrastructure at a heap leach operation. It should include an analytical section (for ore control) and a metallurgical testing section that regularly runs column leach tests on production samples. In addition to the standard production column leach tests, periodic test programs should be run to check the effects of chemistry, crush size, and agglomeration. For a small operation processing fewer than 5,000 t/d of ore, staffing should involve two or three technicians for sample preparation and assaying, and one metallurgist to conduct process tests. Large operations may have a laboratory staff of 10 to 15 people and around-the-clock operation.

HEAP PERMEABILITY AND FLOW EFFICIENCY

The key element in a successful heap leach project is a heap with high and uniform permeability. In any heap there are three zones of different flow regimes:

1. Coarse channels, which allow direct short-circuiting of solution from top to bottom
2. Highly permeable zones, in which solution is efficient at contacting the rock and washing the values downward in "plug flow"
3. Zones of low permeability where high-grade solution or unleached ore may be trapped

Efficiency of Solution Displacement

If the heap was "ideal" (i.e., moving in true plug flow), then when one displacement volume of solution was placed on top of the heap, it would fully replace the solution in the heap. This would be 100% wash efficiency. In practice, the "best" heap leaches exhibit a wash efficiency of about 70%. At 70% per displacement, three displacement washes are required to achieve a recovery of 95% of the dissolved metals. A fourth "displacement" is required, initially, to saturate the ore. Since a typical heap contains 20% moisture, 95% recovery (of the

dissolved value content) requires that 0.8 t of solution must be applied to each ton of ore. In gold heaps, typical practice is to apply 1.3 t of solution per ton of ore during a 70-day primary leach cycle. This suggests two things: (1) most heap leach operations are able to maintain reasonably good permeability characteristics, yielding at least 50% wash efficiency; and (2) a high percentage of the recoverables is solubilized early in the 70-day leach cycle.

Drainage Base

A drainage base of crushed rock and embedded perforated pipes should be installed above the plastic leach pad and below the ore heap. The importance of this drainage base cannot be overemphasized. Solution should percolate vertically downward through the entire heap and then enter a solution removal system with zero hydraulic head. If the drainage base cannot take the entire flow, solution builds up in a stagnant zone within the heap and leaching within this stagnant zone can be very slow.

To put this in context, a "typical" heap might run 500 m upslope from the solution collection point. All of the on-flow solution in a 1 × 500-m strip must flow out at the downslope edge of the heap through the drainage base, which is typically 0.65 m thick. The design of the horizontal percolation rate through the drainage base is therefore nearly 800 times the design rate of the heap itself. This is not a difficult engineering accomplishment since flow is carried in pipes within the base.

At one Australian copper heap leach operation, three adjacent leach panels were built. The two flanking panels had a good installed drainage base, but the center panel did not. Recovery in the center panel was depressed 20% by the poor drainage base. A similar effect has been seen but not quantified at some gold heap leach operations.

Recovery Delay in Multilift Heaps

As subsequent lifts are stacked, the lower lifts are compressed and the percentage of low permeability zones increases. The first solution exiting an upper lift may have a values concentration of up to three times that of the ore. If impermeable zones have developed in a lower lift, high-grade solution may be trapped, causing a severe reduction in recovery rate and possibly in overall recovery percentage. The highest heap leaches currently in operation are 200 m high, with about 10 lifts of ore. Hard ore, crushed or ROM, can withstand the resulting pressure without significant permeability loss. Many softer ores can be agglomerated with enough cement so that they can perform under a load of 30 m; some agglomerated ores perform satisfactorily to 100 m. These properties can be properly evaluated in advance in laboratory column tests, which are run under design loads.

The delay in recovery as lifts are added to the heap is partly a function of the impermeability of the lower lifts and partly a function of the wash efficiency discussed earlier. The net effect is that average recovery is delayed as the heaps get higher and overall pregnant solution grade decreases (requiring more solution processing capacity).

Intermediate Liners

If the permeability of lower lifts becomes a serious problem, it is possible to install intermediate liners, though this is not recommended. Two problems occur with installing an intermediate liner: (1) the heap below the liner is compressed as the upper lift is placed, resulting in differential settlement and possible tearing of the liner; and (2) the ore below the liner cannot be washed with water, which is sometimes required as part of final heap closure.

SOLUTION APPLICATION RATE AND LEACH TIME

With regard to sprinkling rate, the timing for metals recovery is a function of the following factors:

- The rate at which the metal dissolves. Coarse particles dissolve very slowly, and may not fully dissolve for several months in a heap leach environment.
- The percentage of the ore minerals that exist as free or exposed particles
- The rate of diffusion of the solution into rock fractures, and dissolved metal back out of the rock fractures. Where ore minerals occur on tight fractures or in unfractured rock, the rock must be crushed into fine particles to achieve target rates and levels of recovery.
- The effect of chemical reactions within the heap, or within rock particles, which consume the reagents needed for leaching
- The rate of washing the values off the rock surfaces and out of the lift of ore under leach. This is a complex issue that depends on the overall permeability of the lift and the local permeability variations due to segregation and compaction as the lift is being constructed.

These factors cause wide theoretical differences in the response of various ores to leaching. However, in practice, most heap leach operations apply solution to crushed-ore heaps within a fairly narrow range of flows. The typical range is from 8 to 12 L/h/m^2, though some heaps range far above or below these rates (Van Zyl et al. 1988). Some operations start at higher solution application rates to saturate the ore and decrease the application rate with time. Other operations will start solution application very slowly until the heap is thoroughly wetted, to prevent the formation of preferential channels of water flow and then increase the application rate.

Laboratory columns always respond much faster than field heaps. Generally the cause of this problem is reduced reagent-to-values contact in a production heap. Two major reasons for the reduced contact are: (1) the ore is placed in the laboratory column much more uniformly so that percolation is more effective; and (2) the solution-to-ore ratio (tons of solution per ton of ore in a given time frame) is generally higher in laboratory columns than in field heaps. Both small- (150-mm) and large-diameter (1,000-mm) column tests tend to leach similarly. For some field heaps, notably where the ore is fine crushed and the ore leaches quickly, the solution/ore ratio is a more important factor than overall leach time. However, for the majority of heap leaches, time seems as important as specific application rate. The general target for the solution/ore ratio is between 1:1 and 1.5:1.

For ores with very slow leaching characteristics, an intermediate pond and a recycle stream may be added to the circuit, so that each ton of ore sees 2 t of leach solution during an extended leach period. The process plant treats only the final pregnant stream—1 t of solution/t of ore.

The use of multiple cycles is good operating practice for single-lift heaps of high-grade ore. However, for multilift heaps this is not the case. Heap modeling indicates that after the heap attains a height of three lifts, the intermediate solution contains almost as much metal as the pregnant solution. Recycling results in a significant buildup of dissolved values



Figure 11.3-2 Heap leach installation at Brewery Creek. The open-pit mine is shown in the background. In the center of the photograph is an operating heap leach completely covered with snow.

within the heap, causing a slight overall recovery loss and a cash-flow delay. For multilift heaps, it is often possible to justify an increase in the size of the recovery plant so that only fully barren solution returns to the heap.

It is extremely important to design a heap leach system so that the ore can be leached for a very long time. Unlike an agitated leach plant where the ore can be ground to a fine powder and intensively mixed, heap leaching is not a very energy-intensive process. After a heap is built, one of the most significant variables the operator can employ to solve design or production problems is the leach time. Some operations use on-off leach pads to achieve rapid first-stage recovery and then transfer the ore to long-term heaps to complete the process (Kappes 2002).

SOLUTION APPLICATION

The primary goal in the irrigation of a heap is to apply the solution as uniformly as possible. Solution distribution to a heap is done by flood, spray, or drip irrigation. The choice of irrigation can vary greatly depending on specific heap conditions. In recent years, flood irrigation has become a rarity. Spray and drip irrigation are widely used and frequently both are used on the same heap, depending on the season. The following equipment has become standard in heap leaching.

Drip Irrigation

Drip emitters, which issue drops of water from holes every 0.5–1.5 m across the heap surface, are very common. Drip emitters are small in-line devices spaced along a tube that distributes the solution evenly by forcing the water through a complicated path at each emitter, creating an equal pressure loss for the first and last emitter in a line. Drip emitters are easy to maintain and minimize evaporation and cyanide loss. In very cold climates it is possible to bury the emitters to prevent freezing, as shown at Brewery Creek (Alaska, United States) in Figure 11.3-2. In the winter months, the solution flow is piped directly to the process facility on the lower right. The barren leach solution is heated prior to application on the heap and solution pipes are heat traced. The ponds in the foreground are frozen over but, if necessary, excessive flow may be directed into them. Drip emitters can be advantageous for some copper and uranium heaps because they conserve heat that may be required for biological activity. The main drawback to drip emitters is that they do not provide continuous drip coverage. Thus the top 1 m of the heap may not be leached very well until it is covered with the next lift. Other drawbacks are that emitters, due to their small channels, require intensive (and expensive) use of antiscalant, and the use of in-line filters.

Wobbler Sprinklers

Wobbler sprinklers are used at a large number of operations. Their main advantages are that they issue coarse droplets, which control but do not eliminate evaporation, and they deliver a uniform solution distribution pattern that ensures uniform leaching of the heap surface. The coarse droplet size has another advantage in gold heaps. Cyanide is readily oxidized by air and sunlight, and the wobbler-type sprinkler minimizes this loss (but not as well as drip systems). Wobblers are typically placed in a 6×6-m pattern across the heap surface. A disadvantage of all sprinklers is that they require continual servicing, and personnel spend extended periods working in a "rainstorm." Occasional skin contact with cyanide solution does not pose a health problem, but an environment that encourages repeated skin/solution contact is not recommended. Sprinkler maintenance personnel, especially on acid copper heap leaches, wear full rain gear to eliminate any exposure problem, but the working environment (especially in cold weather) is not as pleasant as with drip emitters. Because of the impact of the water on the surface of a heap, wobblers, or any sprinkler system, can lead to the breakdown of delicate agglomerates and migration of fines into a heap. In a tropical climate it may be necessary to use sprinklers because of evaporation issues, but they can cause permeability problems. The influence of the sprinklers on the breakdown of agglomerates and migration of fines can be minimized by placing screen material over the heap to dissipate the impact energy of the water droplets.

Reciprocating Sprinklers

Reciprocating sprinklers shoot a stream, typically 5–8 m long, of mixed coarse and fine droplets. They are not considered ideal for heaps but often find application for sprinkling side slopes since they can be mounted on the top edge to cover the entire slope. If emitters and wobblers are used on side slopes, they must be installed on the slope, which is a difficult and sometimes dangerous place for service personnel to operate from.

High-Rate Evaporative Sprinklers

High-rate evaporative sprinklers typically operate at high-pressures with an orifice designed to produce fine droplets and shoot them in a high trajectory. Evaporative blowers using compressed air to atomize and launch the droplets can also be used. The drifting of the fine particles with wind can cause a concern with chemicals entering the surrounding environment. This type of equipment is not normally used at heap leach operations, but it will become more common as more heaps enter the closure mode where rapid evaporation is needed.

Regardless of the systems used for solution application and management, capital and operating costs for solution handling are usually small.

WATER BALANCE

Since many heap leach operations occur in desert areas where water is scarce, and others occur in environmentally sensitive areas where water discharge is not acceptable, the balance between water collection and evaporation is important. Fortunately, by adjusting the method and scheduling of solution application, it is usually possible to meet the local conditions.

The evaporation of water, regardless of its mechanism, requires a heat input of 580 cal/g of water evaporated. A heap leach gets this heat input from three sources: direct solar heating on heap and water surfaces, latent heat in the shroud of air within the "sprinkler envelope," and latent heat in the air that is pulled through the heap by convection.

The average 24-hour incident solar radiation on a flat horizontal surface ranges from 3,000 kcal/m^2/d in the central United States to about 7,600 kcal/m^2/d in the equatorial desert, which could theoretically evaporate 5–12 L of solution/d/m^2. With a typical heap application rate of 10 L/m^2/h, incident solar radiation could account for an evaporation rate of 2%–5% of applied solution when using sprinklers. Evaporation would be somewhat less when using drip irrigation (1%–4%) because some of the solar energy is reradiated from dry areas on top of the heap. This same heat input would result in pond evaporation of 5–13 mm/d.

Overall evaporative losses include the sprinkler losses, convective loss from air flowing through the heap, and losses due to heating/evaporation from ponds and from other areas not sprinkled. These have been determined at several Nevada operations to be up to 20% of the total solution pumped in summer months, but averaging 10% annually. Thus, direct sprinkler loss accounts for about 60% of the total evaporation. Use of drip irrigation can reduce but not eliminate evaporative loss.

In tropical climates, noticeable losses occur even during the rainy season. On several tropical heap leach projects where rainfall is seasonal and up to 2.5 m/yr, the overall annual evaporative loss from all sources, when using wobbler-type sprinklers operated 24 h/d, is about 7% of the solution pumped. A typical heap application rate is 10 L/m^2/h, or 88 m/yr/m^2. Thus, evaporative loss of 7% is equal to 6.2 m/yr/m^2 on the areas actually being sprinkled. If the heap and pond systems are properly designed, the active leaching area can be up to 40% of the total area collecting rainfall; it is therefore possible to operate in water balance when rainfall is 2.5 m/yr. For these operations, very large solution surge ponds are required.

Where rainfall is high and evaporation rate is low, some operations cover the side slopes with plastic to minimize rain collection. Others have tried to cover the entire heap during the rainy season, but this has not worked very well because of the mechanical difficulties in moving the covers.

In western Africa and Central America, it is acceptable practice to treat and discharge excess solution during the rainy season. Typically, excess process solution is routed through a series of ponds where cyanide is destroyed using calcium hypochlorite [$Ca(ClO)_2$] or H_2O_2, followed by adjustment of the pH to near neutral. The INCO SO_2 system, using copper-catalyzed hyposulfite to destroy cyanide, is also employed for this purpose. Cyanide-free solution is further treated in controlled wetlands (swamps) to remove heavy metals prior to discharge.

The worst water balance situation occurs in cool, damp climates such as in high-altitude operations. In such climates, rainfall and snowfall may be significant and evaporation is minimal. Generally such heaps can stay in water balance with an aggressive program of summer sprinkling. Arctic heap leaches have been able to stay in water balance because precipitation is lower than the total water requirement needed to saturate the ore (Kappes 2002).

LEACH PADS

The leach pad below the heap is a significant element of a heap leach design. The ideal location for the heap is a nearly flat (1% slope), featureless ground surface. Usually some earthwork is required to modify contours, but it is not necessary to eliminate all undulations. It is only necessary that all the solution will flow across the surface toward the collection ditches on the base or the sides of the heap. Where the slope exceeds 3%, the front edge of the heap (30–50 m) should be graded flat to provide a buttress to prevent heap failure. In the western United States, where the water table is often far below the surface, the current practice is to construct the leach pad of 1.5-mm thickness high-density polyethylene or very-low-density polyethylene on a 30-cm-thick layer of compacted clay. A leak detection/water seepage system of pipes is installed below the liner.

HEAP HEIGHT

As discussed previously, for multilift heaps, there is a delay in the recovery and grade. This is not only true for multilifts but also for single lifts as the lift gets taller. A lower recovery grade means that either a larger recovery plant needs to be installed or a lower cash flow will have to be tolerated. The delay in cash flow caused by delayed recovery also needs to be taken into account for additional lifts or height. Depending on the permeability of the material and lift height, the delay per lift can range from 3 to 30 days. It is common to see a delay of 7 days per lift. As a single lift gets taller, there is the added concern of particle size distribution. As the ore is stacked, it rolls down the side slope of the heap. This cascading action naturally segregates the fine particles near the top of the lift, and the large particles near the bottom. The taller the lift the more stringent the controls on this problem need to be. On the other hand, the advantage of taller lifts is that there is less high-grade solution flowing through already-leached ore beneath it. The larger the quantity of high-grade solution flowing through leached ore, the more opportunity there is to lose values. For copper heaps there are additional factors that must be considered when choosing lift height. Copper oxide heaps are frequently limited in height to maintain the pH in the range that keeps the copper soluble. Copper sulfide heaps will create acid within the heap that may cause a pH problem, and these can also easily become oxygen deficient. Because of these issues and permeability problems, copper ores may be leached in "thin layer" heaps where the ore is stacked only 3–4 m high and leached for a relatively short period (weeks instead of months).

With harder ore and quality agglomeration, permeability is not always the driving factor in the ultimate heap height. It may be good operating practice to reduce the heap height in order to maintain a high, consistent solution grade, which is important for planning purposes, especially at very large operations.

MINING, ORE PREPARATION, AND STACKING

Mining of ore for heap leaching employs the same techniques and equipment as mining of ore to feed any other process



Figure 11.3-3 Agglomerating drum and conveyor stacking system with 6-m-high heap

method. Where uncrushed (ROM) ore is placed on the leach pad, ore may be blasted very heavily in order to reduce rock size and improve gold recovery. In high-rainfall environments when processing clay-rich material, it is very important to practice a mining routine that minimizes the amount of rainfall absorbed by the ore.

Ore preparation varies widely. ROM ore may be hauled from the mine and dumped directly onto the heap. At the other extreme from ROM leaches, some heap leaches crush the ore and dry grind it to more than 50% passing 150 μm (100 mesh) and agglomerate the fines (AusIMM 1991). At times, the high-grade ore will be ground and subsequently reblended with the coarse low-grade ore going to the heap. This process is called "pulp agglomeration" and is practiced at Ruby Hill in Nevada.

Ores high in clay (such as saprolites) are typically processed by two stages of crushing using toothed roll crushers, then agglomerated in drums and stacked using a conveyor stacking system. Many ores are crushed and then either truck-stacked or conveyor-stacked without agglomeration. For these harder ores, crushing is usually achieved by a jaw crusher, followed by one or two stages of cone crushing.

Agglomeration

The term *agglomeration* means different things to different operators.

The simplest application of agglomeration is practiced where the ore is hard but contains a large percentage of fines. Agglomeration means simply wetting the ore with water so the fines stick to the coarse particles and do not segregate as the heap is built.

A more involved form of agglomeration is practiced where the ore contains amounts of clay or fines that may begin to plug a heap of untreated ore. Belt agglomeration may be employed. In this technique, cement and water are mixed with the ore at a series of conveyor drop points, and the mixture tends to coat the larger rock particles. The primary goal is stabilization by mixing and contact. A typical conveyor stacking system involves 10 or more drop points, so belt agglomeration may occur as a normal part of the process.

Where ores are nearly pure clays, such as the laterite/saprolite ores in tropical climates, drum agglomeration is usually employed. Figure 11.3-3 shows a typical agglomerating drum. The ore is first crushed finely enough (typically 25–75 mm) to form particles that can be a stable nucleus for round pellets. For gold and silver ores, cement and water are then added and the ore is sent through a rolling drum. The fines and the cement form a high-cement shell around the larger particles, and the rolling action of the drum compacts and strengthens the shell. Copper ores cannot use cement because the sulfuric acid will break it down. Concentrated sulfuric acid is used as a binding agent instead, although the binding effect of sulfuric acid is not very good. Where copper ores are very high in clay or in minerals that decompose in the heap to clay, heap leaching may not be very effective. Drum size and throughput are a function of several factors, but typically a 3.7-m-diameter 10-m-long drum can process 750 t/h. A 2.5-m-diameter drum can process 250 t/h. At the Tarkwa mine in Ghana, two 3.7-m drums were installed to process up to 20,000 t/d of ore. For multilift gold heaps, it is often necessary to use a higher cement addition on the lower lifts, and this is usually determined by laboratory tests in which the ore is leached under the full heap load. Maintaining permeability of the lower lifts is extremely critical to the success of a multilift heap (Kappes 2002).

Truck Stacking

Where rock is hard and contains very little clay, it is possible to maintain high permeability even when ore is crushed and dumped with trucks (Figure 11.3-4). Truck dumping causes segregation, of the ore as it cascades down the slope. To control the degree of this segregation, the ore may be partially agglomerated (wetted to cause the fines to stick to the coarse material) prior to placing in the trucks.

Truck dumping can also result in compaction of roadways on top of the heap. Several studies have indicated large trucks noticeably compact ore to a depth of 2 m. To mitigate this problem, most operations rip the ore after stacking but prior to leaching. The number of ripper passes is important; usually



Figure 11.3-4 A truck is dumping an upper lift onto a lift that is actively under leach. The road formed by haul truck traffic on the heap can be seen on the right.

it is four passes in a crisscross pattern. Some operations practice building an elevated truck roadway that is later bulldozed away. However, this requires substantial bulldozer traffic on the heap surface, which can lead to permeability problems.

Stacking the ore with trucks can result in the tie-up of a large tonnage of ore below the truck roadways. This is a bigger problem for small operations than for large ones, because the roadway width is usually the same regardless of the daily production rate. For a heap leach of 5,000 t/d, the roadways on the heap can tie up one month's ore production. A conveyor system that stacks ore from the base of the lift can reduce the unleached inventory to a few days' production. Because of this inventory reduction, at smaller operations where the ore is crushed, it is usually less capital intensive to install a conveyor stacking system. Conversely, for operations of 100,000 t/d or more, truck stacking is more flexible and may be less capital intensive than a conveyor system.

Conveyor Stacking

Two major types of conveyor stacking systems are used on heaps: radial stacking systems and spreader conveyor systems.

Radial conveyor stacking systems commonly include the following equipment:

- One or more long (overland) conveyors that transport the ore from the preparation plant to the heap. Typically these consist of conveyors up to 150 m or longer.
- A series of 8–15 "grasshopper" conveyors to transport the ore across the active heap area. Grasshoppers are inclined conveyors 20–50 m long, with a tail skid and a set of wheels located near the balance point.
- A transverse conveyor to feed the stacker-follower conveyor
- A stacker-follower conveyor, typically a horizontal mobile conveyor that retracts behind the stacker
- A radial stacker 25–50 m long, with a retractable 5–10-m conveyor at its tip called a stinger. Wheels, discharge angle, and stinger position are all motorized and are moved continuously by the operator as the heap is built.

Radial stackers are usually operated from the base of the lift but may be located on top of the lift, dumping over the edge. Figure 11.3-5 shows a heap that is not only stacked with conveyors on top of the heap but is also stacked in a novel manner: a spiral of one continually climbing lift instead of multiple individual lifts. Inclined conveyors can be installed up the sides of the lower lifts, and the stacking system can be used to build multilift heaps. Stackers for this purpose should have very-low-ground-pressure tires and powerful wheel drive motors to cope with soft spots in the heap surface.

Radial stacking systems can be used for heaps processing up to 50,000 t/d of ore, but beyond that, the size of the stacker (and the bearing pressure that is exerted by the wheels) becomes prohibitive. For operations stacking very high tonnages, large stackers can be mounted on caterpillar tracks to reduce ground pressure.

Spreader conveyors can be used on very-high-tonnage heaps. On on-off (dynamic) heaps, spreader conveyors span the entire width of the heap and continually travel back and forth, distributing the ore on the heap. A bucket-wheel excavator can be installed to remove the ore after leaching is complete. Several recent dynamic heap installations have been built in the form of a ring or oval with a moving "slot" from which the ore has been removed. Ore is removed on the advancing face of a slot and new ore is placed on the trailing edge. The "slot" can be seen in Figure 11.3-6 between the new ore stacked by a spreader conveyor on the right and leached ore removed by a bucket-wheel excavator on the left. The slot between old and new material with a short distance into each face of the heap for the two conveyors to work is the only area of the heap that cannot be under leach.

Mobile spreader conveyors up to 400 m long, mounted on multiple caterpillar tracks so they can travel in any direction, are used on multilift, very-high-tonnage heaps, with notable examples being in the Chilean copper industry.



Figure 11.3-5 Valley-fill heap leach at Ocampo, Mexico

SOLUTION COLLECTION

After the values have been dissolved, it is necessary to collect the pregnant solution. A wide variety of arrangements are used to collect the solution and direct it to the pregnant pond. On some valley-fill heap leaches, all of the solution is drained to a single point on the pad and directly into the pregnant liquor pond. As mentioned earlier, a portion of the solution may even be stored inside the void spaces in the heap, thereby minimizing the need for a large pregnant pond. Pregnant solution is usually collected in lined ditches that run adjacent to the heap. More than one inlet to the ditch may be used if it is desired to observe the recovery of different areas on the heap. To further increase the ability to watch the recoveries of different parts of the heap, small ridges may run the length of the leach pad, perpendicular to the collection ditch, to separate the heap into "cells." It is common for a weir to be placed at the point where the pregnant solution exits from each cell, or from the entire pad, to measure the flow draining from the heap. With a heap designed with cells, it is possible to segregate solutions into an intermediate pond. Multiple ditches or pipes run parallel to the heap, one running to each pond. Figure 11.3-7 illustrates this process at the Tarkwa heap leach in Ghana. The three large gray pipes on the right distribute the barren and intermediate leach solutions to each cell of the heap while the black pipes collect the intermediate and pregnant heap runoff solutions. The box at the base of the cell is a solution control point to select which solution will be used for leaching and to which pond the runoff will go. There is also a weir in the distribution box for an instantaneous measurement of solution from the heap. The pad below the heap is commonly a single plastic liner above compacted clay, but the ditches and ponds (and flooded areas of valley-fill heaps) are usually constructed with a double plastic liner with intermediate leak detection.

METALLURGICAL BALANCE

If the values available are not well tracked, the amount of recovery will never be known. The following are some of the values that are important to obtain to be able to calculate a good metallurgical balance:

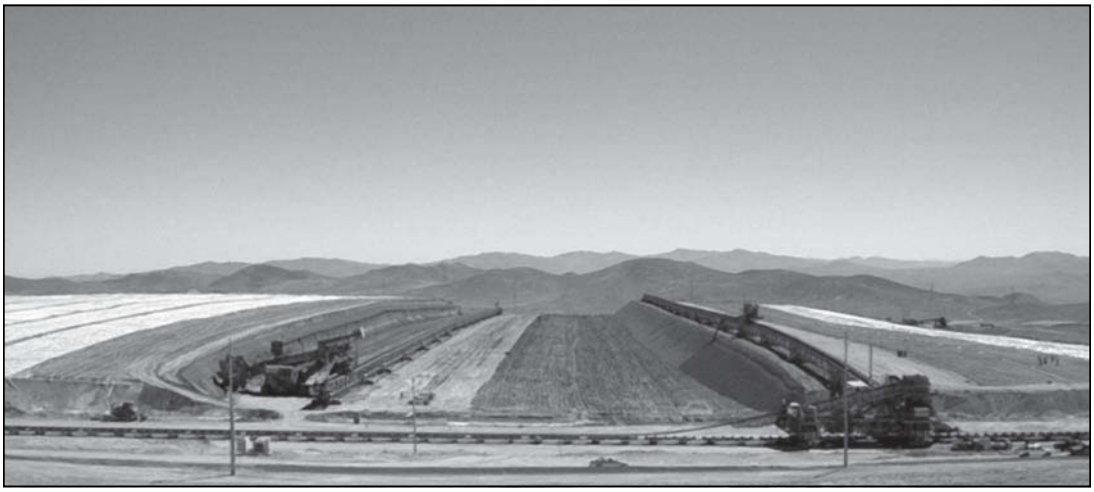
- Sampling of the ore to be stacked
- Knowing when, where, and how much ore is stacked
- Conducting quality column leach tests to monitor the continually changing ore, and modeling the test results to apply to the heap
- Sampling and flow measurement of the barren and pregnant solutions to and from the heap
- Good assay procedures employing fire assay of the ore samples for gold ores, and total and soluble copper assays for copper ores
- A good model that accounts for delays caused by adding lifts
- Patience

It is important to collect good grade, tonnage, and column test data from day one. It cannot be assumed that the entire ore body will act the same as the initial sample taken for design. To maintain a representative model, the recovery curves will change with the grade, ore type, variations of dilution from waste, and other factors that may affect the recovery. It is difficult to produce a stable and accurate metallurgical balance early in the life of a heap because of the slow reaction time. As a heap ages, the intricacies of how the permeability and leach rates of that particular heap will become easier to predict. A metallurgical balance is a continually changing tool for managing a heap and predicting the cash flow. A good model will keep bankers, stockholders, and your managers happy. A poor one can put a mine in dire straits when the predicted cash flow does not materialize.

DESIGN CONSIDERATIONS FOR RECLAMATION AND CLOSURE

After the heap leaching operation is completed, the facility must be closed in accordance with local environmental requirements. Closure activities are highly variable depending on the environmental sensitivity of the site and on the regulatory regime. In general, heaps are washed for a short period of time (commonly 3 years), during which time 1 t of wash water or recycled treated process solution per ton of ore is applied. Heaps are then capped, and ponds are filled and covered with suitable materials.

The easiest heaps to reclaim are single-lift heaps because the older heaps are abandoned early in the life of the operation and can be washed while production operations continue. In



Courtesy of FLSmidth RAHCO, Inc.
Figure 11.3-6 Slot between new ore (right) and leached ore (left)



Source: Kappes 2002.
Figure 11.3-7 Solution collection at the Tarkwa heap leach

valley-fill heap leaches, nearly all the ore ever placed on the pad is situated directly under active leach areas. Thus, washing of the entire heap must wait until operations are completed. Larger operations may have two or more valley-fill leach areas and can therefore appropriately schedule closure activities.

Environmental regulations usually applied in the United States call for reasonably complete washing of the heap to reduce pH, to remove cyanide, and to partially remove heavy metals. Cyanide is fairly easy to remove because it oxidizes naturally, but pH and heavy metals are more difficult to control. Regulators are recognizing that a better approach is to conduct a "limited" washing program and then to cap the heap with a clay cover and/or an "evapotranspiration" cover of breathable soil with an active growth of biomass. These covers are designed to prevent infiltration of water into the heap. After several years of active closure activities, the flow rate of the heap effluent decreases to a manageable level (or to zero in arid environments). After the flow rate has reached an acceptably low level, heap closure is accomplished by installing a facility for recycling collected effluent back to the heap. A relatively small "cash perpetuity bond" is maintained such that the interest on the bond covers the cost of maintaining and operating the intermittent pumping facility for as long as necessary.

Worldwide practices range from simple washing and abandonment to the more complex procedure described above. Environmental design is an industry unto itself, and the simplistic concepts discussed here may not be universally applicable. Heap closure needs to be addressed in the feasibility stage of the project.

TROUBLESHOOTING

As with most things in life, the best troubleshooting for a heap is to do it right the first time. One of the primary challenges in operating a heap is the delay in metallurgical response. With long leach times, the results of a change in operations will

not be seen for months. A common mistake people make is to watch a heap on a day-by-day basis, which is an insignificant amount of time. Arguably the most critical time in the construction of a multilift heap is the base lift of the heap, when operators are least familiar with the equipment and the characteristics of the ore. If the first lift is built poorly and has poor permeability or chemistry, this can have a lasting effect throughout the life of the heap.

Permeability

When permeability problems arise, there are few options to improve it. Often the first sign of low permeability is puddles of standing water on the surface of the heap, called ponding. Apart from ripping the top couple of meters, there are few viable options, although several have been tried. Some heaps are rehandled to improve permeability, which may be profitable on a higher-grade heap as pockets of ore will not have leached. Heaps have also been blasted to try to "fluff up" the ore, though with limited success. Allowing the water to pond might help to force the solution through. Efforts to improve the permeability of already-stacked ore are typically cost prohibitive.

Stability

Low permeability can also cause the ore to become saturated with water and lead to stability issues. The primary cause of stability issues in a heap is the complete saturation of a structurally significant portion of the heap. At the first signs of solution discharge from the side of the heap (more than 1 m above the base) or small blowouts, it is best to stop leaching that area immediately and assess the problem. The problem may be a general issue of poor permeability, or it may be that during stacking of an upper lift, the top surface of the lower lift was made impermeable. If this problem occurs, it usually implies that there will be an overall loss of recovery in the entire heap. To solve the immediate problem, the only practical solution is to reduce leaching near the heap edge, but it is possible to drill the area for installation of vertical drains. The problem indicates that the operation must do a better job of agglomerating new ore, or of ripping and then limiting traffic on the surface below a new lift.

Liner Leaks

Most pads have leak detection pipes below the pad, and leaks will report to the secondary containment sump. Some leakage through the primary liner is generally permitted, but excessive leakage will require abandonment of the heap. Several geophysical technologies have been developed to help locate a hole in a liner, but their accuracy is limited to a circle on the pad with a diameter equal to the vertical height of the heap. It is expensive to repair a liner under an operating heap, as the simple act of exposing it can complicate the issue by creating more holes. It is critically important to prevent liner leaks during construction.

Poor Recovery

A common problem that a heap leach will run into is that the ore takes longer to leach than predicted by the laboratory tests. To eliminate this risk, heap leach design must allow for total flexibility to extend leach times. Many heaps, especially in the copper industry, have achieved economically acceptable performance even though leach times were very long.

It is common practice to minimize the use of cyanide on a heap to save money. In reality this can lose money if done incorrectly. Initial laboratory tests can indicate starting concentrations and operators may periodically explore different cyanide levels to see the effects on recovery. The problem with this approach is that the results of any changes in a heap take a long time to notice. A better approach is to maintain an active production support laboratory and to periodically run a series of parallel column leach tests at different cyanide addition levels to find the best addition concentration. The cyanide concentration in a heap is controlled by the addition of cyanide to the leach solution, but the cyanide in the pregnant solution coming from the heap should also be monitored.

Acid Production/pH

For cyanide heap leaches, lime is usually used to maintain a pH above 10. For ores that generate acid slowly, limestone can be blended in with the ore as well. Caustic can be used but is normally not economic and can result in chemical reactions that plug the heap or are detrimental to the recovery process. If the pH of the pregnant solution flowing from the heap drops because of the oxidation of pyrite and other sulfides, there is generally no economic remedy. Such acid zones should be watched carefully as they can cause gold loaded in upper lifts to precipitate. That area of the heap may have to be abandoned. Future ore should be stacked with the correct amount of pH modifier.

For copper heaps, H_2SO_4 can be generated by the ore itself, but in most heaps, extra acid must be added. However, it may be needed in such quantities as to make the availability of this reagent the key economic issue in evaluating the project.

CAPITAL AND OPERATING COSTS

Heap leaching normally has significant capital cost advantages over agitation leaching. The capital costs for a heap leach can vary widely, from $2,500 to $8,000 per daily metric ton of ore treated depending on many variables including, but not limited to, process rate, location, infrastructure, and company policy. The capital costs of a typical agitated leach circuit per daily ton of ore treated is up to eight times that of a typical heap leach.

Operating costs for an agitated leach circuit may be several times that of a heap. In other cases where a large amount of cement is required for agglomeration or where the ore needs to be fine-crushed, the operating costs of agitation leaching are not necessarily higher than for heap leaching. Heap leach operating costs are not very sensitive to the size of the operation because as some things get less expensive (i.e., general and administrative expenses), other things get more expensive (recovery plant operation). This relationship can be seen in Table 11.3-1.

CONCLUSION

Although the concepts of heap leaching are simple, the practices have substantially evolved over the past 40 years. Early choices for pad materials, sprinkler systems, and stacker designs have been discarded under the pressure of operating experience and cost-reduction factors. Overall operating costs have continually declined as "superfluous" activities and controls have been eliminated.

In spite of the apparent simplicity of the heap leach process—or perhaps because of it—there were many failures in the early years. Now there is a large number of successful operations from which to draw the experience needed to optimize the process. Heap leaching is expected to maintain its

Table 11.3-1 Heap leach operating cost distribution

Operation	3,000 t/d	15,000 t/d	30,000 t/d
Mining, %	27.8	26.0	24.2
Crushing, primary, %	3.7	2.6	2.9
Crushing, second plus third stage, %	4.6	5.1	2.9
Crushing, fourth stage, %	7.4	10.4	11.4
Agglomeration/stacking, %	1.9	1.3	1.4
Leach operations (including sprinkler supplies), %	4.6	3.9	2.9
Recovery plant operations, %	13.9	16.9	20.0
General site maintenance, %	5.6	3.9	4.3
Cement for agglomeration (10 kg/t), %	9.2	13.0	14.3
Cyanide, lime, other reagents, %	2.8	3.9	4.3
Environmental reclamation/closure, %	4.6	6.5	7.1
General and administrative, support expenses, %	13.9	6.5	4.3
Total operating costs, %	100	100	100
Total operating costs, $	10.80	7.70	7.00

place as one of the principal tools for extracting gold, silver, and copper from their ores for both large and small deposits. The challenge for the future will be to remember and apply the experiences of the past.

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CHAPTER 11.4

Surface Techniques of Solution Mining

W. Joseph Schlitt

Most categories of solution mining represent alternatives to mechanical excavation of ore. For example, dredging is simply mechanical excavation from the bottom of a river or other body of water. Hydraulic mining uses a high-pressure jet of water to dislodge material from a working face and even transport it to the processing plant. In-situ mining extracts values chemically or by simple dissolution, leaving the host rock in place and essentially undisturbed. In contrast, surface (leaching) techniques are forms of ore processing not unlike milling and flotation. All rely on mechanical excavation of ore to provide the feed to be treated.

Most types of solution mining represent the primary method of production, including mine-to-leach operations. However, surface leaching can also provide a secondary or supplemental form of production, with leaching limited to sub-mill-grade mine waste. Usually such material is of too low a grade to justify the cost for some degree of crushing, even though crushing could improve recovery or at least the rate of recovery. Hence, leach material is typically truck-hauled in run-of-mine (ROM) condition and dumped in lifts on the leach pile. Supplemental leach operations are fairly common in the copper industry, where higher-grade ore is usually milled and floated. They also exist in the gold industry, where higher-grade ores are typically ground- and agitation-leached.

Cost accounting usually differs significantly for primary and secondary leach operations. In a primary mine-to-leach scenario, the leach operation carries the full cost of mining. This is not unlike typical milling and flotation operations, which also carry the full cost of mining. Conversely, secondary operations typically carry no mining costs, as the leach material must be mined anyway to expose mill-grade ore. In effect, secondary operation lowers the strip ratio and provides better utilization of the resource, as some production is derived from material that is otherwise wasted. However, some incremental costs may be assigned to secondary leaching. An example is the extra cost needed to construct one or more leach-lined dumps with profiles that optimize recovery, rather than simply dumping material at the shortest haul location.

Both primary and secondary operations must meet two general sets of criteria, one that focuses on the characteristics of the ore and one that involves the development of efficient operating practices.

Consideration must first be given to the characteristics of the deposit and the leach material itself. Parameters include location of the resource, extent of the mineral inventory, mineralogy and mode of occurrence of the metal values, and chemical and physical nature of the host rock. Nature must provide a sufficient tonnage of material with adequate recoverable metal values per ton; otherwise the leach project never reaches the point where operating practices are of any consequence. For a proposed leach operation, the same type of exploration and development program is required as for any other mining project. The difference is that metallurgical testing is directed toward leaching instead of ore comminution and beneficiation.

This chapter describes the establishment of leach operating practices for treating whole ore, as opposed to some type of concentrate. Coverage begins with a section on ore–lixiviant systems for the different commodities, as well as bioleaching and metallurgical testing. This is followed by a section on materials handling that includes mining, leach systems, ore preparation, and emplacement. The next section covers solution management, including environmental considerations. The chapter concludes with shorter sections on downstream operations and infrastructure requirements.

Throughout this chapter, the following terms are used:

- *Heap leaching* refers to a mine-to-leach operation involving crushing, possible pretreatment, and careful stacking to optimize metal recovery.
- *Dump leaching* refers to an operation in which low-grade ROM material is placed in mine dumps and leached to obtain additional metal values.

ORE–LIXIVIANT SYSTEMS

One of the first decision points in a proposed leach operation is to select the optimum lixiviant with which to treat the ore. Clearly, if the mineralization is too refractory, leaching is not viable and other process options need to be considered.

The two key requirements for any lixiviant are high selectivity and rapid leach kinetics. The kinetics should always be

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such that dissolution at the surface of the ore is the most rapid step, so that the overall leach rate is controlled by transport of reagents or products to or from the mineral surface. Other desirable characteristics for the lixiviant include low price, low toxicity, ease of handling, and capacity for regenerating or recycling. The lixiviant should also allow for easy concentration and recovery of the metal values.

In spite of many years of research on possible lixiviants, only a few are used widely in commercial operations. The principle reagents used to extract each major commodity are discussed in the following subsections.

Gold and Silver

Alkaline sodium cyanide solution is used almost exclusively to extract gold and silver during heap and dump leaching. Studies on the mechanism of dissolution indicate that two reactions are involved in gold dissolution (Dorey et al. 1988). Most of the gold dissolves according to the following reaction:

$$2Au + 4CN^- + O_2 + 2H_2O \rightarrow 2Au(CN)_2^- + H_2O_2 + 2OH^- \quad (11.4\text{-}1)$$

However, a small but still significant portion of the gold dissolves according to the Elsner reaction:

$$4Au + 8CN^- + O_2 + 2H_2O \rightarrow 4Au(CN)_2^- + 4OH^- \quad (11.4\text{-}2)$$

In both cases, the balancing cation is sodium (not shown, for simplicity), and oxygen is required. For any given ore, the gold dissolution rate depends on cyanide concentration, oxygen content, and solution alkalinity. The optimum pH is about 10.3 (Dorey et al. 1988), but many operators tend to run at somewhat higher values as a margin of safety. Because most ores are not so basic, alkaline reagent must also be added to the system to raise the pH. This is generally done by blending lime (CaO) with the ore, or adding milk of lime [$(Ca(OH)_2$] or caustic (NaOH) to the solution.

In theory, nothing should prevent dissolution of virtually all of the gold in a heap or dump. In actual operations, extraction seldom exceeds 85% and recoveries as low as 40% occur. The problem is caused by poor accessibility of the gold to the lixiviant. One cause for this is poor heap construction that leaves portions of the ore blinded off so that solutions do not wet the entire rock mass. Another cause is gold encapsulated in silica or other refractory minerals such as pyrite or arsenopyrite. The processing of these so-called refractory ores is an area of technology that is still developing but is often applied to concentrates rather than whole ore.

Silver, although commonly extracted along with gold, is generally not recovered as effectively. Its dissolution reactions with cyanide are analogous to those for gold, but a lesser chemical driving force is involved. In addition, silver is much less noble than gold and tends to associate with various refractory minerals in addition to pyrite and arsenopyrite.

In spite of its almost universal usage on low-grade material in heaps or dumps, cyanide has a number of inherent problems, including the following:

- High potential toxicity
- Environmental concerns such as aquifer contamination
- Relatively slow kinetics for dissolution of both gold and silver
- Generally poor extraction of silver
- Requirement for alkaline reagents when leaching acidic ores
- High reagent consumption due to poor selectivity when leaching complex ores containing various base metals

The presence of copper is particularly troublesome, as most oxide and secondary sulfide minerals leach well and form highly soluble cyanide complexes. In addition, copper usually reports with gold and silver as they are stripped from the leach solution. The presence of copper then complicates the final parting and refining of the precious metals.

In spite of the problems with cyanide leaching, operators have learned to deal with issues such as cyanide destruction in effluent streams and detoxification of heaps and dumps at closure. Nonetheless there remains a strong impetus to find an alternate lixiviant. Most attention has been focused on thiourea [$(CS(NH_2)_2$] (Hiskey 1981, 1984). Its potential benefits, compared to cyanide, include the following:

- Up to 12 times faster leach times, resulting in leach cycles of hours rather than days or months
- Functional in an acidic rather than an alkaline medium
- Better selectivity for gold and silver
- Lower toxicity

Unfortunately, however, after three decades of development, thiourea has not lived up to its potential, and its use generally remains confined to high-grade materials. The main problems are high unit costs and high reagent consumption.

Copper

Only three acidic lixiviants have been used in commercial copper leaching operations on ores and mine wastes: (1) acidic sulfate solution is the dominant choice and can be used on both oxide and sulfide mineralization, including ores that contain a mix of the two types; (2) acidic chloride solution has also been used; examples involve the use of sea water or saline groundwater for the leach solution and do not involve additions of hydrochloric acid; (3) sulfuric acid is used for leaching, even with a chloride solution, and actually produces a mixed sulfate–chloride lixiviant. Of these, sulfuric acid offers the following advantages:

- It is relatively benign with regard to materials of construction, with well-known handling characteristics.
- It is compatible with recovery of copper by conventional solvent extraction–electrowinning (SX-EW) technology.
- It is inexpensive and commonly available as a by-product from copper smelters that are often located nearby.

In addition to acidic systems, an alkaline ammonia-based lixiviant has also been used commercially in specialized cases, including the leaching of both ores and concentrator tailings containing malachite and azurite (copper carbonates). For many years, the Calumet and Hecla Mining Company ran ammonia leach plants to recover native (metallic) copper from mill tailings (Benedict and Kenny 1924). Today, no copper ores or tailings are treated in ammonia leach plants. The technology is therefore not covered further in this chapter; for more information, see Chase (1980).

The copper mineralization dictates how the ore should be leached. Oxidized ores, including most oxides, silicates, carbonates, hydroxides, and chlorides, can be dissolved effectively with a nonoxidizing acid—that is, sulfuric acid. The general reaction is a straightforward, irreversible dissolution

that consumes acid. The specific reaction for the common mineral chrysocolla is

$$CuSiO_3 \cdot nH_2O + H_2SO_4 \rightarrow CuSO_4 + SiO_2(H_2O)_{n+1} \quad (11.4\text{-}3)$$

However, unoxidized ores, mainly the various sulfide minerals and native copper, do require an oxidant. As described below, the chemistry is more complex and relies on natural oxidation processes to accomplish leaching. The oxidant may be either dissolved oxygen or an ionic species such as ferric iron in a sulfate-based system or ferric or cupric ions in a chloride system. Regardless of the lixiviant, the leach solution must be either acidic enough or basic enough to carry copper in solution, as copper salts tend to precipitate from near-neutral solutions.

In the absence of chloride input, the natural lixiviant for copper sulfide minerals is acidic ferric sulfate. An example for the leaching of chalcopyrite is

$$4Fe^{+3} + CuFeS_2 \rightarrow Cu^{+2} + 5Fe^{+2} + 2S^\circ \quad (11.4\text{-}4)$$

Here the balancing anion is sulfate (not shown). Although small amounts of elemental sulfur are detected on partially leached mineral surfaces, no accumulation of elemental sulfur is observed in actual operations. Thus, any elemental sulfur is transitory and progressively oxidizes all the way to sulfate in the presence of bacterial catalysts and oxygen in the air within rock-pile voids.

The ferric iron in Equation 11.4-4 can come from several sources, including acid-soluble iron in various gangue minerals. Without going into detail, much of the iron comes from the oxidation of pyrite (Hiskey and Schlitt 1982), which plays an important role in sulfide leaching. Pyrite is typically 3 to 10 times more abundant than the copper species, so it is important simply on a mass basis. Pyrite oxidation is analogous to the chalcopyrite reaction shown in Equation 11.4-4. In addition to supplying iron, pyrite oxidation is an in-situ source of acid generation that helps maintain the acid balance and low pH needed to keep copper in solution (Templeton and Schlitt 1997).

These sulfide oxidation reactions are also exothermic. Heat released during reaction boosts the temperature in the heap or dump and accelerates the leach kinetics. It also produces a buoyancy effect and promotes convective airflow through the rock pile, which provides the oxygen that is needed for reoxidation of ferrous iron back to the ferric state. This reaction is typically bacterially catalyzed, as described later in the "Bioleaching" section.

Uranium

Commercial heap leaching of low-grade uranium ores was first practiced in the United States during the uranium boom in the 1950s. More recent examples include the Los Gigantes acid heap leach in Argentina and the bacterial heap leach on pyritic ore at Urgeiriça in Portugal (now shut down). Commercial operations currently exist in Brazil, China, Niger, and Spain, and both acid and alkaline leach projects are being planned in Australia, Botswana, and Namibia. The list seems likely to grow, given the renewed interest in nuclear power and the economics of heap leaching.

As is also the case for copper, both mine-to-leach and secondary operations can use sub-mill-grade feed. Most of these projects currently produce no more than a few thousand tons of yellow cake per year, and total output is small compared to in-situ operations. In addition to primary production, uranium has also been produced as a by-product of copper leaching operations in a few cases (George et al. 1968).

Regardless of the leaching method, the same lixiviant chemistry is applicable to uraninite (UO_2) and coffinite [$U(SiO_4)_{1-X}(OH)_{4X}$] ores that are commonly treated. Both acidic and alkaline lixiviant choices exist (Grunig 1981). To minimize reagent losses in solution mining, the lixiviants have been tailored to avoid excessive gangue reactions. Thus a sulfuric acid–ferric sulfate leach was developed for sandstone ore, and a sodium bicarbonate or ammonium carbonate leach was developed for uraninite in calcareous ore. The oxidant in the latter system is either hydrogen peroxide or oxygen dissolved in the leach solution.

For both acid and alkaline circuits, dissolution requires oxidation of tetravalent uranium to the hexavalent state. In dilute sulfuric acid (pH 1.6 to 2.0), uranium is oxidized by ferric ion and goes into solution as the very stable uranyl sulfate complex. Similarly, sodium or ammonium bicarbonate lixiviants form uranyl carbonate.

The overall reactions in an acid circuit are (Merritt 1971)

$$UO_2 + Fe_2(SO_4)_3 \rightarrow UO_2SO_4 + 2FeSO_4 \quad (11.4\text{-}5)$$

$$UO_2SO_4 + 2H_2SO_4 \rightarrow [UO_2(SO_4)_3]^{-4} + 4H^+ \quad (11.4\text{-}6)$$

The overall reactions in an alkaline circuit are (Merritt 1971)

$$2UO_2 + 2H_2O_2 \rightarrow 2UO_3 + 2H_2O \quad (11.4\text{-}7)$$

$$UO_3 + Na_2CO_3 + 2NaHCO_3 \rightarrow Na_4\,UO_2(CO_3)_3 + H_2O \quad (11.4\text{-}8)$$

Both types of circuits are well established and have seen extensive commercial application in both solution mining and conventional mills using agitation leaching.

Nickel

Nickel and by-product cobalt are the most recent commodities to be recovered by heap leaching. The target ore types are laterites, which contain more than 70% of the world's nickel but account for only ~40% of the world's production (McDonald and Whittington 2008a, 2008b). Unlike gold, silver, copper, and uranium, virtually no discrete nickel or cobalt oxide minerals are found in laterite ores. The occurrence of both nickel and cobalt is quite variable, as they substitute for iron in goethite (limonitic ore) and smectite group clays or for magnesium in saprolite (serpentine) ore. A typical example is the hydrous magnesium silicate falcondite [$(Ni,Mg)_4Si_6O_{15}(OH)_2 \cdot 6H_2O$)].

As of this writing, the technology for heap leaching of laterites is still developing and competes with the more established high-pressure acid leaching. Key issues for heap leaching have recently been reviewed (Steemson and Smith 2009). For both types of leaching, sulfuric acid is the overwhelming choice of lixiviant. For heap leaching, the acid concentration is typically 50–75 g/L. Because nickel and cobalt substitute in the structure of various complex minerals, the entire mineral must be solubilized to release the metal values. As a result, sulfuric acid has poor selectivity, and mass loss can be as high as 30%. Acid consumption is also high and varies from 10 to 75 kg/kg (lb/lb) of nickel or 100 to >500 kg/t (200 to 1,000 lb/st) of ore.

Because of high acid consumption, heap permeability can be a problem. As a result, laterite heap leaching is typically done in shallow, single-lift heaps run in on/off mode, as described later in the "Leach Systems" section.

Other Metals

Other than the commodities covered previously, zinc is the only metal that is currently produced commercially by heap leaching. Teck Cominco (now Teck Resources) reportedly pilot-tested the heap leaching of low-grade sphalerite (ZnS) ore, but results have not been reported, nor has a commercial project been announced. However, a zinc silicate heap leach is reportedly operating in Namibia with sulfuric acid as the lixiviant.

Low-grade manganese ores should also be amenable to solution mining, although this has never been done commercially. The lixiviant would be aqueous sulfur dioxide (SO_2), which reacts with manganese dioxide (MnO_2) as follows:

$$MnO_2 + SO_2\,(aq) \rightarrow MnSO_4\,(aq) \qquad (11.4\text{-}9)$$

Less-oxidized forms of manganese require an oxidant such as dissolved oxygen, hydrogen peroxide, or perhaps ferric sulfate. Tests reported by Potter et al. (1982) suggest pregnant-leach-solution (PLS) grades ranging upward of 10% to 12% $MnSO_4$, implying that manganese can be recovered as the sulfate by evaporation or as the carbonate or hydroxide by neutralization and precipitation.

Polymetallic Systems

Other than gold/silver production, leach operations recovering more than one metal have been rare. In addition to the by-product uranium mentioned previously, efforts have also been made to extract cobalt from copper leach liquors in the United States. Cobalt recovery from some African copper leach operations seems certain, as does recovery of cobalt from nickel laterite heap leaching. Another possibility is dual-circuit recovery of copper and zinc. Recovery of cobalt, copper, manganese, and zinc was planned for the mine-to-leach operation at the Sanyati mine in Zimbabwe; however, the current status of this project is uncertain. The technology for cobalt recovery already exists for both copper and nickel circuits. However, when multiple recovery circuits operate in series, problems always exist with coextraction of metal values and carryover of reagents from one circuit to the next.

Bioleaching

Only the leaching of sulfidic ore and mine waste has a bacterial component; the leaching of nonsulfide materials is based on straightforward chemical control. Years of development work have been spent on bacterial or bioleaching, but commercialization has generally been limited to high-grade ores and concentrates.

The exception is Newmont Mining Company's two-stage heap-leach process for treating refractory (pyritic) gold ore. In the first step, bacterial leaching with forced aeration partially oxidizes the pyrite, liberating or at least exposing the gold inclusions. When at least 40% of the pyrite is oxidized, the heap is thoroughly rinsed to remove acid that was formed. In the original plan, the material was to be off-loaded, blended with lime, restacked on a second pad, and leached with cyanide to recover the gold. In actual operation, the first heap is off-loaded and hauled to the carbon-in-leach mill, where bio-oxidized material is neutralized in a semiautogenous grinding mill. This process has not found widespread use, but details are available in Logan et al. (2007).

Other commercial leach operations having a bacterial component all rely on indigenous microbial species for the necessary activity. The natural bacterial oxidation provided by the chemoautotrophic bacterial strains *Thiobacillus ferrooxidans* and *Thiobacillus thiooxidans* has been well known for years. *Thiobacillus* are generally active at temperatures below 40°C (104°F). Large sulfidic leach dumps typically operate at higher temperatures, with indigenous moderate or extreme thermophiles playing the same roles.

The exact mechanisms of bacterial action are still in dispute, but the accepted theory is that the bacteria obtain energy for growth from enzymatic oxidation of ferrous to ferric iron and elemental sulfur to sulfate.

T. ferrooxidans bacteria catalyze the oxidation of ferrous to ferric iron (Brierly 1978; LeRoux et al. 1973):

$$2Fe^{+2} + \tfrac{1}{2}O_2 + 2H^+ \rightarrow 2Fe^{+3} + H_2O \qquad (11.4\text{-}10)$$

Similarly, *T. thiooxidans* bacteria catalyze the oxidation of sulfur to sulfate:

$$S^\circ + 3/2O_2 + H_2O \rightarrow 2H^+ + (SO_4)^{-2} \qquad (11.4\text{-}11)$$

The rates of these reactions can increase by several orders of magnitude when catalyzed bacterially. With the addition of carbon, nitrogen, oxygen, phosphorous, potassium, and other elements as nutrients, the bacterial metabolism can be maintained to sustain a stable population.

The key factor to keep in mind in such an operation is that microbial populations are living organisms. Actions that cause sudden changes to their preferred environment should be avoided—otherwise the change can render the population inactive or even kill it outright, negatively impacting production. An example is sudden introduction of chloride ion via use of magnesium or calcium chloride as a dust suppressant or deicer on haul roads.

Metallurgical Testing

Regardless of the values to be recovered, a proper metallurgical test program is required for any new leach project. Test results establish expected operating parameters and set the design criteria needed to engineer and construct leaching and recovery facilities.

The most important single factor in any test program is the selection of samples to be tested. Great care must be taken to ensure that samples are representative of the material to be leached. If a deposit contains more than one type of ore, in terms of either mineralogy or lithology, then each type must be tested thoroughly. When a mine plan is available, samples representing various mining periods should also be tested to avoid surprises in recovery or reagent consumption. A common approach is to test quarterly composites for the first 3 years of operation and annually thereafter.

Test Levels

At least two and sometimes three levels of testing must be undertaken (McClelland 1988):

1. Preliminary tests: small-column percolation or bottle roll tests
2. Detailed tests: larger column tests to optimize feed size and other parameters
3. Large- (pilot-) scale tests: large-column or field tests

Preliminary tests accomplish the following: (1) provide insight into the amenability of the material for leaching; (2) allow selection of the preferred lixiviant, should the choice not be obvious; (3) provide information on the chemical aspects of the proposed ore–lixiviant system; (4) indicate whether or not agglomeration is needed to avoid permeability problems with clays or fines. With respect to the chemical aspects of the proposed system, chemical parameters include a first estimate of reagent consumption and the rate and ultimate level of extraction of the one or more metals. Differences in the behavior of the various ore types become evident.

Care must be taken in extrapolating preliminary test results to predict design parameters in the eventual operating system. Preliminary results are almost always misleading in the following ways:

- Leach rates are higher than in commercial operation. Material is crushed much more finely in small-scale tests than in a full-sized system. Finer crushing maximizes liberation of the metals and minimizes diffusion distances to any mineralization that is not exposed. Thus metals are solubilized rapidly and completely, assuming that results are not impacted by complete reagent consumption in typical batch-type tests.
- Reagent consumption is higher than in commercial operation. This reflects not only the high extraction of metals but also the high host-rock surface area. In particular, fine rock size tends to maximize reagent consumption when the lixiviant is not highly selective for the metals and attacks host-rock (gangue) minerals as well. Such behavior is particularly common for acidic lixiviants, which tend to be less selective than alkaline reagents. For example, in acid leaching of copper, on a mass basis there is generally much more reaction with gangue constituents than with copper minerals.

Detailed tests are run in larger columns. These tests are generally intended to determine whether or not the ore should be leached in ROM conditions, crushed to some smaller size, or crushed and then agglomerated. Extraction-rate curves and reagent-consumption figures are determined as a function of particle size and degree of agglomeration.

Detailed tests can be run in either of two ways: single- or multiple-feed sizing. In the former, medium-sized material is column-leached until the extraction rate levels off to near zero. The contents of the column are screened and assayed to determine recovery as a function of particle size. Generally a second confirmatory test is run on material crushed to the selected size. This approach is less costly but more time-consuming than running parallel tests on multiple samples crushed to a range of sizes. With either approach, the last step is to balance projected recoveries and revenues against the cost of progressively finer crushing and/or agglomeration. The most economical operating scenario can then be selected.

Similar detailed tests can be done to optimize other operating parameters as well. These parameters include agglomerating conditions, heap height, leach-solution irrigation rate, reagent concentration, pH, and, in some cases, oxidation potential. Along with crush size, these parameters define the operating conditions.

Large-scale tests are done less frequently now than in the mid- to late 1900s. They typically involve large columns or square cribs that represent a slice of the expected commercial heap. Cribs are frequently loaded in the same way that the commercial heap will be stacked, replicating ore emplacement with its natural angle of repose and size segregation with fines at the top. Typical large-scale column tests are run with 18–36 t (20–40 st) of gold/silver ore (McClelland 1988) and as much as 180 t (200 st) of copper ore (Murr 1980). In many cases, large-scale column tests are more meaningful than tests performed on small field heaps because of edge effects caused by the large percentage of material contained under the sloping heap faces.

Large-scale tests are often omitted, and test programs stopped at the detailed-test stage, for the following reasons:

- Considerable expense and an extended schedule are required to complete the tests and evaluate the results.
- Obtaining a representative sample can be difficult. Even driving an adit into the middle of the ore body may be insufficient if the deeper ore has different characteristics.
- For deposits that have only been drilled, there may simply not be enough core samples available for large-scale tests.
- Permitting and environmental requirements are likely to be as onerous as for full-scale operation.
- Extensive databases for actual projects, together with advanced computer-modeling techniques, now make projection of expected operating performance much more accurate than it has been in the past. However, this effort still requires a comprehensive test program that accurately replicates the proposed operating parameters.

Logistical considerations and the availability of representative leach material often limit really large-scale field tests to existing mine operations. Use of a large-scale heap or dump is required only when assessing the impact of a parameter that cannot be replicated in smaller-scale testing. A good example is the million-ton ROM test heap that Kennecott Utah Copper Corporation built at the Bingham Canyon mine (Utah, United States) (Schlitt 2006). The primary objective was to assess the impact of severe winter weather on performance of the heap and the 6 t/d (6.6 st/d) SX-EW plant used to recover the leached copper. A secondary objective was to provide a facility for operator training. By the end of the program, the project had produced 771 t (850 st) of cathode copper that was sold as a credit against the cost of the project.

Test Results

Regardless of the test scale, at the completion of metallurgical testing, results are typically presented as plots of cumulative metal extraction or reagent consumption versus a test variable.

One obviously useful plot is cumulative extraction versus time under leach (Figure 11.4-1). The leach curve has a pseudo-parabolic shape, indicating that the reaction rate is fast initially but slows progressively with time. The slope of the tangent to the curve at any given time gives the instantaneous leach rate at that time. For the curve shown in the figure, the rate is in units of percent extraction per day. A linear initial leach rate implies that the initial rate is controlled by either

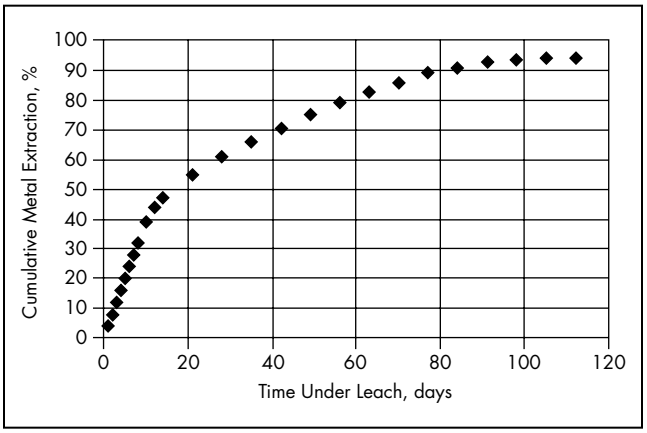


Figure 11.4-1 Typical leach curve for a heap leach

reagent availability or reaction rate at the mineral surface. As leaching continues, metal-bearing minerals exposed on the rock surfaces are consumed and the rate decreases as diffusion to or from less-accessible mineralization becomes rate-controlling. The rate continues to decrease until all accessible mineralization has reacted and the ore is leached to exhaustion.

Another useful plot is recovery versus cumulative solution flux (total quantity of leach solution applied per metric ton of ore under leach). Such a plot has a normalizing effect, enabling comparison of, for example, results for columns of different heights. Using solution flux also compensates for periods when flow to the heap is interrupted. Other plots may be appropriate for special situations. For example, for sulfuric acid lixiviant, a plot of recovery versus acid flux may be informative.

In addition to the general test criteria outlined previously, there are many commodity-specific aspects to metallurgical testing. These cannot be addressed in detail here but do require that a well-planned research program be conducted by knowledgeable metallurgists and engineers to get meaningful results. To cite just one commodity-specific example, bottle roll tests on cyanide extraction of gold should be done in open bottles. If the bottles are sealed, poor results may be due to oxygen deficiency rather than the presence of refractory mineralization.

MATERIALS HANDLING

This section covers everything from mining of the leach material to its emplacement on the heap prior to leaching. This includes a description of the leach systems, with alternatives to conventional heap and dump leaching. The section concludes with a description of ore preparation, which covers crushing, agglomeration, and the various methods of emplacement.

Mining

Surface heap and dump leaching operations are almost invariably associated with open-pit mining operations. These require essentially the same planning functions and use the same unit operations as those described in Chapter 10.2. However, two areas may be impacted by leaching operations: mine planning, and drilling and blasting.

If heap or dump leaching supplements conventional milling, mine planning can become quite complex. One complexity lies in the need for more segregation of mined materials. Material must be divided into not just two categories, mill ore and waste, but three: mill ore, leach material, and waste. Both leach heaps and waste dumps need to be designed, and movement of all three types of material must be scheduled. In addition, water distribution and collection systems must be designed and installed for the leach system.

Supplemental production of leach copper also affects calculation of the ore cutoff grade and ultimate pit limits. Since leaching permits partial recovery of copper from material that would otherwise be wasted, the effect of a leaching operation is always to raise the cutoff grade on ore going to the mill. This, in turn, lowers unit costs for the mill output. Also, since leaching boosts total recovery of values from the mineral resource, it also increases the tonnage of material that can be profitably mined. Stated another way, a supplemental leach operation expands the ultimate pit limits. The exact impact of leaching is site specific and depends on the relative costs and output of leach production and mill production.

Drilling and blasting is another area that may be impacted by leaching, particularly when a mine-to-leach operation uses ROM ore. The drill-and-blast pattern must be designed to give a well-fragmented product with minimal quantities of oversized rock, about plus 200–250 mm (8–10 in.) top size. The powder factor also requires control to avoid generation of excessive amounts of fines that could cause permeability problems in the pile. The well-fragmented product should stack and percolate well while providing good mineral exposure needed for effective leaching. Because rock characteristics are very site specific, costs for an optimized drill-and-blast program that gives the desired as-mined leach product may need to be compared with costs for a low-cost drill-and-blast program that includes primary crushing to produce material for leaching.

Leach Systems

Heaps and dumps are by far the most common types of surface leaching operation. There are two types of heap leach systems to consider: a permanent multilift system and an on/off single-lift system. Each has advantages and disadvantages.

Permanent Multilift Leach System

An advantage of the permanent multilift system is that it minimizes material-handling costs. After a lift is leached, fresh ore is stacked over the spent ore. If the original lift is leached to exhaustion, an interlift liner can be used to separate the old and new ore. This prevents high-grade leach liquor from percolating down into the old material where it will be diluted by the residual moisture in the heap. However, in most cases, the original lift still contains residual metal values that can be extracted. In this case, no interlift liner is used and the solution percolating down through the new ore continues through the original lift, recovering additional metal. Stability issues and installation costs require additional consideration when employing interlift liners.

A disadvantage of the permanent multilift system is that the base of the heap must be impervious to solution flow to avoid contamination of the environment and avoid loss of metal-bearing solution. As a result, the lined-pad area must be expanded as the heap footprint grows. At the end of operation, the lined area must be sufficient to contain the entire leach ore output. A related problem is that loading will develop at the base of the heap as additional lifts of ore are added. This may compact the lowest ore and cause permeability problems at the bottom of the heap. The result can be development of a perched water table within the heap, causing solution to break out at intermediate heights or even heap instability. Even in

the absence of permeability problems, the amount of residual moisture in the heap system increases as ore tonnage in the heap increases. This residual moisture typically varies from 6% to 12% of the ore's dry weight, increasing as ore size becomes finer. This moisture contains soluble metal values that were extracted from the ore but are not available for recovery. Thus, they represent in-process inventory and must be taken into account when forecasting production. These in-heap metal values will not be recovered until the operation is closed and the heap system is given its final rinsing as part of the closure exercise.

On/Off Single-Lift Leach System

The on/off single-use leach pad is typically used for ores that leach quickly or have serious permeability problems. The size of the lined pad area is fixed and is much smaller than that for a permanent multilift system. Ore is stacked on the pad, leached to exhaustion, rinsed, and removed to a permanent impoundment. Thus the pad must be large enough to hold material being stacked, leached, and off-loaded, and some additional area must be available for loading or to provide some flexibility in the leach cycle. If the residual ore is not acid-generating, the impoundment area may not require a liner. Because of the rinsing and small active ore volume, in-heap inventory is not a serious issue.

An on/off pad has two major disadvantages. One is the cost of double-handling the ore, once to stack it and once to remove it to the residual impoundment area. The other is the lack of opportunity to recover any values remaining when the leach cycle is complete and the spent ore is removed.

Alternate and Variant Leach Systems

Some alternatives or variations on these systems are appropriate for special circumstances.

Dump leaching is equivalent to a permanent multilift system but involves low-grade material handled in ROM condition. It is subject to the same issues as is the permanent multilift system.

Sand–slime separation is an alternate means for handling ore with a high fines content. Separation typically is done at 2 mm (10 mesh). The coarse fraction can be heap- or vat-leached and the fines can be agitation-leached in some type of countercurrent system. This circuit is currently in use at the Tintaya mine in Peru, where the coarse fraction is heap-leached and the fines are agitation-leached. It was previously used at the Ray mine (Arizona, United States), where the coarse fraction was vat-leached and the fines were leached in air-agitated Pachuca tanks.

In-place systems represent a special case of materials handling, intermediate between surface heap and dump leaching and in-situ mining. Broken or rubblized material is leached more or less in place. In-place leaching clearly involves no loading or hauling of broken ore to heaps or dumps. However, unlike in-situ mining, in-place leaching still relies on downward percolation of solution through unsaturated leach material. (In contrast, in-situ mining is characterized by pressure-driven solution flow in a saturated medium).

In-place systems are so site specific that discussion is limited to brief descriptions of a few commercial operations. Although broken uranium ore has been leached in underground settings, copper appears to be the main target to date. Probably the largest-scale operation was undertaken by Ranchers Exploration and Development Corporation when they drilled and blasted the entire upper portion of the Old Reliable mine (Arizona) ore body and then acid-leached the rubblized zone in place (Longwell 1974). Technically they were able to fracture and rubblize the ore, but leaching yielded less copper and produced a solution that was lower in copper concentration or tenor than expected. Thus the project was not the commercial success that was hoped for (Catanach et al. 1977).

Pit-wall leaching has yielded greater success. It involves progressive rubblization and leaching of abandoned pit walls. Since most porphyry ore bodies contain a halo of low-grade mineralization around the economically mineable zone, pit-wall leaching often recovers a portion of the copper that is otherwise left behind. Examples of pit-wall leaching include the Mineral Park and Silver Bell mines in Arizona. Examples of underground leaching of broken, remnant ore left behind in block-caving operations include the El Teniente mine in Chile and the San Manuel mine in Arizona.

Ore Preparation

Material scheduled to go to heaps or dumps can be handled in ROM condition or prepared for leaching by crushing, with or without agglomeration. Size reduction should be undertaken when the increase in production achieved with finer product generates sufficient incremental revenue to cover the capital and operating costs for crushing. In general, low-grade mine waste that must be removed to expose mill-grade ore does not merit much extra material handling or preparation. Such material is often dumped in ROM condition at locations that minimize haulage costs rather than maximize leach output. However, there has been a move to optimize at least the dump profiles to enhance recovery. Then the dump is leached to recover some of the values that would otherwise be lost.

Agglomeration

Agglomeration is widely applied in precious metals operations. It can provide at least two benefits:

1. Agglomeration provides strong, porous, and coarsely sized material that otherwise would be unleachable because of poor percolation characteristics when placed in heaps. This is particularly true when at least 10% of the material is finer than 420–500 μm (35–40 mesh). Such conditions usually lead to low metal extraction because of slow and uneven percolation (channeling) of solution and development of impermeable (dead) zones within the heap or dump. Clays are probably the leading source of fines, although fines produced by poor crushing practices are undesirable as well. Even tailings can be agglomerated and subsequently heap-leached, as was done for uranium–vanadium recovery from salt-roasted tailings at Naturita (Colorado, United States) (Scheffel 1981).
2. Agglomeration uniformly blends the lixiviant into the crushed ore. Leach reactions have a chance to start before the ore is even placed on the heap, which provides for quick release of metal values and high PLS grades. However, experience indicates that agglomeration is likely to have little or no effect on the final extraction level.

Agglomeration follows one of two pathways: it causes either adherence of fines onto coarser particles or conversion of fines into stable pellets.

For gold and silver, important process parameters include quantity of binder (typically 5 kg/t or 10 lb/st), moisture content (10% to 20% of total), and agglomerate

curing period (≥8 hours). Since the subsequent cyanide leach requires alkaline conditions, binders are typically lime or portland cement. These binders generally have two important properties: (1) clay permeability is improved by the exchange of sodium ions in the clay with calcium ions in the binder, and (2) the binders have a cementing or pozzolanic effect that strengthens the agglomerates. Leach extraction rates can also be accelerated by using a barren cyanide solution as the source of moisture.

For copper ores and other materials that use an acidic lixiviant, the purpose of agglomeration is often primarily to make the ore acidic. The acid-cure dosage is normally less than half the amount of acid consumption expected for the particular ore. A higher dosage does little to improve extraction but raises the amount of acid consumed by gangue mineralization. In addition to accelerating the initial release of copper, acid agglomeration prevents a high pH in the initial drainage from the heap—that is, the PLS. If the pH is higher than 3.5 to 4.0, copper can precipitate from solution and be difficult to resolubilize.

Other process parameters include feed characteristics (particle size and presence of fractures), process control, and equipment selection (Milligan 1984; McClelland and van Zyl 1988). For gold and silver, the most popular types of agglomeration equipment are belts, drums, and pans. For copper, the most popular type is the drum, which thoroughly blends the acid and water throughout the ore charge. To save on capital costs, however, acidulation is sometimes achieved by spraying acid on the ore while it is transported by conveyor belt, although significant blending occurs only at the conveyor drop points and transfer stations.

Alternate Methods

As an alternative to agglomeration, some operations perform sand–slime separation as described in the previous section. For dump leaching, size reduction is generally not accomplished unless primary (in-pit) crushing is performed so that the material can be transported and stacked by conveyors. For heap leaching, primary, secondary, and tertiary crushing are often used to produce a final product that is nominally 25 to 12.5 mm (1 to 0.5 in.). Screening requirements depend on the nature of the ore, extraction sensitivity to crush size, and equipment manufacturer recommendations. Modeling the crushing circuits based on the various rock properties can enable selection of the optimum equipment configuration.

Ore Emplacement

How material is stacked for leaching—how it is emplaced—is a major factor in the success of a surface leaching operation. The approach to emplacement represents a principal difference between dump leaching and heap leaching.

For dump leaching, dumping practices reflect topographic constraints and efforts to minimize haulage costs. Consideration of leach recovery is usually limited to practices such as waste segregation, modification of the heap profile, and dump-surface preparation. Segregation usually involves efforts to avoid mixing barren or mineralogically unleachable materials into the leach dumps. Surface preparation before addition of the next lift is typically limited to leveling and deep ripping to break up the compacted zone associated with the truck traffic pattern. However, dump profiles can be adjusted to enhance leach conditions. For example, sulfide leaching normally requires some type of aeration, and dump dimensions can often be adjusted to enhance natural convective airflow through the rock mass. Optimization of the profile for different conditions has been modeled extensively and provides a guide to dump design for a particular ore.

Waste dumps are often high to minimize haulage costs; elevation differences of 90 to 230 m (300 to 750 ft) between toe and crest are common. This profile exacerbates two types of segregation:

1. Size segregation, which occurs when ROM material is dumped over the crest. This allows the coarsest material to slide or roll to the bottom while the fines tend to remain at or near the crest. As a result, the least permeable and most easily compacted zone of material is found at the top. Leach solution then percolates slowly and unevenly at the top of the dump and tends to channel and short-circuit through this layer. These poor percolation patterns normally persist at depth so that a significant volume of material in the dump is leached poorly if at all. The presence of leaching material below the sloping face may also be a problem.
2. Layer segregation, which creates alternate layers of coarse and fine material that lie parallel to the angle of repose. The coarse zones act as conduits and carry the bulk of the lixiviant flow. Hence they are usually well leached in spite of their large particle size. In contrast, the fine material has a low permeability and so undergoes little solution penetration or leaching unless proper solution management practices are followed.

These factors in waste emplacement are responsible for the generally poor extractions observed in waste dumps. For ROM material, recovery of metal values seldom exceeds 50% after many years of leaching. Only 10% to 15% extraction during the first 1 or 2 years is not uncommon, at least for copper sulfide dumps.

For heap leaching, careful emplacement of rock is important to maximize extraction of values. This generally involves construction of much shallower lifts, typically 3 to 10 m (10 to 35 ft) high to minimize size segregation. A further way to minimize size segregation is to use crushed or agglomerated ore. In turn, solution percolation is more uniform and solution-rock contact is improved, resulting in faster and more complete metal recovery.

Several additional approaches to heap construction, described by Muhtadi (1988), warrant consideration due to their widespread use. Foundation requirements are covered later in the "Environmental Aspects" section.

Truck dumping and dozing. Emplacement by means of truck dumping and dozing is essentially as described previously for waste dumps. However, more care must be given to emplacement of material. Typically, a ramp of waste is constructed to the height of the first lift. Ore is dumped from the end of the ramp down onto the pad or foundation of the heap. As ore builds up, the heap extends away from the ramp. To minimize compaction, an elevated roadway can be extended as the heap expands, with haulage trucks restricted to this path. When the heap is completed, the roadway is dozed off and the surface of the heap is graded and ripped to ensure good permeability and uniform percolation of solution.

In general, this method of heap construction is restricted to ores that do not undergo compaction or generate fines. This requires a strong, competent material handled either in ROM or crushed condition. Agglomerated materials cannot generally be handled by truck dumping, as they are too soft and friable. In addition, truck dumping still causes coarse/fine segregation.

Plug dumping. Emplacement by plug dumping also involves trucks but is not characterized by the rough handling noted previously. Hence it is suitable for softer ores and agglomerates. The first step is to bed the liner or foundation with crushed rock as a protective measure to ensure liner integrity. Then each haul truck dumps its load as closely as possible to the previously dumped pile until the entire pad is covered with overlapping mounds. The maximum heap height is about 2 to 3 m (7 to 10 ft), slightly less if a dozer is used to level the tops of the mounds.

A variation of this method is to replace the trucks with large rubber-tired front-end loaders. These are very efficient and can stack material to heights of ≥5 m (16 ft) without the need to first bed the liner.

Conveyor system. Emplacement by conveyor stacking has gained widespread acceptance in heap leaching. The concept is not new, however. It started early in the 20th century when high-grade copper ores were bedded and removed from leach vats by means of a conveyor-tipper and a clamshell bucket on a traveling gantry crane, respectively. Indeed, one of the first stacking systems in gold heap leaching (Gold Fields Mining Corporation, Ortiz, New Mexico, United States) used a traveling bridge that spanned a reusable pad and used a moving tripper to continuously add ore to the face of the heap. Spent ore was removed from the asphalt pad by front-end loaders.

Now conveyor hauling and stacking of ore on heaps has evolved into a highly mobile operation. The front end of the system usually consists of a short fixed conveyor that receives prepared ore from a crusher, agglomerator, or a stockpile via a feed hopper. The conveyor transfers material to a mobile, radial-arm conveyor/stacker by means of a system of movable intermediate conveyor sections. The stacker has great flexibility and can bed the heap uniformly and without compaction to almost any height desired. Thus conveyor systems can handle anything from primary crushed ore to agglomerated tailings. For on/off systems, front-end loaders or bucket-wheel excavators are used to remove the leach residue.

SOLUTION MANAGEMENT

Materials handling and solution management are interactive, with rock preparation and stacking having a substantial effect on solution flow. For example, maximum recovery of metal values can be achieved only if the heap/dump design permits good distribution and collection of leach solution. The design must also provide for movement of air into the piled rock, as oxygen is needed as a reactant in solubilizing many values. Likewise, the rock must be broken and stacked so that the entire rock mass is accessible to solution percolating down from the surface of the heap or dump.

Nevertheless, after the rock has been prepared and put in place, there is little that an operator can do except optimize the way the leach solution is managed. Doing so accomplishes two things: (1) it transports the lixiviant to the mineralized values throughout the piled rock, and (2) it washes the solubilized values from the heap or dump to the recovery plant.

Fluid Flow Phenomena

Fluid flow phenomena include both static and dynamic aspects. The latter covers the optimum application rate and the potential benefits of leach/rest cycles and pulsed leaching.

Static Considerations

The first step in understanding solution flow through rock mass is to categorize the space within the dump or heap. There are at least three distinct regions:

1. Solid rock (including isolated pores). By far the largest, this completely stagnant space typically involves about 60% of the total volume.
2. Space associated with openings created by rock blasting and handling. This space exposes pores and creates fractures or cracks on the faces of the rock fragments. Generally only 2% to 4% of the total volume, nevertheless, this space greatly increases the exposed surface area and makes the contained mineral values more accessible to the lixiviant.
3. Void space that occupies the remaining volume within the rock pile. This space results from the swelling that is a consequence of rock fracturing and stacking during mining and handling. For newly prepared heaps, void space normally constitutes about 40% of the volume of the rock pile. This value decreases as weathering and leaching consolidate the material and typically drops to <30% if a mine dump is leached over a period of years.

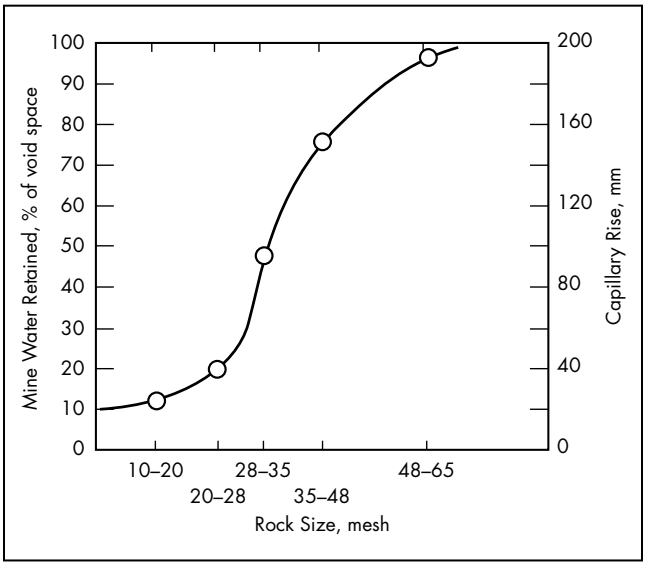
Obviously, void space can be filled with either leach solution or air. However, because the water–rock interface has a lower surface energy than does the air–rock interface, all the pores within the rock fragments fill with solution. The rocks themselves are covered with water under conditions of percolation. The water-filled void space represents the area for active water flow. In contrast, mass transport within the water-filled fractures or pores occurs only by diffusion. A consequence of the difference in interfacial energies is the location of the air-filled void space. Under conditions of percolation, only the larger voids and channels contain air, and the smaller openings are flooded.

Except under conditions of flooding, competition between air and water for available void space is almost independent of the rate at which leach solution is applied. Instead, the controlling factor is the effective particle size of the rocks in the pile. This is a consequence of the limitations imposed by capillary forces. As shown in Figure 11.4-2, capillary rise or drain-down height is related to effective rock size. The figure shows the correlation between rock size and percentage of the total void space that remains filled with water after leaching ceases and drain-down is complete. At this point, capillary rise is balanced by the force of gravity, and only evaporation can further reduce the water-filled void volume.

The experimental results shown in Figure 11.4-2 are significant when ingress of air is needed to help solubilize the metal values. For material that is finer than ~320 μm (48 mesh), the pile does not drain down at all, and little or no air can penetrate the pile. For material that is coarser than ~2.0 mm (10 mesh), the pile drains down almost completely, allowing air to permeate throughout the heap or dump.

Dynamic Considerations

When leaching begins at a low application rate on a thoroughly drained dump, some additional air-filled void space



Source: Schlitt 1984.

Figure 11.4-2 Solution retention and corresponding capillary rise as a function of rock size

becomes water-filled at the solution–air interfaces. Flow then begins. As the application rate increases, the only change is to thicken the film of flowing water in the larger voids and channels where there is a solution–air interface. No substantial change in the volumes occupied by air and liquid occurs until the application rate increases to the point where the air-filled voids begin pinching off. When such bottlenecks occur, the open voids flood upward from the pinch points and the air-filled void spaces begin to disappear.

Experimental column data demonstrating this phenomenon are shown in Table 11.4-1. At typical application rates (4–40 mm/h or 0.1–1.0 gal/ft^2/h), there is considerable air-filled void space. Thus, adequate aeration should not be a problem unless the material being leached is very fine (e.g., flotation mill tailings).

Regardless of the application rate, the velocity of solution flowing through a water-filled void is controlled by the pressure gradient associated with the hydrostatic head and the permeability at that location. This relationship is expressed by Darcy's law:

$$V°_L = k/\mu\ (dp/dx) \qquad (11.4\text{-}12)$$

where

$V°_L$ = superficial velocity of the leach solution
k = permeability
μ = fluid viscosity
dp/dx = pressure gradient or hydrostatic head of the leach solution

Optimum Application Rate

From the standpoint of the leach operator, the maximum effective application rate (q_{max}) is limited by the zone of minimum permeability in the system under leach. This may be the bulk permeability of the rock pile. However, care should be taken to ensure that the solution application rate does not exceed the surface acceptance rate of the heap or dump. This is an important consideration because several factors can combine

Table 11.4-1 Effect of solution application rate on water-filled void space

Application Rate		Water-Filled Void	
mm/h	gal/ft^2/h	Space, %*	Comment
0	0	8.0	Drained
1.2	0.03	9.6	Capillaries filled
6.0	0.15	9.7	Typical range of rates
24.0	0.60	10.1	
120.0	3.0	10.3	
3,050.0	75.0	10.5	
4,070.0	110.0	27.6	Flooded

Source: Schlitt 1984.
*Expressed as a percentage of the total space within the rock pile.

to make surface permeability lower than the bulk value. These factors include the following:

- Accelerated decrepitation or weathering of the surface layer
- Breakup and compaction of the surface rock due to equipment movement, particularly haulage truck traffic
- Segregation of fine material at or near the top of the heap where dumping occurs
- Formation or accumulation of chemical precipitates such as gypsum or hydrated iron oxides due to evaporation and crystallization of the leach solution

Where surface permeability is lower than the bulk value, the application rate should be reduced to avoid ponding (surface flooding), which prevents air circulation into or through the rock. Ponding also provides ideal conditions for continued chemical precipitation that further seals the rock-pile surface.

To select the maximum effective application rate, the leach operator needs some knowledge of the permeability range in the system. This information is probably best obtained by performing one of the standard water permeability tests. Unfortunately, there is little published information on heap and dump permeability to serve as a guide. Limited field measurements on copper leach dumps suggest a value of q_{max} in the range 20–40 mm/h (0.5–1.0 gal/ft^2/h). Higher flows cause increasing amounts of water simply to short-circuit through the rock without recovering additional values.

The importance of controlling application rates is further demonstrated in Kennecott data for copper dump leaching (Jackson and Ream 1980). In a comparison of sprinkler leaching at 20 mm/h (0.5 gal/ft^2/h) and trickle leaching at 80 mm/h (2 gal/ft^2/h), sprinkler leaching produced 11% more copper and a 15% higher copper content in the PLS. Overall, 13% less water was pumped per pound of copper produced at the lower application rate, significantly reducing power costs per unit of copper produced.

Leach/Rest Cycles

Leach/rest cycles are a factor in solution management when the active lixiviant (such as ferric iron) is generated in situ. However, such cycles have little value when the lixiviant is delivered externally (such as by acid in the irrigating solution). For lixiviants generated in situ, when a volume of rock has been thoroughly wetted with solution, the lixiviant distributes throughout the rock mass. Then, during the rest cycle, dissolution of the values continues without further solution application. For lixiviants delivered externally, as soon as the externally

supplied reagent is consumed, leach reactions stop. Flow must resume supplying reagent before reactions can continue.

For appropriate ores, rest cycles can extend the economic life of older heaps and dumps, as they permit maximum recovery of metal values from material that was mined. Rest cycles also permit an operator to reduce costs by maintaining a limited flow of high-grade PLS to the plant. In both cases, the key factor is switching application of water from one leach area to another. While standing fallow, chemical and diffusional processes dissolve metal values and cause their concentrations to build up in the water-filled void space. Then a relatively short period of active solution application washes the solubilized values from the heap or dump. As an example, a 1-week leach cycle might be followed by a 4-week rest cycle. On older material, a longer rest cycle may be required to build up an economic concentration before washing.

Similarly, recycling barren solution to the oldest material first, then advancing the outflow to fresher material, reduces the volume and increases the grade of the pregnant solution being treated in the plant. This is sometimes referred to as *solution stacking*.

A third option is to reduce the irrigation rate progressively, which minimizes the solution flow when extraction is slow and helps to maintain a high-PLS metal concentration.

Thus, leach/rest cycles should not be overlooked as a way of optimizing a leach operation and minimizing pumping and facility costs.

Pulsed Leaching

Recent developments in vadose (unsaturated) zone flow (Breckenridge and Henderson 2005) may have application in situations where a heap or dump has a wide particle size distribution that includes a high fines content. Typically a leach dump would likely form a low-permeability zone that restricts leaching at higher flows. Under this condition, the capillaries are all filled and the strongest force is gravity, which leads to the familiar downward percolation of solution through the larger partially water-filled voids. However, as flow rate decreases, the point is reached where matric forces become dominant and most of the flow is via capillaries in the finer material.

Thus, for material with both coarse and fine sizes, leaching by cycling the irrigation rate between high and low values may improve recovery of metal values from the full spectrum of particle sizes, leading to higher overall production. For perspective, Breckenridge and Henderson (2005) give an example for a gold ore heap for which the high and low flows are 1.45 and 0.48 mm/h (0.036 and 0.012 gal/ft^2/h).

Production Aspects

The production aspects of solution management include the pumping systems and the pipeline requirements. Also covered are the types of equipment used to irrigate the heaps and dumps: either spray systems or drip emitters.

Pumping Systems and Pipeline Requirements

Selection of a pumping system depends on several factors: hydrostatic head on the system, line pressure needed to drive the solution distribution equipment on the leach surface, flow volume, and corrosion potential of the lixiviant. Because there are so many site-specific factors involved, only general guidelines are provided here.

In many operations, the recovery plant is located at a lower elevation than are the heaps or dumps. Thus, PLS generally flows to the plant by gravity in lined ditches or low-pressure pipelines. Any pumping from a heap or dump outflow or from a plant feed pond generally requires only a low-pressure horizontal pump, which can be mounted beside the pond with the pump intake suspended from a float in the pond to readily accommodate rising or falling water levels.

For the situation noted previously, recycling of barren (tail) solution back to the heaps or dumps requires a high-head pumping system, typically involving multistage vertical pumps. Pumps can be mounted on a fixed structure such as in a pump house where pumping is from a sump. Alternatively, they can be mounted on a barge and allowed to float in the barren solution pond.

If system pressure is not too great, flexible pipe or couplings can be used to tie the pumps into the shore-based pipeline system, allowing them to rise and fall with the pond level. The type of pipe is dictated by the elevation difference between the plant and the leach surface, plus the line pressure needed to drive the solution-distribution equipment on the heap or dump. The elevation difference can be as little as 30 m (100 ft) or as much as 600 m (2,000 ft). The latter requires sophisticated designs starting with heavy-walled metal pipe, shifting gradually to lighter schedules. The entire pipe system must be well anchored, yet allow for expansion and contraction due to temperature changes. In addition, the entire header system must almost certainly be higher than the 70–345 kPa (10–50 psi) pressure needed on leach areas, requiring installation of outlet stations on the main header line to serve as pressure-reduction points as well. When the pipeline profile and other flow parameters are defined, sophisticated computer programs can be used to aid in pipeline design.

Pipe materials must be selected with the chemical nature of the lixiviant in mind. For higher line pressures that dictate metal pipe, 316 L stainless is a frequent choice for acidic copper leach liquors that contain ferric and cupric ions. Polyvinyl chloride (PVC) or other liners have been tried on black iron or steel pipe, but the linings tend to deteriorate with time, particularly at joints, leading to corrosion and eventual failure at the joint. For alkaline solutions or concentrated sulfuric acid, black iron may be satisfactory. For lower line pressures, high-density polyethylene (HDPE) pipe has become the standard of choice. This is rugged, easy to cut and fuse in the field, resistant to scaling, and essentially corrosion-proof. PVC and other rigid plastic pipes may be less expensive but are not as rugged and tend to shatter. One of the few problems with HPDE pipe is its lack of rigidity: it twists and turns, and may require stakes or anchor blocks to keep it in place. However, this flexibility can also be an asset when routing the pipe during initial installation.

Solution Application Systems

As pointed out previously, careful distribution of solution on the heap or dump surface is required to optimize the leach operation. Development of such a solution management system requires a clear understanding of three concepts:

1. *Application rate,* calculated by dividing the leach solution volumetric flow rate by the surface area to which the solution is actually being applied
2. *Surface acceptance rate,* the maximum application rate that can be employed without encountering noticeable ponding on top of the heap or dump
3. *Irrigation rate,* a measure of the intensity of leaching, calculated by dividing the total volumetric flow rate by

the total area planned for leaching, including areas under active leach and areas that are being allowed to rest

For sulfidic copper minerals, typical irrigation rates are only 20%–25% of the application rate, and may be only 10% on old areas needing a long rest cycle. For gold, silver, and oxide copper, the percentages are much higher because the extraction rates are higher.

Only two classes of irrigation system are in widespread use today: spray or leach sprinklers and drip emitters. Pond leaching and high-intensity trickle systems have generally been abandoned, because their application rates normally exceed the optimum levels, leading to short-circuiting of solutions and dilution of values in the PLS. The desired application rate should be based on *q*.

Spray or sprinkler leaching devices include the following:

- **Sprinklers.** These are of two types: (1) rotating impulse sprinklers, which are much like typical "rain bird" agricultural sprinklers except that they are constructed of acid-resistant plastic or stainless steel for all wetted parts; and (2) Senninger wobblers, which are off-center rotary-action sprinklers that provide a coarse spray pattern to minimize evaporation. However, the off-center design creates a thrust problem, and wobblers must be staked down.

  Both designs provide a pattern of reasonably uniform solution coverage at an application rate of 20 mm/h (0.5 gal/ft^2/h), which is safely below *q*. With their own pressure regulators, wobblers are sophisticated enough to provide even lower rates (such as 12 mm/h or 0.3 gal/ft^2/h), the level typically cited as optimum for gold. Ranges of operating parameters for several types of sprinklers are shown graphically by Schlitt (1984) and are available in vendor literature.
- **Pressure emitters.** These are equivalent to drip-irrigation devices used in agricultural applications. They operate at low gauge pressure (100–140 kPa [15–20 psi]) and produce a steady seepage of leach solution that gives about the same application rate as sprinklers.

In spite of their similar delivery rates, each system has a number of relative advantages or disadvantages with respect to the following considerations:

- **Droplet impact.** Pressure emitters minimize droplet impact, reducing particle decrepitation and fines migration. This is particularly valuable when leaching soft, friable ore or agglomerates.
- **Winter weather.** Pressure emitters are effective even when buried. Thus they can be used in areas that experience severe winter weather and can cause sprays to ice up.
- **Wind.** Pressure emitters are immune to wind drift. Under extreme conditions, sprays may have to be shut down to avoid spreading acid or other chemicals beyond the boundaries of the leach system.
- **Evaporation rate.** Pressure emitters have low evaporation rates, only 1% to 2% of total flow, whereas spray losses have rates in the range of 5% to 10%. Which type has the advantage depends in part on a site's water balance. In dry areas where makeup water must be purchased, emitters are favored. In net precipitation areas, even on a seasonal basis, sprays are favored, since a high evaporation rate can benefit the water balance. In fact, special sprays called "foggers" are designed to maximize water removal by evaporation.
- **Blockage.** The major drawback to pressure emitters is the likelihood of blocking, or plugging, the small flow channels with precipitated salts. To avoid plugging, conditioning of the recycle solutions to prevent scale formation or remove suspended solids must receive careful attention. Conditioning can be done by means of settling ponds, filtration systems, or chemical additives.
- **Coverage.** Sprinklers give complete coverage of the top surface. In contrast, pressure emitters, which are actually arrays of point sources of water, create a conical volume of wetted ore below each emitter and leave some near-surface material unleached. For a permanent multilift system, this is not a serious issue, because any unleached ore will be wetted when the next lift is added and placed under leach. However, for a shallow on/off heap, as much as 10% of the ore can remain unleached.

Environmental Aspects

At least in the United States, obtaining the necessary environmental permits is probably the single most time-consuming activity in the development of a heap or dump leaching project.

Permitting requirements vary from state to state and may differ depending on whether the project is to be sited on private, state, or federal land. Because of these differences and constantly changing regulatory requirements, a detailed guide to the appropriate permitting process cannot be presented here. The best approach is usually to hire knowledgeable environmental consultants who practice in the area where the project will be located. These consultants should be quite familiar with applicable regulations and procedures and perhaps even know the local and state regulatory personnel.

Although specifics may differ from one site to another, regulatory agencies generally have two major concerns regarding surface leaching projects: (1) ensuring the safe handling of toxic chemicals and reagents, and (2) preventing contaminated runoff or seepage to groundwater. Detoxification of the heaps or dumps may also be a concern for all future operations and is already having an impact on the gold heap-leaching segment of the industry. However, groundwater restoration, a major factor for in-situ operations, has little impact on surface operations unless leaks have contaminated the subsurface aquifer.

Other environmental concerns are common to most mining operations. These include control of particulate emissions (dust) from the mining and any crushing or processing operations as well as topographic restoration to preexisting profiles and revegetation of mine and leach areas. Restoration activities usually have some sort of bonding requirements. In terms of costs, solution containment and heap or dump stability are the major considerations. Each is discussed in the following subsections.

Geotechnical Design

The design of piled rock structures that are stable under conditions of water flow, especially in seismically active areas, is a complex undertaking. Entire books have been written that address the necessary design features for both heaps (van Zyl 1987) and dumps (McCarter 1985). Because of this complexity, a detailed analysis of the problem cannot be presented here.

The first consideration is the method of heap or dump construction. Alternative methods involve use of reusable

pads, expanding pads, and valley dumping. Major requirements and considerations in each case are discussed by Dorey et al. (1988).

Reusable pads are best in terms of both containment and stability. This method involves a sloping base that is highly impermeable and can easily support the weight of a single shallow lift of ore. In the past, the base has been asphalt or even concrete. However, today a multilayer base with plastic liners over a clay foundation is more common. From an environmental point of view, the biggest concern is the integrity of the final impoundment area for the leached ore. This requires a properly designed retaining structure and one or more liners to prevent seepage losses of residual reagents. A monitoring system is also required to detect any leakage. Backfilling an abandoned pit area should not be overlooked for creating a possible containment site.

Expanding pads and valley dumping do not entail the material rehandling required with reusable pads. Rather, use of expanding pads involves periodic extension of the prepared heap foundation and liner system, which is then loaded with an initial lift of ore. This method is characterized by a gently sloping impervious liner that is overlain with a network of drainage pipes and a coarse drainage layer. The intent is to drain off the PLS as soon as it percolates down through the ore. Rapid removal minimizes the hydrostatic head on the liner to help minimize leakage. As more ore lifts are added, loading on the liner system can become an issue.

Valley dumping uses a natural or constructed retaining structure such as a ridge or dam located at the bottom of a canyon or valley. Ore is stacked upstream behind the structure in progressive lifts. Total land area required is usually less than that required for expanding pads, but lining the underlying surface can be difficult because of slope and terrain features. Solution containment for this type of system is addressed in the following subsection, while stability is covered here. In a true valley-fill system, such as that of Carlota Copper Company in Arizona, the PLS is retained within the heap system as a saturated zone. Submersible pumps then advance the collected PLS to the recovery plant.

Stability depends on a number of factors, including strength and slope of the retaining structure, load imposed by the stacked ore, and buildup of a saturated PLS zone at the foundation or elsewhere in the rock mass. The latter is critical, as more heap and dump failures have been caused by buildup of excessive pore pressure than by any other cause (Caldwell and Moss 1985). Without going into mathematical details, the reason for this is that pore pressure within a saturated zone reduces the shear strength and frictional resistance to movement of the stacked material. As pore pressure builds, especially near a free face, it can cause shear strength to drop below that needed to withstand the loading imposed by the rock mass. At this point, the structure fails, often catastrophically.

Solution Containment

An effective liner is the principal barrier to seepage loss. This is true not only for heap or dump foundations but also for catchment basins, flow channels (including ditches and trenches), and holding ponds. In addition, no containment system is perfect. Some seepage always occurs and can be measured by Darcy's law (Equation 11.4-12). The factors that control seepage are liner permeability and hydrostatic head. The engineer must design the system so that the expected seepage level is within environmentally acceptable limits.

Strachan and van Zyl (1988) cover liners in detail, including liner selection, system design, and installation. Selection is based on the type of application, nature of the solution to be contained, physical loading, and exposure to the elements. Liner materials include clays, clay or chemically amended soils, and geomembranes (PVC, Hypalon, HDPE, asphalt, etc.), used alone as single liners or in combination. When used in combination, the upper liner is usually a geomembrane and the lower one is a soil material. On double or triple liners, a leak-detection system is often installed in a porous zone between the layers. The topmost liner is overlain with drainage-collection piping and a coarse rock drainage layer. If a forced aeration system is to be used to boost the sulfide leach rate, then under-heap aeration piping should also be installed during construction of the leach-pad overliner system.

The use of systems that minimize seepage loss not only meet regulatory requirements, but also make good sense from an operational perspective. Even a small leak, if undetected, can cause a major loss in metal values and reagents over time. Such losses are equivalent to lowering the ore grade or getting poorer metallurgical recovery. In many cases, the dollar value of lost production can equal or exceed the cost of a good solution containment system.

COMMODITY RECOVERY SYSTEMS

Systems used to recover metals from heap and dump leaching operations are the province of hydrometallurgists. Although the recovery systems themselves depend on the lixiviant used in leaching, they are essentially independent of ore-emplacement or solution management practices. Thus the unit operations are commodity specific and covered in standard metallurgical texts and reference volumes. Current recovery systems are briefly described in the following subsections.

Copper

Historically, copper has been recovered as an impure metallic mud or "cement" by cementation (precipitation) on scrap iron. However, today, virtually all copper is recovered by means of SX-EW, a process that produces directly salable cathode copper at the mine site.

SX involves two stages. In the extraction stage, copper in the leach solution is transferred to the lean organic liquid at a low acid concentration ($\leq$10 or 15 g/L) and the two immiscible liquids are allowed to settle. The less-dense organic phase rises to the top and is drawn off. In the strip stage, the acid level in the lean electrolyte is much higher ($\geq$150 g/L) and copper transfers from the organic phase to the electrolyte. The rich electrolyte is then subjected to electrowinning to produce copper cathodes.

Gold/Silver

Pregnant solutions from gold/silver heaps and dumps are currently treated by either of two methods.

One method involves absorption on activated carbon granules packed in columns. The gold and silver cyanide complexes load on carbon, which is periodically stripped into a concentrated cyanide solution. The gold and silver are then recovered by electrolytic deposition from the strip liquor.

The other method involves stripping the gold and silver from solution as a sludge by means of zinc precipitation (the

Merrill–Crowe process). The precipitate is dried and smelted to produce doré metal for refining. Because carbon is less effective for silver than for gold recovery, high silver solutions are usually treated by the Merrill–Crowe process.

Uranium

The relatively low-grade leach liquors from uranium heaps and dumps are usually concentrated by ion exchange, then purified and concentrated further by solvent extraction. Yellow cake (U_3O_8 [triuranium octoxide]) is precipitated from the solvent-extraction strip liquor.

Nickel

As for copper, nickel and any by-product cobalt are normally recovered by SX-EW.

Solution Conditioning

The only potentially unusual feature of the various recovery systems is conditioning of the barren solution before recycling it to the heaps or dumps. Because most of these facilities must operate under conditions of zero discharge, use of a bleed stream to control buildup of undesirable impurities may not be possible. Thus, conditioning may be needed to control scale-forming constituents, remove potentially harmful impurities, replenish reagents that have been consumed, adjust solution Eh or pH, or simply make up the water lost to evaporation.

Conditioning is highly site specific and depends on the ore–lixiviant system, gangue reactivity, and even chemical makeup of the available process water.

Overall conditioning requirements for heap and dump leaching are less than for in-situ mining. This is due to the sensitivity of the in-situ formation to loss of permeability caused by suspended or precipitated solids building up and blocking the flow paths between the injection and recovery wells.

ANCILLARY AND INFRASTRUCTURE REQUIREMENTS

Ancillary and infrastructure requirements for surface leaching are much the same as for any open-pit mining operation. This is because leaching is merely the ore-processing operation associated with mining. The main requirements are for electric power, water, a transportation system, and access to a sufficient labor force. Fuel (e.g., natural gas) is not a concern, as there are generally no large furnacing or other heating operations. Building heat, small precious-metal furnaces, and driers can easily be fired electrically or with bottled gas if a natural-gas line is not available.

In general, ancillary and infrastructure requirements are lower for primary heap and dump leaching operations than for mine-and-mill operations. Leaching plants are thus well suited to operations that are small or remotely located.

Site Requirements

A mine-and-mill operation requires a rail line, heavy-duty haul road, or both to move ore from the mine to the mill and concentrate from the mill to a smelter. In contrast, a leach operation requires merely an all-weather access road, since shipping is limited to receipt of reagents and other supplies and shipment of small tonnages of crude or refined product. Acid trucks or similar vehicles are likely to be the largest units entering or leaving the plant on a regular basis.

Labor Requirements

Labor requirements are lower for leaching operations than for conventional mills, as most workers perform actual mining activities. Exclusive of mining, labor requirements for a leaching operation should represent no more than 15% of the direct operating costs. Thus, much of the work force can be recruited locally, even in rural areas. However, a camp arrangement may still be required at very remote sites. Specialized services such as instrument and computer-system maintenance or even assaying may have to be done on a contract basis.

Power Requirements

Power requirements vary widely, depending on the type of operation.

Gold plants generally have low requirements, mainly for pumping relatively small solution flows and running small heaters and furnaces. Many remote gold plants use diesel generator sets or other types of on-site generation, as power so generated is cheaper and more dependable.

However, large copper leach operations are power-intensive. With flows as high as 3.2 m^3/s (50,000 gpm), water-distribution systems may include multistage pumps with 37.5–110 kW (500–1,500 hp) motors. Copper electrowinning is also power intensive, requiring at least 2.2 kWh/kg Cu (1 kWh/lb Cu). Power is therefore usually purchased from a utility company. This purchased power is a big factor in the operating costs and the operation may incur a high capital cost if an extensive pole line with transformers, rectifiers, and substations is required.

Water Requirements

Water requirements vary according to operational stage. Start-up requirements are relatively high, principally because the various catchment basins and recovery-plant feed and tail-water reservoirs must be filled. The heaps and dumps must also be saturated with water before flow occurs. This represents a significant solution inventory, since the rock mass retains 8% to 12% moisture by weight, which is about 13 L/m^3 (0.1 gal/ft^3) of the heap or dump material.

Steady-state requirements are relatively low, because they cover only evaporation and other miscellaneous losses. Since most plants must operate under zero-discharge permits, no net makeup is required to compensate for bleed-off. The exact makeup requirement generally varies seasonally with the local evaporation rate. For copper in the southwestern United States, a typical annual average figure is 35 to 50 L/min/t (10 to 15 gpm/st) of daily production.

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CHAPTER 11.5

In-Situ Techniques of Solution Mining

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INTRODUCTION

In-situ solution mining is the selective dissolution and recovery of target mineral(s) without the use of conventional excavating techniques. Instead, the minerals are dissolved where they reside within the ore body. The dissolved mineral solution is then pumped to a processing plant located on the surface, where the desired metals are produced for market. This method can be considerably more economical than conventional mining methods, because massive tonnages of gangue materials are simply left where they exist, resulting in elimination of waste handling efforts, tailing piles, and waste brines.

More importantly, low-grade deposits can potentially be economic, because solution mining typically is cheaper than conventional mining methods. Additionally, through careful engineering, drilling, and monitoring, in-situ mining can have a greatly reduced environmental impact compared to mining by conventional methods. Mineral commodities exploited by in-situ techniques include salts such as sodium chloride, potash, magnesia, soda ash, borates, sodium sulfate, and lithium, as well as metals such as uranium and copper.

Deposit Characteristics

The primary requirement for in-situ solution mining is that the ore body has amenable characteristics to allow the lixiviant (the dissolving fluid employed to extract the valuable mineral) to come into contact with the target mineral throughout the ore body in sufficient quantity that an economical recovery rate of the mineral can be achieved. Beyond this requirement, metallurgical and ore host issues must be acceptable so the mineral can be leached from the ore. The deposit may have characteristics that prevent efficient leaching in a sufficient time frame. The general category of minerals where in-situ solution mining has proven to be economically successful includes both salts and metals.

The two main types of deposits associated with soluble salts are evaporites and brines. Evaporites may be classified as either marine or alkali; both types accumulate as a result of concentration through the evaporation of inland brines. Natural brines may occur as surface or subsurface deposits and are often associated with playas.

Marine Evaporite Deposits

Marine evaporite deposits are typically impermeable and located in large plastic salt domes. Often, they are found in a series of horizontal beds and in anticline deposits. Some deposits are hard to mine from a conventional perspective because of the deposit depths and the plastic behavior of the salt under stress. These deposits, however, can be quite attractive for a solution mining application. Marine deposits mirror seawater composition to a degree, which has about 31% sodium, 4% magnesium, 1% calcium and potassium as cations, and 55% chlorides, 8% sulfates, and less than 1% bicarbonate as anions. Therefore, halite, sylvite, and sodium sulfate are commonly found in marine deposits.

Alkali Evaporite Deposits

Alkali evaporite deposits are typically found in inland seas such as the Great Salt Lake in Utah and Searles Lake in California (both in the United States), and the Dead Sea in Israel/Jordan. Evaporation of an inland sea leads to formation of a playa or salt bed. If the salt bed formation is followed by geological burial, then a subsurface alkali basin is formed. Typical deposits that are mined include halite, potash, trona, borates, sodium sulfate, lithium (as lithium chloride), and soda ash.

Natural Brine Deposits

Brine occurs naturally as surface brine, such as in the Great Salt Lake or the Dead Sea, and as subsurface brine, such as at the Hector deposit in California. The Searles Lake deposit has both surface and subsurface brines at various times of the year. Where groundwater or surface water runoff congregates in a depression, subsurface brines typically exist in the permeable matrix below the depression. The water percolates through the depression or playa, dissolving water-soluble salts along the way. The salt-laden water then flows to a location where it is stable, creating a surface or subsurface brine deposit.

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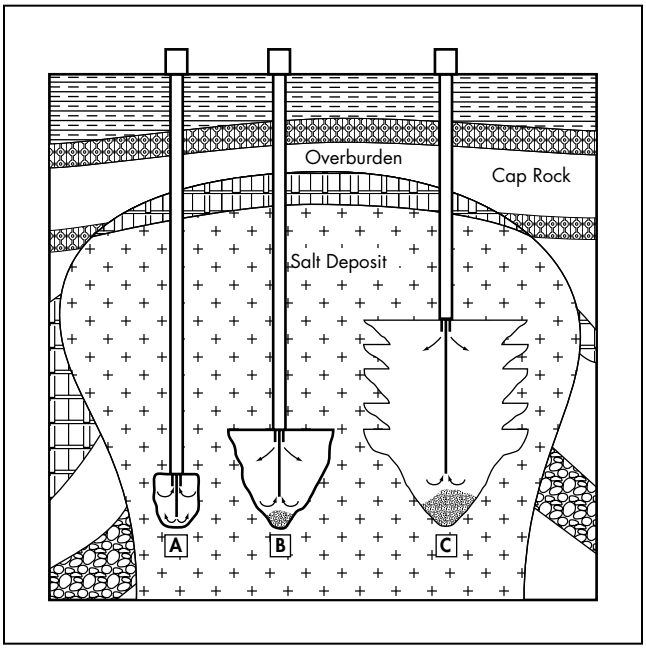


Figure 11.5-1 Leaching methods in impermeable salt deposit: (A) direct circulation solution mining—cavity establishment phase, (B) reverse circulation solution mining—brine production phase, and (C) reverse circulation—brine production continues

Metal Deposits

For metal deposits, in-situ leaching has been successfully employed in sandstone-hosted uranium deposits. The characteristics of these ore deposit types are discussed in depth later in the chapter, but in-situ recovery production for uranium requires ore within the water table so that a regime can be employed to move fluids through the sandstone in order to extract the uranium from the host rock.

Solution Mining Methods

Mining Impermeable Salt Deposits

Impermeable deposits are typically mined by introducing hot water or other solutions into the deposit to create a cavity. The formation of the cavity through dissolution of the target mineral produces a salt brine containing the mineral. The brine is pumped from the cavity to the ground surface where it is processed further in a plant to achieve the desired marketable product. Prior to the creation of the underground cavity, the potential for the collapse of overlying rock into the cavern must be considered. Salt-solution mining operations must be designed and carried out with the structural integrity of the resultant caverns in mind. A layer of rock or salt (beam) must be designed and left intact above the cavern to support the cavern. The minimum thickness and dimensions of this beam are designed based on the characteristics, strength, and cavern dimensions within the salt structure and the overlying geologic structure above the cavern. The cavern dimensions can be monitored with sonar, and its volume can be estimated from continuously recorded mass balance readings during production of the salt.

In order to develop a salt cavern, a vertical hole is drilled from the drill collar to a point near the base of the salt formation. Coaxial tubing is then inserted into the drill hole to provide for flow of lixiviant brine, product brine, and blanket fluid. Lixiviant brine chemically bonds with the target salt to make the product brine, which is extracted to harvest the target salt. The fluid blanket is used as an immiscible surface that rides on top of the salt brines to control vertical cavern creation. Alternately, tubing is inserted side by side rather than in a coaxial fashion. In either case, the depth of injection for each fluid is set at an appropriate vertical point for the leaching method and cavern design being implemented.

Using the direct circulation leaching method shown in Figure 11.5-1, an initial cavity (sump or sedimentation cavern) is formed at the bottom of the drill hole, which may extend below the base of the target salt bed. This sump creates an area designed to collect insoluble material as it drops out of the salt matrix in the cavern above, as the salt goes into solution. Lixiviant, as fresh water or brine, is pumped into the cavern at a designed elevation at a calculated flow rate. A blanket fluid, selected to float on top of the other liquids, or compressed air is injected at the top of the forming cavern to maintain the roof elevation of the developing cavern and drive the dissolution process horizontally rather than upward. As salt dissolves, a density gradient forms within the brine. During start-up, the lower density solution flows to the top of the cavern, becomes more saturated along the way, and is pumped out at the top of the cavern. As the overall brine saturation increases, the brine production flow rate is also increased. This method allows for the formation of a relatively wide lower cavity.

After the sump cavity is formed, the fluid injection levels are adjusted so the lixiviant is introduced near the top of the cavern. In some cases, heat may be added to the system to assist in facilitating dissolution. The new lighter-density lixiviant spreads horizontally under the blanket and removes the target salt in the ceiling of the cavern. The saturated brine, or pregnant solution, is pumped from the lower cavity region to the surface for processing. There is no need for a blanket to protect the cavern floor, because the brine at this level is already saturated. Periodically, the tubing is raised upward in successive steps, thereby controlling cavern diameter and mining of individual horizontal cuts upward through the height of the cavern. This reverse circulation leaching method is also illustrated in Figure 11.5-1.

The lixiviant can be tailored to selectively dissolve the target salt while leaving behind undesirable salts. However, the remaining salt matrix can drastically decrease dissolution rates and in some cases can *blind off*, stopping fluid flow altogether. Selective lixiviant can be prepared by recycling spent product brine from an established processing plant. The blanket can be any material that is immiscible with the lixiviant and brine, and has a specific gravity less than the lixiviant and brine—for example, compressed air or oil. Typical cavities can be 80 to 150 m (87 to 164 yd) in diameter.

Depending on the ambient surface temperature, *salting out* may occur when a change in brine concentration, due to temperature change or brine influx, causes premature crystallization. This is detrimental in a situation where, as the production brine nears the surface, the temperature drops enough to significantly decrease the solubility of a salt, causing crystallization of a potentially desirable product before it reaches the surface. The salt then comes out of solution prior to plan, which causes production problems—whether in the product brine line within the well, the pipeline on the surface, or in any facility accepting the brine prior to processing. Conducting dissolution and bench-scale studies prior to implementation

of a plan in the field will be helpful in determining the overall design of the system to avoid (or deal with) salting out issues.

Mining Permeable Salt Deposits

Permeable salt deposits, such as occur below playas or partially dissolved salt deposits containing natural brine, are deceivingly simple to mine in situ. This method does not require a long period to establish saturated brine, because the natural brine is already at equilibrium. To extract the salt, a series of wells are placed within the salt deposit to inject brine being used as a lixiviant. This lixiviant flows through the salt deposit to a pump that is placed within a centrally located production well, so the product brine can be collected, extracted to the surface, and processed.

The flow rate of fluids extracted/injected must be carefully designed to achieve maximum production while avoiding short circuiting. This occurs when the recycle/injection brine is not allowed enough time to resaturate, thus not reaching the desired target salt content. This then results in a lower density than desired, and flow of the injection brine within the formation (or cavern in an impermeable deposit) is compromised.

It is important to understand the initial permeability of the salt deposit and the salt characteristics to ensure that the well field, well, and flow rates are properly designed for production operations. These types of production schemes can be employed to feasible depths based on economic conditions that are dictated by commodity price, mineral grades, capital costs, and production costs.

Mining Permeable Metal Deposits

Permeable deposits are those that have sufficient space between the particles or grains (pore space) to allow a lixiviant to be passed through the deposit so contact with the desired mineral can be accomplished. For example, where uranium is found in permeable sandstone, a lixiviant can be injected into the sandstone to allow the lixiviant access to the uranium particles so the uranium can be chemically attracted by the lixiviant for pumping to the surface and on to the processing plant. Often, sandstone of appropriate character will have a pore volume of 20% to 30%. To accomplish this, a well field is drilled with a number of injection wells and a much fewer number of production wells. Wells are drilled to surround a single production well that has been established within the ore body and are specifically designed and cased so that a lixiviant can be pumped into the injection wells. The lixiviant then flows through the ore body to the central production well, in which a pump is installed to control the movement of lixiviant from the injection wells to the production well.

In some cases, the deposit needs to be submerged within an aquifer so that solution flow can be controlled. Often, the aquifer will already be tainted because of the ore body's chemistry, but, nonetheless, there are groundwater issues to consider and manage. In more developed countries, regulations concerning in-situ mining are quite evolved, providing good guidance on acceptable operating practice. It is also of great value to have aquitards or aquicludes (an impermeable layer that prevents fluid flow) above and below the target ore body, which helps manage production solutions. Generally, the solutions are overproduced from the production well, so that an excess of solution is pumped from the production area of the ore deposit. This bleed is performed as a protective measure to capture all the production solution placed in the aquifer.

New innovations include modifications to standard vertical drilling methods. By employing practices perfected in the oil sector, inclined and horizontal drilling methods are now being considered. Production monitoring has also become much more sophisticated. This is achieved by using monitor wells to check for and identify possible loss in the production solutions within the aquifer. New process-control instrumentation and logic controllers can now be employed in the complete production system. These methods are of solid value in ensuring that the water supply and quality are properly managed at the site.

Mining Abandoned or Flooded Conventional Mines

When conventional underground mining methods have reached their feasible or economic limits, solution mining can provide for the additional recovery of minerals. These conventional mines are commonly established in bedded deposits that typically used a room-and-pillar (R&P) mining method. With this method, pillars of valuable ore are left to support the roof as the mining operation is conducted. Valuable ore left in the walls of the mining openings can be extracted by flooding the mine, thereby introducing a lixiviant to extract the ore. As solution mining is employed, it is essential to consider existing groundwater regimes, undesirable ore types, lixiviant flow paths throughout the mine openings, recovery of saturated brine, and any issues relative to induced subsidence of the natural ground layers if the pillars have already collapsed (or will collapse) as solution mining is implemented.

Whether the mine is flooded deliberately or by natural means, extraction wells are drilled into the lowest elevations of the mine, and injection wells are placed at higher elevations. Gravity does its work, with the more dense saturated brine collecting at the bottom of the mine from where the pregnant solution is pumped to the surface for harvesting.

Hydraulic Fracturing

Hydraulic fracturing is the process of expanding or creating artificial fractures around a well to enhance its permeability or the communication between adjacent wells. It is typically used where a section of the well is segregated, and a liquid such as brine, water, or oil is used to pressurize the target zone and induce high pressures, which will either extend fractures or generate new fractures. In the oil industry, acid has been used for decades to etch the fracture sides, while sand/glass beads have been included in the fracture fluid to prevent fracture closure when the hydraulic pressure is bled off. The fracturing system is dependent on inherent weaknesses, such as bedding planes, within the host structure; regional stress fields; and other geologic phenomena. Two of the primary difficulties of the technology are projecting the fracture propagation direction, intensity, and growth; and estimating the pressure requirements for successful fracturing. However, the technique has been successfully employed for several decades in the oil industry, and hydraulic fracturing has been shown to be a valuable technique.

SALTS EXPLOITED

Salt

Salt is mined utilizing three methods:

1. Conventional underground mining, which produces rock salt
2. Solution mining, which produces evaporated or vacuum pan salt
3. Solar evaporation of ocean or lake brine, which produces solar or sea salt

Desirable Geologic Conditions and Chemistry

Salt occurring in underground salt domes is typically exploited using a solution mining method. Many salt deposits have been found in a thick bed located 6 to 8 km below the earth's surface. The high temperature and pressure encountered at these depths allow the salt to flow in a plastic mass. With the salt having a lower specific gravity (2.168 for pure sodium chloride [NaCl]) than the surrounding rock, the salt naturally pushes upward to reach equilibrium near the surface. These salt domes are generally pure halite (the mineral form of NaCl), typically have a circular footprint in the horizontal plane, are 1.5 to 3 km (0.9 to 1.9 mi) in diameter, and extend for a vertical distance of 6,000 m (6,562 yd) or more. The deposits become more commercial when the top of the dome is within a few tens to a few hundreds of meters below the ground surface.

Mining Method

Salt dome deposits (impermeable-type deposits) are typically solution mined by employing multiple wells. Sometimes, hydraulic fracturing or lateral drilling is used to link several wells together within a well field, or *gallery*. Employing hydraulic fracturing technology has been successful and has greatly improved the competitiveness of mining operations by

- Enhancing the stability of underground openings by changing the cavity configuration (to eliminate the unstable "morning glory" shape) to a design based on hemispheres and ellipsoids, which increases the stability of underground cavities. This is quite beneficial in reducing the potential for subsidence at the surface;
- Cutting the costs related to brine generation by approximately 50%;
- Greatly increasing product extraction by roughly 40%;
- Reducing well maintenance costs by about 95%; and
- Increasing the well productive capacity by a factor of 10 (Feldman 2006).

Processing Method

Salt brine must be evaporated to extract the entrained salt values. With the proper climate and available land, this may be achieved through solar pond evaporation followed by further upgrading using flotation and centrifuges, much like the processes employed for potash. Alternatively, high-grade salt is obtained through vacuum pan evaporation. This process uses a series of three to five steam-heated multiple-effect evaporators in which a vacuum is pulled to lower the boiling temperature of the brine. Each successive evaporator has a lower pressure than the previous, and the superheated vapor proceeds through the series, helping heat the subsequent evaporators. In either case, before processing, the brine is often chemically treated to remove impurities that may yield a lower-grade product or cause scaling.

Production

Much of the brine produced through solution mining in the United States is never crystallized but is used directly by chlor-alkali plants in the manufacturing of such products as chlorine, caustic, and soda ash. When the desired product is food-grade salt, vacuum pan evaporation can yield fine-grained salt that is 99.8% to 99.95% pure.

Worldwide, the chemical industry is the largest user of salt, consuming about 60% of the total salt produced. Salt for human consumption takes up 30%, and the remainder goes to road deicing, water treatment, and other minor applications.

Potash

There is a long history of producing potash from brines collected from mixed salt deposits on the earth's surface. Commercial production at Wendover, Utah, dates from 1937 (Garrett 1996). Production of potash-bearing brines from underground deposits that had no prior conventional mining production began in the early 1960s. Potash has also been recovered from flooded conventional mines since the early 1970s. In 2008, construction began on another operation to deliberately flood and recover potash by solution mining from mines in New Mexico (United States) that were closed due to prior reserve exhaustion using conventional mining methods.

Desirable Geologic Conditions and Chemistry

Typically, the types of deposits exploited are natural brines and underground evaporite deposits. In many parts of the world, natural brine sources contain sufficient amounts of potash to justify commercial production. Some operations produce potash from brine lake deposits such as the Dead Sea, Searles Lake, and the Great Salt Lake. Natural surface evaporite deposits dominated by halite at Bonneville Salt Flats in Utah contain a brine with 2.5% (potassium chloride or potash [KCl]). Potash has been produced by solution mining from the generally flat-lying, bedded sylvinite (mixed KCl-NaCl) underground evaporite deposits found in North America, particularly in Saskatchewan in Canada, and in Utah and New Mexico. Some properties have been developed solely as solution mines, while others started life as conventional underground mining operations and have been converted to solution mining after cessation of traditional mining operations.

The chemistry of the salt deposits is important in both naturally occurring brines and in minerals. Naturally occurring brines containing potash are found in many locations around the world. The most common anion in those brines is chloride, but sulfate and other anions are known. The potassium levels are typically low (often less than 1% KCl), with sodium and magnesium being the most prevalent of the other metals.

The most commonly exploited mineral containing potassium is sylvite (KCl), which is found in the mineral sylvinite, a mixture of sylvite and halite. However, some leaching operations exploit carnallite ($KCl{\cdot}MgCl{\cdot}6H_2O$) for its potash content.

Mining Method

Solution mining for potash employs three of the mining methods previously discussed: (1) mining of permeable salts; (2) mining of impermeable salts; and (3) mining abandoned or flooded conventional mines.

Mining of permeable salts. This includes naturally occurring brines containing potash that are recovered from brine lakes, salt beds, and wells. Brine lake operations, such as those at the Dead Sea and Great Salt Lake, pump brine from the lake into a solar evaporation pond system. The pond system is either created from a portion of the lake bed or uses nearby shore land that is separated from the main body of brine by human-made dikes. Initially, the brine can simply flood the solar ponds through holes in the dikes, but pumping arrangements are necessary for continued recharge of fresh brine into the system.

The Intrepid Potash operation at Wendover, Utah, gathers brine from the Bonneville Salt Flats via a system of collection canals. Intrepid has a network of canals that extends for miles into the salt flats. About 3 m (10 ft) deep and 5 m (16 ft) wide

at the bottom, these canals are dug with hydraulic excavators to allow brine to flow to pumping stations near the solar evaporation ponds. Cantilever pumps are used to transfer brine from the pumping stations to the solar ponds. This system of canals is effective, but it requires constant and extensive maintenance. Intrepid Potash has also installed brine wells drilled into the salt beds. About 10 m (33 ft) deep, these wells provide a more constant baseload flow to the solar ponds than do the collection ditches.

Mining of impermeable salts. Underground evaporite deposits can be classified as impermeable-type deposits. Two operations, one in Saskatchewan and one in the Netherlands, exploit potash-containing salt beds by means of single hole—a combined injection and extraction well—without any other preparation. Large-diameter drill holes are drilled, and downhole, multistage extraction pumps are lowered to the bottom of the target leaching area. The leaching fluid is often pumped down the annulus, and pregnant brine is pumped up a pipe hanging in the center of the well. Some wells are designed to use parallel injection and production lines contained within a large casing.

Beds have been exploited at depths ranging from 1,400 to 1,700 m (1,530 to 1,860 yd) (Garrett 1996). Leaching begins at or below the bottom of the potash bed. These operations use a blanket or pad of an immiscible fluid floating on the brine in the cavity. The immiscible liquid limits the dissolution of minerals in the roof of the cavity, which causes the cavity to expand laterally. The cavity diameters typically grow to more than 100 m (109 yd) and may even reach 150 m (164 yd). Initial leaching is conducted with water to create an open cavity. Selective leaching, using saturated NaCl brine, leaves a matrix of salts that slows diffusion of the lixiviant into the bed and the brine out of the bed. The salt matrix also can blind off completely, preventing further dissolution.

Recovery of potash minerals from an impermeable type of deposit varies. Depending on overburden characteristics and thus well-field design, the mineable area becomes limited; extraction of minerals can be as low as 30% to 40% (strictly before processing.)

As reported by Garrett (1996), Kalium (now a subsidiary of the Mosaic Company) produces potash from a sylvinite bed about 9 to 15 m (10 to 16 yd) thick, at depths of up to 1,650 m (1,804 yd) in Saskatchewan. Formation temperature is 57°C (135°F), which increases solubility. Water is injected to dissolve both the potash and salt until the cavity is more than 100 m (110 yd) in diameter. As the cavity size increases, the large surface area allows selective leaching despite the slower diffusion rates. Annual production is more than 1,800,000 t/a (2,000,000 stpy).

BHP Billiton in the Netherlands exploits a deposit dominated by carnallite, with other salts that are bounded at the top and bottom with halite. The bed averages 100 m (328 ft) thick with depths of 1,400–1,700 m (1,531–1,859 yd). Directional drilling is used to create a *seven-spot* pattern of wells in the salt bed from a single drilling site on the surface. Both the leach solution composition and the flow rate must be carefully adjusted and maintained to ensure dissolution of the potash. Production is approximately 820,000 t/a (900,000 stpy).

The Potasio Rio Colorado project in Argentina, now owned by Vale Inco, proposes to use solution mining to produce up to 5.1 Mt/a (5.6 million stpy) of potash.

Mining abandoned or flooded conventional mines. There are many examples of such mines being exploited for solution mining operations. Texas Gulf developed a conventional underground potash mine near Moab, Utah, in the 1960s, which is now owned by Intrepid Potash. Before the solution mining was initiated, mining conditions were difficult because of methane problems, a weak roof, and an undulating ore body. After several years of R&P mining, a decision was made to flood the mine and operate it as a solution mine. Extraction wells were drilled into lower elevations of the mine, injection wells were placed at several higher elevations, and the mine was flooded with lixiviant through the injection wells. Saturated brine is pumped from the production wells within the mine to solar evaporation ponds for potash recovery.

The Potash Company of America's (purchased by Potash Corporation of Saskatchewan in 1993) Patience Lake mine in Saskatchewan was a conventional R&P operation that experienced repeated problems with major water inflows that closed the mine for 6 years early in its life. Eventually, this caused the company to abandon conventional mining techniques and convert the operation to a solution mining methodology in the late 1980s. A recovery well was located at the deepest point in the mine, and 10 injection wells were drilled into the old workings at higher elevations to establish the flow through the old workings.

In 2008, Intrepid Potash began a project to convert an abandoned mine in the Carlsbad, New Mexico, district to solution mining. The former Eddy Potash Company mine had been closed for many years after exhausting the resources available for R&P mining that it controlled on its leases. As with the mines previously described, extraction wells will be drilled into lower elevations of the old workings and injection wells drilled into the higher elevations. In this operation, because of the large surface area available in the old workings, the leach solution is proposed to be NaCl-saturated brine, to selectively leach the KCl from the pillars, floor, and roof of the old mine. Pregnant brine will be treated in a solar pond system.

Processing Method

Two basic approaches are employed to treat potash containing brines: solar evaporation ponds and evaporator-crystallizers. The low concentration of potash typically found in most naturally occurring brines makes solar evaporation pond treatment a necessity because of its low cost. The use of evaporator-crystallizers greatly increases the capital costs as well as the operating costs because of higher energy requirements. Solar evaporation ponds are limited to operations having locations with sufficient land area and suitable climate for the solar pond application. In northern climates, producers typically recover potash by crystallization rather than solar evaporation, because of low ambient temperatures.

The brines produced from the underground deposits may contain high enough concentrations of potash to allow the use of vacuum evaporators and crystallizers without pretreatment. This system has the advantage of being able to produce finished potash without need for further upgrading, but there is an increased cost in the form of additional energy requirements. When a potash product of sufficient grade is produced, regardless of the method employed, common operations include debrining in centrifuges, drying, compacting, and screening to produce the various market products.

As described, naturally occurring brines are universally processed by solar evaporation as a concentration step. Some brines containing relatively high concentrations of sulfate will be treated with lime to remove most of that anion. The

low-sulfate brine is pumped into a series of shallow, flat-bottomed ponds, which are often unlined and are simply lakes that have been divided or contained by dikes. The brine is sequentially transferred from pond to pond as the water evaporates and the concentrations of the contained salts increase. With careful monitoring, different salts will saturate and precipitate in different ponds. This allows operations, such as Searles Lake and the two Dead Sea plants, to make several different products. Evaporation rates vary with the environmental conditions, so solar pond operations are generally found in areas that enjoy warm, sunny, and dry conditions for long periods during the year. Rainfall is a risk, because high rainfall can slow production, and unusual rain events have even shut down operations for a year or more.

Generally, relatively pure NaCl is precipitated in one or more stages. Depending on location, this may be a marketable product, rather than a waste. When the potash approaches saturation in the brine, it is typically sent to harvesting ponds where the potash precipitates along with sodium chloride if there is little magnesium present. At the Israeli Dead Sea operation, the potash precipitates as carnallite. Harvesting ponds are typically constructed to have a 300-to-600-mm (12-to-24-in.) thick layer of hard NaCl deposited first to build a solid floor for the harvesting process and to protect the liners, if present.

A variety of methods are used to recover the precipitated salts from the evaporation ponds. Floating dredges are used in a few operations (Dead Sea Works), but dry harvesting is more common. In some dry harvesting operations, the salt mixture is plowed into windrows with graders, loaded with elevating scrapers, and hauled to a dump pocket. In others, specialized salt-harvesting machines dig up the salt or collect it from windrows and load it into dump trucks. The harvested solar salt typically contains 24% KCl and 75% NaCl. This material is commonly upgraded with froth flotation to produce a high-grade KCl concentrate. The concentrate is leached with a saturated KCl brine to remove the remaining NaCl and is then de-brined in centrifuges and dried. Potash for fertilizer or feed supplements is usually 93% or more KCl, which is also called muriate of potash.

Brines from underground mines are exemplified by the Intrepid Potash mine at Moab, where they are treated in a system of ponds similar to those for naturally occurring brines. However, the total area of the ponds at Moab is considerably smaller than the area at a natural brine operation with a similar production rate. This is due to the higher potash concentration in the brine from the mine. These ponds are also lined with synthetic material to prevent leakage into the local groundwater. Additionally, dye is also added to the brine in some ponds at Moab to increase evaporation rates. The dyed brine absorbs more light and heat than a clear brine over a reflective white salt bed.

The Patience Lake operation uses a pond to recover potash from the mine-produced brine, but by cooling rather than evaporation (Garrett 1996). The brine pumped out of the mine is warm and, when pumped to a pond near the plant, it cools by natural convection, especially during nighttime. Potash crystallization rates in this pond are highest in the wintertime as temperatures drop. This is one operation that recovers potash by dredging. The precipitation is fairly selective, and the dredged potash can be upgraded to market standards by leaching.

Some brines with high potash content from underground mines and some solar evaporation operations are treated by a combination of evaporators and crystallizers. Triple-effect evaporators remove much of the water from the brine and in many cases precipitate NaCl. The resulting slurry is separated and the salt is washed with potash brine to recover dissolved potash. The strong potash brine is fed to vacuum crystallizers where the potash is precipitated. Although these plants are compact, compared to the solar pond operations, the energy consumed in the evaporators and vacuum crystallizers results in higher operating costs. The potash from the crystallizers generally does not need further upgrading before it is dewatered, dried, compacted, and screened into final products. The brine from the crystallizers is recycled in the plant, but a bleed stream is generally required to eliminate deleterious elements.

The most prevalent waste product from potash solution mining is sodium chloride. In many natural brine operations, the sodium chloride is a co-product with the potash. In others, the sodium chloride can be returned to the brine source. Some of the salts associated with potash production, notably magnesium chloride, have virtually unlimited solubility in water. These will produce *bitterns,* which are brines of high concentration. Some bitterns can be treated and used as a source of magnesium chloride or even magnesium metal. In many cases, disposal of the bitterns is managed by returning them to brine lakes, deep-well injection, or storage in ponds.

Production

Worldwide consumption of potash was 52 Mt (57.3 million st) in 2005. Two-thirds of its production comes from Canada, Russia, and Belarus, with only nine other countries contributing to production. Primary consumers of potash include the United States, China, Brazil, and India. The most prevalent use of potash is as a fertilizer, primarily in the form of muriate of potash, KCl. Muriate of potash is also sold as an animal feed supplement.

Magnesium

Magnesium is found in about 60 minerals, of which only a few are of commercial value. The third most common element in seawater, it is found in deposits of magnesite and dolomite, and, in soluble form, in lake brines, mineral waters, and springs. Magnesium is produced from minerals, seawater, wells, and lake brines.

Desirable Geologic Conditions and Chemistry

Much like many other soluble salts of value, magnesium is produced from evaporites and brines. The brines are typically sourced from concentrated seawater, wells drilled into salt deposits, and concentrated inland salt lakes. In addition, brines are produced from saline aquifers, oil-field production brines, and mineral springs. In salt beds, magnesium is found in minerals such as carnallite found in New Mexico, kainite found in the Stassfurt mines in Saxony, Germany, or bischofite. The primary water-soluble magnesium minerals are shown in Table 11.5-1.

Mining Method

Solution mining of magnesium, practiced for decades, has been proven to have substantial advantages over conventional mining methods, given the right deposits. Heated water is used as the lixiviant and injected into impermeable deposits, as described previously in this chapter. Naturally occurring brine simply needs to be collected from the source. US Magnesium LLC uses

Table 11.5-1 Principal magnesium minerals

Mineral	Composition
Carnallite	$KMgCl_3 \cdot 6H_2O$
Kainite	$MgSO_4 \cdot KCl \cdot 3H_2O$
Bischofite	$MgCl_2 \cdot 6H_2O$

Table 11.5-2 Principal soda ash minerals

Mineral	Composition
Trona	$Na_2CO_3 \cdot NaHCO_3 \cdot H_2O$
Nahcolite	$NaHCO_3$
Dawsonite	$NaAlCO_3(OH)_2$
Thermonatrite	$Na_2CO_3 \cdot H_2O$
Natron	$Na_2CO_3 \cdot 10H_2O$

solar evaporation to concentrate natural brine from the Great Salt Lake prior to processing and production of magnesium metal.

Recovery is based on the ore deposit characteristics, the mining method employed, and the volume of nonproduct minerals such as alumina, silica, lime, and iron oxide.

Processing Method

The processing method varies by desired product. Magnesium hydroxide ($Mg(OH)_2$) is used to produce magnesia (MgO). Magnesium chloride ($MgCl_2$) can be used as is or further processed to produce magnesium metal. Magnesium can also be a by-product of other soluble salts such as salt, potash, and borates. Because of its very high solubility in water and other brines, magnesium is typically the last salt to precipitate in solar evaporation processes.

Magnesium hydroxide. Magnesium hydroxide is recovered from brines by precipitating the dissolved magnesium as magnesium hydroxide with the addition of dolime (calcined dolomite CaO·MgO). After precipitation of calcium carbonate, the product-bearing solution is treated with sulfuric acid to remove any remaining calcium bicarbonate and then pumped to thickeners where concentrated magnesium hydroxide slurry is generated. This slurry is then filtered to produce a filter cake containing about 50% solids. The filter cake is either directly calcined to produce refractory or caustic-calcined magnesia, or it is calcined and pelletized before dead-burning to produce the desired size and density characteristics.

Magnesium chloride. To recover magnesium chloride from brines, the brine is pumped to a series of solar evaporation ponds where water evaporates, thus concentrating the brine. Generally, the first salt to precipitate as the water evaporates is sodium chloride. Double salts containing potassium and magnesium precipitate next and, dependent on other factors, kainite ($MgSO_4 \cdot KCl \cdot 3H_2O$), schoenite ($MgSO_4 \cdot K_2SO_4 \cdot 6H_2O$), and carnallite ($KMgCl_3 \cdot 6H_2O$) can be precipitated. Sodium sulfate is then precipitated, while the final product to be precipitated from the brine is magnesium chloride.

Although magnesium chloride is a directly marketable product, it can be used to produce magnesium metal by feeding high-quality, anhydrous magnesium chloride to a direct-current electrolytic cell with a very high amperage to reduce the magnesium chloride feed to magnesium metal and chlorine gas.

Soda Ash

Egyptian writings mention crude soda ash (Na_2CO_3) as early as 5,000 years ago. Natrun, crude trona, has been used as a valuable resource well before 3500 BC from the Wadi Natrun lakes in Egypt. Although natural deposits are rare, soda ash has been historically obtained by leaching wood or plant ashes. The uses of soda ash as a cleansing compound and as a flux in the making of glass were eventually discovered, and these are now its primary uses.

In 1792, the Leblanc synthetic soda ash process replaced the leaching of wood ashes. In 1865, the Solvay process made the Leblanc process obsolete because Solvay was simpler and more efficient and its by-products were less difficult to manage. In the late 1930s, the discovery of the large Green River deposit in Wyoming made natural soda ash less expensive in the United States than the synthetic soda ash obtained via the Solvay process.

Desirable Geologic Conditions and Chemistry

Using natural soda ash deposits rather than synthetic generation of soda ash from salt and limestone is preferable because the synthetic process is usually more expensive and produces more waste. About 60 natural sodium carbonate deposits are identified globally, only some of which have been assessed. Natural soda ash deposits can be defined in two manners: solid evaporites and brines.

Solid deposits of trona mineral are found in the extensive Green River Formation in Wyoming and at Lake Magadi in Kenya. Sodium-carbonate-rich deposits, in the form of nahcolite and dawsonite minerals, are located in the Piceance Creek basin in Colorado (United States). Other deposits occur as trona in dry lakes such as Searles Lake in California. The Rift Valley of northeast Africa has a series of lakes and dry lake beds that contain both solid trona and brines of various minerals, such as Lake Natron in Tanzania and the Sua Pan deposit in Botswana. The principal soda ash minerals are shown in the Table 11.5-2.

The Green River Formation is host to the world's largest deposit of trona. It is believed that about 45 billion t (49.6 billion st) of soda ash resources can be recovered through mining processes. Bed thicknesses are typically greater than 1.8 m (6 ft) thick and occur as beds that are primarily trona or with trona and halite intermixed. The Green River basin contains geologic formations created by inland lake sediments with many saline minerals.

The brines of the Searles Lake and Owens Lake deposits in California contain a series of dissolved minerals of sodium, calcium, magnesium, and sulfur, and are said to contain about 740 Mt (815 million st) of soda ash. The nahcolite deposit located in the Piceance Creek basin of northwestern Colorado contains reserves estimated at 26 billion t (29 billion st). Solution mining was practiced by American Soda for 4 years but was halted in 2004 because of economic conditions. Other deposits, such as those in northeastern Africa, have been created by the carbonatite leaching of volcanic ash or lava.

Mining Method

The conventional in-situ solution mining method most commonly employed is to create a cavity within the target ore formation and use a blanket on top of the lixiviant to control the shape of the cavity. Different methods of hydraulic fracturing of the beds are also used to increase permeability within

Table 11.5-3 Principal boron minerals

Mineral	Composition	B_2O_3, wt%
Boracite	$MgB_7O13{\cdot}Cl$	62.2
Sassolite	$B(OH)_3$	56.4
Kernite	$Na_3B_4O_7{\cdot}4H_2O$	51.0
Colemanite	$Ca_2B_6O_{11}{\cdot}5H_2O$	50.8
Hydroboracite	$CaMgB_6O_{11}{\cdot}6H_2O$	50.5
Priceite	$Ca_4B_{10}O_{19}{\cdot}7H_2O$	49.8
Probertite	$NaCaB_5O_9{\cdot}5H_2O$	49.6
Tincalconite	$Na_3B_4O_7{\cdot}5H_2O$	47.8
Meyerhofferite	$Ca_2B_6O_{11}{\cdot}7H_2O$	46.7
Ulexite	$NaCaB_5O_9{\cdot}8H_2O$	43.0
Szailbelyite	$MgBO_2(OH)$	41.4
Inyoite	$Ca_2B_6O_{11}{\cdot}13H_2O$	37.6
Kurnakovite	$Mg_2B_6O_{11}{\cdot}15H_2O$	37.3
Borax (Tincal)	$Na_3B_4O_7{\cdot}10H_2O$	36.5

the formation. The average recovery, using solution mining techniques, is about 30%, whereas recovery with conventional mining techniques is about 45%. Techniques are now being investigated to improve this recovery percentage.

Mining operations began in the Green River basin in 1949 using conventional underground methods. As economic conditions changed, new methods were considered. For example, one operation, the Granger plant, was shut down as a conventional mining facility in 2001 because of poor markets but was reopened in 2005 employing a solution mining method.

Solution mining at the American Soda project at Yankee Gulch in the Piceance Creek basin injected adequately pressurized high-temperature (177°C to 216°C [351°F to 421°F]) water to thermomechanically fracture the nahcolitic oil shale and generate a bicarbonate liquor that was further processed on the surface. Ore-body depths were 670 to 800 m (733 to 875 yd), and the cavities were approximately 150 m (164 yd) tall with diameters up to 60 m (66 yd).

Trona and nahcolite ore are solution mined with aqueous sodium hydroxide and aqueous hydrogen chloride. After processing, the brine is regenerated in an electrodialysis cell before being returned to the solution mining cycle. After the trona is captured, it is clarified, filtered, recrystallized, and placed in a centrifuge to dewater the product. After this, it is dried and packaged or shipped in bulk to the market.

Processing Method

In cases where solution mining has been used, the pregnant liquor is a carbonate solution that has been carbonated to form sodium sesquicarbonate or sodium bicarbonate. In some instances, both carbonates are formed. These carbonates are then calcined to produce monohydrate or anhydrous sodium carbonate, depending on the required soda ash product. The extraction solution is then treated with slaked lime and water to produce a weak sodium hydroxide solution to recirculate back to the mine.

Borates

Although borates are commonly found, economic deposits only occur in the Alpine belt of south-central Asia, the Andean belt of South America, the eastern part of Russia, and the Mojave Desert in California. Most borate production is by conventional mining methods.

Several types of deposits are mined to produce specific products that typically have stringent standards. Approximately 70% of global borate supplies come from sodium borates produced in North and South America, while nonsodium borates are produced in Turkey. The primary borate minerals are shown in the Table 11.5-3. Borate deposits occur as natural brines and as crystallized underground deposits. Underground ore bodies may contain natural brine within void spaces.

Mining Method

Searles Lake employs permeable-type techniques to solution mine natural subsurface brines in addition to processing weak lake brines. The Fort Cady Minerals project in the Mojave Desert mines a deep permeable-type deposit consisting of colemanite ore beds permeated with naturally occurring brine. The deposit is situated more than 400 m (437 yd) below the surface, has an average thickness of 36 m (39 yd) that extends across 110 ha (272 acres) and appears to be completely encapsulated in clay. This encapsulation allows hydrochloric acid to be used as a lixiviant to further saturate the existing brine, allowing 50% boron content to be achieved. The operation is permitted for a production of 90,000 t/a (99,000 tpy) (Hartman 1996).

In-situ mining of an impermeable deposit was tested at the Kramer deposit in Death Valley, California, in 1948. This deposit was 72% borax with an 8.8-m (28.9-ft) thick layer at a depth of 107 m (351 ft). Water was injected at 102°C to 110°C (216°F to 230°F). The experiment ran for 38 days and leached 230 t (250 st) of borax (Garrett 1998).

Processing Method

Most commercial deposits have borax, colemanite, kernite, or ulexite as the primary boron mineral in the deposit. The chemical makeup of these minerals dictates the required different processing methodology. At Fort Cady, the leach liquor is sent to solar evaporation ponds where crude boric acid is crystallized or lime is added to precipitate synthetic colemanite, with the option to regenerate hydrochloric acid with sulfuric acid, for re-injection (Garrett 1998).

At the Searles Lake Trona plant, currently operated by Searles Valley Minerals, solvent extraction is used to recover boric acid from weak lake brines and produce anhydrous borax (Pyrobor) and borate decahydrate. The Searles Lake Westend plant manufactures anhydrous borax (Pyrobor), borax pentahydrate (V-Bor), and sodium sulfate. Borax production methods can be quite varied when the brines contain a complex of soluble salts. At Searles Lake, two processing methods are used: evaporation and carbonation.

In the evaporation process, the brine is evaporated to crystallize sodium chloride, licons, and burkeite, thereby saturating the potash. Rapid cooling then precipitates filterable potash. The property of slow borate crystallization assists in separation of the potash from the borates. The liquor from the potash recovery operations is pumped to crystallizers where the boron is recovered as boron pentahydrate. The sludge from the crystallizer is thickened, filtered, washed, repulped, and redissolved to prepare a liquor, which is sent to polishing filters and vacuum crystallizers to make a high-purity deca- or penta-hydrate, which is then carefully dried and prepared for market. Boric acid and anhydrous borax are also made.

In the carbonation process, the brine with high borax and sodium carbonate is loaded into the top of carbonating towers. It is then subjected to compressed flue gas, which is

Table 11.5-4 Principal sodium sulfate minerals

Mineral	Composition
Thenardite	Na_2SO_4
Mirabilite (Glauber salt)	$Na_2SO_4 \cdot 10H_2O$
Glauberite	$Na_2SO_4 \cdot CaSO_4$
Astrakanite (bloedite)	$Na_2SO_4 \cdot MgSO_4 \cdot 4H_2O$
Burkeite	$2Na_2SO_4 \cdot Na_2CO_3$

controlled with very fine bubbles and rotating screens from distributors in the base of the towers. Carbon dioxide causes sodium carbonate (Na_2CO_3) to convert to sodium bicarbonate ($NaHCO_2$), and since the latter has very low solubility in the brine, it crystallizes, further purifying the brine. The liquor from the carbonation towers is then sent to the boron vacuum crystallizers, where borax seed crystals serve as nucleation points for growth.

Sodium Sulfate

Sodium sulfate has been used for more than 5,000 years. Records show that thenardite was used in ancient Egypt to preserve mummies while later uses included soap production and a mixture for glass. Its many applications and convenient location in playas and lake beds has made it a common industrial mineral for centuries.

Geologic Deposits

Sodium sulfate deposits can range from very large to quite small. Deposits in the Gulf of Kara-Bogaz in Turkmenistan, Searles Lake, Laguna del Rey in Mexico, Great Salt Lake, Lake Kuchuk in Russia, and the Aral Sea in Central Asia fall into the large-deposit category. Some of the smaller deposits of glauberite or thenardite in nonmarine deposits are less pure than the larger deposits. About 95% of the global sodium sulfate production comes from the larger deposits, which employ more sophisticated processing plants.

Many of the sodium sulfate deposits are found in playas, saline lakes, or brine deposits that have been developed along with a complex of other salts. One would think that sodium sulfate deposits would be formed within a mixture of salts, but, because of the steep solubility curve of sodium sulfate, mirabilite selectively crystallizes during cool evenings to form a more pure deposit that can be quite massive. In many deposits, mirabilite is then further enhanced to produce thenardite, astrakanite, burkeite, or glauberite. The chemical formulas for these minerals are shown in Table 11.5-4.

Mining Method

Although conventional mining methods have been used, solution mining is also being employed on suitable projects. Typical lixiviant is injected at moderate temperature (around 40°C [104°F]) with the mining operations determined by mineral solubility rather than by a chemical methodology of extracting the desired mineral.

One of the world's largest glauberite–thenardite deposits is located in Spain where 2.5 billion t (2.7 billion st) of equivalent sodium sulfate has been estimated. These resources are located in the central Madrid and Tajo basins along with the northeast Ebro basin.

Quimica del Rey S.A., a subsidiary of Industria Peñoles, produces sodium sulfate from subsurface brines, which are pumped from the subsurface to the surface and processed to produce sodium sulfate. The spent brine is further treated to produce magnesium oxide.

Processing Method

Equipment used to process naturally occurring sodium sulfate varies widely and includes air-cooled ponds, spray pipes, spray towers, polished-tube coolants within agitated tanks, scraped surface coolers, multicompartment vacuum crystallizers, and growth-controlled vacuum or heat exchange crystallizers. Generally, all operations are quite different up to the point where the evaporative process begins.

The production of a pure product requires that a minimal amount of insoluble material remains in the brine and that other soluble salts be minimized as well. To accomplish this, the feed brine is filtered, and an intermediate crystal sludge is prepared and then washed. The evaporators must be thermally efficient, have high capacity, and be able to operate in an economic manner in order to hold down costs. Multi-effect evaporators should be used to ensure lower operating costs, and heat sources should be selected in order to provide for lower costs. The design should reflect the fact that uniform crystals produce a higher-quality product.

Lithium

Lithium carbonate production from alkali brines started in 1966 at Clayton Valley, Nevada (United States), by Foote Mineral Company, which has been producing lithium carbonate from brines ever since. This is the only primary lithium carbonate-producing plant in North America.

Three types of brines are usually associated with lithium deposits: continental, geothermal, and oil-field. Continental brines occur where the lithium content is mainly derived from the leaching of volcanic rocks. These brines vary greatly in regard to lithium content, largely as a result of the extent to which they have been subjected to solar evaporation in concentrated solutions in lakes. They range from 30 to 60 ppm in the Great Salt Lake, where the evaporation rates are modest and dilution is constant because of the high volume of fresh water inflow. Examples of subsurface brines are Searles Lake (a former location of lithium production), Silver Peak, Nevada (a current source), and the high-altitude salars in Bolivia, Argentina, Chile, Tibet, and China, where lithium concentrations can be very high.

Geothermal brines occur as small quantities contained in brines at Wairakei, New Zealand (13 ppm Li), at the Reykanes Field (8 ppm) and other areas in Iceland, and at El Tatio in Chile (47 ppm). The most attractive known occurrences are in the Brawley area south of the Salton Sea in Southern California. Large tonnages of lithium are contained in U.S. oil-field, brines in North Dakota, Wyoming, Oklahoma, east Texas, and Arkansas where brines grading up to 700 ppm are known to exist. Other lithium brines exist in Paradox Basin, Utah.

Lithium salts are produced from brine processed in both North and South America. Lithium salts crystallize out of solution after the precipitation of sodium, potassium, and magnesium compounds. In marine evaporites, the Li:Na ratio is generally equal to or less than a 0.0001:1 ratio due to natural concentrations of natural seawater. In contrast, the average lithium concentration in seawater is about 0.174 mg/L, or 0.174 ppm. In nonmarine evaporites, the Li:Na ratio is substantially enhanced.

Lithium deposits with higher lithium and lower magnesium concentrations are more economic than other alkali

Table 11.5-5 Chemical composition of lithium brine lakes

	Salar de Atacama, Chile	Salar del Hombre Muerto, Argentina	Salar de Rincon, Argentina	Salar de Uyuni, Bolivia	Clayton Valley	Great Salt Lake	Zhabuye Salt Lake, Tibet	DXC Salt Lake, Tibet	Taijinaier Salt Lake, China	Seawater
Lithium content, %	0.15	0.062	0.033	0.035	0.023	0.004	0.12	0.04	—	0.000017
Mg:Li ratio	6.4:1	1.37:1	8.61:1	18.6:1	1.43:1	250:1	Low	0.22:1	—	7,000:1
Height, m (AMSL*)	2,300	3,700	3,700	3,653	1,300	1,280	4,422	4,475	—	0
Recoverable lithium carbonate, Mt	5.23	2.1	1.3	3.2	0.62	—	4.0	0.42	1.4	—

Source: Tahil 2007. Recoverable LCE updated based on Tahil 2008.
*AMSL = above mean sea level.

evaporites (Bartlett 1998). The ratio of magnesium to lithium is a critical factor in recovery of lithium from brines. If the level of magnesium in the brine is too high, the evaporation rate is slowed and the lithium yield is reduced. The average magnesium concentration in seawater is about 1,200 ppm, or a Mg:Li ratio of 7,000:1, making recovery of lithium from seawater uneconomical.

Mining Method

Subsurface brines are produced from wells, whereas lake brines are simply pumped to solar evaporation ponds. The brines have high concentrations of chloride ions, which are corrosive to most common metals, so materials of construction for pipe and pumps are generally plastic, polyethylene, and stainless steel. In the past, these solar evaporation ponds were not lined. The evaporation of the brine produced a layer of salt that sealed the ponds. However, present operations have to consider liners to reduce leakage of bitterns and loss of solution. Characteristics of several lithium-contained lake brines are shown in Table 11.5-5.

The depth of the ore deposit, thickness of the zone with high lithium concentrations, and the permeability of the material are of primary importance to the successful extraction of brine from the deposit. Typically, in production environments, the subsurface solutions flow by gradient into the extraction wells. Experience has shown that if brines are pumped into these systems to accelerate the capture of the lithium, they typically short-circuit, which causes the production system to fail.

A driving factor for the production of lithium from brines is the evaporation rate in solar evaporation ponds. High rainfall, high humidity, and high altitude are all detrimental to rapid evaporation of the water and subsequent concentration of the salts in the starting brine. Based on the measured lithium content of the brines, recovery of lithium carbonate is typically only 50%.

During mining, there is limited potential for gas generation, with the exception of working with oil-field brines, which underlie deposits of oil and natural gas. Because of the association with petroleum products, these brines must be carefully handled. Caution must be exercised during lithium extraction so that combustible gases are properly vented.

Most, if not all, solution mining and solar-evaporation-based lithium carbonate operations have yet to start production, so little history is available on the final reclamation of lithium production operations. However, reclamation requirements will be similar to most salt production operations and most likely will require removal of any residual sodium chloride, potash, or other materials in the ponds, removal of the dikes and liners, and leveling the area for natural drainage. The groundwater status in the immediate area will also have to be addressed and the site recontoured and revegetated to regulatory standards.

Processing Method

The process used to obtain lithium carbonate (Li_2CO_3) is known as the lime soda evaporation process. Brine is pumped from the evaporite deposits as a naturally occurring solution with little or no recharge or makeup with fresh water from the operator. The brine then flows through a series of evaporation ponds. Sodium chloride is the first to precipitate and is often harvested from the first set of ponds in order to avoid either raising the height of the pond dikes or adding ponds as they fill with precipitate. The sodium chloride may also be a by-product that can be sold, if there is a nearby market. The next series of ponds produce a mixed sodium/potassium chloride precipitate that may also be recovered for reprocessing to separate the potassium chloride (potash or KCl) as a by-product. At this point, the lithium content is 40 to 60 times the original concentration of the evaporite brine. The bittern is further processed to remove the soluble magnesium and boron, if present. Though the boron is removed by solvent extraction, the magnesium is precipitated using lime. The magnesium hydroxide and magnesium sulfate are then settled and filtered. The final bittern may be further concentrated by evaporation or, if in sufficient concentration, is precipitated with soda ash. The lithium carbonate precipitate is then filtered, dried, and prepared for market.

The primary product from in-situ mining of lithium, which is followed by solar evaporation and precipitation, is lithium carbonate. Other products that can be produced are lithium hydroxide, lithium chloride, and mixed lithium-magnesium sulfates.

Table 11.5-6 summarizes the main lithium resources, recoverable reserves, and potential chemical grade lithium carbonate production to 2020.

METALS EXPLOITED

Uranium

Because uranium was recognized in the 1940s as having significant value in several applications, underground and surface mining were employed to recover this material in quantity. As the industry was founded in the western United States, ores were treated in a number of conventional surface milling facilities. In the late 1960s in Wyoming and Texas, the uranium industry pioneered the commercial use of in-situ acid leaching. In the mid- to late 1970s and 1980s, in-situ leaching (ISL)

Table 11.5-6 Lithium carbonate production

Deposit Lithium Brines	Resource (Li metal), Mt	Reserve (Li metal), Mt	Current Li_2CO_3 Production, t, 2007	Probable Li_2CO_3 Production, t, 2010	Optimum Li_2CO_3 Production, t, 2015	Optimum Li_2CO_3 Production, t, 2020
Salar de Atacama	3.0	1	42,000	60,000	80,000	100,000
Hombre Muerto	0.8	0.4	15,000	15,000	20,000	25,000
Clayton Valley	0.3	0.118	9,000	9,000	8,000	8,000
Salar de Rincon	0.5	0.25	—	10,000	20,000	25,000
Salar de Uyuni	5.5	0.6	—	—	15,000	30,000
Zhabuye	1.25	0.75	5,000	10,000	20,000	25,000
Taijinaier	1.0	0.5	10,000	20,000	40,000	50,000
DXC	0.16	0.08	—	5,000	5,000	5,000
Salar de Olaroz*	0.32	0.16	—	—	5,000	5,000
Salton Sea	1.0	—	—	—	5,000	10,000
Smackover†	1.0	—	—	—	10,000	25,000
Bonneville	—	—	—	—	—	—
Searles Lake	0.02	—	—	—	—	—
Great Salt Lake	0.53	—	—	—	—	—
Dead Sea	2.0	—	—	—	—	—
Total			81,000	129,000	228,000	308,000

Source: Tahil 2008.
*Argentina
†Arkansas (United States).

techniques were largely perfected and proven to be economically feasible, and the technology became universally acceptable and employed in sandstone-hosted uranium deposits. At the same time, ISL techniques were also developed in the Soviet Union, utilizing similar approaches in slightly different geological environments. A few decades later, Australia also started to use the ISL technology for uranium recovery, and presently Kazakhstan is a major producer of uranium by ISL. The overall process of solution mining is now largely known as in-situ recovery (ISR).

Several types of uranium deposits are suitable for exploitation by ISL/ISR technologies. Of these, sandstone-hosted and paleochannel-type (as mined in Australia) deposits are amenable to ISR mining. These deposits typically occur as roll-front or tabular deposits and account for about 18% of known uranium resources. Ore bodies that can be economically recovered range in size from a few million pounds to hundreds of million pounds and are typically low- to mid-grade (0.04% to 0.15% U_3O_8), although higher grades are certainly possible. Typical uranium minerals in sandstone deposits include uraninite (UO_2), coffinite ($U(SiO_4)\cdot 0.5(H_2O)$) and carnotite ($K(UO_2)_2(VO_4)_2\cdot H_2O$). Often, vanadium that occurs with the uranium can also be processed and recovered. Conversion factors for uranium species are given in Table 11.5-7.

In the United States, the ISR of uranium from sandstone deposits started in the 1950s with research into the heap leaching of run-of-mine ores. Eventually, it progressed to working subsurface deposits of uranium when mining conditions were favorable. Generally, flat-lying deposits with groundwater within confined ore zones and with good permeability are necessary prerequisites. Economical ore zones can range from a meter to tens of meters thick. Good permeabilities are typically considered to be better than 0.5 darcys, but deposits with lower darcy ratings have been successfully operated. After the ore deposit is drilled to determine thickness and uranium content, maps are generated to depict the grade multiplied by the thickness in order to prepare a grade-thickness map

Table 11.5-7 Uranium species conversion factors*

Species	Multiply by
U to U_3O_8	1.1792
UO_2 to U_3O_8	1.0395
UO_3 to U_3O_8	0.9814
UO_4 to U_3O_8	0.9294
UO_2SO_4 to U_3O_8	0.7667
$UO_2(SO_4)_2^{\{2-\}}$ to U_3O_8	0.6074
$UO_2(SO_4)_3^{\{4-\}}$ to U_3O_8	0.5028
$UO_2(CO_3)_2^{\{2-\}}$ to U_3O_8	0.7196
$UO_2(CO_3)_3^{\{4-\}}$ to U_3O_8	0.6237
$UO_4\cdot 2H_2O$ to U_3O_8	0.8303
$UO_4\cdot 2H_2O$ to U	0.7041

*For example, 1 kg U is equivalent to 1.1792 kg U_3O_8.

(in Kazakhstan, a similar method is used to develop a *productivity* map, which shows metal per square meter) that is used to guide evaluation and planning exercises. The minimum acceptable economic grade thickness is determined by evaluating economics, including the depth of ore zones, price variations, and establishing what ore can be mined by in-situ techniques at that time after considering overall project economics. After this minimum is established, the ore resources exceeding this amount, if the ore bodies are of sufficient size, are included in the mining plan.

Because it is critical to understand the leachability of the target uranium ore zones, metallurgical tests are conducted. Simple bottle roll tests, or column leaches, can be used to obtain an idea of potential optimal leachability, but this needs to be followed by complementary groundwater studies to establish groundwater quality, permeability, and other in-place characteristics. The overall geologic framework must also be fully defined to understand the structural geology, fracturing, definition of confining layers for the target groundwater regime, and, of course, both valuable and gangue mineral

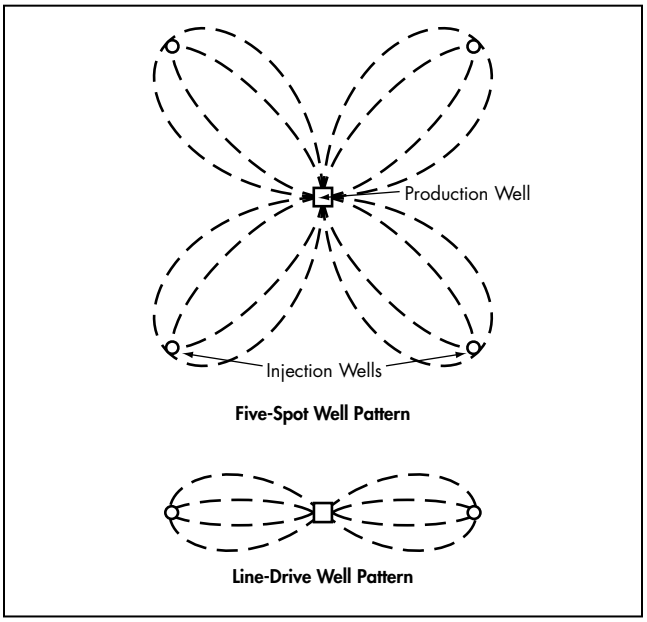


Figure 11.5-2 Well patterns

content. It may be necessary to conduct more in-depth mineralogical and metallurgical studies to define other constituents, types of clays, types of grain cement, and other items of interest. For example, titanium might be found, which is indicative of branerite; or clays or gypsum that create permeability problems; or highly reducing environments that must be overcome for effective in-situ leaching to be conducted.

Mining Method

The mining concept is to drill holes into the ore body and inject a lixiviant and then extract the pregnant lixiviant from a central production drill hole that is surrounded by the injection drill holes. Figure 11.5-2 provides a graphic of a five-spot and a line-drive production pattern, while Figure 11.5-3 shows the injection and production concept for a single target bed with a line-drive pattern. The type of pattern is dependent on the deposit shape, access, and the specifics regarding geohydrology. The specific design of the well-field pattern is dependent on the ore zone width, which can be tens of meters to hundreds of meters wide; the number of ore zones in the vertical ore zone to be mined; and the thickness of these ore zones. A third pattern, which is similar to the layout of the five-spot pattern, uses six injection wells so that a seven-spot pattern is created. This pattern is generally used in the United States when permeability of the ore zone, and therefore lixiviant flow, is low and more wells are needed for better lixiviant distribution throughout the ore zone. In Australia, seven-spot patterns are often used in very high permeability and/or high-grade zones. The additional wells help to ensure better contact and allow for the ability to reverse flow directions to stimulate production rates and obtain the highest uranium recovery possible. Figure 11.5-4 shows the transmission of pregnant and fresh/barren lixiviants to and from the ore body.

Economic ISR overall uranium recovery factors can range from 50% to 90%, depending on the deposit and time allowed for the mining process. Typically, recovery factors in the range of about 60% to 80% are common in the United States, while Kazakh well fields have often been operated with much higher recoveries, despite being uneconomic, and lower recoveries have historically been achieved in Australia. However, evaluation of recovery factors is not a simple task and requires in-depth knowledge of the ore calculation methodology to ensure that the ore in the margins of the roll front, the nose, and the tails, which may be included in the mine plan, can in fact be economically recovered. Some portions of a deposit might not be accessible to leaching because the uranium could be contained in lower-permeability units and/or clays. To further complicate this issue, detailed drilling is necessary to establish the most accurate location and characteristics of an ore body.

Both acidic and alkaline lixiviants are used to extract uranium from sandstone and paleochannel deposits. Alkaline lixiviants are prevalent in North America, while acidic lixiviants are common in Kazakhstan, other former Soviet Union countries, and Australia. Alkaline lixiviants place fewer metals into solution, and leave the well field easier to reclaim once the mining process has been completed. These lixiviants are generally composed of sodium carbonate and sodium bicarbonate, and many include oxygen or an oxygen-bearing compound that will oxidize the uranium to a soluble state.

Sulfuric acid is generally used as an acidic lixiviant, which is more powerful and places more metals into solution, generally leads to faster leaching rates, and can be necessary in specific types of deposits. The choice of lixiviant depends on the metallurgical characteristics of the deposit (specifically acid-consuming gangue), the economics associated with recovery rates, the groundwater regime, and the sociopolitical acceptability of using an acidic lixiviant. In addition, it is necessary to consider other groundwater uses and class of use within the region, as well as the potential for self-amelioration of the deposit once mining has ceased.

Processing Method

Uranium contained in the pregnant lixiviant extracted from the ore body is pumped to the processing plant where it first undergoes ion exchange to load the uranium onto the resin; the now barren lixiviant is recycled to the well field. The loaded resin is stripped of the captured uranium through an elution process. The eluate is subjected to precipitation and filtering processes, which generate yellow cake slurry that is dried in a low-temperature vacuum dryer, dehydrating the product while controlling emissions by capturing particulates. After drying is completed, the yellow cake is packaged and stored for shipment. In some cases, where salinity precludes the use of ion exchange, solvent extraction methods may be utilized.

The recovery of uranium within the processing plant is about 98%. The design of the plant is typically specific only to uranium, but other by-products can be produced. Uranium deposits typically contain other metals, such as molybdenum, vanadium, and copper, and it may be possible to economically extract these metals from lixiviant solutions. The overall processing plant can be designed to include modules to either remove or recover these metals as is economically feasible.

The uranium production process generates waste products. Under U.S. regulations, these solid waste products are classified as 11(e)(2) material and must be disposed of in a Nuclear Regulatory Commission–licensed disposal facility. The primary methods for liquid waste disposal are by deep disposal wells or by evaporating the liquids and disposing of the solids under the directives of regulatory requirements. In Australia, liquid wastes are disposed of in the mining zones

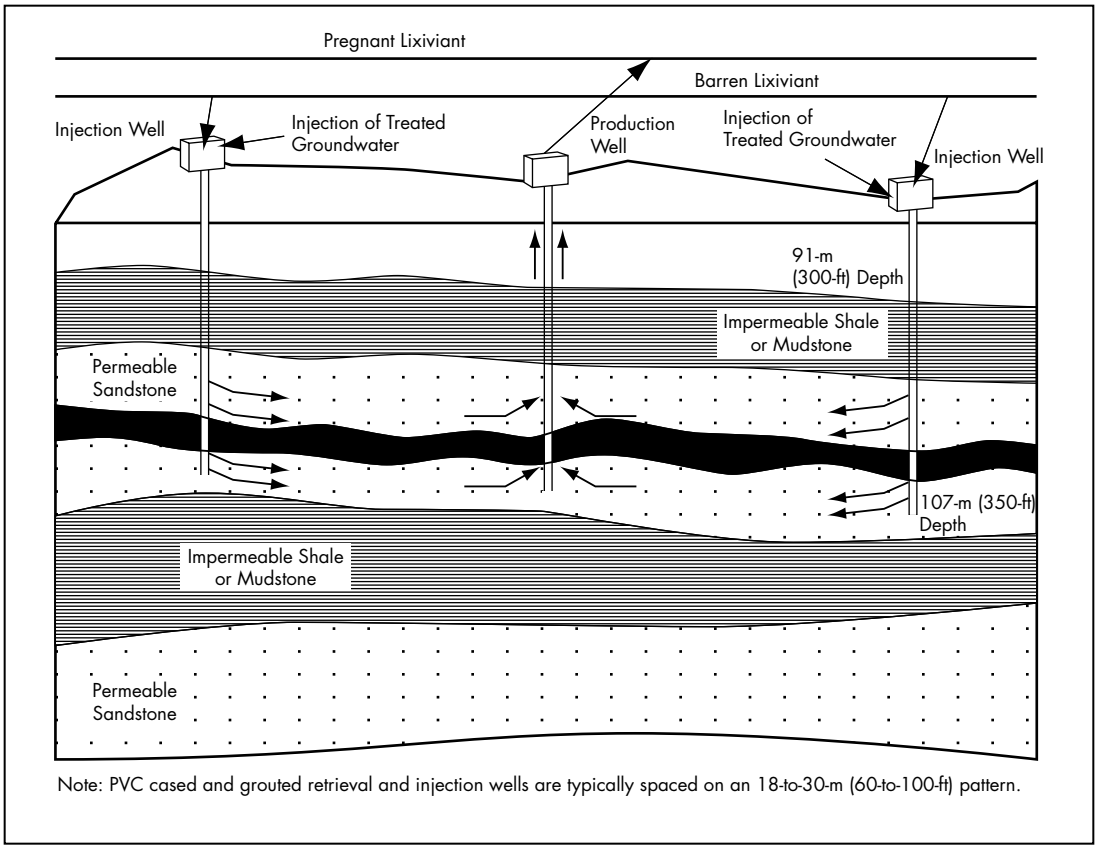


Figure 11.5-3 Cross section of uranium production wells

and solid wastes are disposed of in lined disposal facilities under the directives of the state or territory governments.

Groundwater Restoration

The ideal goal of groundwater restoration is to return the production zone (the area of an aquifer where uranium extraction occurs) to premining conditions. In the event this is not feasible, the Nuclear Regulatory Commission regulations allow for the achievement of restoration to the aquifer's quality of use class prior to mining operations. These standards have been deemed to be protective of public health and the environment.

The type of reagent used determines types of contaminants to the aquifer. ISR using acidic reagents causes a drastic decrease in pH and dissolution of other minerals in addition to the target uranium ores, concentrating most of the elements present in the surrounding rock to levels above the maximum admissible for drinking water supply. Microbes in the sulfide environment of the leach zone can produce a reducing environment, which may cause metals to precipitate in their sulfide forms. Meanwhile, bicarbonate reagents selectively dissolve the uranium ore, and contamination is noticeably lower. However, radium, selenium, and arsenic are still issues that must be addressed during remediation.

Two approaches can be used for groundwater restoration, direct cleaning and self-cleaning, as described in Table 11.5-8.

In Kazakhstan, the mining aquifers are allowed to attenuate naturally, whereas in Australia, some aquifers have been considered unusable for human or animal consumption and hence require no restoration. More recently, however, some of the proposed ISR projects are in areas that may have better quality and more valuable groundwater resources. In these areas, it will be necessary to perform some measures of restoration to at least initiate increases in pH and return to class of use.

Current Production

According to the World Nuclear Association, primary global uranium production for 2008 was approximately 52,000 t (114 million lb) of U_3O_8 (World Nuclear Association 2010). Primary producers include Canada at 10,600 t (23.4 million lb), Kazakhstan at 10,000 t (22.1 million lb), Australia at 9,900 t (21.9 million lb), Namibia at 5,200 t (11.4 million lb), Russia at 4,200 t (9.2 million lb), Niger at 3,600 t (7.9 million lb), and the United States at 630 t (1.4 million lb). Most of the production from the United States and Kazakhstan comes from ISR production, which is recognized as the most cost-effective and environmentally friendly method of uranium production.

Uranium is primarily used in nuclear reactors throughout the world. Total demand forecast, which for 2009 was 77,000 t (170 million lb) U_3O_8, is met by both primary production from mining operations and secondary supply, which includes drawdown from inventories and highly enriched uranium sources. Nonetheless, most researchers forecast that demand will continue to outstrip supply unless more uranium mines are placed into production or expanded.

FRASCH SULFUR MINING

The process by which elemental sulfur is produced from a geological formation is singularly credited to Herman Frasch.

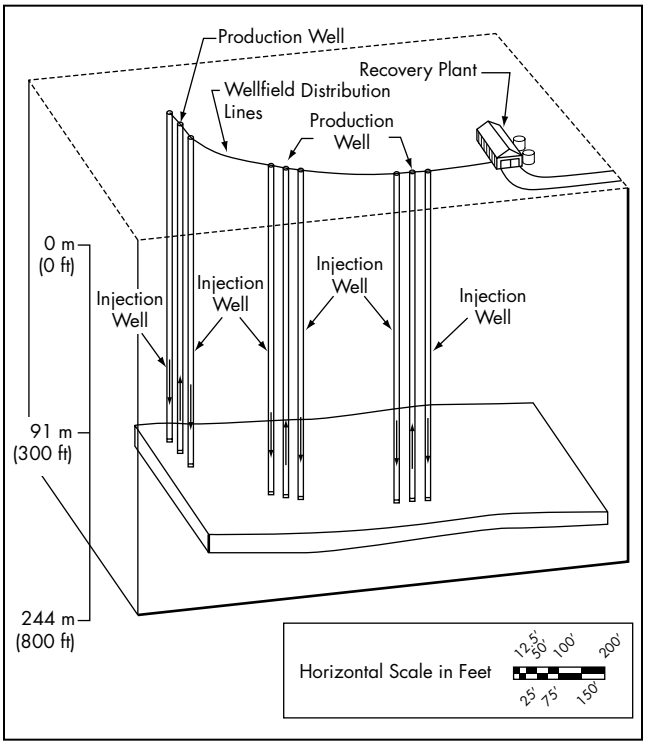


Figure 11.5-4 Schematic of ISR facility

Table 11.5-8 Groundwater restoration methods

Method	Description
	Direct Cleaning
Cleaning by reverse osmosis (RO)	• Varying number of pore volumes drawn to surface RO unit • Permeate recycled back underground • Brine sent to disposal • Most common method in United States
Washing with formation water	• 5–10 pore volumes drawn to surface • Depleted volume restored by adjacent natural groundwater
Cleaning by precipitation with reagents	• 5–10 pore volumes pumped to surface for uranium recovery and precipitation of undesirables with lime pulp • Saline residuals buried • Only recommended when other methods not suitable
	Self-Cleaning
Method of natural attenuation	• Given time—tens to hundreds of years—natural groundwater current restored to mined area • Rate dependent on natural flow rate, residual compositions, minerals present in rock, and acid versus alkaline reagent
Methods for accelerating natural attenuation	• Increase flow rates by pumping groundwater from beyond ore-body limits; introduce material to encourage natural bacteria growth; e.g., cheese whey, and molasses

Source: Data from IAEA 2001.

His concept of the mechanism of melting sulfur and then airlifting the molten liquid to the surface required a persistent effort in technical development and an expenditure of time and capital for a span of more than 10 years before realizing significant success.

The initial production of sulfur by the airlift (Frasch) process began in 1896 with the operation of three wells that produced 2,134 t (2,352 st) of sulfur (Haynes 1942). From a worldwide standpoint, however, the Frasch process is applicable to situations where fuel and water are available and the geological circumstances are favorable—for example, the Middle Eastern petroleum production areas, Poland, and Russia.

Basic Requirements for Frasch Sulfur Mine

Three basic resources are needed to develop a profitable Frasch sulfur mine:

1. Large deposit of ore of good grade (5% sulfur is suggested as a lower limit)
2. Adequate, reliable, and inexpensive source of water, preferably containing little dissolved solids
3. Low-cost source of fuel

Other factors such as market price, royalty payments, tax liabilities, and transportation costs also influence the economic feasibility of the project. Utilities that mine sulfur have been quoted requirements for

- Average water: 19.2 m^3/t (5,000 gal/long ton) of sulfur mined;
- Fuel: 20,000 to 22,300 Mm^3 of gas/m^3 (2,700 to 3,000 million ft^3 of natural gas/million gal) of mine water; and
- Power: 118 to 138 W/m^3 (600 to 700 hp/million gpd) of mine water (*Oil and Gas Journal* 1967).

Reliable and economic transportation systems are also essential for distribution of the bulk sulfur. In this regard, transportation and storage of molten sulfur has been growing, particularly where low-cost water transport can supply shore-based distribution terminals. Sophisticated production personnel are needed to operate and maintain the production and delivery systems.

Ore Deposit

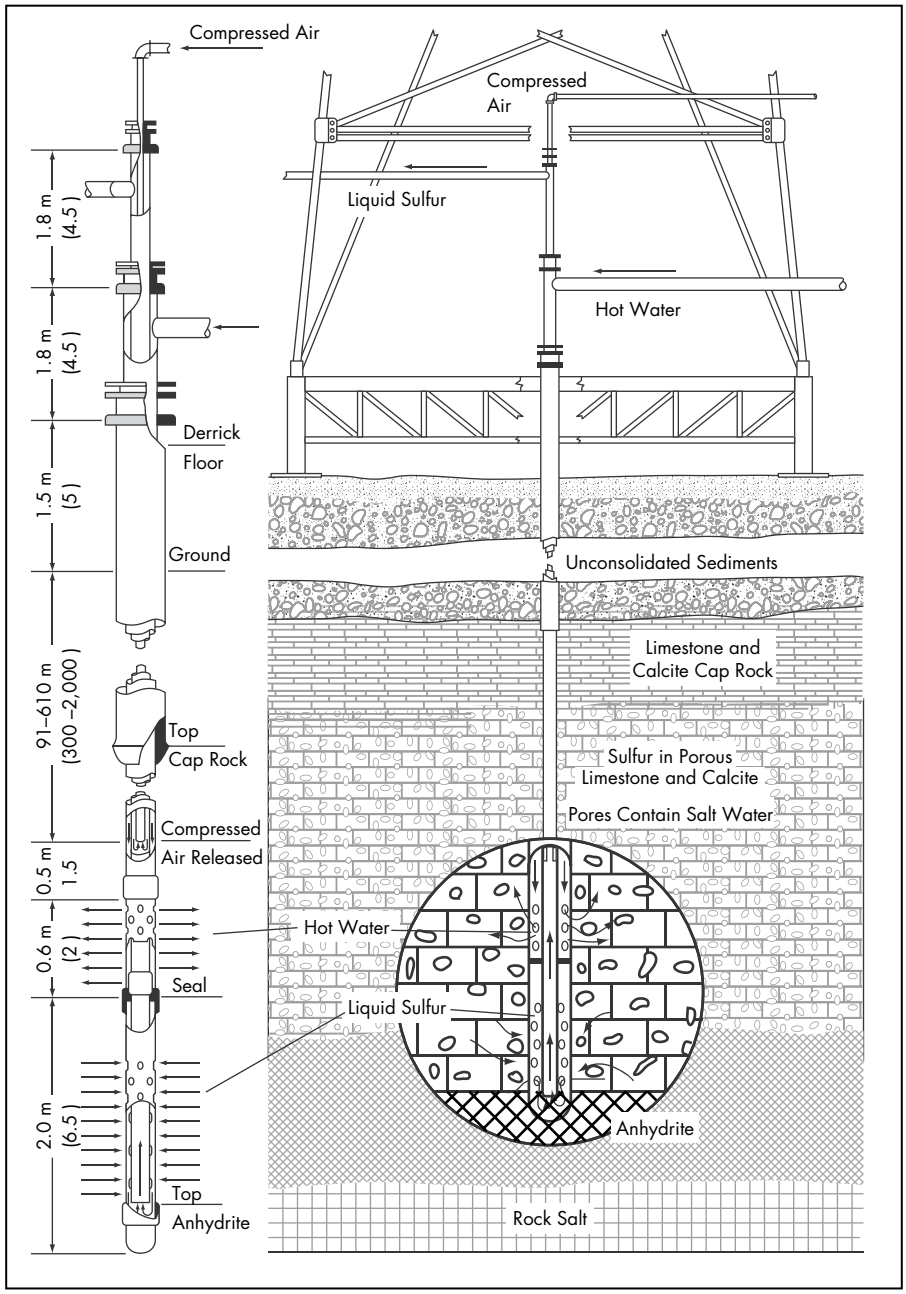
The Frasch process was originally applied to removing sulfur from deposits associated with salt domes formed by the intrusive flow of deep-bedded salt that penetrates the upper layers of sediments. Typically, the sedimentary layers are mud, shales, limestone, gypsum, and anhydrite. The salt plug normally is covered by an insoluble mineral cap, below which the sulfur is found in the vuggy porosity of the limestone. The sulfur location in the cap rock of the domes is highly variable, and not all salt domes contain sulfur-bearing rock.

Sulfate-reducing bacteria in conjunction with hydrocarbons and water are clearly capable of producing elemental sulfur. Thus sulfur can be formed from the gypsum and anhydrite and be precipitated in the porous limestone with the progression of water circulation.

The west Texas deposits of sulfur are associated with layers of gypsum, anhydrite, limestone, and dolomite. These deposits are not associated with salt plugs, and the porosity and permeability of sulfur-bearing beds follow the natural layered porosity of the strata and fracture system (Zimmerman and Thomas 1969; Hentz et al. 1987).

Field Development

Efficient extraction of sulfur from the deposit depends on several interrelated factors. The hot water has to be introduced in such a way and at such a rate as to melt the sulfur so it will

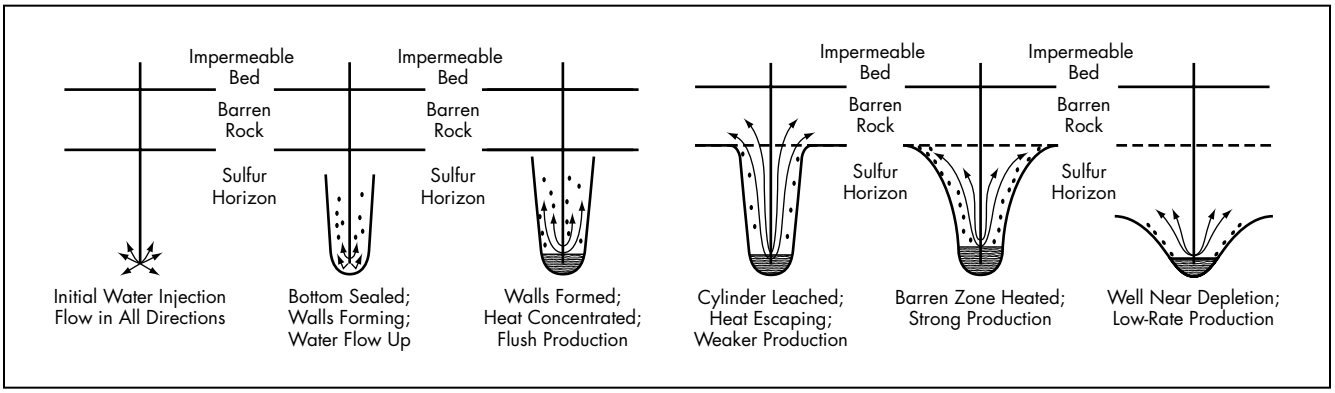


Source: Adapted from Ellison 1971.

Figure 11.5-5 Operation of a sulfur well

form a pool below the hot water, which is then airlifted to the surface. The piping system consists of concentric pipes which extend into the mineable formation (Figure 11.5-5). The operating strings are a series of these concentric pipes. The outer casing is set into the cap rock above the sulfur-bearing strata. The next string conducts the hot water and is set through the sulfur-bearing strata and is perforated at the bottom and also at some distance up the pipe. Another pipe is set concentrically within this pipe, and an internal seal is placed between the upper and lower perforations so that the flow of hot water is above the lower level. At a higher level, another pipe is run concentrically. The piping diagram in Figure 11.5-5 shows how hot water is introduced first, melting the sulfur, which is heavier, sinks to the bottom, runs into the lower pipe, and fills the second string. The molten sulfur is then airlifted with the compressed air and emerges as shown in the figure.

The molten sulfur is sent to storage, which can be handled in a number of ways. Since it may contain solid materials, or even small amounts of acid hydrocarbons, further processing may be required. Steam-heated leaf filters are commonly used for sulfur filtration. Precoating, usually with diatomaceous earth, is almost universal. The precoat should do the actual filtration to minimize blinding and will also neutralize some acid and reduce corrosion. However, small amounts of hydrated lime or ammonium bicarbonate may be added to the filter feed to neutralize acid. Such a system will produce



Source: Adapted from Donner and Wornat 1973.

Figure 11.5-6 Sequence during operation of sulfur well

sulfur that is free of ash and acid. If hydrocarbons must also be removed, activated clay may be added to bleach the dark sulfur. Other filter aids are also available to remove hydrocarbons. Filtration rates and precoat requirements will depend on the quality of the sulfur coming from the wells.

The sequence of the development of the well is shown in Figure 11.5-6. The initial injection melts a pool of sulfur and seals the bottom, as shown. Hot water is forced farther up the string, and a larger cylinder of rock is heated, melting more sulfur, which is airlifted to the surface. This yields a concentrated flush production. This scenario is simple in concept, but in practice, where variability of permeability, sulfur content, heat flow problems, and the influence of other wells exist, the actual operation is complex. It is necessary to reinject water produced in bleed wells to control flow, save heat, and dispose of wastewater. The mining process is highly variable because of the nature of the deposits, and the uncontrollable nature of the reservoir system makes the process something of an art that is gained by years of experience. Attempts have been made to apply reservoir engineering by substituting paradichlorbenzene as a meltable material for sulfur with some interesting insights (Rayne et al. 1956).

As mining proceeds, subsidence and collapse of overburden become a problem that can sometimes be helped by reducing the volume of hot water necessary, but may also shear tubing and cause many difficulties. Wells are often separated into groups to help control these problems and avoid total loss of field control. Corrosion due to molten sulfur contact is quite negligible. When the sulfur pool is drawn down to where the hot water and air come up together, the water flashes as steam, and a *blow* takes place. Hot water rising up the sulfur production string produces a corrosive environment, and, with a blow, it becomes appreciable. The salinity of the water influences the rate markedly. If seawater is used and a blow occurs, the corrosion rate is disastrous (Hackerman and Shock 1947; Shock and Hackerman 1949). One of the practical solutions to controlling corrosion has been to use specially produced cement-lined pipes as well as limiting the salinity of the water. Treatment of the water to minimize corrosion and scale problems is essential.

Bleed wells are required to balance the water pumped into the formation at the production wells. Location of the bleed wells will depend on such site-specific factors as the depth, permeability, and hydrology of the sulfur-bearing structure. Because of environmental concerns, bleed water should be recycled to the maximum extent possible. When required, disposal of bleed water needs to be carefully planned because of the high concentration of dissolved hydrogen sulfide and other sulfur-bearing compounds. Aeration in ponds, chemical treatment, and use of deep disposal wells are all methods of handling these waters.

Facilities Needed at Frasch Sulfur Mine

When a sulfur deposit has been located and defined by exploration drilling, a water supply located, and a fuel supply established, the design and construction of the needed facilities can proceed. Typical layouts showing both the well field and surface facilities are provided in the literature (*Oil and Gas Journal* 1970). Most mines will require some or all of the following:

- Pumps, pipelines, and reservoir for the water supply
- Power plant for heating and treating the water, compressing the air, and generating the electricity and steam
- Equipment for drilling and servicing wells
- Pipeline system for carrying the mine water and air to the wells and the sulfur to collecting stations
- Control stations for the producing wells
- Sulfur collecting, loading, and storage facilities
- Purification equipment needed to meet customer specifications
- Shipping facilities for solid and/or liquid sulfur
- Bleed-water disposal system
- Equipment needed for the maintenance of these facilities
- Employee transporting, dining, and housing facilities if the mine is remotely located
- Offices and buildings for material storage, laboratory, engineering, personnel, administration, and supervisory staff

ACKNOWLEDGMENTS

The authors recognize the work of D'Arcy A. Shock for his contribution of the section on Frasch sulfur mining, which has been reprinted (with minor reductions on history) from the previous edition of this handbook. Minor alterations have been made to convert to the International System of Units.

Additionally, coworker Edwin Bentzen, project manager at Lyntek, Inc., is thanked for his contribution of the section on lithium, as are the draftsmen turned illustrators who helped with the figures; Rachel Hawkins who offered moral support; and Nick Lynn, owner of Lyntek, who gave us the opportunity to complete the work.

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CHAPTER 11.6

Coal-Bed Methane Production

Pramod Thakur

INTRODUCTION

Coal seams were formed over millions of years by the biochemical decay and metamorphic transformation of the original plant material. This process, known as coalification, produces large quantities of by-product gases. The volume of by-product gases increases with the rank of coal and is the highest for anthracite at about 765 m^3/t (Hargraves 1973). Most of these gases escape to the atmosphere during the coalification process, but a small fraction is retained in the coal. The amount of gas retained in coal depends on a number of factors such as the rank of coal, the depth of burial, the immediate roof and floor to the coal, geologic anomalies, tectonic pressures, and temperatures prevailing at the end of the coalification process. In general, the higher the rank of coal and the greater the depth of the coal seam, the higher the gas content. Gas contents of coal seams generally range from 0.1 to 25 m^3/t.

Methane is the major component of gas in coal, accounting for 80%–95% of the total gas content. The balance is made up of ethane, propane, butane, carbon dioxide (CO_2), hydrogen, oxygen, and argon. Because the coal-bed gas is mainly methane, it is generally called coal-bed methane (CBM). Methane recovered from active and abandoned coal mines is called coal mine methane.

Coal-bed methane production (CMP) from longwall gobs started in Europe in the 1940s, but methane drainage from virgin coal prior to mining began in the United States in the 1970s. The primary need for CMP was to make the coal mines safe from explosion hazards. The methane gas control techniques are discussed in Chapter 15.4 in detail. As the CMP process developed and became widespread, other benefits became apparent. In moderately gassy and very gassy mines, immediate improvement in coal productivity was realized. In some very gassy mines in the United States, the productivity improved from a low 15 t/worker-shift to 40 t/worker-shift. It significantly reduced the cost of mining by minimizing the gas delays caused by excessive gas emissions. In the 1980s, the volume of CBM produced prior to mining and postmining continued to increase, eventually leading to gas processing and commercial marketing. Extensive drilling of nonmineable coal deposits, such as CMP activities in the San Juan Basin of the United States, also produced a large volume of CBM. At present, nearly 10% (51 Gm^3 [billion cubic meters]) of U.S. natural gas production is from CMP. This number is likely to increase to 20% in the near future because CMP is less cost-intensive and financially less risky than deep drilling for natural gas production. The CBM, after processing to remove impurities, is exactly like natural gas with a calorific value of 35.8 MJ/m^3 (million joules per cubic meter).

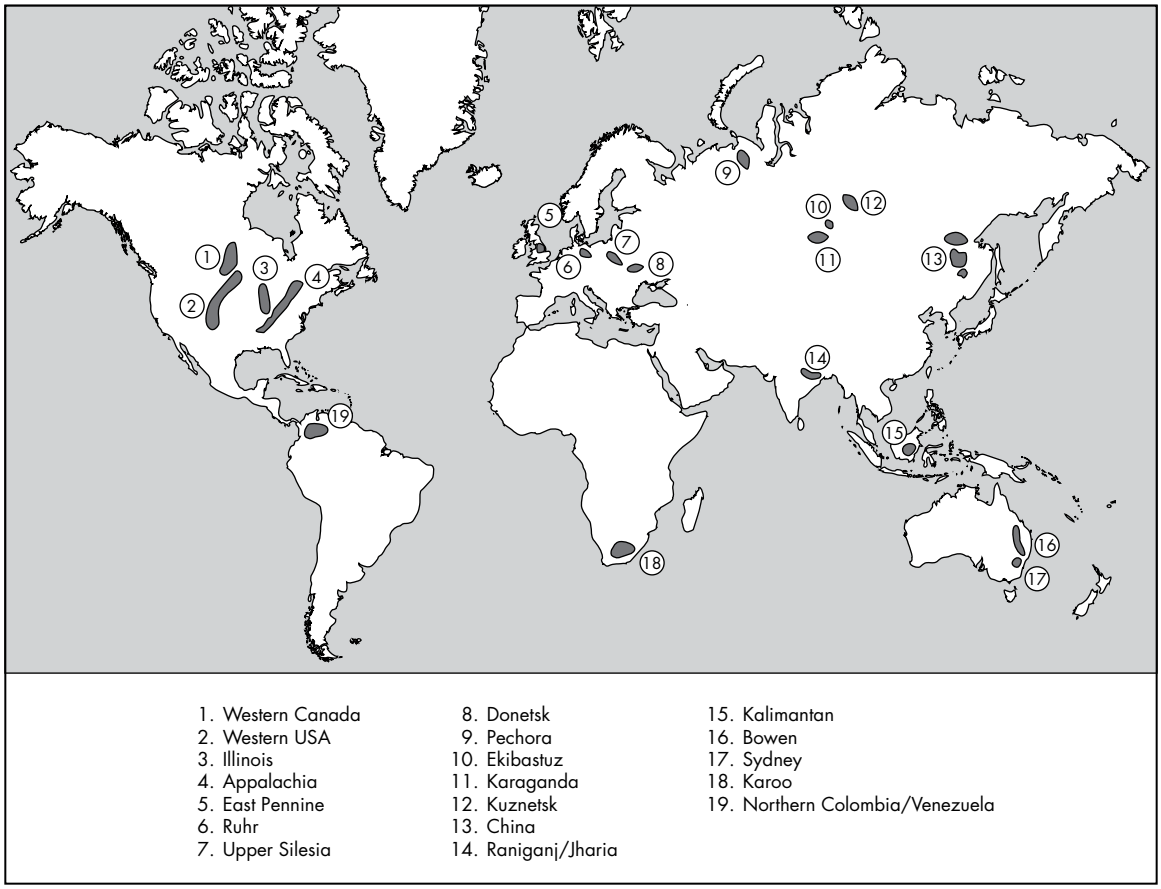
Another financial incentive is driving CMP forward. Methane gas is a powerful greenhouse gas (GHG) with a radiative forcing that is 21 times higher than carbon dioxide (*radiative forcing* is a phrase used to measure the capacity of a gas to trap infrared energy). In many industrialized countries (e.g., the United States and Australia), a financial incentive is provided to capture CBM from active mines. The current U.S. incentive is $6.50/t of carbon dioxide. Thus, 1 t of CBM can qualify for a carbon credit of $136.50 if it is captured and marketed.

CMP today is a standalone, profitable enterprise. Besides saving lives and reducing the cost of coal production, it provides a clean-burning fuel and reduces GHG emissions. With adequate financial incentives, it can become a major supplier of natural gas in many countries such as the United States, China, India, and Australia.

COAL-BED METHANE RESERVES

As illustrated in Figure 11.6-1, coal deposits occur in 19 major basins around the world, and global reserves of coal and CBM are large. Coal has been a global energy source for a long time, but for the past 200 years it has played a vital role in the growth and stability of world economy. At present, 70 countries around the world annually mine about 6,500 Mt of coal. China, the United States, and India, the three largest coal producers, derive most of their electricity from coal. The total proven world reserve of mineable coal is 1 Tt (trillion metric tons), but indicated reserve to a depth of 3,000 m ranges from 17 to 30 Tt (Landis and Weaver 1993). Estimated reserves of CBM range from 78–959 Tm^3 and are shown in Table 11.6-1.

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Source: IEA 1993.

Figure 11.6-1 Major coal basins of the world

Table 11.6-1 Global coal-bed methane reserve estimates*

Country	1990 Estimate, Tm^3†	1997 Estimate, Tm^3‡
Australia	8–14	—
Canada	6–76	92
China	20–35	31
Germany	3	2.83
India	1	0.7
Poland	3	0.4–1.5
Russia	20–166	720–790
South Africa	1	—
United States	11	30–41
Other countries	5–10	—
Total	78–320	877–959

*Recoverable reserve: 30%–60% of the total gas in place.
†Data from Kuuskraa et al. 1992.
‡Data from Cairn Point Publishing Company 1997.

CMP techniques were originally developed in the United States but are now prevalent in the rest of the world. At present, most CMP is realized from relatively shallow (<1,000 m deep) mineable coal deposits. However, coal deposits reach a depth of 3,000 m in many basins. Deep coal deposits are rich sources of CBM, but production techniques are not very successful yet because of very low permeability. New techniques are currently being developed (Thakur 2002). The vast global deposit of deep coal can be used in three ways:

1. CMP from deeper horizons (1,000–3,000 m)
2. Sequestration of carbon dioxide, a GHG. Production of CBM from deeper formation is assisted by CO_2 flooding. Coal seams have a much higher affinity for CO_2; hence, they absorb CO_2 and at the same time release methane.
3. Underground coal gasification (UCG). Coal seams not used for CO_2 sequestration can be burned in situ with a limited supply of oxygen to produce low-calorific-value combustible gases. In-situ combustion of coal also greatly enhances methane production by heating the coal formation.

Thus, there is a synergy in methane production from deep coal seams, CO_2 sequestration, and UCG. The subject will be discussed in the order that a typical CMP project is undertaken:

- Reservoir properties
- Wire-line logging
- Production technology
  - Shallow coal
  - Medium-depth coal
  - Deep coal
- Well servicing and maintenance
- Produced water disposal

- Gas gathering and processing
- CO_2 sequestration
- UCG

COAL-BED RESERVOIR PROPERTIES

The most important reservoir properties that not only influence the gas production rate but also determine the production techniques are

- Gas content of coal seams and their gas isotherms;
- Permeability;
- Reservoir pressure;
- Diffusivity of coal;
- Water content and quality of water; and
- Ground stresses and elastic properties of surrounding strata (e.g., compressive strength and Young's modulus of elasticity).

Coal porosity is generally low (1%–4%) and does not have a significant influence on gas production or production techniques.

Measurement of Gas Contents

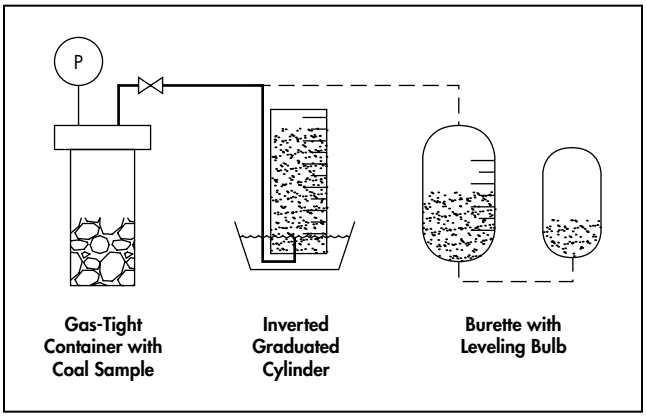
The volume of gas contained in a ton of coal is termed *gas content* of the coal and is expressed in cubic meters per metric ton (m^3/t). It is generally accepted that the gas is stored in a monolayer on the micropore surfaces of coal. The volume of gas contained in coal is dependent on the rank, temperature, and pressure or depth of the coal seam. The micropore surface in coal is large; a ton of coal has a surface area of approximately 200 Mm^2. One cubic meter of coal can store two to three times the amount of gas contained in a typical sandstone reservoir for natural gas of the same volume.

For commercial gas production, it is best to core drill the entire field on a grid pattern and do a direct measurement of gas contents of all coal seams that comprise the gas reservoir. A typical spacing for core holes is one in every 200 ha.

Gas content measurement methods are classified as (a) conventional and (b) pressurized desorption techniques. In the conventional technique, coal cores or drill cuttings are retrieved from the core hole and immediately put in a sealed container to measure the desorbed gas. The method suffers from the uncertainty in the estimation of gas lost during sample retrieval and handling. To eliminate this problem, the pressured core desorption technique has been used. In this method, gas loss is minimized by sealing the coal sample down the core hole.

Both methods provide a positive proof of gas presence and data on the gas desorption rate that can be used to calculate diffusivity (to be discussed later). Desorbed gases are chemically analyzed to determine the composition and calorific value of the seam gas.

The industry standard for direct measurement of gas content is the U.S. Bureau of Mines method shown in Figure 11.6-2 (Kissell et al. 1973). Coal cores or drill cuttings are deposited in a sealed vessel and desorbed gases are measured periodically until the desorption rate is insignificant. Total volume of gas thus produced is known as the desorbed gas. The cumulative volume of desorbed gas plotted on the y-axis against the square root of time yields a straight line, and its intercept on the y-axis is a measure of the gas lost from the core before it was deposited in the sealed container. Subsequently, a portion



Source: Kissell et al. 1973.

Figure 11.6-2 Direct measurement of coal-bed gas content

of the core is ground in a hermetically sealed mill to release the residual gas. The sum of the three component volumes—desorbed gas, lost gas, and the residual gas—is the total gas contained in the sample. This total volume of gas is divided by the weight of the sample to obtain the gas content of the coal seam.

Total gas reserve (gas-in-place) for the coalfield must be established before any commercial undertaking is started.

$$\text{gas reserve, } G, = \wp AHV$$

where

$\wp$ = density of coal, t/m^3
A = area of deposit, m^2
H = height of coal seam, m
V = gas content, m^3/t

If there is more than one coal seam in the reserve, as is the normal case, the total gas reserve is the sum of individual coal seam reserves.

Permeability

The permeability of a coal seam is the most important reservoir property next to gas content. Permeability of a porous medium is defined as 1 darcy when a fluid with a viscosity of 1 cP (centipoise) flows with an apparent velocity of 1 cm/s under a pressure gradient of 1 atm/cm (atmosphere per centimeter). The permeability is usually expressed in millidarcies (1 mD = 1/1,000 of a darcy).

Coal seams can have a permeability range of 1 microdarcy to 100 mD depending on the depth of the seam. Shallow coal seams have higher permeability, but deep coal seams show very low permeability.

Reservoir Pressure

The reservoir pressure can be calculated from the depth of the coal seam, but it is best measured directly. A vertical well is drilled and cased just above the coal seam. The coal seam is cleaned up with high-pressure water jets. A commercially available recording pressure gauge is installed in the well and the borehole is shut off at the surface. The gas pressure begins to build up, and in 4 to 6 days the final pressure is reached. The pressure gauge is retrieved and the final pressure is read from

the chart. The coal seam reservoir pressure is typically less than the hydrostatic head (about 0.7 times).

Thus a 300-m-deep coal seam in the Northern Appalachian Basin of the United States has a reservoir pressure of approximately 2 MPa, whereas 600-m-deep coal seams in the Central Appalachian Basin have a reservoir pressure of 4–5 MPa. The reservoir pressure will rarely exceed the hydrostatic head, but if it does, it is a very fortunate situation for commercial gas production. The "fairway" area of the San Juan Basin in the United States is an example of an overpressurized basin. Many highly productive CBM wells have been drilled and produced in this area.

Diffusivity of Coal Seams

The diffusivity of a coal seam is a parameter that measures how fast the coal seam will desorb the gases when the confining pressure is reduced to the atmospheric pressure. It is commonly measured by a parameter called "sorption time." It is the time a coal seam needs to desorb 63% of the original gas content and is measured in days. Sorption time for some coal seams in the San Juan Basin is less than 1 day, whereas it can be as high as 900 days for coal seams in the Northern Appalachian Basin (Rogers et al. 2007).

Coal seams with high sorption time will take a long time to yield a high percentage of in-situ gas and should be drilled at a closer spacing and produced for a long time. On the other hand, coal seams with shorter sorption times should be drilled at a wider spacing (greater than one well per 20 ha) but produced for a shorter time (usually fewer than 10 years).

Water Content and Quality of Water

The coal seams are usually saturated with water. Actual fluid flow in coal seams is a two-phase flow. Invariably, the water phase is produced first. As the water content depletes, the gas production begins to increase, reaching a peak when water is almost drained out. A typical CBM well can produce from a few cubic meters per day to hundreds of cubic meters per day of water.

The amount of water produced and its quality have a serious impact on the economy of a CBM project and will be discussed later in the chapter.

Ground Stresses and Rock Properties

Every point in a coal seam lies in a stress field that comprises the vertical stress (σv), the major horizontal stress (σH), and the minor horizontal stress (σh). The comparative magnitudes of these stresses are the determining factors for a successful "hydrofracing" of a vertical well. For successful hydrofracing, the sequence $\sigma H > \sigma v > \sigma h$ must be true. This results in the most favorable vertical fracture because the fracture is always orthogonal to the least stress. It will also travel in the direction of σH; that is, the azimuth of the fracture is parallel to σH. Major directional permeability of a coal seam is also parallel to σH (refer to Chapter 15.4).

Rock mechanics properties such as compressive strength, elastic modulus, and Poisson ratios are also needed for a successful design of gas-production techniques.

WIRE-LINE LOGGING

The vertical boreholes drilled to establish the gas reserve and determine reservoir properties are also used for wire-line logging. The process involves lowering special sensors down the hole to measure various characteristics of the coal seam and sending the data to the surface using electrical wire line, hence the name. The most commonly used logging techniques to evaluate coal seams for CMP are

- Gamma ray logging (passive and active),
- Resistivity logging,
- Sonic logging,
- Neutron logging,
- Density logging, and
- Induced gamma ray spectrometry logging.

All these techniques had their origin in the oil and gas industry. They are only slightly modified, if needed, for coal-seam logging. Details of their application are described in literature (Scholes and Johnson 1993).

Figure 11.6-3 shows basic log responses for multiple coal seams in a vertical well. For coal seams, the gamma ray response is low, the resistivity response is high, the density response is high, and the neutron density response is also high. Most commonly, only gamma ray logging is used to precisely locate coal seams in a borehole. Some tools measure the natural gamma ray emissions from the formation, whereas others use an active gamma source and measure the scattered gamma rays.

The only other logging technique commonly used in CMP projects is sonic log, particularly the cement-bond log technique. Several acoustic sources mounted on a tool are lowered into the borehole. The sound wave is reflected by the steel casing and the cement in between the casing and the formation. The velocity of sound and/or the amplitude of sound waves indicate whether the cement is there or only a void is present.

These two logs are highly recommended to locate the coal seam and to make sure the steel casings are well set. All other logging techniques are optional. New logging tools that measure the gas contents, gas isotherms, and directional permeability in situ are being developed, but they are not yet commercially available as of 2010.

PRODUCTION TECHNOLOGY

The CBM resource as shown in Table 11.6-1 is huge. Production strategy varies from basin to basin, and even in the same basin based on local reservoir properties. Since all coal seam reservoir properties are depth dependent, the CBM resources can be classified into three broad categories to identify the best production technique for different depth ranges: shallow, medium-depth, and deep. The main characteristics of these reservoirs are summarized in Table 11.6-2.

Shallow Reservoirs

Shallow reservoirs are characterized by high-permeability, low gas content; low reservoir pressures; and low diffusivity. Vertical stresses (<3 MPa) are smaller than the horizontal stresses. Hydrofracing of a vertical well in these reservoirs results in horizontal fractures that are inefficient for gas production.

The following are preferred production techniques:

- Mine shafts with multiple laterals (horizontal boreholes) at the bottom of the shaft. A plan view of this drilling pattern is shown in Figure 11.6-4. A 3-m-diameter shaft is drilled in the target coal seam and the coal seam is reamed out to a larger size to accommodate drill rigs. In active coal mines, these shafts can be 6–8 m in diameter and are later used as ventilation shafts. The entire mine property may need six to eight shafts to recover coal. They can be sunk 10 to 20 years ahead of mining for CBM production. Depending on the price of gas, the marketed gas can

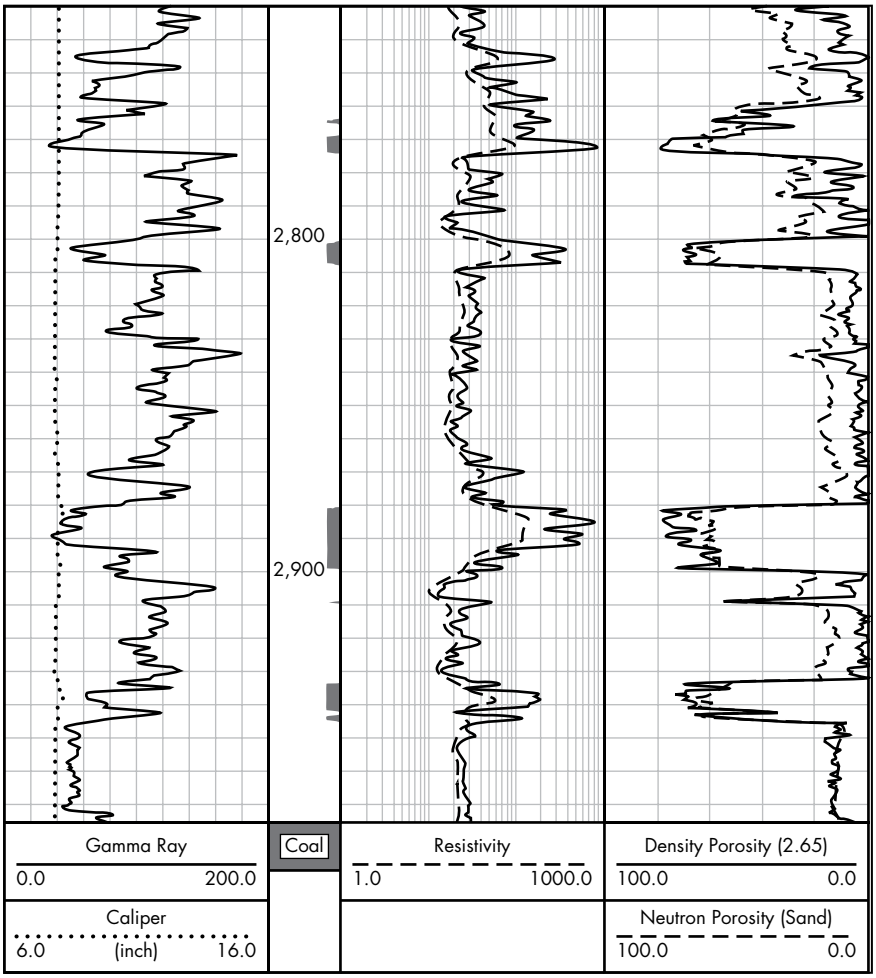


Figure 11.6-3 Basic responses to various logs in coal

defray all costs associated with shaft sinking. Specific gas production from these shallow coal seams ranges from 10–25 m^3/d/m of the borehole. The specific gas production for a coal seam is a characteristic of the coal seam and is a measure of the rate of gas production per unit length of a horizontal borehole drilled for gas production. Assuming six horizontal boreholes drilled to 1,000 m, an initial production of 60,000–150,000 m^3/d can be realized.

- Another production technique uses surface-based rigs to drill horizontal laterals (see Chapter 15.4 for details). The property is drilled with "production wells" on a grid pattern. Typically, four to six horizontal laterals are drilled from access wells 1 km away to intersect the production wells to realize commercial CMP (Figure 11.6-5).
- A special case of gas production from a shallow reservoir is the Powder River Basin of Wyoming (United States). Coal seams in this basin are very thick (30–60 m) and shallow; the maximum depth is <200 m. The gas is of biogenic origin (unlike deeper seams where gas is of thermogenic origin) and gas content is quite low (<3 m^3/t). Permeability and water content are quite high. The production technique used here is a vertical well with hydrojetting but no hydrofracing. A typical well completed with a cost of $50,000 can yield gas production rates of 7–10 km^3/d, making it a profitable venture. More than 10,000 wells have been drilled and completed this way in the Powder River Basin so far. The technique can work successfully in similar deposits in Montana (United States) or other places in the world.

Medium-Depth Reservoirs

Medium-depth reservoirs are characterized by low to medium permeability, high gas contents, relatively higher reservoir pressures, higher diffusivity, and the coal tends to be of higher rank. Vertical wells with well-designed hydrofracing are ideally suited to such reservoirs for gas production. Specific gas productions for vertical fractures created in these seams range from 4 to 8 m^3/d/m. The hydraulic fracturing process basically involves pumping a fluid (such as water, nitrogen foam, or a high-viscosity gel) with fine sand (0.1–2 mm diameter) at a pressure high enough to hydrofrac the coal seam. Under ideal circumstances ($\sigma h < \sigma v < \sigma H$), a vertical fracture is created in the coal from the floor to the roof (Figure 11.6-6). Depending on the volume of the frac fluid pumped, the total frac length can be 300–600 m. The fracture will typically have two wings of 150–300 m that travel in the opposite directions but on the same azimuth that is parallel to the direction of σH.

For commercial gas production, all coal seams with a thickness >1 m are hydrofraced in the vertical interval 500–1,000 m. Multiple coal-seam fracing techniques are well

Table 11.6-2 CBM reservoir characteristics

Reservoir Type	Approximate Depth, m	Permeability, mD	Remarks*
Shallow	≤500	10–100	$v < h < H$
Medium-depth	500–1,000	0.1–10	$h < v < H$
Deep	≥1,000	<0.1	$h < v < H$

*v is the vertical stress; h and H are the minor and major horizontal stresses.

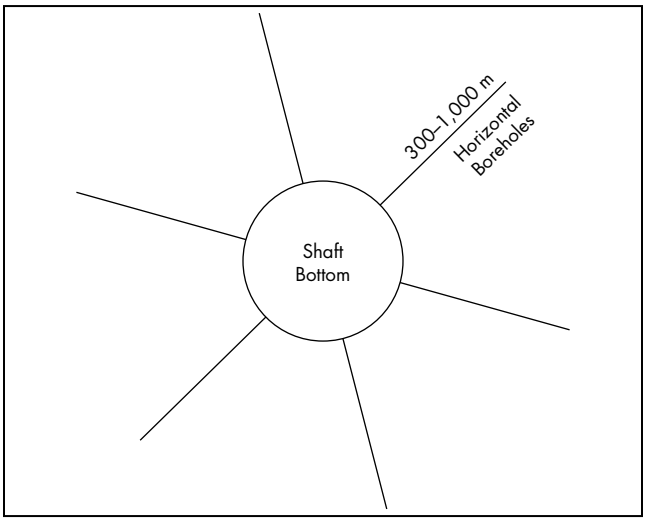


Figure 11.6-4 Mine shaft with horizontal boreholes

established but beyond the scope of this chapter. A typical gas production from each horizon of 1.5–2-m-thick coal seam is 3,000 m^3/d. If three or four horizons of this thickness are hydrofraced in a well, gas productions of 9,000–12,000 m^3/d can be realized.

Observations of nearly 200 hydrofraced wells and mapping of the subsequent mining of the fracture created can be summarized as follows:

1. The fracture volume (length × width × height, or LWH) is always proportional to the fluid volume, Q, pumped in:

   $LWH = CQ$

   where C is the leak-off coefficient; it is typically 1 or 2 orders of magnitude higher than that for sandstone or limestone reservoirs.
2. The fracture plane is always orthogonal to the least stress (σh or σv, whichever is smaller).
3. The fracture azimuth is always parallel to the major horizontal stress, σH.
4. The fracture always travels up into the roof but not in the floor, following the law of physics that the fluid will always flow down the pressure gradient.
5. When a coal seam bounded by shale or sandstone is fraced, the formation with the smallest modulus of elasticity, E, will hydrofrac first. In fact, strong shale and sandstone provide a confining barrier containing the fracture in coal.
6. The fracture width is proportional to (fluid viscosity/E) raised to the power of 0.25.

Figure 11.6-7 shows two methods of completion, namely (coal-bed A) open-hole and (coal-bed B) through perforations.

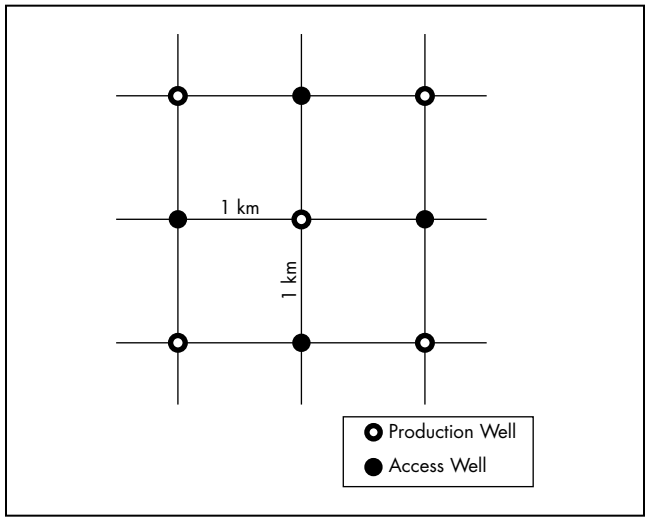


Figure 11.6-5 Horizontal laterals drilled from the surface

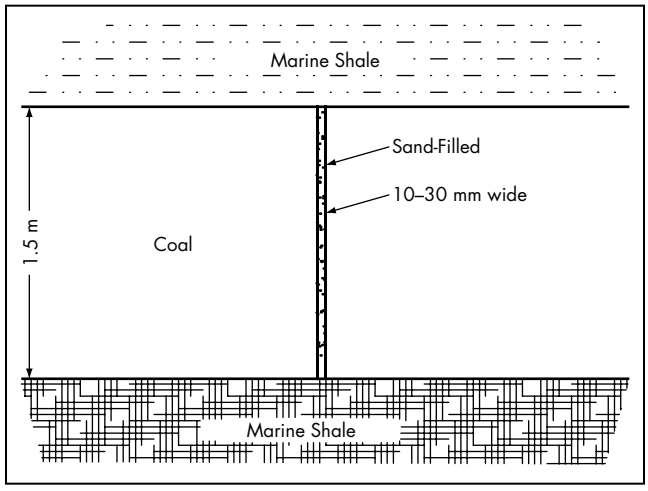


Figure 11.6-6 Sectional view of sand-propped hydraulic fracture

If the coal seam has a high compressive strength, the lowest coal seam can be fraced open-hole. The upper coal seams are fraced through perforations in the steel casing. If the coal seam has low compressive strength, it is preferable to hydrofrac it with perforated casings.

The choice of frac fluid varies from basin to basin. In the depth range of 500 to 750 m, gelled water or nitrogen foam, 70% quality, has given good results in the Central and Southern Appalachian Basin of the United States. On the deeper end (750–1,000-m depth) and for thick coal seams (>10 m), cross-link borate gels with very high viscosity have produced better results. An example is the San Juan Basin of New Mexico and Colorado. Sand concentrations in the frac fluid vary from 0.1 t/m^3 to 0.5 t/m^3 depending on the viscosity of the fluid. Fluids with higher viscosity can carry higher concentrations of sand.

A special case of CMP from deep and thick coal seams is the cavitation technique. Figure 11.6-8 shows a completion technique called cavitation through a perforated casing. If the coal seam has high compressive strength, cavitation can be

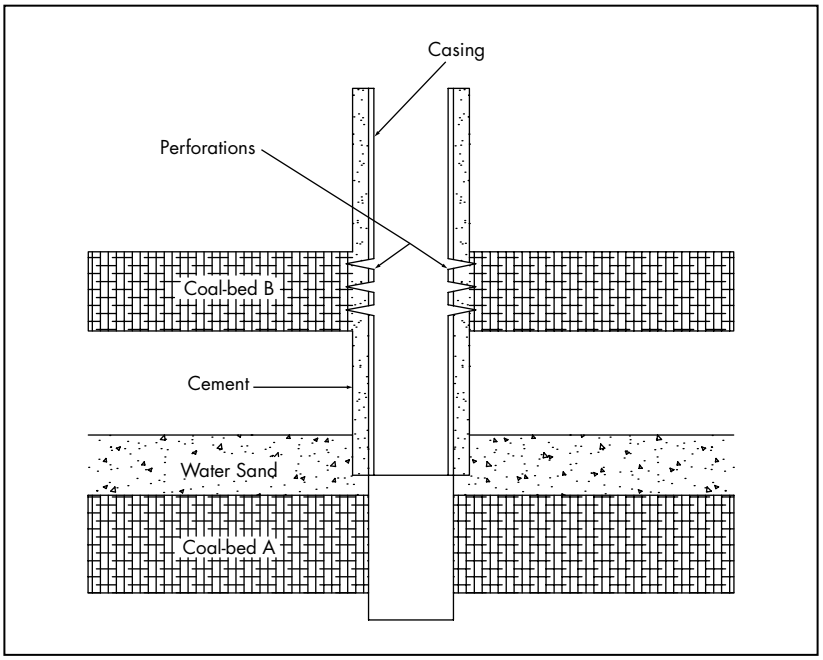


Figure 11.6-7 Two completion procedures for vertical wells in thin coal seams

done in an open hole. In the cavitation technique, the borehole is pressurized with water and the pressure is released. The process is repeated several times in a quick sequence with the hope that coal will cavitate in the depressurizing phase and enhance permeability leading to good gas production. Cavitation completions have worked well yielding 30,000–60,000 m^3/d of gas production in 10–20-m-thick coal seams.

Deep Coal Reservoirs

The bulk of the worldwide CBM reserves are in deep coal seams. These reservoirs are characterized by high gas content and high reservoir pressures but extremely low permeability (a few microdarcies). Coal is a viscoelastic material. An extreme reduction in permeability with increasing depth (and hence pressure) is a quite natural phenomenon. Because of depth, no horizontal drilling from a mine or from the surface has been done. As such, specific production of horizontal boreholes is not known but it is likely to be low, <1 m^3/d/m.

Mine shafts are obviously too expensive for these seams. Even vertical wells with hydrofracing are very inefficient. Actual hydrofracing efforts in Virginia (United States) at a depth of 1,400 m and in eastern France at a depth of 1,800 m ended in failure.

The only feasible technique at present is to drill long horizontal laterals from the surface. Figure 11.6-9 shows a plan view of 12 horizontal laterals drilled in four groups from the same surface location. The outlying horizontal laterals in each subgroup are hydraulically fractured at intervals of 300 m. Hydrofracing should be designed such that the fractures can extend at least 300 m on each side of the vertical well.

In the beginning, all hydrofraced horizontal laterals will have a high gas production rate, but after the gas production rate has significantly declined, the middle lateral in each subgroup should be injected with carbon dioxide. Coal seams have a much higher affinity for CO_2. It will be preferentially adsorbed by the coal matrix releasing methane at the same time (Thakur 2002).

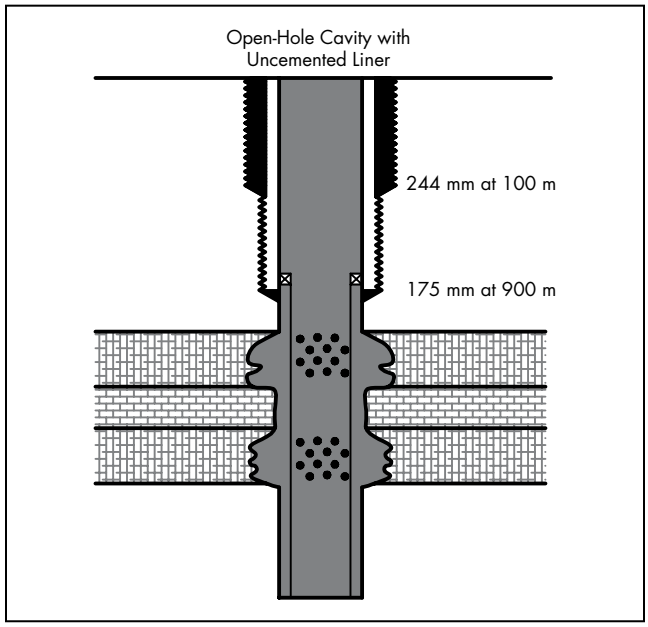


Figure 11.6-8 Cavitation completion procedure for thicker coal seams

WELL SERVICING AND MAINTENANCE

After drilling and completion, all CBM wells will produce water and gas, in that order. All wells need some servicing and maintenance. Figure 11.6-10 shows a typical gas and water production profile. Depending on the water content of the coal seam, a CMP well can produce moderate to large quantities (1–100 m^3/d) of water. As the reservoir pressure is reduced and more water is produced, the water production declines and gas production rises, reaching a peak. After that, the gas production rate also declines. Horizontal and vertical wells need different kinds of servicing and maintenance.

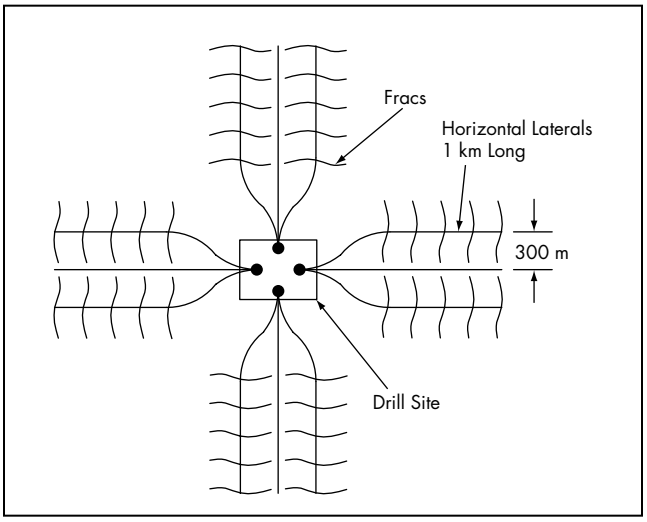


Figure 11.6-9 Gas production from deep coal seams

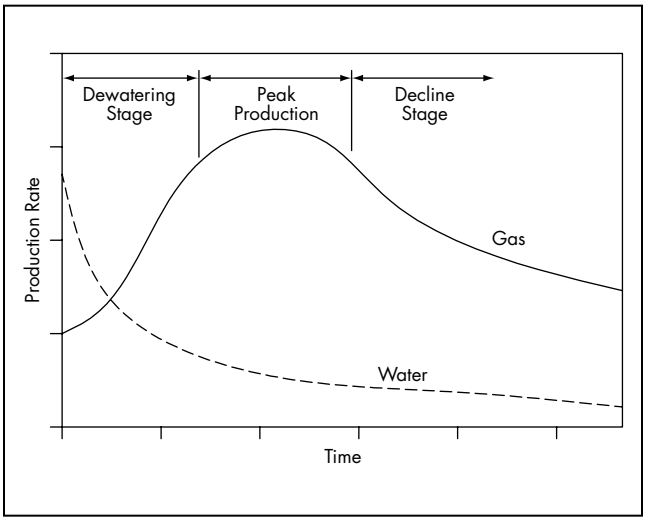


Figure 11.6-10 CBM well production profile

Horizontal Laterals Drilled from the Surface

These wells do not need much maintenance. The produced water collects at the bottom of the production well. If several horizontal laterals discharge water in one production well, the volume is large enough to warrant the use of a progressive cavity pump. As water production declines, the pump can be put on a timer. The gas pressure in the reservoir is enough to clean up the borehole.

This method of well completion still suffers from two drawbacks: an inability to stay in the coal seam and an inability to completely dewater the well. The first problem is due to drilling-instrument limitations. The borehole exits the coal seam and often enters the roof or floor, limiting gas production and the ability to completely dewater. Hence the specific production (cubic meters per day per meter) is generally only 50% or less compared to in-mine drilling, which is more precise (see Chapter 15.4). It is also difficult to reenter a horizontal lateral for maintenance, especially if multiple laterals are drilled from the same vertical well.

Vertical Hydrofraced Well

The servicing procedure and equipment for these wells is borrowed from the oil and gas industry.

After hydrofracing, a well is generally swabbed clean to remove any debris, and a dewatering pump is installed. Depending on the depth, gas and water production rates, and reservoir pressure, one of the following systems is used:

- Sucker rod pump
- Progressive cavity (Moyno) pump
- Submersible electric pump
- Gas lift

A sucker rod pump is most often used when water flow is less than 20 m^3/d. For heavier flows, a progressive cavity pump is used. In the dewatering phase of gas production, the pump is usually kept slightly above the coal seam and a 30–60-m water column is maintained above the coal seam to prevent a blowout (i.e., a sudden discharge of gas and proppant sand). Submersible electric pumps are used when the water production is excessive and progressive cavity pumps are unable to handle the flow. Gas lift is used when no electric power is available to operate a dewatering pump.

PRODUCED WATER DISPOSAL

All produced water must be metered to determine the flow rate. If the water is clean, a turbine meter is used to precisely measure the water flow. If the produced water contains some solids, a positive displacement water meter is used. The latter is more durable and commonly used. Where the water flow is intermittent and low, a 20-L bucket is used to collect the pump discharge, and the time of discharge is measured by a stopwatch from which the flow rate is calculated.

Gathering water and its proper disposal is often a significant part of the CMP. The following are the possible options for water disposal:

- Surface disposal
- Deep-well injection
- Evaporation in a pond or a boiler
- Land application
- Off-site disposal in abandoned mines
- Reuse for hydraulic stimulations

The choice of the technique depends on the volume of water produced per day and its quality. Permitting from national and state authorities is often needed and it is a prolonged process. Most commonly, only the first two choices are used, and they are the only ones discussed here. Figure 11.6-11 shows a layout of various options.

Surface Disposal

The simplest method of surface disposal comprises pH (a measure of water acidity) adjustment of water to meet the permit requirements, aeration of water with compressed air, and addition of some polymers to facilitate settling. The clean water is used for land applications or simply discharged into a local stream. The total dissolved solids (TDS) and chloride concentrations are monitored to match the quality of treated water with that of water in the stream.

If the produced water has a higher concentration of contaminants, it is pretreated, and the reverse osmosis process is

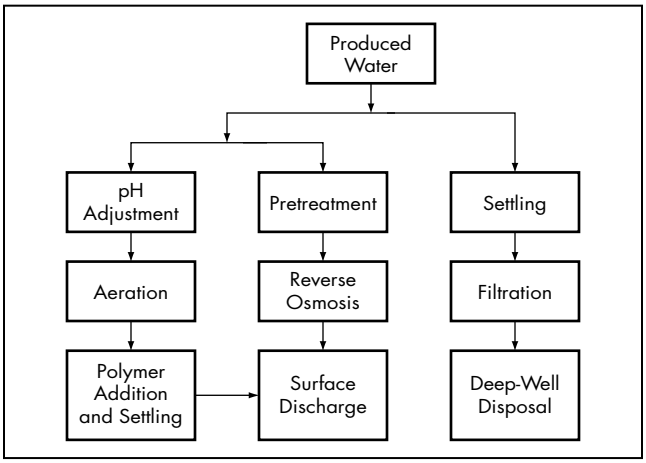


Figure 11.6-11 Options for treating produced water

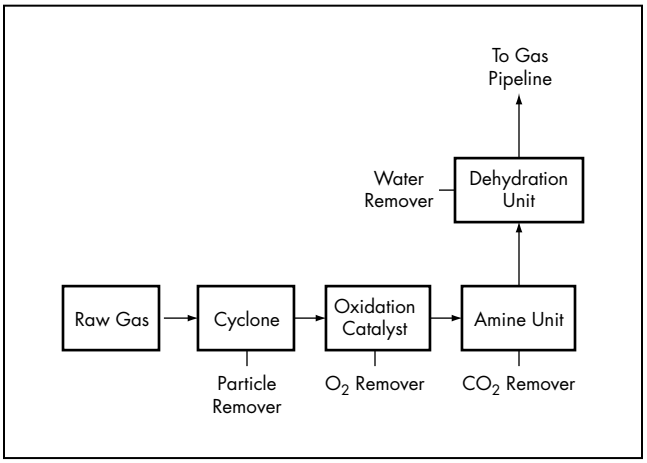


Figure 11.6-12 Gas processing flow

used to remove the TDS. Average efficiency of TDS removal is in excess of 95%. Only in the presence of bicarbonates does the efficiency go down to 50%–80%. The main limitation of this process is membrane fouling caused by suspended solids or oils in the produced water.

Deep-Well Injection

If there are no streams in the neighborhood of a CBM field or the necessary permits to discharge into the local streams cannot be obtained, the treated water can be injected into a deep well. Deep-well injection also requires a permit but it is easier to get. Typically these injection wells are old gas wells with a depth of 3,000 m or so. They are often hydrofraced to create good permeability.

Produced water must be properly treated before injection to keep the injection pressure below the permitted limits. Thick sandstone or limestone reservoirs with high porosity and permeability are ideal for this purpose. Preinjection filtration is almost always essential. Scale and corrosion inhibitors and bactericides are often added to the injected water to increase long-term use of the injection well. Finding or even drilling an injection well in the vicinity is crucial because the cost of gathering and hauling water can be quite substantial.

GAS GATHERING AND PROCESSING

CMP must be gathered to a central location for processing and commercial marketing.

Gas Gathering and Measurement

Gas flows from individual wells must be measured. The primary measurement instruments are either an orifice meter or a turbine meter. The orifice meter measures the differential pressure upstream and downstream of a measured orifice. The pressure drop is a function of the flow rate, gas pressure, and temperature. The pressure differential is continuously recorded on a chart that is calibrated to give the gas production. This is the most common method of measuring individual well production.

The turbine meter measures the flow rate and total volume of gas produced. Gas flows over rotary blades, and magnets in the meter housing pick up signals that are proportional to gas velocity. The area between the blades has fixed volumes so that the number of turns equates to the volume of gas produced. The turbine meter is more accurate than the orifice meter, but it is more expensive and needs frequent maintenance.

Gas from each producing well is gathered using polyethylene pipes and compressors. Usually a cluster of wells is connected to a small compressor. These small compressors deliver the compressed gas to a central compressor facility where it is processed for marketing. In the design of the gas-gathering system, the main object is to maintain a low wellhead pressure, usually <3–5 kPa. Pipe diameters and compressor sizes are designed to minimize line losses. In general, one may start with 50-mm-diameter pipe at a single wellhead, but progressively the pipe diameter may increase to 250–300 mm at the main gas-gathering and processing plant.

Gas Composition and Processing

For commercial marketing, the CBM composition must meet the pipeline specification. All major pipeline companies require a minimum calorific value and put a limit on oxygen, moisture, and noncombustible concentrations. CBM composition varies from basin to basin, but generally it contains 85%–95% methane. Impurities must be removed to raise the calorific value to about 37 MJ/m^3. Other requirements are that the noncombustible content of gas should be <4%, moisture should be <0.11g/m^3, and oxygen is restricted to 20 ppm. In many applications, CBM is mixed with coal mine methane that contains higher oxygen and nitrogen, in which case excess nitrogen is removed by a cryogenic process.

Figure 11.6-12 shows a typical gas processing flow diagram. First, the raw gas is passed through a cyclone to knock solid particles out. Next, the gas is passed over an oxidation catalyst where methane is oxidized that consumes all oxygen and produces water and carbon dioxide (CO_2). Next, the CO_2 is removed by passing the gas through an amine unit. Finally, the moisture in gas is removed by dehydration units containing glycol. If there is excess nitrogen in the gas stream, it is generally removed by a cryogenic process. The clean, pipeline-quality gas is compressed to pipeline pressure, typically 7–8 MPa for delivery into a commercial gas pipeline.

Small rotary compressors are used for primary gathering because they are more efficient in compressing somewhat dirty but large volumes of gas. The main station compressors are generally reciprocating piston-type compressors that are more efficient for duty. The prime mover for compressors can be electric, diesel, or even a CBM engine to keep the cost low.

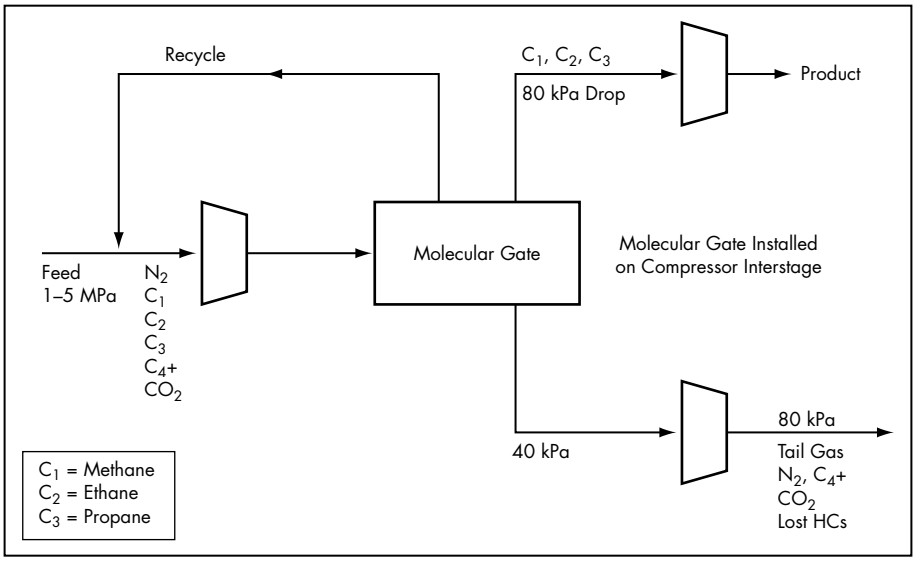


Figure 11.6-13 Molecular gate separation

Molecular Gate Separation System

The molecular gas separation system is a recent breakthrough for processing gases where gas molecules differ in size by only a fraction of an angstrom (Å). A new family of titanium silicate molecular sieves has a pore size that can be controlled down to 0.1 Å (0.01 nm).

In upgrading CBM and coal mine methane, the desired product, methane, has a molecular diameter of 3.8 Å whereas the impurities such as nitrogen (3.6 Å), carbon dioxide (3.3 Å), and oxygen (3.5 Å) are all smaller in size and easily fit within the pore of an adsorbent, where they can be adsorbed and removed in one step from the product stream (Mitariten 2001).

Figure 11.6-13 illustrates the basic flow sheet for a molecular gate separation process. The feed gas is compressed to 1–5 MPa and fed into the molecular gate system. Several repetitions/cycles are made to achieve high purity. The product gas comprising methane, ethane, and propane is recovered at practically the original pressure, which avoids recompression. The tail gas contains most impurities but often has enough methane that it can be used for steam production. The process has a methane recovery efficiency of 95% at 2.4 MPa, but it becomes more efficient at higher pressures. It can remove CO_2, N_2, and O_2 in one step. It is available in sizes of 30,000–900,000 m^3/d of methane. The cost of gas processing declines significantly with increase in the flow rate.

CARBON DIOXIDE SEQUESTRATION

It was mentioned earlier that coal has a great affinity for CO_2 and it can displace methane stored on the micropores of the coal matrix. This property of coal has been used to produce methane from deep coal seams and sequester CO_2 in it simultaneously. Laboratory experiments to displace methane from coal using nitrogen, helium, and carbon dioxide show that the latter is the most effective flooding agent. The storage capacity of coal for CO_2 is at least twice the capacity it has for methane (Collins et al. 2002).

Figure 11.6-9 shows an ideal layout for CO_2 sequestration. When gas production begins to decline, CO_2 injection can start. The optimum parameters of CO_2 sequestration have not yet been established, but many research projects are in progress to determine the following:

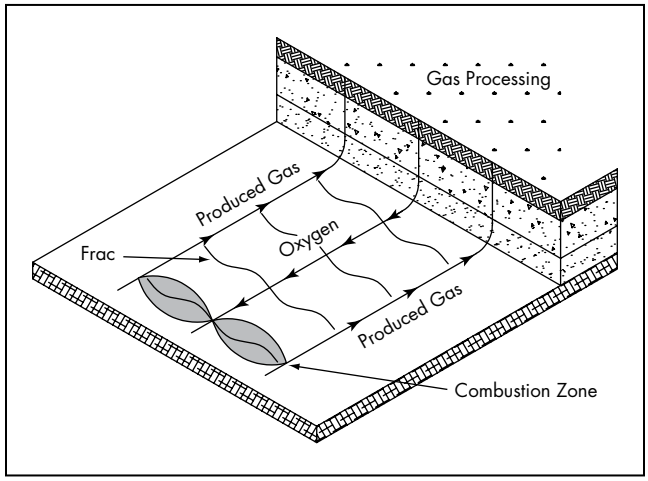


Figure 11.6-14 Underground coal gasification

- Optimum storage capacity for CO_2 in various coal seams
- Rate of injection and travel velocity of CO_2 in coal
- Optimum pressure of injection; generally the reservoir fracturing pressure should not be exceeded
- Ideal mix of gases; CO_2 by itself is slow moving and nitrogen is often added to accelerate the process
- Economics of CO_2 sequestration

CO_2 sequestration research is encouraged by many countries because it is considered a GHG. In many countries, there are financial incentives for sequestering CO_2. The current carbon finance instrument in the United States is fixed at $6.50/t of CO_2. Two sources of CO_2 can be related to coal: (a) power plants that burn coal, and (b) CBM itself that may contain 10%–50% CO_2 in some coal seams. CO_2 can be stripped using gas-processing techniques discussed earlier and its sequestration may become a profitable venture.

UNDERGROUND COAL GASIFICATION

CMP from deep coal seams can also be enhanced by heating the coal matrix. The drilling layout shown in Figure 11.6-14 is a schematic for UCG. Heating of the central horizontal boreholes can be done with steam or radio frequency devices, but the most synergistic approach is to ignite the coal with a limited supply of air/oxygen-combustible gaseous blends. The UCG process is, in fact, already well established, but it has not been done in conjunction with methane recovery. In this process, the outer horizontal laterals are used for gas production. When most methane is driven out, the coal matrix is combusted in situ to produce a lower calorific value gas. The process is also called in-situ gasification of coal.

The product gas composition depends on coal composition and process parameters such as operating pressure, outlet temperature, and gas flow. These parameters are not only monitored but constantly adjusted to optimize the process.

The UCG process is quite flexible. It can be used in coal seams with a thickness of 1–60 m, and calorific values of 10–30 MJ/kg including lignites and other low-quality coal. The coal deposits shown in Figure 11.6-1 have a reserve of 17–30 Tt. Only 1 Tt is amenable to actual mining. Thus, the remaining reserve can be economically exploited by the UCG process.

UCG experiments carried out in the 1970s by the U.S. Department of Energy in West Virginia and Wyoming (United States) produced only low calorific value gases (10–20 MJ/m^3), but new processes are being developed to improve the calorific value. Products of different UCG projects can be mixed with gases with higher calorific values to meet a consumer's specific need.

CMP and UCG processes supplement conventional mining by recovering energy from coal seams that are abandoned or cannot be economically mined. The gaseous product of the UCG process can be used in a variety of ways, such as

- Boiler fuel to produce steam for electric power generation,
- Feed to chemical plants to produce Fischer–Tropsch liquids or fertilizers, and
- A clean gaseous fuel after the raw gas is processed using techniques discussed earlier in the chapter.

The U.S. Environmental Protection Agency is making a great effort to export CMP technology to all major coal-producing countries. This is mainly being done to make mining safer and more productive. But it also creates a clean source of energy that supplements the natural gas supply. Additional incentives such as the carbon credit can provide a great impetus to develop this valuable resource even further. The combination of CMP with CO_2 sequestration can be used to enhance CMP from deeper coal seams and eventually, using the UCG process to convert coal into natural gas, can greatly increase the world supply of natural gas. Further research is needed to make the UCG process more efficient. In the future, the in-situ coal may also be converted into liquid fuel to increase the petroleum supply.

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PART 12

Underground Development

CHAPTER 12.1

Introduction to Underground Mine Planning

Richard L. Bullock

GENERAL BASIS OF MINE PLANNING

Many details go into the planning of a mine. The information gathered must come from several sources. First is the geological, structural, and mineralogical information, combined with the resource/reserve data. This information leads to the preliminary selection of potential mining method and sizing of the mine production. From this the development planning is done, the equipment selection is made, and the mine work force projections are completed, all leading to the economic analysis associated with mine planning.

Planning as just described, however, will not necessarily guarantee the best possible mine operation, unless the best possible mine planning has been done correctly. Any sacrifice in mine planning introduces the risk that the end results may not yield the optimum mine operation. Planning is an iterative process that requires looking at many options and determining which, in the long run, provide the optimum results.

This chapter addresses many of the factors to be considered in the initial phase of all mine planning. These factors have the determining influence on the mining method, the size of the operation, the size of the mine openings, the mine productivity, the mine cost, and, eventually, the economic parameters used to determine whether the mineral reserve should even be developed.

PHYSICAL AND GEOTECHNICAL INFORMATION NEEDED FOR PRELIMINARY MINE PLANNING

Many pieces of engineering and geologic data must be gathered before mine planning can take place. These are covered in the sections that follow. This section draws heavily from Chapter 2 of *Underground Mining Methods* (Bullock 2001).

Technical Information

Assuming that the resource to be mined has been delineated with prospect drilling, study will be based primarily on information supplied through exploration. The results of the exploration are recorded in a formal report for use in project evaluation. The exploration report should contain the following information with appropriate maps and cross sections:

- Property location and access
- Description of surface features
- Description of regional, local, and mineral-deposit geology
- Review of exploration activities
- Tabulation of geologic resource material
- Explanation of resource calculation method, including information on geostatistics applied
- Description of company's land and water position
- Ownership and royalty conditions
- Mining history of property
- Rock quality designation (RQD) values and any rock mass classification work that has been done
- Results of any special studies or examinations the exploration group has performed (metallurgical tests, geotechnical work, etc.)
- Report on any special problems or confrontations with local populace of the area
- Any other pertinent data such as attitude of local populace toward mining, special environmental problems, availability of water and hydrologic conditions in general, and infrastructure requirements

This critical information should be established to assist the mine planning. If the exploration project is one that has been drilled out by the company exploration team, this information should have been gathered during the exploration phase and passed to the mine evaluation team or the mine development group. More information on each of these subjects may need to be obtained, but if the inquiry can be started during the exploration phase of the project, time will be saved during the feasibility/evaluation and development phases of the project. If this is an ongoing mine operation, then most of the technical information will be available from other mine planning projects.

Geologic and Mineralogic Information

Knowledge of similar rock types or structures in established mining districts is always helpful. In developing the first mine in a new district, there is far more risk of making costly errors than in the other mines that may follow. The geologic and mineralogic information needed includes

- The size (length, width, and thickness) of the areas to be mined within the overall operations area to be considered, including multiple areas, zones, or seams;

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- The dip or plunge of each mineralized zone, area, or seam, noting the maximum depth to which the mineralization is known;
- The continuity or discontinuity noted within each of the mineralized zones;
- Any swelling or narrowing of each mineralized zone;
- The sharpness between the grades of mineralized zones within the material considered economically mineable;
- The sharpness between the ore and waste cutoff, including
  - Whether this cutoff can be determined by observation or must be determined by assay or other means,
  - Whether this cutoff also serves as a natural (physical) parting resulting in little or no dilution, or whether the break between ore and waste must be induced entirely by the mining method, and
  - Whether the mineralized zone beyond (above or below) the existing cutoff represents submarginal economic value that may become economical at a later date;
- The distribution of various valuable minerals making up each of the potentially mineable areas;
- The distribution of the various deleterious minerals that may be harmful in processing the valuable mineral;
- Whether the identified valuable minerals are interlocked with other fine-grained mineral or waste material;
- The presence of alteration zones in both the mineralized and the waste zones;
- The tendency for the ore to oxidize once broken; and
- The quantity and quality of the ore reserves and resource with detailed cross sections showing mineral distribution and zones of faulting or any other geologic structure related to the mineralization.

Structural Information

Required physical and chemical structural information includes the following:

- The depth of cover
- A detailed description of the cover, including the
  - Type of cover;
  - Approximate strength or range of strengths;
  - Structural features in relation to the proposed mine development; and
  - Presence of and information about water, gas, or oil that may have been encountered
- The quality and structure of the host rock (back, floor, hanging wall, footwall), including the
  - Type of rock,
  - Approximate strength or range of strengths,
  - Noted zones of inherent high stress,
  - Noted zones of alteration,
  - Major faults and shears,
  - Systematic structural features,
  - Porosity and permeability,
  - The presence of any swelling-clay or shale interbedding,
  - RQD throughout the various zones in and around all of the mineralized area to be mined out,
  - The host rock mass classification (Rock Mass Rating [RMR] or Barton's Q-system),
  - Temperature of the zones proposed for mining, and
  - Acid-generating nature of the host rock
- The structure of the mineralized material, including all of the factors listed previously, as well as the
  - Tendency of the mineral to change character after being broken (e.g., oxidizing, degenerating to all fines, recompacting into a solid mass, becoming fluid, etc.);
  - Siliceous content of the ore;
  - Fibrous content of the ore;
  - Acid-generating nature of the ore; and
  - Systematic fault offsets.

PLANNING RELATED TO PHYSICAL PROPERTIES

The physical nature of the extracted mass and the mass left behind are very important in planning many of the characteristics of the operating mine. Four aspects of any mining system are particularly sensitive to rock properties:

1. The competency of the rock mass in relation to the in-situ stress existing in the rock determines the open dimensions of the unsupported roof unless specified by government regulations. It also determines whether additional support is needed.
2. When small openings are required, they have a great effect on productivity, especially in harder materials where drill-and-blast cycles will be employed.
3. The hardness, toughness, and abrasiveness of the material determine the type and class of equipment that can extract the material efficiently.
4. If the mineral contains or has entrapped toxic or explosive gases, the mining operation will be controlled by special provisions within government regulations.

In countries where appropriate regulations do not exist, best-practice guidelines must be sought.

THE NEED FOR A TEST MINE

From this long list of essential information required for serious mine planning, it becomes evident that not all of this information can be developed from the exploration phase. Nor is it likely that it can all be obtained accurately from the surface. If this is the first mine in this mining area or district, then what is probably needed during the middle phase of the mine feasibility study is development of a test mine. While this may be an expense that the owners were hoping to avoid up front, the reasons for a test mine are quite compelling. They include opportunities to

- Confirm, from a geologic point of view, the grade, ore continuity, ore configuration, and mineral zoning;
- Confirm, from the engineering viewpoint, rock strengths and mass rock quality; verify mining efficiencies; confirm water inflows; and demonstrate waste characteristics;
- Pilot-test the metallurgical process;
- Enhance the design basis for cost estimates, improve labor estimates, build more accurate development schedules; and
- Lower the mining investment risk.

Having a test mine in place will shorten the production mine development and will serve as a training school for the production mine.

LAND AND WATER CONSIDERATIONS

Information needed about the property includes

- Details on the land ownership and/or lease holdings, including royalties to be paid or collected (identified by mineral zones or areas);
- Availability of water and its ownership on or near the property;
- Quality of the water available on (or near) the property;
- Details of the surface ownership and surface structures that might be affected by subsidence of the surface;
- The location of the mining area in relation to any existing roads, railroads, navigable rivers, power, community infrastructure, and available commercial supplies of mine and mineral processing consumables (drill steel, bits, explosives, roof bolts, steel balls and rods for the mills, chemical additives for processing, steel liners for all mining and processing equipment, etc.); and
- The local, regional, and national political situations that have been observed with regard to the deposit.

OTHER FACTORS INFLUENCING EARLY MINE PLANNING

Many operational decisions must be made in planning the mining operation. None is more important than sizing the mining operation. However, it is not an easy or obvious decision to make and several factors must be considered.

Sizing the Production of a Mine

A considerable amount of literature is available on the selection of a production rate to yield the greatest value to the owners, including works by Tessaro (1960), McCarthy (1993), Runge (1997), and Smith (1997). Basic to all modern mine evaluations and design concepts is the desire to optimize the net present value (NPV) or to operate the property in such a way that the maximum internal rate of return is generated from the discounted cash flows. Anyone involved in the planning of a new operation must be thoroughly familiar with these concepts. Equally important is the fact that any entrepreneur who is planning a mining operation *solely from the financial aspects of optimization*, and who is not familiar with the issues associated with maintaining high levels of production at low operating cost per metric ton over a prolonged period, is likely to experience disappointment in years with low (or no) returns.

Other aspects of the problem for optimizing mine production relate to the effect of NPV. When viewed from the purely financial side of the question (i.e., producing the product from the mineral deposit at the maximum rate), optimization that yields the greatest return is the case often selected. This is due to the fixed costs involved in mining, as well as to the present-value concepts of any investment. Still, there are "...practical limitations to the maximum intensity of production, arising out of many other considerations to which weight must be given" (Hoover 1909). There can be many factors limiting mine size, some of which are listed here:

- Market conditions
- Current price of the product(s) versus the trend price
- Grade of the mineral and the corresponding reserve tonnage
- Time before the property can start producing
- Attitude and policies of the local and national government and the degree of stability of existing governments and their mining policies, taxes, and laws
- Availability of a source of energy and its cost
- Availability of usable water and its cost
- Cost and method of bringing in supplies and shipping production
- Physical properties of the rock and minerals to be developed and mined
- Amount of development required to achieve the desired production related to the shape of the mineral reserve
- Amount and complexity of mineral processing required
- Availability of nearby smelting options (if required)
- Size and availability of the work force that must be obtained, trained, and retained
- Availability of housing for employees (in remote locations)
- Potential instability of the government in the future, which might cause a company to develop a smaller, high-grade mine in the beginning until they have received their objective return, then use the income from the existing property to expand and mine out the lower-grade ores

Although all of these factors must be taken into consideration, another approach to sizing the mine is to use the Taylor formulas (Taylor 1977). Taylor studied more than 200 mining properties and then used regression analysis to determine the formulas for sizing a mine. Taylor notes, however, that the formulas are not very applicable to steeply dipping mineral reserves or when mining from deep shafts. The formulas are

$$\text{life of mine} = 0.20 \,\#\, (\text{expected ore metric tons [tons]})^{0.25} = \text{life of mine (in years)} \pm 1.2 \text{ years} \quad (12.1\text{-}1)$$

$$\text{daily production} = \text{reserve ore metric tons (tons)}/\text{expected life/operating days/year} \quad (12.1\text{-}2)$$

Assume 37.2 Mt (41 million tons) of mineable resource:

$$\text{life of mine} = 0.20 \times (37{,}200{,}000)^{0.25} = 15.6 \text{ yr } (\pm 1.2 \text{ yr or } 14.4 \text{ to } 16.8 \text{ yr})$$

$$\text{daily production} = 37{,}200{,}000/16 \text{ yr}/365 = 6{,}525 \text{ Mt/d } (6{,}059 \text{ Mt/d to } 7{,}068 \text{ Mt/d})$$

or

$$\text{life of mine} = 0.20 \times (41{,}000{,}000)^{0.25} = 16 \text{ yr } (\pm 1.2 \text{ yr or } 14.8 \text{ to } 17.2 \text{ yr})$$

$$\text{daily production} = 41{,}000{,}000/16 \text{ yr}/365 = 7{,}020 \text{ tpd } (6{,}530 \text{ tpd to } 7{,}590 \text{ tpd})$$

The two formulas tend to overestimate the production of small-vein-type mining deposits where a lot of vertical development must be completed compared to the daily tonnage that can be extracted. For mineral resources that are steeply dipping, and for deeper mining, the shape of the resource must be considered, as well as how much development is necessary to sustain the desired production. According to McCarthy (1993), for Australian underground narrow-vein mines, approximately 50 vertical meters (165 vertical feet) per annum is currently

economically appropriate for modern mechanized mines. Thus for example, if given certain reserves for mining 10,000 metric tons per vertical meter, then the production would be 500,000 t/a. Properties that were above this "best fit" trend line, McCarthy says, are usually overly capitalized or have higher-than-average operating costs.

Not only does the resource's tonnage affect the mine size, but the distribution of ore grade can certainly affect the mine planning. Unless a totally homogeneous mass is mined, it may make a considerable economic difference as to which portion is mined first or later. Furthermore, no ore reserve has an absolute fixed grade-to-tonnage relationship; trade-offs must always be considered. In most mineral deposits, lowering the mining cutoff grade means that more metric tons will be available to mine. But the mine cutoff must balance the value of each particular block of resource against every type of cash cost that is supported by the operation, including all downstream processing costs, as well as the amortization of the capital that was used to construct the new property.

Even in bedded deposits such as potash or trona, the ability or willingness to mine a lower seam height may mean that more ore can eventually be produced from the reserve. In such cases, the cost per unit of value of the product generally increases. Similarly, narrow-seam mining greatly reduces the productivity of the operation compared to high-seam and wide-vein (or massive) mining systems.

In evaluating the economic model of a new property (after the physical and financial limits have been considered), all of the variables of grade and tonnage, with the related mining costs, must be calculated using various levels of mine production that, in the engineer's judgment, are reasonable for that particular mineral resource. At this point in the analysis, the various restraints of production are introduced. This will develop an array of data that illustrate the return from various rates of production at various grades corresponding to particular tonnages of the resource. At a later stage, probability factors can be applied as the model is expanded to include other restraining items.

Timing Affecting Mine Production

For any given ore body, the development required before production start-up is generally related to the size of the production, as well as the mining method. Obviously, the necessary stripping time for most large porphyry copper deposits is very large compared to the stripping time for even a large quarry. For an underground example, very high production rates may require a larger shaft or multiple hoisting shafts, more and larger development drifts, the opening of more mineable reserves, as well as a greater lead time for planning and engineering all aspects of the mine and plant. The amount of development on multiple levels for a sublevel caving operation or a block caving operation will be extensive compared to the simple development for room-and-pillar operation. In combination, all of these factors could amount to considerable differences in the development time of an operation. In the past, the time for mine development has varied from 2 to 8 years. Two indirect economic effects could result from this:

1. Capital would be invested over a longer period of time before a positive cash flow is achieved; and
2. The inflation rate-to-time relationship in some countries in the past has been known to push the costs upward by as much as 10% to 20% per year, thereby eliminating the benefits of the "economy of scale" of large-size projects.

To aid the engineer in making approximations of the time it takes to develop shaft and slope entries related to the size and depth of the mine entry, the reader is referred to Bullock (1998). For example, sinking and equipping a 6.1-m (20-ft) diameter shaft ranging in depth from 305–914 m (1,000–3,000 ft) will take approximately 64–113 weeks, depending on the production capacity that the shaft must be equipped for. Driving a slope to the same depths will take approximately 52–151 weeks, respectively.

For mines that are primarily developed on one or two levels and have extensive lateral development, the speed of the development can vary considerably. The lateral development on each level of a room-and-pillar mine opens up new working places, and the mine development rate can be accelerated each time a turnoff is passed, provided that there is enough mining equipment and hoisting capacity available. This is in contrast to a vein-type mine that has a very limited number of development faces, as the vein is developed for each level and each level requires deepening the main hoisting shaft or slope. Experience has shown that the mine's development only progresses deeper at the average rate of about 50 m/a. Chapter 4.7 of this handbook discusses several examples of the timing of mine developments.

The timing of a cost is often more important than the amount of the cost. In a financial model, timing of a development cost must be studied in a sensitivity analysis. In this respect, any development that can be put off until after a positive cash flow is achieved without increasing other mine costs should certainly be postponed.

Government Considerations Affecting Mine Size

Government attitudes, policies, and taxes generally affect all mineral extraction systems and should be considered as they relate to the mining method and the mine size. One can assume that a mine is being developed in a foreign country and that the political scene is currently stable but is impossible to predict beyond 5–8 years. In such a case, it would be desirable to keep the maximum amount of development within the mineral zones, avoiding development in waste rock as much as possible. This will maximize the return during a period of political stability. Also, it might be desirable to use a method that mines the better ore at an accelerated rate to obtain an early payback on the investment. If the investment remains secure at a later date, the lower-grade margins of the reserve might later be exploited. Care must be taken, however, that the potential to mine the remaining resource, which may still contain good grade ore, is not jeopardized, whatever the stability situation. There is merit in carefully planning to mine some of the higher-grade portions of the reserve while not impacting the potential to mine the remaining reserve.

Some mining methods, such as room-and-pillar mining, allow the flexibility of delaying development that does not jeopardize the recovery of the mineral remaining in the mine. In contrast, other mining systems, such as block caving or longwall mining, might be impacted by such delays.

Similar situations might arise as a result of a country's tax or royalty policies, sometimes established to favor mine development and provide good benefits during the early years of production; in later years the policies change. Again, as in

the preceding case, the flexibility of the mining rate and system must be considered, but not to the extent of jeopardizing the remaining resource.

PREPLANNING BASED ON GEOLOGY AND ROCK CHARACTERISTICS

In the previous section, the influence of geology from a macro point of view was considered. In this section, the geology from a micro point of view of the rock and mineral characteristics must be considered in mine planning.

Geologic Data

Using geologic and rock property information obtained during preliminary investigations of sedimentary deposits, isopach maps should be constructed for all potential underground mines to show the horizons to be mined and those that are to be left as the roof and floor. Such maps show variances in the seam or vein thickness and identify geologic structures such as channels, washouts (called "wants"), and deltas. Where differential compaction is indicated, associated fractures in areas of transition should be examined. Areas where structural changes occur might be the most favored mineral traps but are usually areas of potentially weakened structures. Where possible, locating major haulage drifts or main entries in such areas should be avoided; if intersections are planned in these areas, they should be reinforced, as soon as they are opened, to an extent greater than that necessary elsewhere. Further details on development opening in relation to geologic structures and other mine openings appear in Spearing (1995), reproduced by Bullock (2001).

Again referring to flat-lying-type deposits, extra reinforcing (or reducing the extraction ratio) may also be necessary in metal mines where the ore grade increases significantly, since the rock mass usually has much less strength. Where the pillars were formed prior to discovering the structural weakness, it will probably be necessary to reinforce the pillars with fully anchored reinforcing rock bolts or cables. It is advisable to map all joint and fracture information obtained from diamond-drill holes and from mine development, attempting to correlate structural features with any roof falls that might still occur.

Characteristics of Extracted Material

The hardness, toughness, and abrasiveness of the material determine whether the material can be extracted by some form of mechanical cutting action, by drilling and blasting, or by a combination of both methods.

The mechanical excavation with roadheaders of the borate minerals from the Death Valley (California, United States) open-pit mine operated by Tenneco, and the later use of the same type of roadheader at the nearby underground Billie mine (California) illustrate the point. Bucket-wheel excavators have been used extensively in Germany and Australia for stripping overburden from the brown coalfields.

Technological advances in hard-metal cutting surfaces, steel strengths, and available thrust forces allow increasingly harder and tougher materials to be extracted by continuous mining machines. The economics of continuous cutting or fracturing, as compared to drill-and-blast, gradually are being changed for some of the materials that are not so tough or abrasive. However, for continuous mining—other than tunnel boring machines—to be competitive with modern high-speed drills and relatively inexpensive explosives, it appears that the rock strengths must be less than 103,400–124,020 kPa (15,000–18,000 psi) and have a low abrasivity. Rock that is full of fractures, however, is also a great aid to mechanical excavation. In one case that was covered in an article about the gradual trend toward mechanical excavation in underground mining (Bullock 1994), a roadheader was being used in a highly fractured, welded volcanic tuff, even though the rock strength was well over 137,800 kPa (20,000 psi).

At times, reasons other than the cost of extraction favor one mining system over another. Using a mechanical excavation machine is nearly always advantageous in protecting the undisturbed nature of the remaining rock where blasting might be prohibited. Likewise, the continuous nature of mechanical excavation can be used to speed mine production. This was seen in the openings driven by Magma Copper in developing the Kalamazoo ore body in Arizona, United States (Chadwick 1994; Snyder 1994) and by Stillwater Mining Company in developing their original ore body as well as for their East Boulder ore body in Montana, United States (Tilley 1989; Alexander 1999). A continuous boring tool may also be desirable for extracting an ore body without personnel having to enter the stoping area. Where their use is applicable, continuous mining machines are certainly much easier to automate than the cyclical drill-and-blast equipment. The automation of the 130 Mobile Miner at the Broken Hill mine in Australia is one case in point (Willoughby and Dahmen 1995). Another more dynamic case is the complete automation of the potash mines of Potash Corporation of Saskatchewan, Canada (Fortney 2001).

PLANNING THE ORGANIZATION

The amount of equipment or numbers of personnel required to meet the needs of all mines cannot always be given in absolute and precise terms. The purpose of this discussion is to mention some of the general problems that may be encountered and actions to mitigate those problems.

Work Force and Production Design

When planning the details of a mining work force, it is necessary to consider several factors:

- Is the supply of labor adequate to sustain the production level dictated by other economic factors? If not, can the needed labor be brought in, and at what cost?
- What is the past history of labor relations in the area? Are the workers accustomed to a 5-day work schedule, and if so, how will they react to a staggered 6- or 7-day schedule?
- Are the local people trained in similar production operations, or must they be trained before production can achieve full capacity?
- Will a camp have to be built and the workers transported in on weekly schedules?
- If long commutes are required to reach the property daily, are company-contracted busses for all shifts an option?
- If long commutes are required, are shifts longer than the normal 8 hours an option?
- Can people with maintenance skills be attracted to the property, or will the maintenance crew have to be built up via an apprenticeship program?

Apprenticeship programs are very slow in terms of producing results. Accordingly, some state laws in the United States restrict the annual number of people who can be trained in such programs. This one item could cause a mine designed

and equipped for a very large daily production to fall far short of its desired goals.

Work-force issues and local community resources need to be investigated at the same time that the property is being evaluated and designed. This will provide adequate time for specialized training, minimize unexpected costs, and also prevent economic projections based on policies that, if implemented, could negatively impact employee morale or community relations. The productivity and profitability difference between an operation with high morale and good labor relations and an operation where such parameters are poorly rated can be drastic. Such matters can make the difference between profit or loss. Of all the items involved in mine design, this one is the most neglected and can be the most disastrous.

Equipment Selection

Field-Tested Equipment

The equipment selected should be produced by manufacturers that field-test their equipment for long periods of time before they are marketed to the industry. Too many manufacturers build a prototype machine and install it in a customer's mine on the contingency that they will stand behind it and make it work properly. Eventually, after both the end user and the manufacturer redesign, rebuild, reinforce, and retrofit this model, a workable machine is obtained. The cost in lost production, however, is imposed on the mine operator, not the manufacturer. The manufacturer can then proceed to sell the "field-tested" retrofitted model to the entire industry, including competitive mines. Beware of taking on equipment in underground mines that has not been thoroughly field tested in severe applications by the manufacturer, unless it is going to be used for an isolated research application.

Equipment Versatility

The equipment selected for the mining operation should be as versatile as possible. Normally, for large surface mines, this is not much of a problem. For small quarries and for most underground mines, however, it can be a problem if the equipment can only be employed on a limited number of mining operations. For example, in one room-and-pillar mining operation (Bullock 1973), the same high-performance rotary percussion drilling machines used for drilling the bluff or brow headings were then mounted on standard drill jumbos for drilling holes for burn-cut drifting and stoping rounds and for slabbing rounds in the breast headings. Because the drills on these jumbos have high penetration rates, they were also used to drill the holes for roof bolts and, in some cases, the holes for the reinforcing pillars. The same front-end loader was used to load trucks in one stope and to perform as a load-haul-dump unit in another stope. By switching working platforms, the same forklift tractors served as explosive-charging vehicles and as utility service units for handling air, water, and power lines. They also served as standard forklifts for handling mine supplies. This equipment philosophy results in the following advantages:

- Less equipment must be purchased and maintained.
- Less training is required for operators and maintenance personnel. In addition, all personnel have a better chance of becoming more efficient at their jobs.
- Having fewer machinery types means having a smaller equipment inventory.

Possible disadvantages of this approach are the following:

- A more efficient machine may be available to do the task currently being performed by the versatile—but less efficient—machine.
- The mine may become too dependent on a single manufacturer to supply its equipment needs.

Equipment Acceptance

Equipment selected should have a very broad acceptance and must be in common use throughout both the mining and construction industries. However, since underground mines impose a headroom restriction not encountered on the surface, this is not always possible. Nevertheless, where headroom is not an issue, selecting a standard piece of equipment means that the components will have endured rigorous testing by the construction industry. Furthermore, equipment parts are normally off-the-shelf items in distributors' warehouses.

Application Flexibility

The selected equipment should be flexible in application. For example, the equipment should be able to accelerate and move rapidly, have good balance and control at high speeds, be very maneuverable, and have plenty of reserve power for severe applications. Both trucks and loaders should have ample power to climb every grade in the mine and be able to accelerate quickly to top speed on long, straight hauls.

PLANNING THE UNDERGROUND MINE SERVICE FACILITIES

Underground facilities such as underground pumping stations, power transfer and transformer stations, underground shops, storage warehouse space, ore storage pockets, skip-loading stations, lunchrooms, refuge chambers, shift and maintenance foremen's offices, and central controlled computer installations and communications all extract a strong influence on engineering study and design. More detailed information appears in Chapter 12.5, where these types of facilities are described and their designed size is discussed.

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CHAPTER 12.2

Hard-Rock Equipment Selection and Sizing

Peter L. McCarthy

INTRODUCTION

Whereas underground equipment should be selected to suit the mine, it is equally true that the mine should be designed to suit the equipment. It is impossible to begin mine planning and scheduling without a good idea of the type and size of equipment available and how it might be applied. Design of stopes or panels, mine accesses, development cross sections, ramp gradients, electric power reticulation, ventilation circuits, and so on must be based on an assumed type and fleet of equipment.

GENERAL APPROACH

In practice, equipment selection for a new hard-rock mine starts with the stoping operation. Stoping equipment should be appropriately sized for the characteristics of the ore body and the stope dimensions. Stope development openings should be sized to suit the stope dimensions and the stoping equipment capabilities and requirements.

Equipment should then be sized to mine the required size of stope development. Separate development fleets for access and stope development may be required. If this logic is not applied, big "efficient" development equipment may be acquired for access development, and then used in the stoping areas. The result is that the stope development is sized to fit jumbo drills, rock-bolter booms, or even the haul trucks, and must be far larger than the optimum, causing inefficiencies in stope design, poor geotechnical conditions, or excessive overbreak and development cost.

Of course, it is often necessary to select new equipment for an existing mine, where the constraints including access dimensions, ventilation, and so forth have to be identified and managed. In a shaft mine, for example, the machine size may be limited by the dimensions of shaft compartments and the component sizes in which the machine can be broken down for transport.

Proposed or possible changes to the mine plan, such as a move from a 5-day working week to continuous working, or a change in the decline gradient, will affect the equipment duty and must be considered. Sometimes a new machine may allow a new way of working. For example, replacement of rear-wheel-drive trucks with four-wheel-drive trucks would allow ramps to be steepened. This could reduce overall costs, and it is this larger context that must be analyzed.

Machines should be compared in terms of capital cost, operating cost, specifications, performance, availability, ease of maintenance, service and parts backup, and service life.

Fleet Description

The first step is to describe the key elements of the equipment fleet. For example, in a hard-rock ramp-access mine, this might include

- Two-boom jumbo drills,
- One-boom jumbo drills,
- Rock-bolting machines,
- Long-hole drills,
- Large load-haul-dump units (LHDs),
- Small LHDs,
- Haulage trucks,
- Integrated tool carriers, and
- Road graders.

Many other machines such as personnel transporters, shotcrete machines, air compressors, raise drills, and others may be needed, but the items in the list will be considered in determining stope and development layouts and dimensions.

Equipment Matching

Individual items of equipment, considered together, form a development system or a production system, and thus they must be physically compatible. Jumbo drills must be capable of reaching the limits of the largest excavation required, usually determined for the production drills and the trucks. LHDs must be able to reach and fully load the tubs (boxes or trays) of the trucks in three to five passes.

Sometimes a workaround will enable incompatible units to be used. For example, a "step" in the floor may enable small LHDs to load large trucks.

Electric machines should all operate from the same electric supply voltage. A decision must be made about whether compressed air will be reticulated, and if not, each pneumatic machine must be self-sufficient for compressed air.

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When items are obtained from a single manufacturer, the manufacturer will be able to determine how well matched they are. If they come from different manufacturers, some research may be needed, including detailed study of the specifications and discussions with staff of other mines about using the units together.

Choice of Manufacturer or Supplier

Manufacturers of underground equipment range from large, international firms with worldwide representation to small manufacturers that generally serve a local market or a specialized niche. In general, the international firms offer robust, reliable, and well-proven equipment at the upper end of the price range. Equipment is usually priced in euros or U.S. dollars, which may appear prohibitive when converted to a local currency.

Small regional manufacturers can offer attractive pricing and may offer a viable alternative to the name brands. Because their level of research and development is limited, and because they are building to a price, machines from such manufacturers generally have lower availability and may have reduced performance. These factors should be assessed carefully in deciding whether lower capital cost outweighs higher operating costs, including the impact of servicing and breakdown delays on the mining system as a whole.

Other manufacturers offer specialist equipment that is not available from the major suppliers. Examples of the products of international niche manufacturers include narrow-vein mining equipment from France and Canada, rock-bolting platforms from Canada, backfill slinger trucks from Germany, raise climbers from Sweden, and so on.

Equipment with similar specifications from the major manufacturers will generally give similar performance. This is often compared with the choice of buying a car from Ford or from General Motors—both will do the job. It is worth enquiring at other mine sites that use the same equipment, because specific models sometimes have problems that are not really resolved during the model run, whereas competing machines are free of those problems.

Apart from operating performance, equipment analysis should include the ease of maintenance. Some equipment models are easier to maintain, which can have a positive impact on the availability of the equipment.

The most important factor in choosing between otherwise similar machines is the level of backup support provided in a region by the manufacturer. Appropriate questions include the following:

- What training will the manufacturer offer to operators and maintenance personnel?
- Does the manufacturer have service facilities with trained personnel within an hour or two of the mine by road? If not, is it willing to establish such facilities on the basis of an order?
- Will the manufacturer stock a comprehensive range of spare parts in-country for any specific model? What is the likely delivery time?
- Will the manufacturer provide spare parts in the mine store on a consignment basis, so that payment is made only when they are used?
- Which major components are not stocked in-country? How long would it take to obtain them, if needed, and what would be the cost of urgent air freight?
- Will the manufacturer offer maintenance and consumables on a fixed cost per hour or per meter basis, and establish its own maintenance work force on-site?
- What is the reputation of the manufacturer for service and spares backup in the region?
- Are components interchangeable between different pieces of equipment such as the same axles in LHDs and trucks?
- Are the driveline components from the same manufacturer?

After these questions have been addressed, it may be advantageous to source the key fleet items, such as trucks, LHDs, and jumbo drills, from a common supplier. It is also attractive to minimize the number of different models of equipment on the mine site, even if from the same manufacturer, to minimize duplication of stores inventories or maintenance skill sets. Mixed manufacturer fleets, especially trucks, may result in unexpected operational conflicts such as different truck speeds, with slower trucks delaying faster trucks, especially when hauling loads up a ramp.

Appropriate Technology

Any new machine should incorporate a level of technology that can be understood and supported by the operators and maintenance team. Although it may be appealing to engineers to acquire the latest technology, the potential performance will not be delivered if the machine spends much of its time waiting for repairs. At the crudest level, there are mines that operate compressed-air rock drills and do not have the workshop facilities or the skills to maintain hydraulic rock drills. Similarly, many mines do not have the electronics capability to maintain the latest fully computerized jumbo drills and prefer to buy the versions with manual controls.

In general, the longer the supply lines to the mine and the less educated or skilled the work force, the more robust and less innovative should be the technology. It is also difficult to mix different technologies within one operation. For example, a mine developed some years ago in southern Africa tried to combine handheld drills and slushers in the stopes with rubber-tired jumbo and truck development and haulage. The two systems were not compatible: large numbers of walking stope miners impeded the movement of trucks; the rock broken in development was too large for the conveyor feeders; and the maintenance workshops had to service large numbers of stope drills and scraper winches, with which they were familiar, and smaller numbers of hydraulic jumbos and diesel trucks, which were new to them. The mine failed for these and unrelated reasons.

Mine systems should be designed to accommodate the technology to be used. For example, if a conventional mine is to be mechanized, not only must the required infrastructure be provided, but materials-handling logistics need to be thoroughly planned and assessed. For example, this could include transportation and storage underground of diesel, spare parts, and hydraulic oils.

Innovative Models

Innovative new models of machines, even from the most reputable manufacturers, may present operational problems. It often takes 2 or 3 years of operation to resolve design problems in trucks, loaders, and jumbo drills, with trucks giving the greatest problems. Usually the machine gets a bad name in the industry and a revised model with a new code name is

released after the problems are resolved. The problems are not predictable and could be as simple as shearing of wheel studs on a high-powered truck.

The promised performance of a new model may be appealing, and some mine has to be the first to try it. When considering such a machine, it is usually possible to negotiate a performance guarantee with the supplier. This should take the form of an operating availability guarantee, possibly with a minimum performance expressed as metric ton–kilometers per hour, meters drilled per hour, or similar. Financial compensation applies if the machine does not deliver the guaranteed performance.

It is usual for the manufacturer and supply agent to be very interested in the success of a new product, and they may provide high standards of operator training and assume responsibility for maintenance for an extended period. The duration of these above-normal levels of service should be ascertained.

Examples of new equipment that have proved disappointing in the past include

- Some high-powered 50–80 t (metric ton) capacity trucks;
- Some combined drilling and cable-bolting machines;
- Some rock-bolting jumbos;
- Many models of scaling machines;
- Undersized impact breakers; and
- Novel surface or soft-rock crushers and sizers, when applied underground in hard-rock mines.

Equipment Replacement

All equipment has a service life, after which it becomes uneconomical and should be replaced. For some items the service life may be equal to or exceed the mine life, in which case replacement is not necessary. Generally, major items such as headframes, winders (hoists), crushers, ball mills, and so on are designed to last the life of the mine and, in practice, may outlast several mines. Worldwide, a number of operations are still getting satisfactory performance from crushers and winders that were built in the 1930s.

Most portable and mobile equipment has a finite service life. Typically, new mobile equipment will operate at low maintenance costs for approximately 10,000 hours or 2 years, after which the cost increases steadily. If the use of the machine is lower than around 5,000 h/yr, this "honeymoon period" may be extended. This cycle may be repeated following one or two major rebuilds, although the subsequent maintenance costs are rarely as low as for a new machine.

If the working environment changes, a machine may be deemed obsolescent and need to be replaced. Examples include a change to the stoping method, requiring a larger machine, or a change in the ventilation standards that requires a cleaner engine. If there is a substantial improvement in the available technology, a machine may be replaced, even though it is still delivering to its original specification. For example, new rear-dump haul trucks with much-improved power-to-weight ratios may displace older, slower units.

A decision to replace an existing machine must consider the entire physical and financial environment. For example, in a struggling mining company, capital may simply be unavailable no matter how well a replacement can be justified. In such circumstances, the equipment life can be extended almost indefinitely but with a much-increased operating cost. To extend useful life, many operations cycle their equipment through two or more duties. For example, haul trucks may be converted to water trucks, or production LHDs become development LHDs, where they are used more intermittently.

A particular problem arises in shaft-access underground mines. The difficulty involved in dismantling and reassembling large mobile units for shaft transfer leads to inflexibility in maintenance, and major overhaul periods may be extended. Operators tend to retain old equipment as backup units long after they have reached the end of their economic life. This creates an environment of low usage and excessive maintenance costs. For example, one Australian shaft mine was able to reduce its mobile equipment fleet from 50 to 11 units after establishing a service decline from the surface.

Service life and maintenance costs also depend on a company's maintenance policy and the level of onboard monitoring installed.

Making Financial Comparisons

The benefit of investment in a new machine should be quantified in advance, and in most companies this is a requirement of the capital approval process. The economic benefit is best measured by comparing the net present values (NPVs, or usually net present costs) of the alternatives. If the expected service lives of the alternatives are different, the analysis becomes complicated. For example, two machines with service lives of 5 and 7 years would need to be compared (artificially) over a period of 35 years to get a meaningful comparison, or an accurate estimate be made of the residual value of the longer-life machine after 5 years. To overcome this problem, costs should be annualized using an approach known as the annual average cost or the equivalent annual value method.

Equipment replacement decisions are not exclusive to mining, but apply throughout all business activities. Detailed discussions of the methods can readily be found in textbooks on engineering economics and on the Internet, and suitable financial modeling software can be downloaded and purchased for a small sum.

Many mining companies require that the internal rate of return for the investment exceed a hurdle rate, a minimum acceptable rate of return that is set from time to time by the administration and is based on the company's cost of capital or desired rate of return on all of its investments. Although this may simplify decision making, it often leads to suboptimal decisions because it prohibits many investments that add some value to an operation, and, in particular, it ignores the many benefits that cannot be quantified in a simple financial analysis. Companies may also limit the timing of capital approvals within the annual budget cycle and take a dim view of "out of budget" applications.

Nonfinancial measures of performance may be important. For example, improvements in workplace safety or environmental quality may not be economically measurable but may be highly desirable. Provision of a new machine with an air-conditioned cab may improve industrial relations even if the machine specifications suggest there will be no increase in productivity.

At many mines, particularly at remote sites, there is a large benefit in reducing labor numbers. Increased mechanization will aid this, and can become a matter of policy even if it is difficult to quantify the gains from a particular decision on an NPV basis. Labor costs are typically >50% of total costs, but when the cost of extras such as airfares and site accommodation are included, they may be higher yet.

Conversely, the social–political climate in some areas may dictate that reduction in labor by introduction of more productive machines is politically undesirable. An example is the large amount of hand labor on ramp roadway maintenance seen in many Mexican mines when a grader would be more efficient.

Information Required for Financial Analysis

The capital cost of a new machine should be estimated including the cost of spares, which can be 20% of the purchase price for a mobile unit. This will depend on whether spares are available on consignment; that is, they can be held in the store and paid for when used. Estimators should use the final or likely negotiated price rather than the manufacturer's list price, which is usually somewhat higher.

If an item is sourced overseas, estimators should check what currency will be used for the transaction and what protection (if any) is available against exchange rate variations. Any payable import duties and the transport and unloading costs including in-country transport to the mine should be checked. For a shaft mine, the transport cost will include shaft transfer and reassembly, which can be substantial for bigger items.

Operating costs can be estimated from

- Detailed cost records and performance of similar equipment,
- Cost records for the new item from another operation,
- Manufacturer's cost and performance estimates, and
- A database of costs from a range of operations.

To support ongoing decision making, good record keeping and cost allocation are required over the life of each unit. The mine accounting system must be able to track each piece of major equipment and not report a combined cost for all LHDs, trucks, jumbos, and so on. Records should include engine hours and, for rock-drilling equipment, percussion or hydraulic pump hours, as well as outputs such as metric tons loaded, metric ton–kilometers hauled, or meters drilled.

Caution must be used when comparing costs from different operations, where different operating conditions may apply and the basis of reported costs may not be well understood. Engineers should be particularly wary of manufacturer's estimates, as they are inevitably optimistic. This is an area where consultants can be very helpful, as a good consultant should have an extensive cost and performance database.

The salvage value may sometimes be important. For mobile equipment it will depend on the age of the unit, its condition, and the operating hours since a major rebuild. A rebuilt 10,000-hour underground mobile item could be worth 60%–70% of new cost. The rebuild itself might cost 20%–30% of the original purchase price. An older unit will generally be saleable for about 20% of new cost, if it is operable. Machinery merchants are generally willing to provide an estimate of value for a used item, and for common items may not require an inspection.

The cost of removing the old item from the mine, particularly if it is a shaft mine, may negate the salvage value, in which case the unit should be pushed into a stope and backfilled.

Taxation effects and the method of financing the new item may affect the analysis.

Leasing and Contracting

Besides outright purchase, other forms of equipment finance are available:

- An operating lease is similar to rental, is usually for short-term requirements, and can be cancelled without penalty. It is relatively expensive, because the lessor assumes the risk of re-leasing and technological obsolescence, but is fully deductible as an operating expense. It does not affect the lessee's balance sheet.
- A finance lease is a contract, whereby payments are made over much of the useful life of the asset. During this time the lease cannot be cancelled, although it may be paid out if the item is disposed of or destroyed. The cost includes capital (less salvage) cost, interest, a risk premium, and the commercial cost of providing the service. Typically 60%–70% of the purchase price is financed, with the balance payable as a residual. The item appears on the lessee's balance sheet.

As an alternative to leasing, a contractor (and contractor's equipment) could be used. Applications include, for example, truck haulage, production drilling, surface crushing, and the entire mine development or production program. Contracting is often appropriate where a small additional increment of capacity is required for 1 or 2 years. It also makes sense where specialist skills such as cable bolting are required for a limited time, or where the necessary expertise is not available in the vicinity of the mine and there is insufficient time to train a work force.

Estimating the Average Annual Cost

The average annual cost is the sum of depreciation, interest, and operating cost. Consider a truck that has an initial cost of $1.0 million and a salvage value of $0.2 million after an estimated 4-year service life. The depreciation is thus $0.8 million over 4 years or $0.2 million/yr.

The interest expense arises, because owning an asset ties up capital. This is true whether all or part of the purchase price is borrowed or existing cash is used. In the latter case, there is an opportunity cost because that money could have been invested elsewhere. The average investment is

$$(\$1.0 \text{ million} + \$0.2 \text{ million})/2 = \$0.6 \text{ million}$$

If the company has a cost of capital (or cost of borrowing) of 10%/yr, the annual interest expense is $0.1 \times \$0.6$ million or $60,000 for each of the four years.

The annual operating cost may be calculated from the estimated hourly operating cost of $200/h (which includes operator, fuel, tires, maintenance, etc., and the cost of a major rebuild at 12,000 hours) and the expected use of 5,500 h/yr, which is a total of $1.1 million/yr. From this, the average annual cost is

$$\$0.2 \text{ million} + \$0.06 \text{ million} + \$1.1 \text{ million} = \$1.36 \text{ million}$$

A similar calculation can be made for each of the trucks under consideration. The equipment selection decision can then be based on these results, together with all of the other important considerations outlined in this chapter.



Courtesy of Atlas Copco.
Figure 12.2-1 Articulated rear-dump truck

A more accurate calculation can be made, if required, to include consideration of tax effects. The capital tax factor (CTF) is used to adjust capital costs for tax effects. It is determined by

$$CTF = 1 - (td / (i + d))$$

where
t = corporate tax rate
d = annual depreciation rate allowable by the taxation authority (average)
i = cost of capital or interest rate

For example, if $t = 30\%$, $d = 20\%$, and $i = 10\%$, then

$$CTF = 1 - (0.3 \times .20)/ (.10 + .20) = 0.8$$

The CTF is applied to capital costs and salvage values, whereas interest and operating costs are adjusted using $(1 - t)$ to get the after-tax cost. In the example,

annual depreciation = \$0.2 million $\times$ 0.8
= \$0.16 million

annual interest cost = \$0.06 million $\times$ 0.7
= \$0.042 million

annual operating cost = \$1.1 million $\times$ 0.7
= \$0.77 million

Hence, the average annual cost (after tax) is \$0.972 million.

Of course, all the trucks must be evaluated in the same way, either before or after tax, and experience shows that the rankings rarely change. Although more-accurate spreadsheet-based evaluations are possible, taking into account the time value of money, the average annual cost is only one of the considerations in selecting the most-appropriate equipment.

HARD-ROCK EQUIPMENT SELECTION

The key items of mobile equipment are loaders, trucks, jumbo drills, and production drills, each requiring a different approach to selection. Specific selection criteria apply to each item of ancillary equipment. In all cases the considerations discussed in the previous sections should be addressed.

Selecting a Truck

Trucks used underground fall into three categories: rigid-body rear-dump trucks adapted from surface mining units; articulated rear-dump trucks that may be adapted from surface mining units but usually are purpose designed to have a low profile (Figure 12.2-1); and tractor–trailer units, some with separately powered trailers, usually side-dumping, which can be assembled as an underground road train (Figure 12.2-2). All have diesel engines except for trolley-electric trucks, which require special infrastructure.

Where trucks are required to travel through development openings, the truck cross section will fix the dimensions of the opening, or vice versa. Allowance must be made for clearance from the load to any pipes, ventilation duct, or other services; and clearance to side walls, which may be a statutory requirement. A typical design profile is shown in Figure 12.2-3.

The clearance to ventilation ducts is influenced by the following:

- The shape of the tunnel profile, whether flat, arched, or shanty-backed, which is determined from the geology and ground-support requirements.
- The duct diameter, which depends on the airflow required. Sometimes oval-shaped ducts are used to improve clearances but at the cost of greater resistance and fan power consumption.
- The type of duct, whether rigid or flexible. Flexible ducts may be more mobile and require greater clearances.
- The size, number, and position of ducts within the cross section. The position could be in the crown or in one corner of the profile. In wider headings, vent ducts could also be along a sidewall.
- The size and number of ducts depends on the planned distance of forced ventilation (before connecting to a return airway) and the ventilation volume required to service the heading. A forced ventilation heading may typically require one loader and one truck. The ventilation volume is usually a relationship based on a factor relative to the



Courtesy of Powertrans Pty Ltd.
Figure 12.2-2 Underground road train

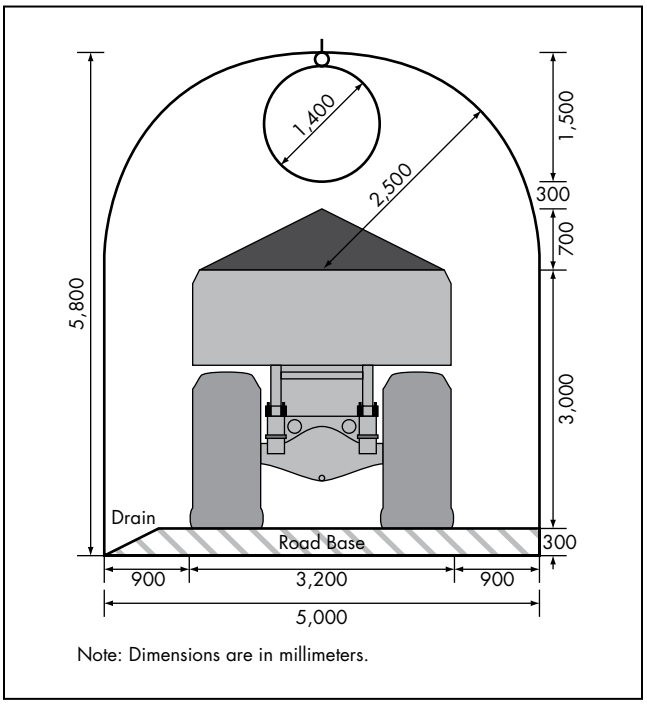


Figure 12.2-3 Typical mine development profile for haul truck

engine power. Duct diameter is then related to required ventilation volume and duct resistance. Duct diameter may become quite large (>1.2-m diameter) for significant forced ventilation lengths, which may require duplication or a mix of types (forced and push–pull systems).

- The expected load profile in the truck. Sometimes trucks are not fully loaded during the development phase, but can be fully loaded after the development is completed, when a ventilation circuit is established and the ventilation ducts have been removed.
- The initial thickness of road base material and any subsequent buildup of road base dressing material that is not removed by grading, and allowance for road spillage that a truck may accidentally ride over.
- The layout of turnouts from the main drive and the way ventilation ducts make the transition through corners.

Sidewall clearance is required to minimize impact damage and for safety reasons. There must be room for pedestrians to walk past a stationary truck and for recovery work to be done in the event of a breakdown or tire failure. It is not acceptable for trucks to pass pedestrians.

Trucks require a minimum turning radius, which generally will not be <20 m due to truck requirements (capability and maintenance) and development issues on grade (overbreak on development rounds). Widening of the heading and/or flattening the gradient may be necessary on bends to maintain required clearances or reduce longitudinal torsion of the truck frame.

Road trains require additional attention to radius and wall clearances due to the variable tracking path of the powered trailers. Development loops are also required for the road trains as they are not designed for reversing, and tipping is usually sideways.

The operating cost per metric ton–kilometer decreases as the truck capacity increases. Thus, there is an incentive to maximize truck capacity, with diesel trucks of up to 60-t capacity being available for underground applications. A trade-off study is required to assess the benefit of the low truck operating cost against the disadvantages that come with size. Factors to be considered include the following:

- The volume of waste rock broken increases with the cross-sectional area of the heading. For example, a 5.5 × 5.5 m drive will produce 53% more waste rock than a 4.5 × 4.0 m drive. If the cost of development breaking and disposal is \$130/m^3 and 10 km of primary development is planned, the cost difference of more than \$1.2 million must be recovered from a reduced trucking cost for all of the ore and waste rock from the mine.
- Ground-support costs generally increase with the square of span.
- Larger, more powerful trucks require more ventilation per truck but not per ton moved.

A larger mine opening with larger trucks will have a greater total production capacity in metric tons per day or per year.

Equipment manufacturers provide comprehensive manuals of specifications of their products, and many provide detailed approaches to equipment selection. The key considerations for an underground truck are

- Physical dimensions and load capacity;
- Exhaust gas and exhaust particulate quality;
- Safety features including falling object protection system and rollover protection system, and whether these comply with local standards;
- Air conditioned, enclosed cabin;
- Ergonomic cabin and seat;
- Power-to-weight ratio, and hence speed on grade;

- Engine braking capability or hydraulic retarder;
- Whether fitted with spring activated, hydraulic release fail-safe brakes;
- Whether the torque converter locks up to provide better power transfer and longer transmission life;
- Centralized lubrication; and
- Centralized fire protection system.

The truck load can be calculated from the truck body volume, the broken density of the rock to be carried, and a fill factor that typically varies from 85% to 95%. The stated tonnage capacity of the truck is typically quoted as "SAE heaped" (where SAE refers to the Society of Automotive Engineers), which has the top surface of the load rising one length unit vertically for every two length units horizontally from the top edge of all sides of the tray. This is rarely achieved in practice, particularly for up-ramp haulage. It is not unusual to find that a 50-t-capacity truck achieves an average payload of 43 t, particularly if there is any problem with back clearance on the haul route. Capacity is maximized by using a well-matched loader and with the use of an onboard weightometer.

The manufacturer's tables will provide torque-speed curves, speed-on-grade curves, and other information to allow cycle times to be estimated. In general, it is sufficient to determine the cycle as the sum of

- Loading time (including any wait for loader), which will depend on the loader capacity and the distance from the muck pile to the loading point;
- Travel time (at an average speed on grade) from the loading point to the dump point, usually including level and uphill segments;
- Dump time (including any delay waiting for other trucks to dump); and
- Travel time (at an average speed on grade) back to the loading point.

The average speed, when loaded, will depend on the condition of the roadway, including the presence of spillage or water, the clearance to the tunnel walls, the road gradient, the number of curves or bends, and any mine-imposed or statutory speed limit. The empty travel speed, if downhill, may be limited for safety reasons to less than the general mine speed limit. Similarly, there may be load or gradient limits for loaded travel downhill.

A typical cycle time calculation for a 50-t-capacity truck in an efficient decline-access mine might appear as follows:

Loading	2.5 minutes
Travel loaded	3.5 km at 7 km/h = 30.0 minutes
Dumping	1.0 minutes
Travel unloaded	3.5 km at 9 km/h = 23.3 minutes
Total cycle time	56.8 minutes
Efficiency factor	90%
Cycle time	63.1 minutes

Thus, if the truck can be loaded to an average 46-t payload, the productivity will be 43.7 t per effective operating hour. If traffic congestion is expected to interfere with the trucking cycle, then a further congestion factor may be introduced to increase the estimated cycle time. One simple way to estimate this factor is to assume that a truck returning empty must pass every other truck on its loaded haul out of the mine. If, as is typical (but not necessarily efficient) practice, empty trucks wait in passing bays for full trucks to pass them in the opposite direction, the cycle time could be increased by the time it takes for a full truck to travel half the average distance between passing bays multiplied by the number of trucks operating, less one.

After the cycle time has been determined for a particular haulage route, the tonnage to be moved, the effective time per shift, and the truck availability must be considered to determine how many trucks are required. It is almost always the case in underground mining that the best outcome is achieved by having an excess of loading capacity, so that truck use is maximized. In other words, it is better to have loaders waiting for trucks than have the trucks waiting for loaders. Often the loaders can spend the time between truck arrivals moving the muck up to a temporary stockpile at the loading point to minimize the loading time and turn the truck around quickly.

If there is in fact little or no queuing of trucks at loaders, the cycle time could theoretically be reduced by having slow full trucks wait for faster empty trucks to pass, rather than the more typical rule noted previously. If this were done, the average delay time in passing bays would be reduced. However, there are other practical operating reasons for why this would not usually be done.

In the example, assume a continuous 24-hour operation with two 12-hour shifts per day. Realistic allowances might include

Fuel up, safety meeting, and travel to job	1 h/shift
Meal breaks	1 h/shift
Job inspections	0.5 h/shift
Planned maintenance (8 h/week)	0.6 h/shift
Breakdowns/unplanned maintenance	0.5 h/shift
Job or efficiency factor	90%
Effective hours per shift (8.4 h × 0.9)	7.56
Effective hours per year (7.56 h × 730 shifts)	5,500
Annual truck production (5,500 h × 43.7 t)	240,350 t

If the target production is 1.5 million t/yr of material brought to surface, then the number of trucks required would be

$$1{,}500{,}000/240{,}350 = 6.2 = 7 \text{ trucks}$$

Traffic congestion occurs when the number of trucks required is high relative to the length of haul, so efforts should be made to provide effectives traffic management through the use of radio communications, block lights, and well-spaced, efficient passing bays. The problem is exacerbated for mixed-fleet systems (different capacity, different loaded speeds).

The difference between the efficiency factor introduced within the cycle time calculation and the job or efficiency factor discussed here is that the former increases the operating engine hours required and, hence, the annual operating cost, whereas the latter reduces the operating engine hours available. A useful check is to see that the annual hours required from the truck are consistent with experience at similar sites. There can be a large difference in annual operating hours for underground haulage trucks ranging from around 2,000 hours up to 5,600 hours depending on the management strategy and mine layout.

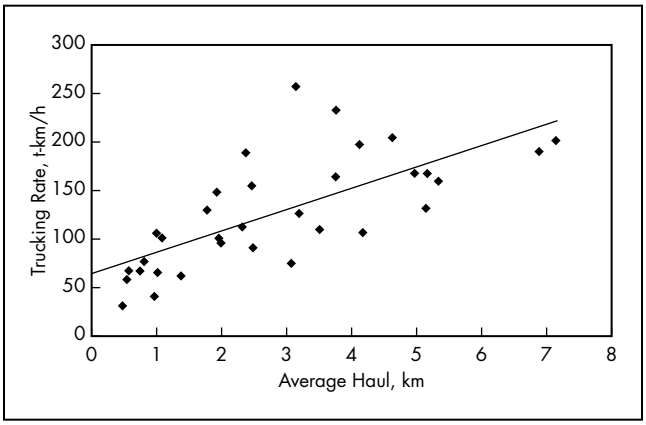


Figure 12.2-4 Truck productivity as a function of haul distance

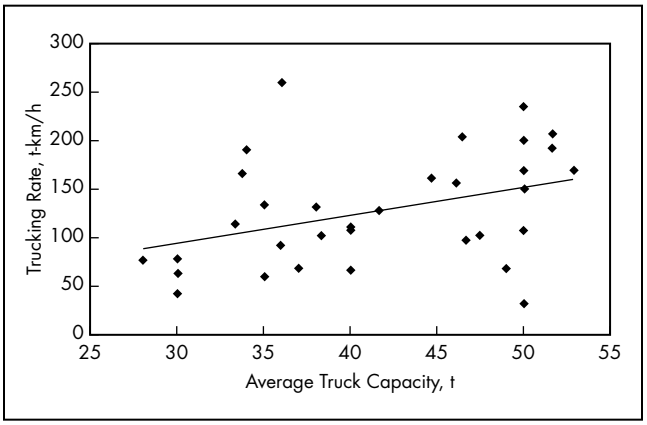


Figure 12.2-5 Truck productivity as a function of truck capacity

Because haulage routes vary, it is often useful to compare truck performance on a metric ton–kilometer basis. This assumes that hauling 1 t for 10 km requires about the same truck capacity as hauling 10 t for 1 km. In the previous example, a 50-t truck hauled 43.7 t/h over a distance of 3.5 km, an output of 153 t-km/h. Actual truck performance, as recorded from 32 operating mines, is shown in Figures 12.2-4 and 12.2-5.

The metric ton–kilometer approach should be used cautiously, because it ignores roadway gradient, which can vary substantially. Whereas some costs, such as tires, vary with load and distance traveled, others, such as fuel, are more closely correlated with engine hours and engine power setting.

When calculating the number of trucks required, allowance must be made for the haulage and disposal of waste rock from development. Often this rock will be backfilled into stopes, requiring a shorter cycle time than a haul to the surface. However, backfilling often causes delays while waiting for a loader or bulldozer to clear the dump point, so that the advantage may not be great. Where trucks must dump fill rock into stopes under a limited back height, ejector-tray trucks may be used. These have an ejector blade, something like a bulldozer blade, which pushes the load from the truck body without having to elevate it into the tipping position.

Ejector-tray trucks have a reduced payload compared to the equivalent standard model, and because of the increased weight in the truck body, they are less fuel efficient. They also have more moving parts to maintain and can suffer damage if large rocks wedge between the sidewall and the ejector.

Trolley-wire electric trucks have been available for many years but have limited application because

- The system has a high capital cost when the trolley-wire installation is considered;
- Although there is no diesel exhaust, they require substantial ventilation for cooling;
- If the mine generates its own power from diesel on the surface, there is no operating cost saving;
- A very high standard of road maintenance is required; and
- Without auxiliary power, trucks cannot travel beyond the fixed trolley wire into actively developing areas.

Despite these limitations, trolley-wire trucks may offer advantages for a mine using low-tariff grid power with a fixed loading point, such as the bottom of an orepass system.

Selecting a Load-Haul-Dump Machine

LHDs are known variously as loaders, muckers, or boggers and differ from front-end loaders in that they have a lower profile and are designed to carry a load in the bucket efficiently for up to 200 m between the muck pile and the truck or orepass that they are loading into. Thus, an LHD spends more time traveling than does a front-end loader, and is engineered accordingly.

A mine may use different-sized LHDs for stoping and for development. Often two or more sizes of LHD are used for stoping, to suit the ore-body width and mining method at different locations within the mine.

Standard models are available in bucket capacities ranging from around 3 to 11.6 m^3. Special machines designed for narrow-vein mining can be as small as 0.5 m^3. Corresponding payloads range from 1 to 25 t. Extremely low-profile machines are also available for working in flat-dipping narrow-vein ore bodies with as little as 1.6 m of headroom.

LHDs may be diesel or electric powered. The diesel units are versatile and can tram quickly from one location to another (Figure 12.2-6). The electric units (Figure 12.2-7) carry a cable drum and rely on a trailing electric cable, so they are tethered to a location during normal production. Electric LHDs have low noise levels and zero emissions, so they require less ventilation. They are highly productive in situations such as block caving, where ore is transported from a series of drawpoints to a fixed orepass location.

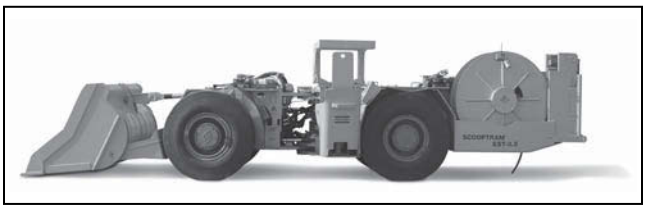
The ergonomic and safety considerations for selecting an LHD are similar to those for trucks, as listed previously. In the past, some operators sustained back injuries due to poor operator location and poor seating. The best operating position is midway between the front and rear wheels, and particular attention should be paid to the quality of the seat suspension.

LHDs often operate in situations with limited forced ventilation, so an enclosed, air-conditioned cab is desirable, and the engine should be the latest-generation, low-emission type. Fail-safe brakes are highly desirable. LHDs can operate effectively in headings as steep as 20% (1 in 5), although they perform best if the gradient is limited to around 14% (1 in 7) or less.

The size of a selected LHD must fit within the planned development and stope openings, and the bucket must be able to reach above and fill a truck efficiently.



Courtesy of Atlas Copco.
Figure 12.2-6 Diesel load-haul-dump unit



Courtesy of Atlas Copco.
Figure 12.2-7 Electric unit showing cable drum

The bucket load can be calculated from the bucket volume, the broken density of the rock to be carried, and a fill factor, which depends on the fragmentation of the rock and may be different for development rock and for broken ore in stope drawpoints. As with trucks, the nominal capacity quoted is typically for SAE-heaped loading, which is also typically difficult to achieve in practice. However, a high fill factor can be achieved with well-blasted rock. For example, an 8-m^3 bucket with a 90% fill factor and broken ore of bulk density 2.2 t/m would carry 15.5 t.

The manufacturer's tables will provide speed on grade and other information to allow cycle times to be estimated. Some manufacturers provide estimates of productivity in metric tons per hour for various haulage distances.

In general, it is sufficient to determine the LHD cycle as the sum of the following:

- Loading time, which depends on the power of the LHD, the muck pile condition, the floor condition, and fragmentation of the broken rock. The presence of large rocks, which require individual attention, can slow the loading cycle considerably.
- Travel time (at an average speed on grade) from the loading point to the dump point.
- Dump time (including any delay waiting for trucks), which is usually faster when tipping into a pass because no attention to the shape of the load is required. However, depending on the fragmentation of the rock and the size of the pass, grizzlies or other methods of size restriction to limit the size of rocks in the pass may constrain the tipping rate into a pass.
- Travel time (at an average speed on grade) back to the loading point.

A more complex cycle may apply. For example, it is usual to cut stockpile (remuck) bays every 120–250 m along a development heading. After blasting, the LHD quickly moves the muck from the round into the stockpile bay to allow ground support and drilling to proceed, after which the LHD loads trucks from the stockpile. It may be necessary to consider the LHD performance to estimate times for the two loading cycles within the overall development cycle.

The average speed, when loaded, will depend on the condition of the roadway, including the presence of spillage or water, the clearance to the tunnel walls, the road gradient, and the geometry of the tramming route. A typical cycle time calculation for an 8-m^3 LHD that is loading trucks might appear as follows:

Loading	30 seconds
Tramming loaded	200 m at 7 km/h = 103 seconds
Dump into truck	30 seconds
Travel unloaded	200 m at 7 km/h = 103 seconds
Delays, waiting	60 seconds
Total cycle time	5.4 minutes
Efficiency factor	90%
Cycle time	6.0 minutes
Cycles per hour	10
Metric tons per hour	155 (using 15.5 t/bucket from previous calculation)

As in the truck example, the effective hours must be determined. For a continuous 24-hour operation with two 12-hour shifts per day, realistic allowances might include the following:

Fuel up, safety meeting, and travel to job	1 h/shift
Meal breaks	1 h/shift
Inspect job, wash down, bar down	0.5 h/shift
Planned maintenance (8 h/week)	0.6 h/shift
Breakdowns/unplanned maintenance	1.0 h/shift
Job or efficiency factor	80%
Effective hours per shift (7.9 h × 0.8)	6.32
Effective hours per year (6.32 h × 730 shifts)	4,614
Annual LHD production (4,614 h × 155 t)	715,170 t

If the target production is 1.5 million t/yr of ore, the number of production LHDs required would be

1,500,000 / 715,170 = 2.1 = 3 production LHDs

The number of production LHDs required could be higher if

- LHDs are locked into production areas, or the tramming distance between production areas is excessive;
- More redundancy is required to ensure that production targets are met in the event of a major breakdown;
- There are periods of time in the stoping cycle when there are insufficient active drawpoints to use the available loaders; and
- LHDs are operated on remote control for stope cleanup, which reduces productivity.

A separate calculation is required to determine the number of development LHDs required.

Some LHD models are available with ejector buckets for discharging under limited headroom, which can reduce the

need for back stripping at loading points, and can be useful for tight backfilling.

Remote Control and Automation of LHDs and Trucks

Remote control of loading units is desirable where conditions at the face or in the stope are hazardous, or where the ground overhead has not been secured. Radio remote control of LHDs has been available for many years, having evolved from cable-remote control of compressed-air-powered loading units. Line-of-sight remote control has proven to be hazardous and has fallen out of favor. Despite training and strict operating rules, operators tend to move into the path of the loading unit and hence are exposed to injury.

Teleremote control, using television cameras mounted on the LHD, allows the operator to be seated in a safe, air-conditioned environment underground or even to operate the machine from the surface. Current developments include employing the operator in a city office, remote from the mine, thus making the job more attractive while greatly reducing on-site costs.

The productivity per hour of an LHD is typically less under teleremote control than with an operator on board, because the teleremote operator does not have the full range of audible, visual, and physical inputs to enable optimum performance. However, most teleremote operations currently use two-dimensional video screens for providing visual information to operators. Such things as three-dimensional (3-D) visual presentation and virtual-reality helmets, available in other industries, have been trialed in mining operations but have not come into widespread use. It is conceivable that in the future, provision of real-time 3-D virtual-reality visual and audio information to remote operators may reduce the difference in productivity between remote control and direct operation. Also, teleremote operation offers the possibility of operating for a greater part of the working shift because delays such as travel time do not apply and because operators can be rotated in a surface office to keep the machines at maximum use.

The major manufacturers now offer autonomous LHDs that can operate with minimal or no human intervention. These machines have sophisticated onboard computers and typically use lasers to scan their environment and determine position. Such a machine can load, tram, and dump autonomously so that a human supervisor can manage two or more LHDs.

The loading part of the cycle requires the muck pile to be well broken. The current generation of autonomous machines is not as effective as humans at assessing the position of large rocks and maneuvering the bucket to displace and lift them. The LHD can be damaged if used autonomously with large rocks, so a common compromise is to load the LHD on teleremote and then switch to autonomous for the remainder of the cycle.

The use of surface teleremote and autonomous LHDs requires a good broadband communications backbone to be installed in the mine, and a high standard of training for maintenance personnel. For safety reasons these machines can only be used in an area that has been locked off from human entry, so the opportunities for their use are limited. They are suited to block caving and some sublevel caving operations where production activities can proceed for a significant time without the need for other activities around them. In many other mines, activities such as development, geological sampling, surveying, and so on interact with production loading, so that it is impractical to create a nonentry zone.

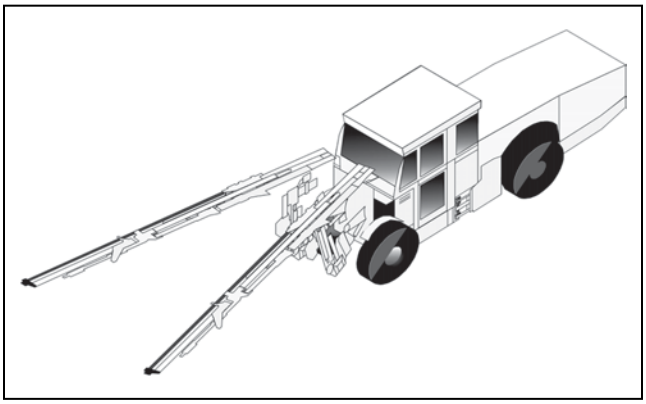


Figure 12.2-8 Two-boom jumbo

For autonomous operations, the mine must be designed to accommodate the level of technology. Where automation has been adapted to an existing mine layout, the results have generally been unsatisfactory. Design of operations needs to include interaction between activities; dealing with spillage, drainage, and road maintenance; blast damage; planned and unplanned maintenance activities; surveying; geological mapping and sampling; and so on.

Autonomous haulage trucks are also available but are not yet widely used. Autonomous trucks must operate on a dedicated haulage route that is locked off from other traffic and from pedestrian traffic, which can rarely be arranged in existing mine layouts. In most cases, duplicate development of a dedicated haulage roadway would be required and may not be justified.

Selecting a Jumbo Drill

A modern jumbo drill has an articulated diesel-powered carrier and an electrically powered hydraulic pump, which operates the boom rams and hydraulic rock drill(s) when the machine is in drilling mode. The jumbo may have one, two, or three booms, depending on the size of openings.

The diesel engine enables the jumbo to tram between working places. It is rare for the diesel engine to provide hydraulic power for drilling because it is inefficient, and because there is usually insufficient ventilation to operate the engine at the face. Some models have a "fast tramming" option that is useful in a large mine where the work places are far apart.

Single-boom jumbos are suited to face areas from 6 to 30 m^2, whereas twin booms may fit in an opening as small as 8 m^2 but can extend 100 m^2 or more. Two booms are sufficient to cover the face area in most mines, with three-boom jumbos being more popular in civil engineering works. Where a three-boom jumbo would, in theory, provide a faster drilling cycle, in practice, the operator is fully busy managing two. A valuable application of the center boom is drilling the large diameter reliever holes in the cut. Many jumbos have a third boom with a basket. A single boom is used in small headings or narrow veins, where there is insufficient room to operate a two-boom machine (Figure 12.2-8).

The manufacturer's data will show the capability of the jumbo to cover the proposed face area. The blast design will determine the hole diameter, generally in the range of 33 to 89 mm, although larger holes can be drilled at greatly reduced penetration rates. Depending on the duty, rock drills (drifters)

with impact power in the range of 10 to 30 kW are available. The length of round to be drilled and fired will determine the length of the drill steel and the boom. In general, longer, heavier booms require a larger carrier.

It is important that the jumbo has adequate reach to enable back holes to be drilled parallel with the axis of the excavation. If an undersized machine is used to drill large profile excavations by angling the back holes out (not parallel to the direction of advance), the result may be a sawtooth profile, excessive overbreak, and unsatisfactory conditions for ground control.

High-powered drifters may require a high-voltage system (1,000 V) to deliver sufficient energy to the hydraulic pumps. The voltage also impacts the trailing cable diameter and, hence, trailing cable length and power extension requirements. This is of relevance for countries where lower-voltage systems (~450 V) are the norm.

An important consideration is whether the jumbo will also be used for rock bolting and meshing, either consistently or occasionally, and the type of rock bolts to be used. For rock bolting, it must be possible to turn the booms at 90° to the axis of the carrier, and to drill holes into the walls and back. The boom length cannot exceed the available width and height. In these situations a split-feed boom may be required, which effectively telescopes the boom to full length as required. This adds complexity, maintenance, and weight and may lead to inaccuracy as parts wear, so the benefit must be weighed against these considerations.

A jumbo may also be used for scaling or "rattling" the tunnel walls and back, as part of the development cycle, in lieu of barring down or using a dedicated scaling machine. This is never recommended by the drill manufacturers because it can cause a lot of damage to the booms from falling rock. Nevertheless, it is a common practice as operators have learned that the convenience and safety benefits can outweigh the increased maintenance. For this application, a robust and well-protected boom should be selected.

The face advance that can be achieved from a jumbo drill depends on the number of faces available to it. In a single-heading situation, such as during the development of a main decline from the surface, a twin-boom jumbo can typically average 45–50 m of advance per week, including the main heading and stockpile bays. This means that the main face will advance, on average, about 40 m/week. This rate can be exceeded for periods of several months if in good ground, but most mines present a range of conditions, including zones of poor ground or faults, which slows progress, so that over a 2-year development program the 40-m/week rate would apply.

A higher advance rate can be achieved using the latest "long round" technology. For example, during the development of Newcrest Mining's Cadia East project in New South Wales, a record for 6.0 m × 5.5 m mine development with full ground support was set with an advance of 283 m in a single month (including stockpile bay development) (Willcox 2008). Long round development has limitations on curve radius and turnout or crosscut development, so is not generally applicable to underground mine development. Higher development rates exceeding 400 m/month have been achieved in civil tunneling works, but the Cadia East performance is a good example for mine development.

The process of estimating the performance of a two-boom jumbo drill is set out in Tables 12.2-1 through 12.2-3, which are based on actual measurements from a mine where each jumbo had at least five faces available to it for drilling, and so could be kept continuously at work. Causes of delays were recorded in detail on shift reports. The jumbo drills were conventional units without computer alignment and were not the latest generation of high-frequency drifters.

Table 12.2-1 Physical development parameters

Parameter	Measurement
Width, m	5.0
Height, m	4.5
Area, m^2	22.5
Rock density (wet), t/m^3	2.8
Overbreak, %	5
Metric tons per meter (including overbreak), t/m	66.2
Drill factor, charged holes/m^2	2.20
Number of charged holes	50
Number of 100-mm cut holes	4
Hole depth, m	3.05
Pull, %	90
Advance per round, m	2.75
Metric tons per round, t	181.6
45-mm-diameter face holes, meters drilled, m	151.0
100-mm-diameter cut holes, meters drilled, m	12.20
Total face meters drilled, m	163.18
Rock bolt length, m	2.4
Friction bolt holes per round	5
Resin bolt holes per round	8
Rock bolt meters drilled per round, m	31.2
Total small-diameter meters drilled per round, m	182.2
Face meters drilled per meter advance, m	59.4
Total meters drilled per meter advance, m	70.8

After this analysis is completed, using performance parameters for the jumbo drill under consideration, the required number of jumbos can be estimated. If the target is 800 m of development per month, then, in this example, three jumbos would be required. Of course, if sufficient faces cannot be made available to fully use the drills, this problem is not solved by adding more drilling capacity. Instead, the materials handling capacity must be addressed.

Selecting a Rock-Bolting Machine

Rock bolting may be done using a handheld rock drill from the muck pile, a work basket, or a scissor-lift platform. Purpose-built articulated, four-wheel-drive scissor lifts are available with or without a bolting boom attached. Alternatively, the development jumbo drill can be used to install rock bolts and mesh with minimal exposure of the operator to the area being supported.

Purpose-designed machines are available for drilling holes and installing both conventional rock bolts and cable bolts. A rock-bolting machine resembles a jumbo drill and may use the same carrier. The working boom carries a rock drill and a carousel that can handle a range of rock-bolt lengths and typically 10 rock bolts. A separate hydraulic arm may be fitted to maneuver and place screen mesh.

A rock-bolting machine may be capable of installing mechanical- or friction-anchored bolts as well as cement- or resin-grouted bolts. Machine selection should address the following points:

Table 12.2-2 Jumbo cycle time

Component	Measurement
Setup	
Tramming speed, k/h	4.0
Average distance between faces, km	1.0
Tramming time, min	15.0
Setup and markup time, min	25.0
Total setup time, min	40.0
Setup time (=engine hours) per cut, h	0.7
Drill meters per engine hour, m/h	272.0
Drilling Cycle	
Instantaneous penetration rate, 45-mm holes, m/min	1.40
Instantaneous penetration rate, 100-mm holes, m/min	0.7
Meters drilled, 45-mm holes, m	182.2
Meters drilled, 100-mm holes, m	12.2
Drilling time, 45-mm holes, min	130.1
Drilling time, 100-mm holes, min	17.4
Total drilling time, min	147.6
Total drilling time, h	2.5
Holes drilled (all types)	67
Boom setup time per hole drilled, s	55.0
Total boom set-up time, min	61.0
Total boom set-up time, h	1.0
Total set-up and drill time (2 booms), h	3.5
Average penetration rate per boom, m/min	0.71
Jumbo drilling time, h	1.74
Ground Support Using Jumbo	
Change centralizers, min	10.0
Rattle backs, min	15.0
Installation time per rock bolt, min	1.0
Total rock bolt installation time, min	5.0
Total ground-support time, min	30.0
Total ground-support time, h	0.5
Total jumbo time per face, h	2.9

Note: Calculations are given in seconds or minutes, and the results are rounded to hours. The final result of jumbo time per face in hours is the sum of setup time plus jumbo drilling time plus ground support time.

Table 12.2-3 Jumbo development performance

Component	Measurement
Roster	
Days per year	365
Days worked per week	5
Days worked per month	21.7
Shifts per day	3
Shift length, h	8
Delays	
Shift change, h	0.8
No face available, h	0.6
No power, h	0.5
Meetings, h	0.2
Face not scraped, h	0.2
No ventilation, h	0.2
Pump/water problems, h	0.1
Maintenance/breakdown, h	0.1
Total delays, h	2.7
Productivity	
Balance of shift available, h	5.3
Faces drilled out	1.82
Face advance per shift, m	5.01
Face meters drilled per shift, m	298
Total meters drilled (including rock bolts) per shift	354.7
Meters drilled per month, m	19,356
Face advance per working day, m	15.0
Face advance per month, m	325.6

- Can the desired type and length of rock bolt be installed?
- Can it handle screen mesh?
- Will it fit into the required minimum back height?
- Can it reach the required maximum back height?
- Is the cab enclosed to protect the operator from resin fumes?
- Is a remote control option available for hazardous situations?

The rock-bolting machine size should be matched to the required heading size, which in turn depends on the purpose of the heading. It is unfortunately quite common to find that limited availability of rock-bolting boom lengths forces headings to be larger than they need to be for the purpose for which they were designed. At best, this might merely produce more development waste than necessary. At worst, it can make the use of the heading for its intended purpose more difficult, more costly, or more time-consuming than it would have been if correctly sized.

Rock-bolting machines are subject to impact damage from falling rock and require tight tolerances on working parts, so that the rock bolt can be indexed to the drilled hole. It is useful to enquire from other operators about their experience with maintenance on the model being considered.

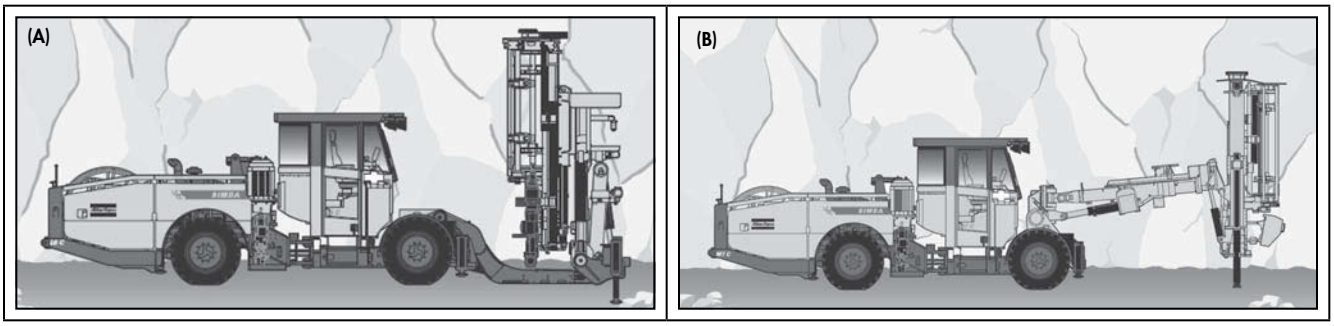
Rock-bolting machines tend to be slower and less mobile than jumbos. A check should be made to ensure that the rock-bolting machine can travel between development faces within the cycle time required.

Selecting a Production Drill

Production drills are used for drilling stope blastholes and service holes. Top hammer drills are generally available in sizes from 5 to 127 mm in diameter, whereas in-the-hole hammer (ITH) drills range from 95 to 178 mm in diameter. The drill-and-blast design will determine the optimum diameter and length of the hole. The selected drill must be able to quickly drill the hole, at minimum cost, with acceptable deviation. Deviation will limit the smaller hole sizes to around 12–15 m in length, whereas holes of around 100-mm diameter with tube drills can be up to 40 m long. Longer holes require larger diameters, favoring ITH drilling.

Skid-mounted, crawler-mounted, and trolley-mounted rigs have generally been replaced by highly mobile diesel-powered rigs. Top-hammer rock drills are hydraulic, using electric hydraulic power packs, whereas ITH drills use compressed air from a stand-alone or booster compressor. High-pressure water-powered hammers are also available.

For narrow-vein mining, production drills that can operate in vein widths of 3.0 m or slightly less are available. In the larger sizes, fully automated operation is possible so that a complete ring of blastholes can be drilled without human



Courtesy of Atlas Copco.
Figure 12.2-9 Production drill rigs

intervention. Considerations in selecting a production drill include

- The performance, including penetration rate, of the drifter at the desired hole diameter;
- Whether the desired hole inclinations can be drilled;
- Parallel hole coverage capability;
- Feed pivoting system (rigid with sliders—more accurate holes or totally variable—highly variable hole alignment);
- Whether reticulated compressed air or an onboard compressor will be used;
- Whether centralized lubrication is fitted;
- Whether automated rod handling is available;
- Whether carousel capacity is adequate for planned hole lengths;
- Whether remote control is available (umbilical cord system for misfire drilling management);
- What onboard surveying system is used for hole positioning;
- Whether cab is closed and air conditioned;
- The level of automation required (for increased use);
- Operator visibility of hole collar and lighting conditions; and
- The ability to collar the hole close to the wall, particularly in narrow-vein applications.

Drill string selection (rods, in-hole stabilizers, tubes) and feed stabilization (front- and/or rear-feed stingers) are also important considerations for managing drill-hole deviation.

Figure 12.2-9 shows production drilling rigs with hydraulic top-hammer rock drills. Both are designed for long-hole ring and parallel hole drilling upward or downward. In Figure 12.2-9A the drill unit faces the canopy and is suitable for holes in the range of 89 to 127 mm. The drill in Figure 12.2-9B is boom mounted for maximum reach and flexibility in both production drilling and bolt-hole drilling and is suitable for holes in the range of 51 to 89 mm.

Other Underground Equipment

The considerations outlined in the opening sections of this chapter apply to a wide range of underground equipment. Specific considerations include the following:

- Integrated tool carriers are based on wheel loaders for greater load-carrying capacity, versatility, and stability than conventional forklifts. Operator visibility and automated quick coupling are important features.
- Robust scalers are becoming more effective in headings between 4 and 6 m high. Backhoes fitted with a scaling pick or lightweight impact hammer and very large scalers have proved less successful. Some limestone mines use a rotary head on an excavator very effectively.
- Shotcrete (or Fibercrete) machines may be designed for wet or dry mix material. Mobility, boom reach, and remote-control capability are important in underground operations. The machine should have a remote-controlled boom nozzle to remove the operator from the unsupported area.
- Explosives charge-up machines may be used for cartridge, emulsion, or ammonium nitrate and fuel oil, and the charging process may be manual or mechanized. As with jumbo drills and rock-bolting machines, the charge-up machine must be sufficiently mobile to service the faces required within the development and production cycle. The reach and load capacity of the basket and the explosives carrying capacity must be suited to the proposed duty.
- Road maintenance equipment, such as graders (with or without rippers), bobcats, water trucks, and rollers need to be fully assessed for the suitability of the road surface medium and dimensions of the underground workings.
- A wide range of four-wheel-drive vehicles are suitable for underground mining, although many of the lower-cost models may not stand the rough conditions of a typical mine and may have a very high maintenance cost. It is a good strategy to buy the best rugged model available secondhand, with around 100,000 highway-kilometers. In corrosive or rough mine environments, the best vehicle life may be around 2 years. Estimators should not underestimate the cost of fitting a vehicle for underground operation, which might include crash bars, engine isolator, and removal of unwanted interior fittings.
- Handheld (airleg or jackleg) drills, rope scrapers (slushers), and rail equipment may still be used in some circumstances, although their use is rapidly diminishing. Proper selection of rail equipment, in particular, requires careful consideration based on cycle times, gradients, and locomotive capability. It is best to consult old textbooks including earlier versions of this handbook to gain an appreciation of the selection process.

REFERENCE

Willcox, P. 2008. Rapid tunnelling technologies at Cadia East. In *Proceedings of the Tenth AusIMM Underground Operators' Conference.* Melbourne: Australasian Institute of Mining and Metallurgy.

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CHAPTER 12.3

Soft-Rock Equipment Selection and Sizing

Jeremy Busfield

INTRODUCTION

Soft-rock mining predominantly applies to coal mining but is also applicable to other bedded soft mineral deposits. Therefore, although this chapter primarily focuses on underground coal mining, certain aspects are relevant to mineral deposits such as potash.

Underground coal mining methods currently fall into the following principal categories:

- Longwall (full-extraction) operations
- Room-and-pillar (partial-extraction) operations
- Room-and-pillar (first-workings) operations

These methods are discussed in the following sections of this chapter. Figure 12.3-1 provides examples of the three categories.

LONGWALL MINING METHODS

Because of its mechanized process, longwall mining generally achieves the highest productivity and reserve recovery and is typically the primary choice for new mines being developed, unless geological, surface, or capital constraints dictate otherwise.

In longwall mining, a set of roadways is driven out from the main entries down each side of the longwall panel to block out a portion of reserves. The mechanized longwall equipment is then retreated through the panel to extract the entire panel of coal. The roads used for traveling and the roads used for coal clearance during panel extraction are termed the *maingate* (or *headgate*) entries, whereas the tailgate entries are generally used for return air. Where possible, a set of parallel adjacent panels is extracted sequentially to allow reuse of one gate road (i.e., the maingate for the first panel becomes the tailgate for the next panel, and so forth). Working in this way maximizes coal recovery, while keeping the quantity of roadway drivage to a minimum. The number of roadways that comprise a maingate varies around the world, with single entries being common in Europe, two roadways common in Australia and China, and three roadways common in the United States. The number of roadways influences the style of coal clearance and the mobile equipment used.

Longwall mining methods vary slightly according to the seam height, which can range from as low as 1.0 m to in excess of 15 m. The height influences the longwall equipment used. Single-pass seam extraction is undertaken in seam heights of up to approximately 5.5 m, for which conventional longwall equipment is used. For seams in excess of 5.5 m, either multislice or top coal caving methods can be used. Top coal caving requires a different style of roof support and an armored face conveyor (AFC) configuration. In very thin seams, the longwall plough system can be used, which uses a different coal-cutting machine (a plough as opposed to a shearer).

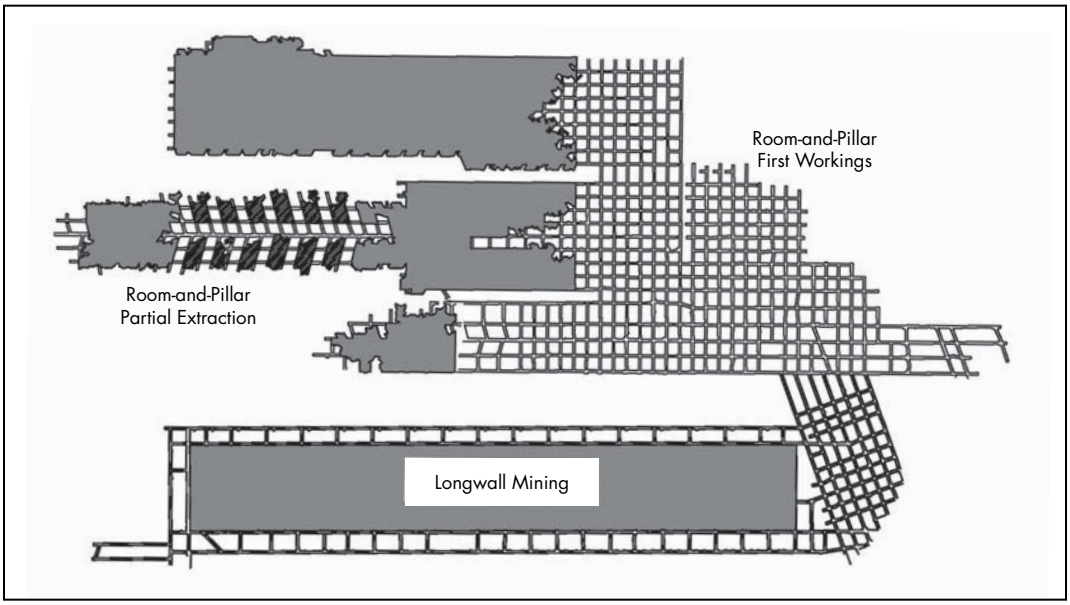
The longwall panel width can vary up to more than 400 m, and this influences the longwall equipment (quantity of roof supports, quantity of AFC pans, hydraulic power, electrical power, etc.). Panel length can be dictated by geology, surface subsidence restrictions, equipment limitations, or other mining constraints. Very short panel widths in the order of 25 to 50 m are typically termed *mini-wall longwalls*, and these use a specific type of AFC arrangement.

ROOM-AND-PILLAR MINING METHODS

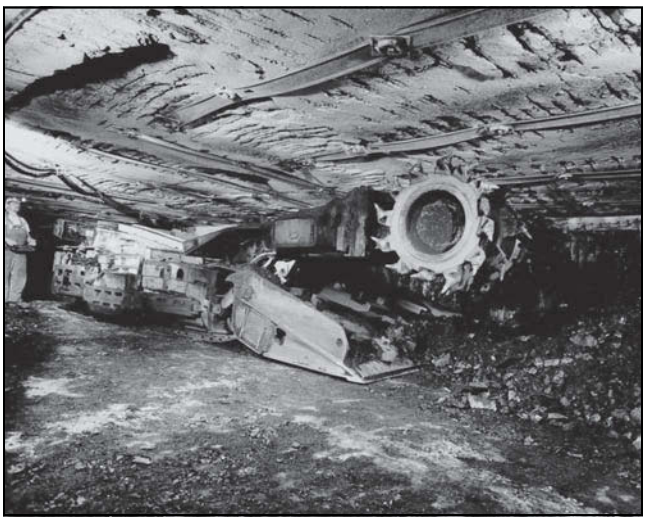
Until the advent of high-production longwall techniques, the majority of underground coal extraction used room-and-pillar (R&P, also termed *development* or *bord-and-pillar*) mining techniques. In many parts of the world this is still the case, with many mines achieving high production rates from R&P methods. The use of the R&P method is often favored because of its low capital requirements compared to the longwall system. The R&P method is also used in small or irregular-shaped deposits, deposits with surface subsidence restrictions, and deposits where geological constraints (such as faulting) preclude the economic use of longwall mining.

R&P methods range from first workings, where only roadways are driven (forming pillars), to second workings, where the pillars are then later either fully or partially extracted. R&P methods are generally deployed in seam heights ranging from 1.5 to 4.5 m. In seam heights of less than 2.1 m, the roof is often excavated periodically or continually to provide an adequate working height. Low seam heights can restrict equipment options, particularly bolting equipment. With greater

Jeremy Busfield, Principal Consultant, MineCraft Consulting Pty Ltd., Queensland, Australia



Courtesy of MineCraft Consulting.
Figure 12.3-1 Mining methods



Courtesy of Bucyrus.
Figure 12.3-2 Continuous miner in a room-and-pillar operation

seam heights, some mines extract the floor as a second working, thus allowing up to 6-m seams to be partially extracted. In these cases consideration is required for roadway gradients and equipment capabilities in regard to negotiating steep grades. In first workings, a benefit can be gained by increasing the roadway width to up to 7.5 m (or more), depending on the spanning ability of the roof. In potash mines, roadway spans typically exceed what can be achieved in coal, with the roadways formed by wide-head continuous miners (CMs) taking several passes. A CM in an R&P operation is shown in Figure 12.3-2.

Secondary extraction methods generally fall into pillar-splitting or stripping methods or fall into split-and-fender methods, for example, Wongawilli extraction in Australia (Figure 12.3-3).

In the standard pillar-splitting technique, the main development roadways are driven toward the boundary of the mining area, forming coal pillars to support the roof. When the limit of reserves or the mining boundary is encountered, the coal pillars can either be split or stripped, thus forming smaller pillars that are then left to support the roof or that collapse in a controlled manner on retreat. This method of mining requires specific equipment that can extract coal and retreat in a nimble and rapid manner.

Split-and-fender mining involves the formation of main access roads into a particular district and the subsequent "blocking out" of an area of coal, typically 60 to 80 m in width and up to 1.5 km in length. A roadway or "split" is driven through to the end of the block using a CM, leaving a thin coal "fender" that is extracted as the CM retreats back toward the access roadways, allowing the roof to collapse behind. When the fender is fully extracted, the mining equipment is retracted, and the entire sequence is repeated until the whole coal block is mined out. Although similar equipment can be used for this method, the use of powered roof supports (mobile breaker line supports) is often used to increase safety and allow for a greater level of roof control.

LONGWALL MINING EQUIPMENT

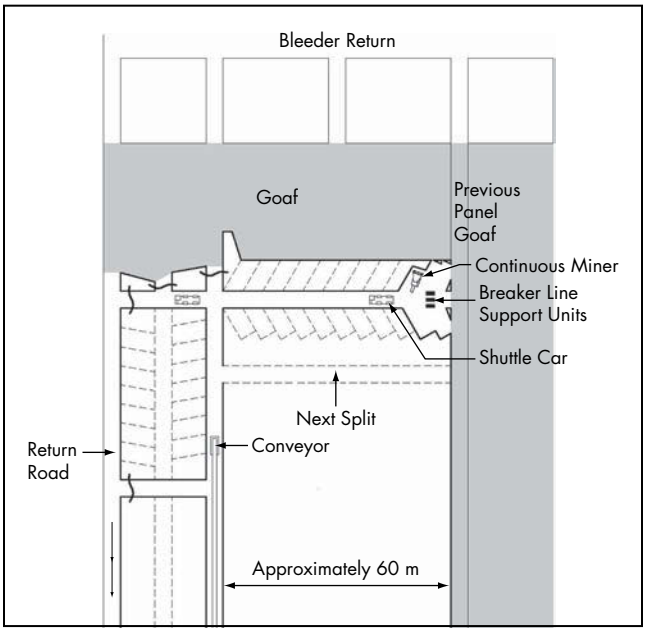
This section begins with a discussion of standard longwall equipment, followed by discussions of longwall equipment variations, selection, specifications, and sizing.

Standard Longwall Equipment

The standard longwall face equipment consists of roof supports; a shearer; an AFC; a beam stage loader (BSL), including crusher and boot end; a monorail; a pump station; and system electrics. Figure 12.3-4 is a schematic of a longwall system showing the relative locations of each piece of equipment.

Roof Supports

Roof supports (sometimes referred to as shields or chocks) hold the exposed roof as the coal is mined, allowing the roof strata to cantilever over the support fulcrum and then break off into the goaf (or gob). In this manner, the operators are



Courtesy of MineCraft Consulting.
Figure 12.3-3 Wongawilli extraction panel

shielded from falling goaf as the face is advanced and the immediate face area is destressed. As the shearer cuts along the face, the AFC is snaked across the new face line using the double-acting sequence rams fitted between the supports and the AFC pans. As the AFC is snaked, the roof supports are lowered and pulled across to the advanced position by the sequence ram and reset to the roof, referred to as the lower-advance-set cycle. In this way the face is automatically advanced as each longwall slice is taken through the panel. The immediate roof falls behind the support, forming a goaf.

Roof supports can be operated in either "conventional mode" or in "immediate forward support" (IFS) mode. Conventional supports cannot be advanced to support the roof until the AFC is advanced (i.e., the roof is exposed and unsupported for some distance behind the shearer), whereas in IFS mode the supports are advanced immediately after the shearer has passed. IFS support is generally needed in weak roof conditions and is the most popular mode of support currently used. IFS supports require longer canopies than "conventional" supports with corresponding larger leg cylinders and hydraulic systems.

Modern longwall supports have been built to operate to approximately 5.8 m in height. Support capacity depends largely on support configurations, although the largest supports available have a capacity of approximately 1,750 t (metric tons), equating to a support density of approximately 125 t/m^2 and weighing up to 65 t.

The roof supports used at the gate ends are configured slightly different to the "run-of-face" supports, in that they have longer canopies so as to extend over the maingate drives. In addition, the gate end supports have higher rated capacity relay bars to enable pushing the maingate drive frames and BSL. Typically, there are five gate end supports at the maingate and three at the tailgate.

Originally operated by valves fitted to each support (lower, advance, and raise), modern supports are controlled by electrohydraulic systems, whereby solenoids operate the valves and the supports can be operated automatically in sequence mode or from adjacent or nearby supports. This automatic sequence also can operate with a positive set mode, whereby each support is set against the roof to a minimum pressure (often between 70% and 80% of the yield pressure), thus guaranteeing effective support when in automatic mode.

When specifying roof supports, various options are available, including the following:

- Four-leg versus two-leg supports (because of advancements in cylinder technology and capacity and because of lemniscate design, two-leg supports are now more popular)
- Front walkway, rear walkway, or both (rear walkways are often used in thick seams to offer more protection against face spall)
- Side shields on one or both sides (used to push supports uphill)
- Face sprags or flippers (used to prevent face spall in thick seams)
- Base lift systems (used to raise shield prior to advance when in boggy floor)

A longwall roof support with face sprag and base lift fitted is shown in Figure 12.3-5.

Shearers

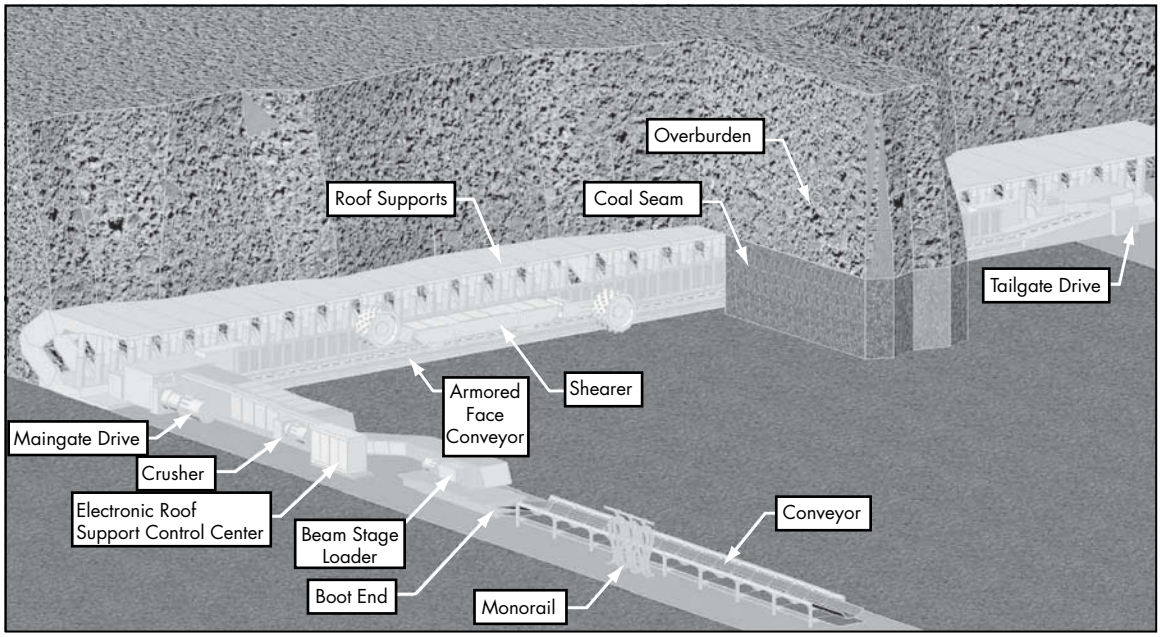
Modern longwall double-ended ranging drum shearers have two cutting drums, one at each end mounted on hydraulically raised ranging arms that are in turn connected to the shearer body (Figure 12.3-6). The shearer is driven by onboard traction units that drive a sprocket wheel that connects to rack bars mounted on the walkway side of the AFC. The face side of the shearer slides along the top (or toe) of the AFC sigma section. The shearer operators control the speed of the shearer and the position of the cutting drum using handheld controls (either radio remote or tethered cable) to maintain a suitable cutting horizon within the seam and minimizing dilution.

Modern heavy-duty shearers are fitted with up to 2,000 kW of available power at the cutting drums, enabling maximum cutting rates of more than 5,000 t/h to be obtained. Shearers can be up to 14 m in length and weigh up to 100 t.

Armored Face Conveyor

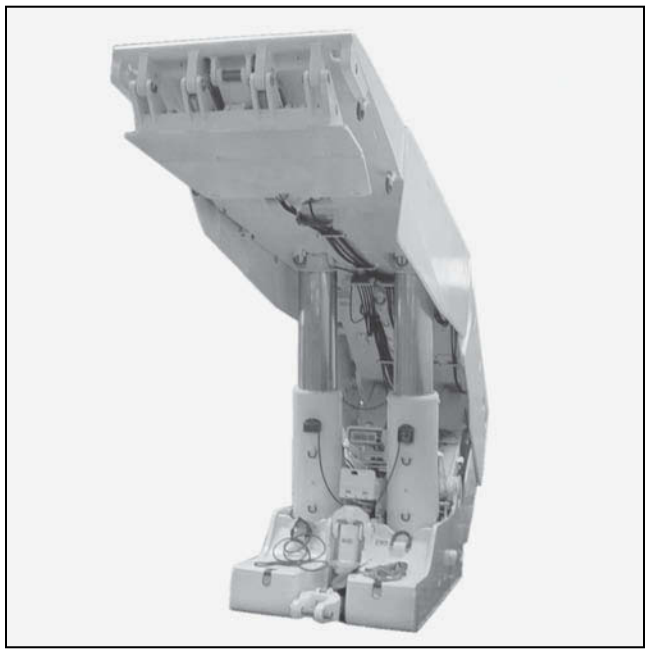
The longwall AFC is used to convey cut coal along the face to the maingate, where it is connected to the BSL and outby conveyor system. Steel AFC pans are constructed in sections typically 1.75 or 2.0 m long to match the width of roof supports. Flexible (dog bone) connectors are used to join the pans together. A clevice bracket is fitted to each pan, by which the roof support relay bar (hydraulic ram) attaches. As each slice is taken on the longwall, the AFC pans are advanced or "snaked" into the newly cut area behind the shearer by activating the relay bars in sequence.

The AFC supports the shearer body as it runs along the face, and the cut coal is conveyed along the AFC by flight bars attached to AFC chains driven by motors at both the maingate and tailgate face ends (Figure 12.3-7). Inspection pans that feature a removable decking plate will typically be fitted as every fifth pan to allow for access to the bottom chain. Transition pans are used at the gate ends to transition the AFC onto the drive frames and BSL. Maintaining tension in the AFC chain is important with a chain-tensioning device fitted at the tailgate drive frame. Often automated using an appropriate control



Courtesy of MineCraft Consulting.

Figure 12.3-4 Schematic view of longwall system



Courtesy of Bucyrus.

Figure 12.3-5 Longwall roof support with face sprag and base lift fitted

system (reed rods, etc.), the tensioner operates by extending a hydraulic ram (1-m stroke) fitted to the tail sprocket assembly. While the tensioner provides fine adjustment on a day-to-day basis, coarse adjustment is achieved by removing links from the AFC chain.

Important aspects of the AFC include the following:

- The drive motors must be able to start the AFC when fully loaded.
- The AFC speed must be sufficiently faster than the shearer's cutting rate to allow coal to be cleared when cutting from tailgate to maingate.
- The AFC chain must be of sufficient strength to handle the applied tensions.

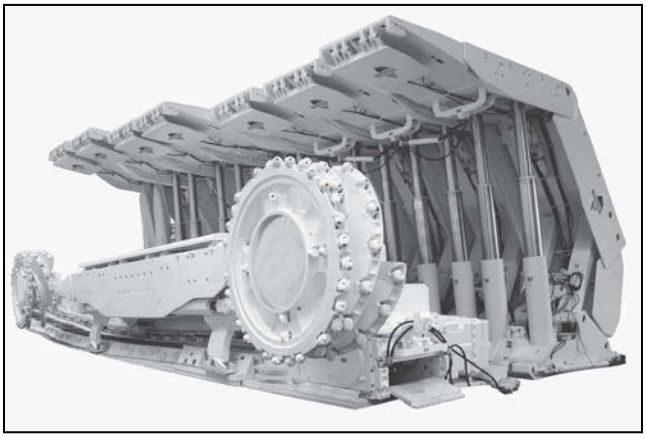
Modern AFCs can be supplied with up to 3,000 kW of installed power in AFC pan sections with 1,300-mm raceways. Coal can be transported at rates in excess of 4,500 t/h. The current limiting factor for face length is the AFC (chain strength versus installed power). Modern chains are being developed so as to allow longer faces.

Currently, the two main AFC drive systems are controlled-slip transmission (CST) systems and fluid-coupling systems. CST systems incorporate clutch systems in the output stage of the AFC gearbox (Figure 12.3-8) with the motor directly coupled to the gearbox. Fluid-coupling systems are fitted between the motor and gearbox and incorporate valves to change the fill volume of the coupling, thus changing the level of torque. Both systems allow for controlled starting and load sharing between drives. An AFC maingate drive frame is shown in Figure 12.3-9.

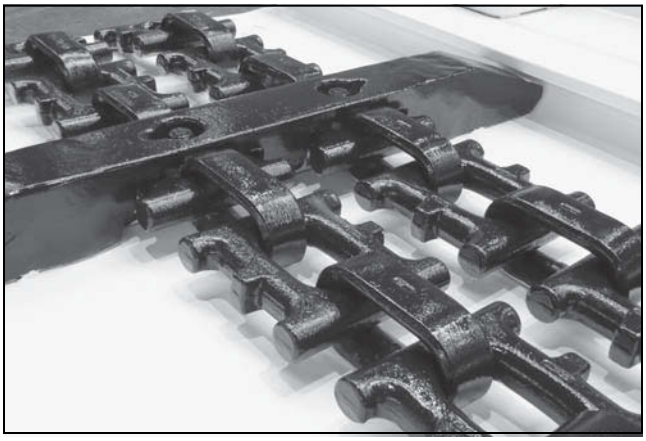
Beam Stage Loader, Crusher, and Boot End

The BSL transfers the coal from the AFC to the conveyor belt and includes a chain conveyor running over deck plates, a crusher, a gooseneck, and a boot end. The BSL is capable of handling a higher volumetric capacity than the AFC to ensure that coal removal from the face occurs without accumulation at the maingate corner (AFC/BSL intersection).

The crusher sizes the cut coal to ensure that oversize lumps are not carried onto the conveyor, and they consist of a high-inertia rotating drum fitted with large "hammers" to break up large lumps of coal that may have slabbed from the face or large pieces of stone from the roof. The gooseneck is an elevated section that raises the coal so it can be transferred onto the belt.



Courtesy of Bucyrus.
Figure 12.3-6 Double-ended ranging drum shearer



Courtesy of Bucyrus.
Figure 12.3-7 AFC chain and pan

The boot end consists of the tail pulley for the conveyor belt fitted to a sliding frame so that the BSL can be retracted a certain distance (up to 3 m) without moving the tail pulley location. The boot end is fitted with tracks or pads for movement and features hydraulic rams so that the tail pulley can be leveled and adjusted for belt tracking.

Monorail

The monorail system provides a flexible services link between the operating longwall face and the pump station and transformer equipment. Services, including electricity, hydraulic fluid, water, compressed air, communications, and control, are carried along the monorail system as either hoses or cables. The purpose of the monorail is to allow the longwall to independently retreat typically 200 m (two pillars) before the need to relocate the services equipment further outby.

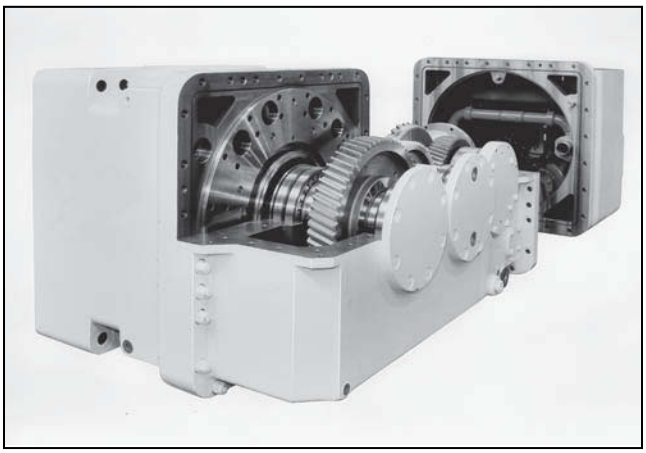
The monorail equipment comprises long-length runs of hosing and cables mounted to trolleys supported by roof-mounted monorail beams located adjacent the gate conveyor in the belt heading.

Pump Station

The pump station provides the high-pressure hydraulic fluid required to operate the longwall face equipment, including the hydraulic emulsion supply for the roof supports and water for shearer cooling, dust suppression, and other auxiliary systems as required. The pump station consists of sleds containing typically up to three pressure pumps, including one high-pressure set pump (if used), a high-pressure water pump for the shearer, a reservoir for the closed emulsion fluid system, and the pump control units. The pump station is typically located in the mine roadways (on a steel-framed sled) or in cut-throughs (trailer mounted to assist movement around the mine). Hydraulic fluid can be either fed from a surface-mounted emulsion-mixing farm (via poly pipe to the pump station) or from mixing tanks located on the sled.

Electrical System

A track-mounted transformer located outby (with the pump station) reduces the panel's incoming power to the required face voltage prior to supplying the distribution and control board (DCB) equipment, which is normally located on the BSL. The DCB provides isolation, control, and monitoring capabilities to the longwall face electrical equipment. The face voltage is typically specific to the country (e.g., 3.3 kV in Australia and 4.4 kV in the United States).



Courtesy of Bucyrus.
Figure 12.3-8 AFC gearbox

The control and monitoring equipment, which is the electrical control center for the longwall system, is typically located on the maingate corner and provides an interface for operators with the longwall equipment.

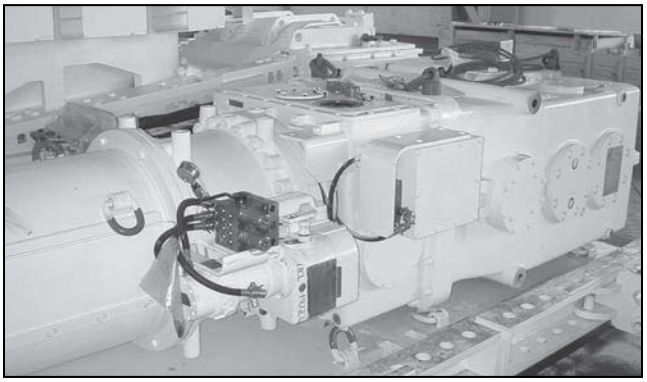
Longwall Equipment Variations

Alternate equipment configurations are available for alternate mining methods and/or methods of operation.

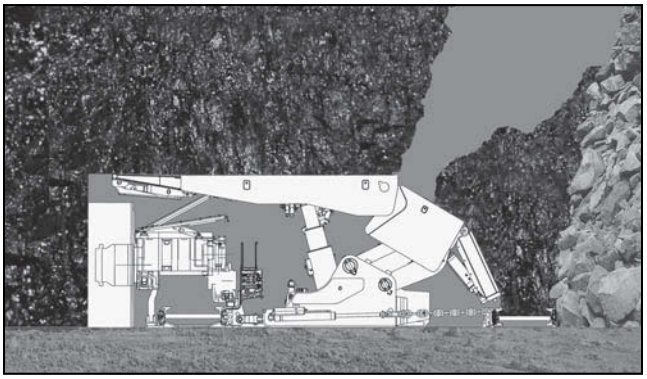
Top Coal Caving

The roof supports for top coal caving feature a hinged rear canopy shield that, when hydraulically lowered, will allow coal that has fallen from the roof to flow onto a rear AFC and thus be collected (Figure 12.3-10). Although this feature can successfully allow for the recovery of coal from thick seams, the caving methodology and sequence (lowering of the canopy) requires careful integration into the overall longwall sequence to ensure that the productivity of the entire operation is not compromised.

Top coal caving requires a rear AFC, which is a second AFC located behind the roof supports along the edge of the goaf and protected by the hinged rear canopy. The rear AFC



Courtesy of Bucyrus.
Figure 12.3-9 AFC maingate drive frame



Courtesy of Bucyrus.
Figure 12.3-10 Top coal caving schematic

is connected to the roof support by a hydraulic relay bar and chain. The rear AFC transfers the coal onto the front AFC at the maingate end of the face. Hence, it requires a short transfer loader that crosses the main front walkway. In addition, the rear AFC drive unit must be located in the maingate area. To fit all this equipment into the maingate area requires a special roof-support configuration (Figure 12.3-11).

Longwall Ploughs

Longwall ploughs are used in place of shearers in thin-seam longwall applications. A plough consists of a vertical steel frame housing a series of cutting picks that sits on top of the AFC and is driven at high speed (2.5 to 3.6 m/s) the length of the face by a chain drive system (Figure 12.3-12). The plough cuts a thin slice of coal (150 to 250 mm) in either one or both directions with the coal falling onto the AFC to be cleared. The longwall supports and AFC are advanced after each cut in a staged manner. Because of the low seam height applications, plough faces are generally fully automatic with the equipment being operated and monitored from the maingate.

Longwall Automation

The longwall mining method is highly mechanized, which lends itself to various aspects of automation. This has been progressively developed over time. A longwall face in operation is shown in Figure 12.3-13.

Until approximately 2003, the level of longwall automation was limited and consisted primarily of inner-equipment automation (e.g., roof support sequencing) and intra-equipment automation (e.g., shearer initiated support advance). Recent Australian industry-funded studies have advanced the capability of longwall automation to now include the following capabilities:

- Face alignment control (use of an inertial navigation system)
- Horizon control (improved memory cut)
- Shearer-state-based automation (shearer speed control based on position)
- Automated fly and straightening cuts
- Automated gate road support operation
- Open communication systems (more user-friendly software systems)

Several mines are now testing these capabilities, and they have been reporting improved productivities.

Longwall Equipment Selection

The detailed specification of a longwall system needs to be determined at the time of tender preparation and is based on the results of detailed studies carried out during the feasibility studies. These studies include detailed mine planning, geotechnical characterization, caving studies, and engineering analysis. Key specification aspects include face length, web depth, nameplate capacity (NPC), and support density.

Face Length

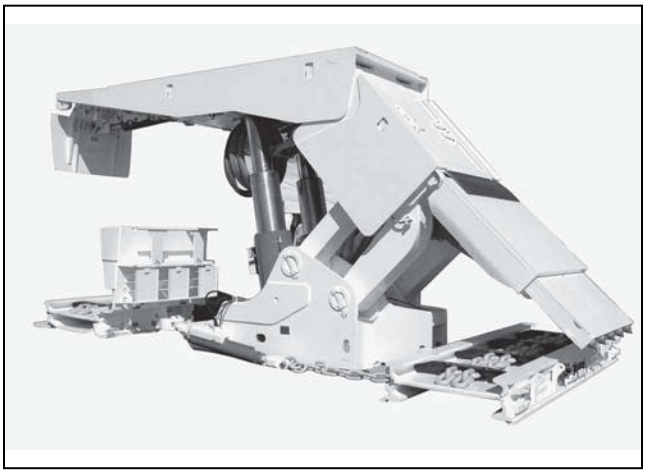
Because of improved technology, longwall face lengths have progressively increased from 200 m in the mid-1980s to 440 m today. Long faces in the world include 400 m at Ulan in Australia, 385 m at Cumberland in the United States, and 440 m at Prosper in Germany. The current limiting factor for face length is the AFC (chain strength versus installed power). Hence, there is a trade-off between face length and NPC. Modern chains are being developed so as to allow longer faces. Current technology indicates that a 400-m face length with 4,500-t/h NPC is the current limit.

Web Depth

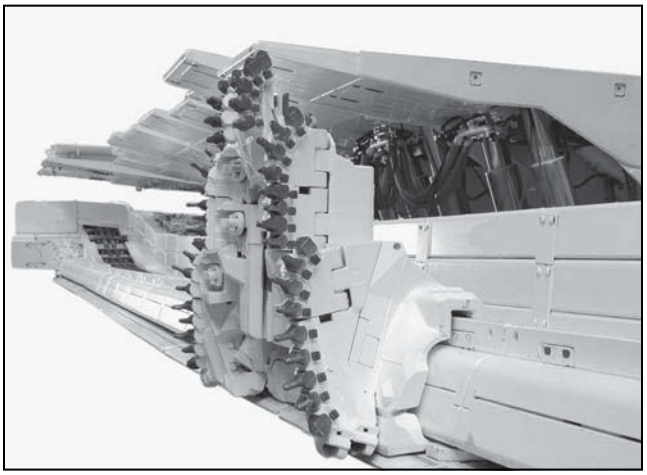
The depth of cut made by each pass of the shearer can range up to 1,200 mm and commonly ranges from 800 to 1,000 mm. There is a significant productivity benefit from having a deep cut, as this translates to more coal per pass of the shearer. However, the greater the web depth, the greater the required shield canopy length to achieve the required tip-to-face distance. Hence, the cautioning aspect to web depth is the risk of face instability. A detailed geotechnical study is required to establish an appropriate web depth that balances the need for high productivity with the need for face stability. Another approach is to use a half-web (or partial-web) system, whereby only half the web depth is cut during each pass, which can lead to better ground control in particular situations. A half-web system requires faster shearer speeds to maintain productivity. However, this has been very successful (e.g., Twenty Mile mine in the United States). If a half-web operation is envisaged, then the longwall specification will need to include partial push capability of the relay bars (half stroke).

Nameplate Capacity

NPC is the rated capacity of the AFC and is a measure of the coal output during the main cutting run of the shearer. The most important aspect of NPC is that it directly impacts the capacity of the entire underground coal clearance system. The longwall



Courtesy of Bucyrus.
Figure 12.3-11 Top coal caving roof support



Courtesy of Bucyrus.
Figure 12.3-12 Longwall plough

belt conveyor must match the longwall NPC, or the longwall will not be able to operate at designed capacity. This is commonly seen in practice for various reasons. Consequently, the shearer speed is commonly restricted to compensate.

Similar to face length, nameplate capacities have steadily increased in line with increasing technology up to the maximum of 4,500 t/h in Australia and 5,200 t/h in the United States. Although high NPCs can theoretically directly translate to higher productivities, this is not always seen in practice, with many of the highest-producing longwalls in the world rated at 3,500 t/h. This is because NPC is not the only aspect of the determination of productivity.

In general terms, an NPC of 4,500 t/h will require 6,000- to 6,500-t/h trunk belts (2,000- to 2,200-mm belt widths), while an NPC of 3,500 t/h will require 5,000- to 5,500-t/h belts (1,600- to 1,800-mm belt widths).

Support Density

Support density is the rated capacity of the roof shield and is expressed in units of metric tons per square meter of canopy area, usually after the shearer has made its pass (after cut). Again, improving technology has allowed this capacity to increase over time, with capacities up to 130 t/m^2 now available. Typical support densities are between 90 and 110 t/m^2. Variables such as support width (1.75 m versus 2.0 m), shield geometry, and leg cylinder diameter have an impact on support density. The required support density is determined using geotechnical techniques such as ground response curve modeling or Fast Lagrangian Analysis of Continua (FLAC) modeling. Modern supports typically have a support density rating between 95 and 115 t/m^2. Typically, the ratio of setting pressure to yield pressure will be specified at 80%. However, there is an increasing trend toward 90% set, which requires use of a high-pressure set hydraulic supply system.

Longwall Equipment Specifications

There are several longwall equipment suppliers throughout the world, many of which offer complete systems. The typical process of selecting suppliers is to conduct a formal prequalification process involving the following:

- **Expression of interest:** All suppliers are requested to formally express their interest in supplying the equipment. The expression of interest may be advertised in a suitable media. However, in most countries, equipment suppliers maintain close relationships with organizations, so they are aware of forthcoming tenders.
- **Technical prequalification:** An indicative technical specification is issued to interested suppliers who are requested to prequalify by submitting a response as to whether they can meet the equipment technical requirements, provide examples of where they have supplied before, and provide sufficient equipment description to satisfy the prequalification.
- **Commercial prequalification:** Often attached to the technical specification, the commercial prequalification requests answers to various questions of a commercial aspect relating to security of supply, financial strength of the supplier, warranty provisions offered, and a budget price for the supply of the equipment.
- **Short listing:** Following the prequalification process, a short list of two to four suppliers can be selected. These suppliers would then be subjected to a formal tendering procedure for the supply of the equipment.



Courtesy of Bucyrus.
Figure 12.3-13 Longwall face in operation

Examples of a typical longwall requirements and specifications are included in Tables 12.3-1 and 12.3-2.

Table 12.3-1 Example of a longwall system's general requirements

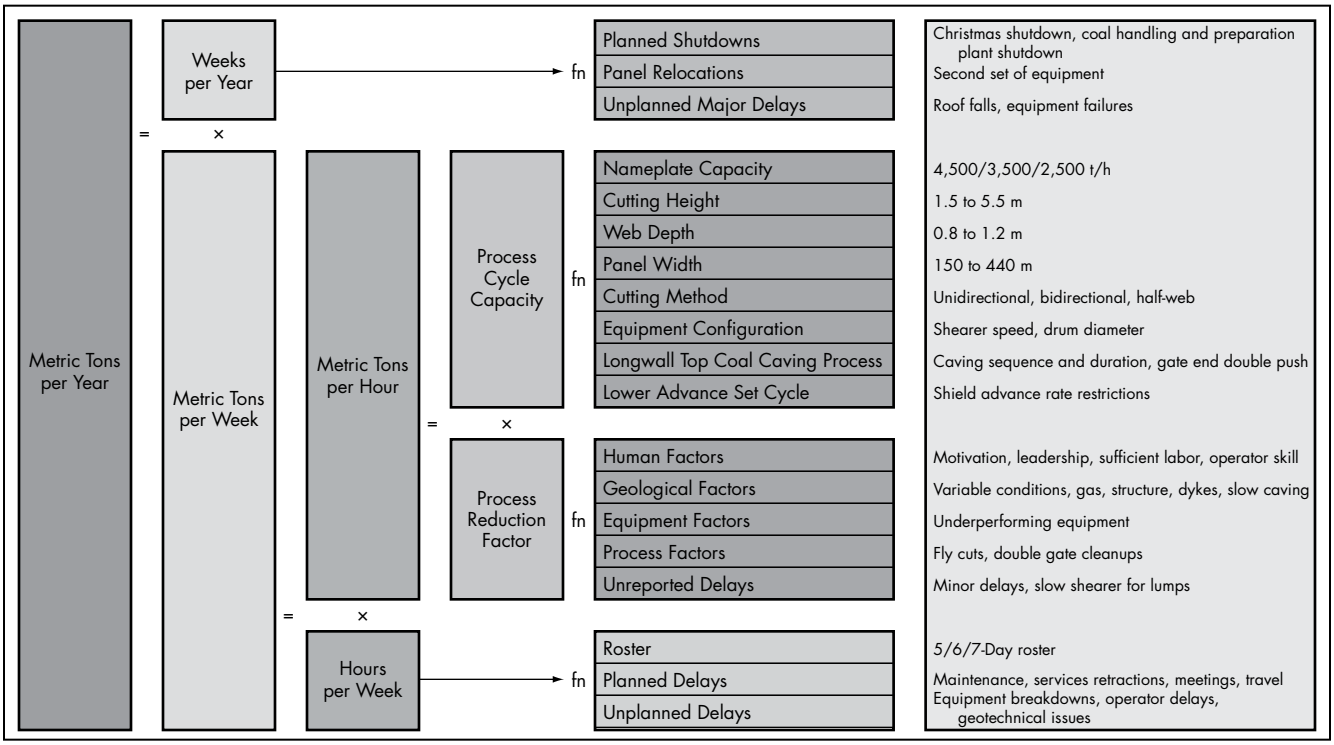
Parameter	Requirement
Mining method	Conventional retreat longwall
Number of maingate roadways	Two
Panel width	305-m centers
Mining extraction range	Nominal 4.5 m, minimum 3.5 m, maximum 4.8 m
Face cross grades	Flat to 1:10 downhill
Panel lengths	1,500 to 4,000 m (total of 16 panels)
Longwall retreat	~50 km and ~100 Mt over the life of the mine
System cutting process capability	Bidirectional and unidirectional sequence mandatory; half-web optional
Automation capabilities	In accordance with the capabilities expected to be supplied at the time of ordering (approximately 2010)
Extraction web depth	Nominal 1,000 mm
Nameplate capacity	3,500 t/h
Process cycle capacity	3,000 t/h

Courtesy of MineCraft Consulting.

Table 12.3-2 Example of longwall equipment preliminary specifications

Equipment	Requirement
Roof supports	• Two-leg immediate forward support (IFS), 2.0-m nominal width roof support • Operating height range, 2.8 to 4.0 m • Single-piece rigid construction IFS-type canopy with single moveable side shield and automated face sprags • Support density, 100 t/m^2 • Tip to face distance of 400 mm before cut • Set to yield ratio of 80% • Rigid catamaran-type base fitted with base lift and telescopic relay bar • Full electrohydraulic system, including filtration • Closed height, 2,200 mm; maximum extension height, 300 mm higher than maximum extraction height • Run of face roof support typical mass, 35 t; gate end, 37 t • In-service life of 45,000 lower advance set (LAS) cycles without loss of function and 45,000 to 60,000 LAS cycles with up to 25% withdrawn from service for maintenance; maximum life, 80,000 cycles
Shearer	• Double-ended ranging arm/drum multimotor shearer • Ranging arms individually powered by electric motors • Haulage via indirect drive sprocket arrangement and track type to suit AFC pans • Drum diameter and arrangement to be optimized to provide acceptable coal loading and equipment undercut/cut-out capabilities; sprays incorporated • Shearer typical mass, 80 t without drums fitted
Armored face conveyor (AFC)	• Power rating to suit 3,500 t/h over full length of conveyor and volumetric capacity to suit 4,000-t/h short-term peaks • Drive units incorporating soft-start and load-sharing capabilities • Automatic chain-tensioning system • Capable of greater than 150-t reserve chain pull for face overload events • AFC pans of length to suit 2.0-m nominal width roof supports fitted with services trough and bretby* trough arrangement
Beam stage loader (BSL)	• Power rating and volumetric capacity to suit 4,000-t/h short-term peaks • Automatic chain-tensioning system • Fitted with ancillary equipment, including spray system, wet scrubber, and monorail removal platform
Crusher	• Power rating, motor inertia, and volumetric capacity to suit 4,000-t/h short-term peaks • Output material sizing to <300 mm
Boot end	• Skid-type advancing boot end with steering, side shift, and leveling capabilities • Travel to suit two BSL advances • Trough and skirting to suit 1,600-mm belt-width gate conveyor belting
Monorail system	• Nominal 200-m services retraction frequency, plus 40-m reserve compression (240 m total) • Located in belt road heading, with capacity to house all services required to support the longwall system • Materials management system to suit 100-m cut through spacing and 1,600-mm belt-width gate conveyor structure
Pump station	• Track-mounted pump and tank stations; cut-through located • Services to be supplied include roof support hydraulic supply, shearer water, roof support dust suppression, plus miscellaneous auxiliary systems • Output capacity to suit hydraulic power requirements plus a reserve of not less than 20% of maximum demand requirements • 350 bar standard pressure, 420 bar high-pressure set • Equipment transport configuration design to suit typical mine size and constraints
Electrical system	• Track-mounted transformer; cut-through located • BSL/monorail-mounted distribution and control board unit and AFC-mounted control and monitoring equipment unit • Incoming supply at 11-kV, 3-phase, AC 50 Hz; longwall motors typically 3.3 kV

Courtesy of MineCraft Consulting.
**Bretby* is a trailing cable attachment, usually made from rigid, hinged plastic, that protects the cable from damage.



Courtesy of MineCraft Consulting.

Figure 12.3-14 Longwall productivity factors

Longwall Equipment Sizing

Longwall equipment size is generally referred to as its nameplate capacity. Selecting the longwall equipment size relates to the required levels of productivity. Hence, it is important to understand the relationships between the longwall system, its NPC, and its productivity. The relationship between the longwall system and annual productivity is shown in Figure 12.3-14. Two important productivity key performance indicators (KPIs) are the average metric tons per operating hour and the operating hours per week. For an operating mine, these statistics can be collected as part of the production monitoring data. For a new longwall operation, these KPIs need to be derived, which requires a combination of modeling and benchmarking.

To model longwall productivity, the following definitions are used:

- **Nameplate capacity (NPC):** NPC is the rated instantaneous capacity of the longwall (e.g., 2,500 t/h). This is the load that is carried by the AFC during its main cutting run and will occur somewhere in the cycle for between approximately 10 and 20 minutes. In some instances the NPC is determined (restricted) by the coal clearance system.
- **Process cycle capacity (PCC):** PCC is the average capacity achieved during one full cycle at maximum efficiency. This is equal to the metric tons produced in cutting maingate/tailgate and tailgate/maingate divided by the time taken (e.g., 1,800 t/h).
- **Actual productivity:** This is the actual operating rate achieved at the mine over an extended period, as determined from the statistical reporting system (e.g., 1,000 t/h). Depending on the analysis period, the actual productivity will fluctuate from very low levels up to, but not exceeding, the PCC.
- **Productivity reduction factor (PRF):** PRF is the difference between the actual operating rate and the PCC. It is a measure of efficiency and reflects various issues such as adverse mining conditions, operator skills, motivation, and organization. For new mines, this can be derived by benchmarking against similar operations and will generally range between 55% and 75%.
- **Operational availability:** This is a measure of the average number of hours in a shift that the longwall is producing divided by the total hours in a shift (e.g., 56%). It is calculated by dividing the actual operating hours recorded over an extended period by the planned operating hours. The difference between the two measures is the amount of unplanned downtime that occurs, either due to equipment breakdowns or process delays. For new mines, the operational availability can be derived by benchmarking against similar operations and will generally range between 50% and 70%.

The longwall model will calculate the PCC, assuming everything operates to its design capacity.

However, in reality, the longwall does not always operate in this manner. Numerous operational issues, such as operator skill and attention, the presence of geological anomalies, taking fly cuts (straightening cuts), slowing for lumps on the AFC, and gate end cleanups, all slow and impact the cycle. The PRF takes this into account and is a measure of how efficient the longwall cycle is performed. The PRF will fluctuate from very low levels up to very high levels but can be measured over an extended period for use as a benchmarking tool.

In addition, numerous unplanned breakdowns (mechanical/electrical) and operational delays occur, which reduce the total amount of operational time available within a shift. This is termed *operational availability* (also called utilization).

PCC is calculated as follows:

- PCC equals the cycle metric tons per cycle time, where cycle metric tons equals metric tons cut from the maingate to the tailgate and the tailgate to the maingate. This is a function of face length, extraction thickness, web depth, and wedge cuts at gate ends.
- Cycle time equals the time taken for the shearer to cut from the maingate to the tailgate and the tailgate to the maingate. This is a function of NPC, shearer speed, shearer acceleration, snake length, cutting mode (bidirectional and unidirectional), and face length.

When calculating the PCC, it is important to conduct some operational reality tests regarding issues such as the following:

- Excessive shearer speed
- Sufficient shearer underframe clearance
- Excessive drum rotational speed
- Adequate ranging arm power
- Adequate AFC power

Typically, the engineer should have some operational experience with the longwall method to perform these calculations.

Longwall Top Coal Caving

Calculating the productivity of the longwall top coal caving (LTCC) method creates an additional level of complexity. However, the same principle can be applied. It is important to accurately model the LTCC process, particularly the caving cycle, as to when this occurs in the process. Typically, the main cut and the cave cannot occur at the same time as the BSL, and the section conveyor cannot handle the dual output. Therefore, the cave either occurs while the shearer is stationary at the gate end or while the snake is occurring. Therefore, the caving cycle is generally the rate-determining step for the process cycle, with several factors to consider, including the capacity of the rear AFC, the number of caving drawpoints, the time required to cave each support, and the method of caving (manual or automatic).

The key benefits of LTCC are resource recovery (more of the coal seam is recovered) and improved development ratio (more longwall metric tons per development meter). However, this can often be at the expense of productivity when compared to a conventional high-capacity longwall, and care is required when designing and modeling the process.

Other Longwall Equipment Considerations

Other engineering considerations when selecting a longwall system include the following:

- **AFC power demand calculations:** These calculations are performed to ensure that sufficient AFC drive power is installed so that one can be able to start a fully loaded AFC with sufficient reserve chain pull.
- **Electrical load flow study:** This study is performed to ensure that adequate electrical power is available to cater for both normal loads and for starting under adverse conditions.
- **Hydraulic flow simulations:** These simulations ensure that the hydraulic system is sufficient to meet the roof support requirements, including routine LAS cycle (up to three supports moving at once), high-pressure set, gate end advance, and to cater for adverse loading events.

ROOM-AND-PILLAR MINING EQUIPMENT

R&P mining can be divided into two categories: first workings (development) and second workings (extraction). Both processes require equipment to cut and gather the coal at the mining face, convey and discharge the cut coal onto a section conveyor, and install ground support at the mining face. Ancillary support equipment is required to provide ventilation and power. There are numerous options of equipment that can be used as shown in Figure 12.3-15.

Standard R&P Mining Equipment

Standard R&P equipment and systems include CMs, roadheaders, haulage units, feeder breakers, mobile bolters, electrical power, ancillary equipment, and mobile roof supports.

Continuous Miners

A CM is a large electrohydraulic machine that extracts the coal to form a rectangular profile roadway or tunnel. It features a rotating cutterhead (drum) laced with rock picks at the front. The cutterhead is driven into the coal face, thus breaking out the coal. The broken coal falls to the ground and is loaded onto a centrally located chain conveyor using a loading apron and gathering arms, spinners, or east–west conveyor. The coal is conveyed through the body of the CM and loaded into coal haulage units (typically shuttle or ram cars), which come to the rear of the CM to be loaded.

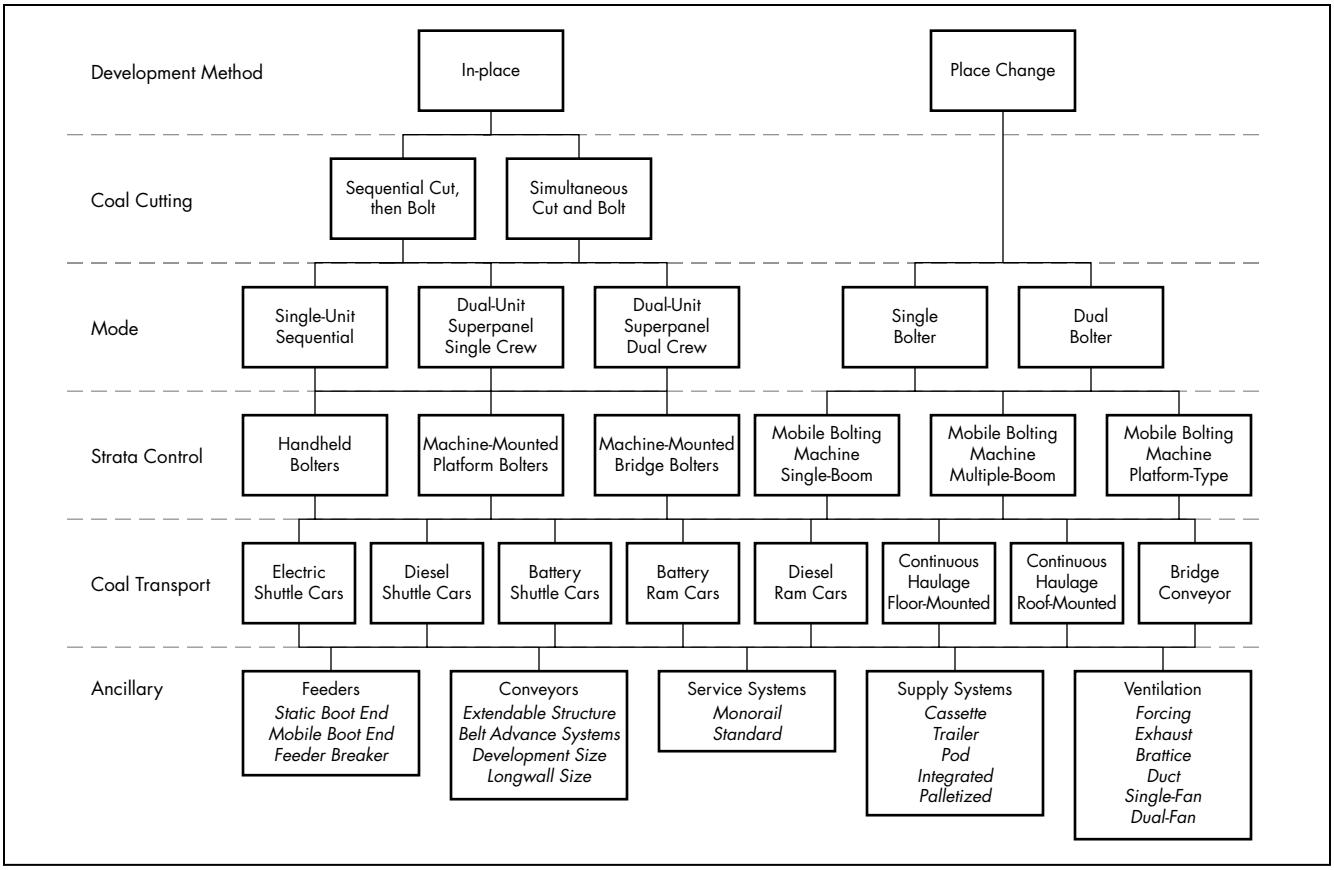
Modern CMs are fitted with hydraulic drill rigs that drill and install the primary ground support as the roadway is formed (roof and rib bolts). Platforms are provided for the drill rig operators to stand on while the CM is cutting coal. Commonly, the drill platforms feature hydraulically operated temporary support mechanisms to protect the operators from roof or rib collapse. The CM is fitted with caterpillar tracks to allow it to be propelled forward and backwards (skid steering). CMs are also commonly operated by handheld radio remote-control units, although older units have pendant controls (cable-connected) or operator cabins.

The three main types of CMs are as follows:

1. **Simultaneous cut and bolt:** This type features a full roadway width cutting head (e.g., 5.2 m) that can sump into the coal face while the body of the CM remains stationary, resulting in the ability to use the drill rigs at the same time as cutting coal (Figure 12.3-16).
2. **Sequential cut and bolt:** This type features a full roadway width cutting head. However, the body of the CM moves forward when cutting coal, and thus drilling cannot occur at the same time.
3. **Place change (or cut and flit):** These CMs are not fitted with drill rigs, often have a narrower cutting head (e.g., 3.5 m), and are used for the place-changing method of mining. Other machines are used to install the ground support in these applications (Figure 12.3-17).

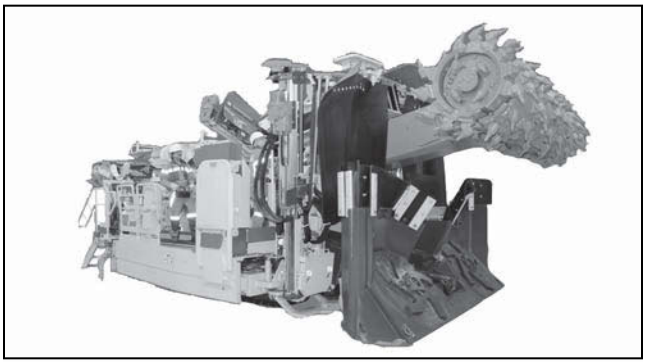
Roadheader

A roadheader can be used as an alternative to the CM. A roadheader is a large electrohydraulic machine that extracts the coal to form an arched profile roadway, often extracting a portion of the stone roof. Roadheaders are fitted with a pineapple-shaped



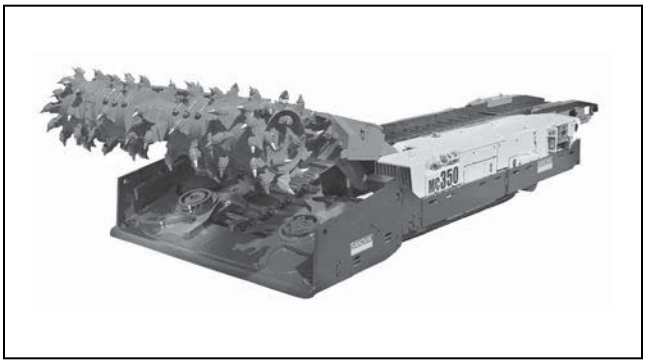
Courtesy of MineCraft Consulting.

Figure 12.3-15 Development process options flow chart



Courtesy of Sandvik.

Figure 12.3-16 Simultaneous cut-and-bolt continuous miner



Courtesy of Sandvik.

Figure 12.3-17 Place-change continuous miner

cutting head attached to a slewing boom. Similar to the CM, the cut material falls on a loading apron equipped with gathering arms or spinners that directs the material to a central chain conveyor that discharges from the rear of the roadheader.

Coal Haulage

Coal haulage units are required to convey the cut coal from the CM to the gate conveyor loading point. Key requirements include a large payload and rapid and flexible maneuverability. Three types of units are available, namely, electric-powered shuttle cars (SCs) with a tethered trailing cable, battery-powered ram cars, and diesel-powered ram cars.

Cable electric SCs (Figure 12.3-18) are commonly used for longwall roadway development and offer the benefits of being fumeless, do not require refueling or battery replacement, do not generate large amounts of heat, and do not require turning around (shunting) to discharge. Their disadvantages include attention to cable handling (cable damage) while in operation and a limited travel distance of approximately 200 m (the length of the trailing cable). Ram cars offer



Courtesy of Sandvik.
Figure 12.3-18 Electric shuttle car

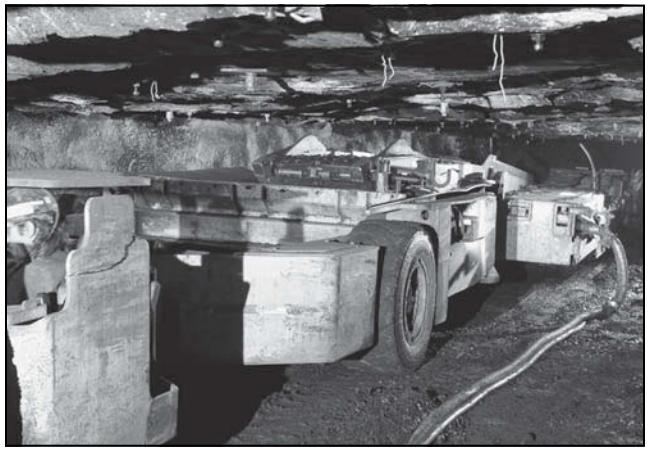


Courtesy of Sandvik.
Figure 12.3-19 Battery ram car

unlimited travel distance. However, they require regular refueling and require shunting to discharge. If battery ram cars are used (Figure 12.3-19), then a battery charge station is required, which is commonly placed some distance from the working face because of their construction and ventilation requirements. A ram car in operation is shown in Figure 12.3-20.

Several SCs can be used behind each CM. However, most Australian mines use a single car in longwall gate roads because of the short shuttle distance, the congestion caused by two SCs, and the need for an additional person in the crew. If ram cars are used, then at least two units are required to provide service while one is being refueled. For pillar extraction methods, several SCs need to be used (between two and four) to rapidly clear the coal when a pillar split has commenced.

As an alternative to coal haulage units, various techniques of continuous haulage have been developed over time, including flexible conveyors, bridge conveyors, and ADDCAR systems. Careful panel design is required for continuous haulage to gain the benefits of this technology. Consideration is required for roadway width and height, cutting sequence, clearances, face ventilation, cut-through angles, access to the face by vehicle, and conveyor extension process. Continuous haulage has been successfully used in several non-longwall coal mines and potash mines throughout the world.



Courtesy of Bucyrus.
Figure 12.3-20 Ram car being loaded at the face

Feeder Breaker

Feeder breaker units are typically required for large-capacity coal-haulage units. These consist of coal hoppers that accept the rapid discharge from the haulage units and then size and discharge the coal at a steady rate onto the section conveyor belt. Feeder breakers are fitted with crawler tracks so that they are self-advancing during panel extensions. However, they are typically left stationary during the mining process.

An advantage of feeder breakers, when using large haul units, is their ability to protect the gate conveyor boot end from knocks/collisions, which otherwise will misalign the conveyor. For a high-capacity development system, a basic requirement is the ability to quickly drive the haulage unit up to the discharge point and rapidly discharge the load. Feeder breakers can provide this requirement, whereas boot ends require a slower approach to avoid damage.

Some suppliers offer mobile boot ends that are a feeder breaker with a tail pulley and sufficient power to extend the conveyor belt (pull belt out of the loop take-up) when they advance. This can assist the belt-extension process. However, care is required to ensure the mobile boot end can withstand the belt tension as the panel extends. Staking props are often fitted to the mobile boot ends to anchor against the roof so that the belt tension does not pull them back.

Mobile Bolters

When the place-changing (cut-and-flit) method is used, roof support is installed by an independent bolting unit that trams into the heading when the CM is flitted out. Modern mobile bolting units are electrohydraulic powered; are fitted with up to four roof bolting rigs, two rib bolting rigs, temporary roof, and rib protection canopies; and have provision for the storage of consumables.

Electrical Power

Electrical power to a development panel is reticulated at the country-specific mine supply voltage (e.g., 11 kV in Australia) to a section transformer via an isolator switch located at the start of the development panel. The transformer, located within 600 m of the working face, steps the voltage down to typically between 950 V (United States) and 1,000 V (Australia) and is then fed to a distribution control box located within 200 m of the face. The distribution control box features up to

seven outlets to which trailing/flexible cables feed the various devices at the face (CM, SCs, fan, feeder, pump, etc.). Cable sizes reflect the various power demands of the equipment, with the CM cable typically 120 mm^2 and other cables typically 35 mm^2. The section feed cables are typically 150 mm^2.

Ancillary Equipment

Auxiliary fans are required to provide immediate face ventilation. Typically, these are centrifugal exhaust fans mounted on relocatable sleds or trailers placed in close proximity to the operating face in conjunction with either lightweight fiberglass or steel ducting (600 to 750 mm in diameter). These fans effectively place the fan's negative pressure suction inby of the CM operators, minimizing dust and methane buildup at the face. The fans commonly feature variable inlet vanes (to control airflow) and methane degassing valves (to prevent high gas levels from passing over the fan blades). The number of fans required for each working face depends on the number of roadways being developed, the method of working, and the level of activity. Commonly, between one and three fans are required.

An alternative method of face ventilation is by forcing ventilation in conjunction with dust scrubbers mounted on the CM. In this case, relocatable axial or centrifugal fans are located in close proximity to the face, with air directed using flexible or rigid ducting or brattice sheeting.

Panel dewatering is typically through pneumatic diaphragm pumps, as these pumps are able to operate in this arduous environment with minimal attention. Because of their relatively low output flow rate and their pressure-head capability, they normally report to a nearby electric pump pod. The electric pump pod units are capable of the greater pumping duties required to reach the central mine dewatering station before delivery to the surface.

Because of the high solids content associated with mine dewatering, system design and pump selection requires sound detail to avoid the settling of solids in the reticulation system (causing pipe blockages).

Mobile Roof Supports

For some of the secondary extraction methods (e.g., Wongawilli), mobile roof supports that are also called breaker line supports are used to provide a level of protection along the goaf (gob) edge. These are electric-powered, four-legged hydraulic roof supports mounted on caterpillar tracks with pendant controls. Typically, three units are used in a section. Breaker line supports are often used during longwall takeoffs to provide roof support as the roof supports are removed.

R&P Development Equipment Selection

The detailed specification of the R&P equipment needs to be determined at the time of tender preparation and is based on the results of detailed studies carried out during the feasibility studies. These studies include mining method analysis, panel configuration design, development process studies, productivity requirement studies, ground support specification, and engineering analysis.

Key specification aspects include mining strategy (development for longwall, first workings R&P, secondary extraction pillar splitting, etc.), face mining method (place change, in-place sequential, in-place simultaneous), type of CM, coal clearance method (continuous haulage or cars), bolting method (mobile bolters, CM-mounted rigs), and panel configuration.

Development Equipment Specification

The key aspects in selecting the most appropriate development arrangement and equipment specification include the following:

- Mining method: development or extraction
- Panel design: number of headings, pillar width, angle of cut-throughs, and roadway width
- Cutting method: in-place or place-change
- Cutting machine: roadheader or CM
- Type of CM: sequential, simultaneous, or place-change
- System of coal haulage: SC, ram car, or continuous haulage
- Number of SCs: one or two
- Number of bolting units: one, two, or three
- Method of face ventilation: method used

Table 12.3-3 summarizes a typical development arrangement for a longwall mine, and Table 12.3-4 summarizes a typical general specification for the development equipment for a first workings development system.

R&P Development Equipment Sizing

Development productivity is fundamentally measured in meters advanced per week or in metric tons per week (for which the conversion is relatively simple). Therefore, development equipment is typically sized to match a desired advance rate (e.g., 8 m/h) or output (e.g., 250 t/h). Unlike a longwall, where the equipment is completely integrated into a highly mechanized system, development is a collection of equipment that requires the mining engineer to integrate it into a process. The key process requirements are cutting, coal clearance, and ground support, and these are conducted by separate and independent items of equipment.

Development productivity can be calculated in a similar manner as longwall productivity, as shown in Figure 12.3-21. However, for development calculating the process advance rate is more complex, as itvneeds to take into account that the CM is continually moving from a fixed conveyor discharge point.

Two example development productivity cases are given in the following discussion. The first example, shown in Table 12.3-5, is for a conventional two-heading gate road entry for a longwall panel using a simultaneous miner bolter with a single shuttle car (100-m cut-through spacing). The second example, shown in Table 12.3-6, is for a three-heading development using the place-change method (60-m cut-through spacing). This example compares three mines with slightly different plunge depths.

Definition of the terms used in development productivity modeling include the following:

- **Nameplate capacity (NPC):** NPC is the peak cutting and loading rate of the CM (e.g., 950 t/h). It is often quoted by the equipment supplier and can be used to determine the time taken to load a shuttle car or to load onto a continuous haulage system.
- **Nameplate advance rate (NAR):** NAR is a measure of the peak advance rate of the CM. It is the time taken to cut and load one car, drive the car to the boot end, discharge its load, and then return, divided by the cut-out distance of the CM (e.g., 0.5 m). NAR is a useful factor when sizing the section conveyor belt.

Table 12.3-3 Development system—general arrangement

Parameter	Requirement
Roadway cross section	Nominal 5.2 m wide by 3.4 m high; rectangular profile
Working grade	Up to 1 in 10 (maximum)
Quantity of development panels	One main (up to seven headings) and two gate roads (two headings)

Courtesy of MineCraft Consulting.

Table 12.3-4 Development equipment—general requirements

Equipment	Requirement
Continuous miner	• Single-pass wide-head track mounted; one per panel; either simultaneous or sequential cut-and-bolt configuration • Fully remote operation with manual override • Four roof and two rib automatic-type bolters; to suit standard or long tendon bolts • Data communications, including real-time bidirectional between equipment and mine communication system • Onboard ventilation system • Dust-suppression system fitted; preferably with wet head design • Hydraulic system with capacity to suit simultaneous operation of all bolting equipment • Onboard roof support materials storage-and-handling equipment • Methane-detection monitors fitted • Fitted with integrated temporary roof-and-rib support systems
Coal haulage	• Electric shuttle car with trailing cable; one per panel • Nominal 15-t capacity; with feed-out rate of 1,400 t/h • Capable of travel speeds up to 8 km/h
Feeder breaker	• Track-mounted mobile feeder breaker; one per panel • Controlled via remote umbilical cable system • Nominal 22-t capacity with a feed-out rate 300 to 1,000 t/h (coal sizing to <200 mm)
Ancillary equipment	• Auxiliary fan; typical output 23 m^3/s • Ventilation ducting; nominal 720-mm diameter; range 610 to 760 mm • Pumping: pneumatic diaphragm face pumps and electric pump panel pods

Courtesy of MineCraft Consulting.

- **Process advance rate (PAR):** PAR is a measure of the maximum advance rate during the cycle, assuming that all activities are completed at their peak rate with no delays. The importance of this factor is that it can be determined by modeling and using actual measurements observed at the mine (or from benchmarking). Therefore, it is a scientific means of calculating productivity. The PAR will be less than the NAR, as it takes into account bolting, extending the ventilation ducting, and any other process-related activity such as repositioning the miner and lowering the canopy. The PAR will only be evidenced in practice over short durations at the mine (e.g., 2 to 6 hours), as it is rare that the process flows at their peak rate over an extended time in the development process.
- **Actual advance rate (AAR):** AAR is the advance rate achieved at the mine over an extended period (e.g., 3.6 m/h). AAR is measured from the mine statistical reporting system for an operating mine. For a new mine, this would be calculated from the modeled process advance rate (AAR = PAR × PRF).
- **Process reduction factor (PRF):** PRF is the difference between the AAR and the PAR. It is a measure of efficiency and reflects various issues such as adverse mining conditions, operator skills, motivation, and organization, and the complexity of the mining process. For new mines, this can be derived by benchmarking against similar operations and will generally range between 45% and 70%.
- **Operational availability:** This is a measure of the average number of hours per week that the development section is operating divided by the total planned operating hours in a week (e.g., 56%). It is calculated by dividing the actual operating hours recorded over an extended period by the planned operating hours. The difference between the two measures is the amount of unplanned downtime that occurs, either due to equipment breakdowns or process delays. The time to extend the panel is not included in planned operating hours. For new mines, the operational availability can be derived by benchmarking against similar operations and is generally between 50% and 70%.

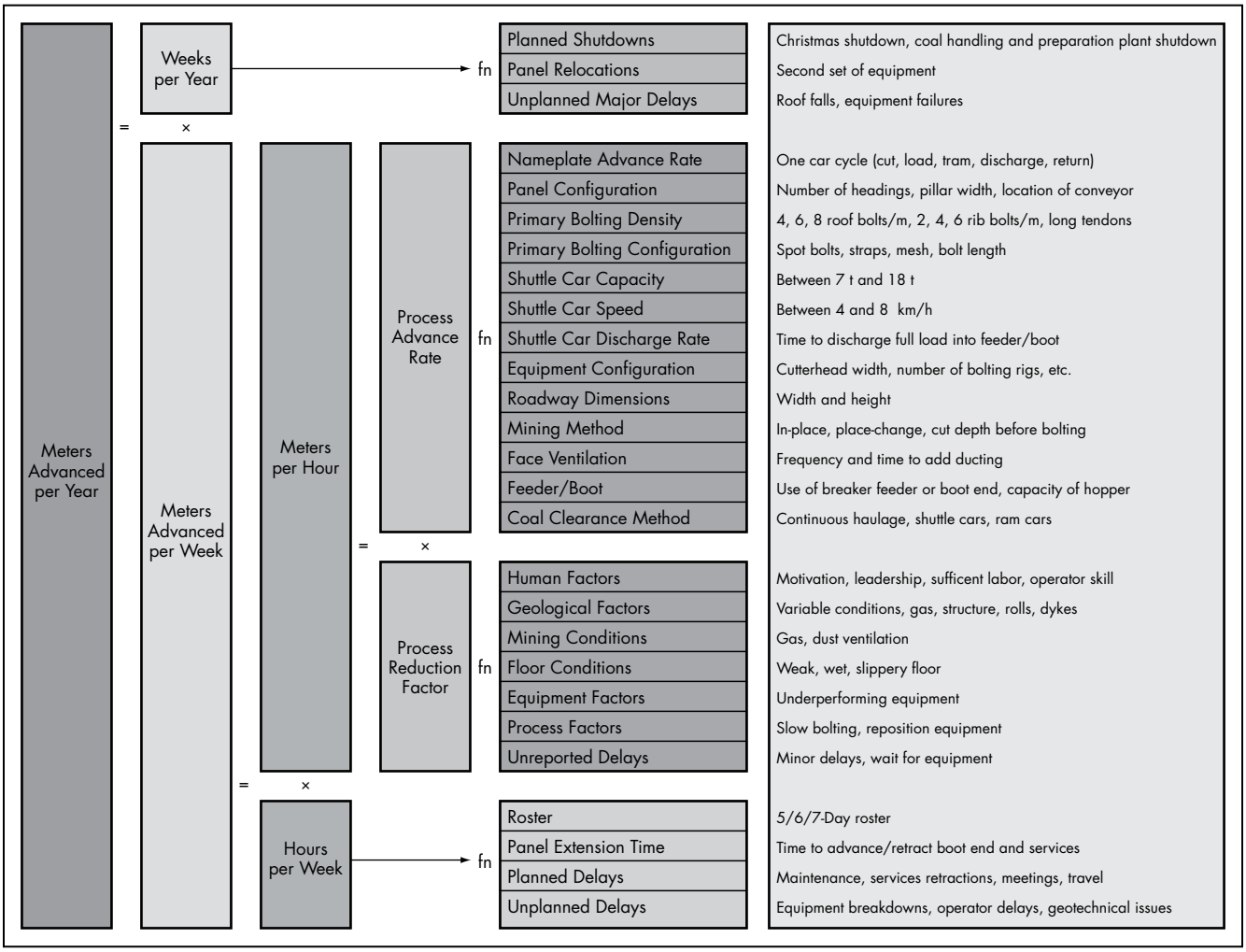
COAL CLEARANCE EQUIPMENT

The coal clearance system provides the means for transporting run-of-mine (ROM) coal from all mining units operating in the underground mine to the surface ROM stockpile. The coal clearance system consists of various types and sizes of belt conveyors, which are designed specifically for the various duty requirements associated with each mining unit. The loading pattern from each production face varies depending on the mining method. For example, longwall mining will produce a fairly constant flow for 15 to 20 minutes while the main cut is taken, punctuated by 10 to 20 minutes of irregular flows as the snake is taken. First workings development will produce very short duration high flows of 1-to-2-minute duration as each SC is discharged with a frequency between cars of 10 to 20 minutes. Occasionally, underground surge systems are used (bins, bunkers, etc.) to smooth the flow and allow for lower-capacity outby conveyors to be used.

The required duty of a coal clearance system depends on the mining system. For example, if the mine employs a 3,500-t/h NPC longwall system and up to three development units operating simultaneously (i.e., one unit completing mains development and two units completing gate road development), then the coal clearance system volumetric capacity duty would be as follows (simple method):

- Longwall gate conveyor: 3,500 t/h (matched to NPC of longwall)
- Development gate conveyors: 1,000 t/h (loading rate reduced to approximately 500 to 600 t/h by use of feeder breaker)
- Main headings trunk conveyor: 5,000 t/h (to allow for longwall plus three development units)

Detailed conveyor calculations are required to refine these duty requirements, which are undertaken during the mine feasibility study. These calculations take into account the panel gradients and panel lengths over the mine life and provide key specification data such as belt rating, power requirements, and the need for tripper drives (mid-panel booster drives). Specialist conveyor duty software is available to perform



Courtesy of MineCraft Consulting.

Figure 12.3-21 Development productivity factors

these calculations. However, some practical experience is required to audit the outputs and to ensure that realistic specifications are produced.

Where conveyors are required to run downhill, braking units are required. Because gate road conveyors are extended as the panel is developed and then retracted as the panel is extracted, care is required in the calculations to ensure that the maximum duty case is calculated. This is especially prevalent where undulating seams are encountered (both up- and downhill sections).

Because the development conveyor duty is commonly significantly lower than the longwall conveyor duty, the following options are available when developing the gate roads:

- Install a small development conveyor (e.g., 1,050 mm) for development and then fully replace this conveyor prior to longwall extraction.
- Install the longwall conveyor drive head and structure (e.g., 1,600 mm), but with a narrow belt (1,200 mm) and minimal drive units, and then replace the belting prior to longwall extraction and fit additional drive units.
- Install the longwall conveyor drive head, structure, and belting (e.g., 1,600 mvm), but twith minimal drive units, and then fit additional drive units prior to longwall extraction.

Preliminary Specification

Table 12.3-7 is an example of the general requirements for the coal clearance equipment for a new longwall mine. Specific details would also be required in regard to requirements for related aspects, including the following:

- Dust control and belt scrapers
- Transfer chute design
- Detection systems for belt wander, belt alignment, belt tear, chute blockage, belt slip, and belt tension
- Belt monitoring, emergency stop, communication systems, gas/fire monitoring (e.g. carbon monoxide detectors)

ANCILLARY EQUIPMENT

Ancillary equipment used in underground coal mines must comply with the local statutory regulations, which are intended

Table 12.3-5 Development productivity model for in-place mining

Process (two-heading gate road)	Australia Mine Calibration 90 m Up Travel Road	Australia Mine Predicted for Belt Road	Australia Mine Predicted Average for Pillar
	Nameplate		
Shuttle car speed, km/h	7.1	7.1	7.1
Time to load car, min	3.40	3.40	3.40
Car cycle time, min	4.60	3.53	3.94
Total car cycle, min	8.0	6.9	7.3
Total cars per hour	7.5	8.7	8.2
Metric tons per car	15.50	15.50	15.50
Metric tons per hour	116.2	134.2	126.7
Nameplate advance rate, m/h	3.8	4.3	4.1
	Process		
Roof bolting cycle time, min	2.80	2.80	2.80
Rib bolting time, min	3.60	3.60	3.80
Lower canopy time, min	0 .20	0.17	0.17
Advance time, min	0 .60	0.60	0.60
Load mesh time, min	0 .20	0.20	0.20
Raise canopy time, min	0.20	0.20	0.20
Prep tube time, min	0.00	0.00	0.00
Install tube time, min	12.50	12.50	12.50
	Process Advance Rate		
Bolting Cycle 1			
Install roof bolts, min	2.80	2.80	2.80
Install rib bolts, min	3.60	3.60	3.80
Lower canopy, min	0.20	0.17	0.17
Load mesh mat, min	0.20	0.20	0.20
Advance 1 m, min	0.60	0.60	0.80
Raise canopy, min	0.20	0.20	0.20
Cutting Cycle 1			
Cut and load car 1, min	3.40	3.40	3.40
SC1 cycle time, min	4.60	3.53	3.94
Cut and load car 2, min	3.40	3.40	3.40
SC2 cycle time, min	4.60	3.53	3.94
Test if still bolting	Wait for SC	Wait for SC	Wait for SC
Cycle 1 duration, min	16.00	13.86	14.68
Bolting Cycle 6			
Install roof bolts, min	2.80	2.80	2.80
Install rib bolts, min	3.60	3.60	3.80
Lower canopy, min	0.20	0.17	0.17
Load mesh mat, min	0.20	0.20	0.20
Advance 1 m, min	0.60	0.60	0.80
Raise canopy, min	0.20	0.20	0.20
Install vent tubes, min	12.50	12.50	12.50
Cutting Cycle 6			
Cut and load car 1, min	3.40	3.40	3.40
SC1 cycle time, min	4.80	3.53	3.94
Cut and load car 2, min	3.40	3.40	3.40
SC2 cycle time, min	4.60	3.53	3.94
Test if still bolting	Wait for bolting	Wait for bolting	Wait for bolting
Cycle 6 duration, min	20.10	20.07	20.07
Total duration of cycles 1 to 6, min	100.10	89.36	93.46
Total cars	12	12	12
Total metric tons	186	186	186
Total advance, m	6.0	6.0	6.0
Process advance rate, m/h	3.60	4.03	3.85
Process reduction factor, %	67	67	67
Actual advance rate, m/h	2.4	2.7	2.57

(continues)

Table 12.3-5 Development productivity model for in-place mining (continued)

Process (two-heading gate road)	Australia Mine Calibration 90 m Up Travel Road	Australia Mine Predicted for Belt Road	Australia Mine Predicted Average for Pillar
Pillar Cycle			
Meters per pillar	237.2	237.2	237.2
Duration, h	98.9	88.3	92.3
Operational availability, %	72	72	72
Actual duration, h	137.9	123.1	128.7
Panel extension duration, h	34.0	34.0	34.0
Total pillar duration, h	171.9	157.1	162.7
Roster			
Rostered days per week	7	7	7
Scheduled production time per day, h	17.6	17.6	17.6
Number of days per pillar	9.8	8.9	9.2
Number of pillars per week	0.7	0.8	0.8
Advance per week, m	170.2	186.2	179.8

Courtesy of MineCraft Consulting.

to ensure that these vehicles are safe to operate in potentially hazardous environments. The following comments relate to Australia, which has strict vehicle requirements.

Mobile Equipment

Mobile equipment refers to the diesel vehicles that are typically used in underground coal mines. They are categorized as personnel carriers, material transporters and utility vehicles, or special-purpose vehicles. Typically, these vehicles were developed over the years into purpose-designed vehicles to suit the rugged environment of the underground coal mine. Statutory regulations require that these machines comply with diesel engine exhaust-emission limits, be flameproof, have wet braking systems, and be fitted with various shutdown monitors (high temperature, low scrubber water, etc.). Recently, nonflameproof personnel carriers (modified Toyota Land Cruisers) have been successfully introduced into some Queensland mines. However, these suffer some restrictions in their ability to travel into working faces, depending on methane levels and statutory zones.

Personnel Carriers

Transport of personnel from the surface to the work site and back is through the use of personnel carriers having typical carrying capacities of up to 14 people, including the driver. Several models are available from suppliers, with some companies requiring additional safety features such as forward facing seats and seatbelts. Some personnel carriers are converted into maintenance utility vehicles with a rear flat tray for carrying tools and spares.

Material Transporters and Utility Vehicles

Material transporters and utility vehicles include a range of vehicles for duties such as equipment and materials transport and other work duties around the mine. Typical configurations include the following:

- **Load-haul-dump vehicles (LHDs):** LHDs are used with attachments including fork tines, loading plates, buckets, jib cranes, elevated work platforms, augers, hydraulic drilling rigs, stone dusters, pipe installation platforms, cable reelers, and trailers. LHDs are typically the workhorse of the modern mechanized mine and are commonly fitted with a quick-detach system. Typically, there are one or two of these vehicles, with a capacity between 7 and 10 t, at every panel. The mine will also have some higher-capacity LHDs, with capacities of between 10 and 15 t. These are used for roadwork and other heavy-duty requirements. A utility loader with a 10-t capacity is shown in Figure 12.3-22.
- **Multipurpose vehicles:** Multipurpose vehicles are specifically designed to pick up and carry modular pods and tubs.
- **Bobcats:** Occasionally, mines will use small LHDs such as bobcats for light cleanup duties.

Special-Purpose Vehicles

These are purpose-built vehicles that perform only one or a few specific duties on a periodic basis, with typical configurations including the following:

- Heavy-duty LHDs (40 to 50 t): Used for transport of longwall equipment and some belt conveyor equipment (Figure 12.3-23)
- Roadway graders and rollers: Used for road construction and maintenance
- Roof support trailers: These are used in conjunction with an LHD to transport longwall roof supports
- Specific shield haulers: Used for larger roof supports
- Shearer transporter: Used to transport the longwall shearer
- Mine dozer: Diesel powered, heavy-lift machine for moving longwall equipment
- Mule: Electric-powered, heavy-lift machine for moving longwall equipment
- Mobile generator: Used for CM flits
- Mobile bolters: Used for outby roof support
- Drilling rigs: Used for gas drainage and/or exploration

Some of this equipment is used infrequently, and there are often a number of leasing companies available to service the mine requirements. The majority of mines in Australia rent this equipment during longwall moves, although if the mine purchases equipment that is not typical (e.g., roof supports weighing more than 35 t), then the mine may be required to purchase its own longwall move equipment. The general

Table 12.3-6 Development productivity model for place changing

Process (three-heading gate road)	Australia Mine Example Four Bolts + Strap at 1.0-m Spacing, 10-m Plunge Depth	U.S. Mine Predicted Four Bolts + Mesh at 1.0-m Spacing	
		10-m Plunge Depth	15-m Plunge Depth
Nameplate			
Time to load car, min	1.0	1.0	1.0
Car cycle time, min	1.5	1.5	1.5
Total car cycle, min	2.5	2.5	2.5
Total cars per hour	24.0	24.0	24.0
Metric tons per car	15	12	12
Metric tons per hour	360.0	288.0	288.0
Nameplate advance rate, m/h	13.5	13.6	13.6
Mining Cycle			
Number of cars advanced	18	17	26
Metric tons mined	270	204	312
Plunge depth, m	10.1	9.6	14.7
Duration of mining cycle, min	45.00	42.50	65.00
Miner flit time, min	28.00	20 .00	20 .00
Bolting Cycle			
Number of rows bolted	17.0	10.0	15.0
Bolting time per row, min	3.50	3.50	3.50
Bolting duration, min	59.50	35.00	52.50
Vent tube duration, min	0.00	0.00	0.00
Stone dust duration, min	0.00	0.00	0.00
Duration of bolting cycle, min	59.50	35.00	52.50
Bolter flit time, min	15.00	20.00	20.00
Longest cycle, min	74.50	62.50	85.00
Total cars	18	17	26
Total metric tons	270	204	312
Total advance, m	10.1	9.6	14.7
Process advance rate, m/h	8.1	9.2	10.4
Process reduction factor, %	44.3	70	70
Actual advance rate, m/h	3.60	6.5	7.3
Pillar Cycle			
Meters per pillar	324.5	501.0	501.0
Duration, h	90.1	77.6	69.0
Operational availability, %	28.5	70	70
Actual duration, h	316.3	110.9	98.6
Panel extension duration, h	46.0	16.0	16.0
Total pillar duration, h	362.3	126.9	114.6
Roster			
Rostered days per week	7	7	7
Scheduled production time, h/d	2.7	18.0	18.0
Number of days per pillar	16.0	7.0	6.4
Number of pillars per week	0.4	1.0	1.1
Advance per week, m	142.3	497.6	550.9

Courtesy of MineCraft Consulting.

requirements for the mobile equipment are summarized in Table 12.3-8.

ELECTRICAL EQUIPMENT AND DISTRIBUTION

Electrical power is typically reticulated in and about an underground mine at the mine supply voltage, which will be derived from the surface main mine substation. Locally positioned substations are then used to transform the electrical power to a voltage suitable for utilization by the various electrically powered equipment. This includes the equipment used in the following areas:

- Development panels
- Longwall panel
- Conveyor systems

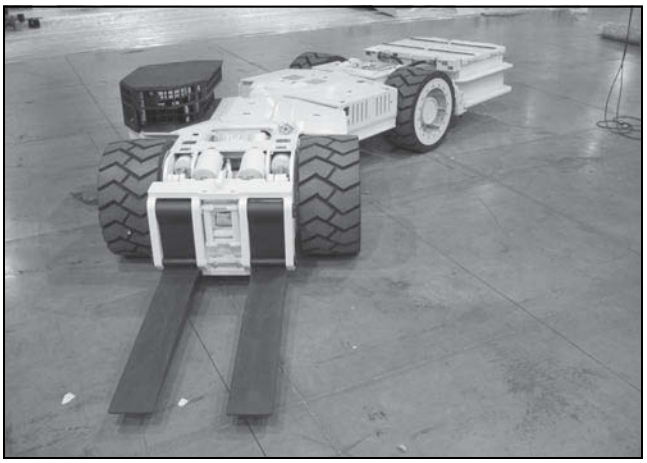
Table 12.3-7 Coal clearance equipment—general requirements

Equipment	Requirement
Construction conveyors	Conveyors are installed during the construction phase of the mine, catering for the lower outputs from the development units, and installed in a suitable location to allow construction of the permanent conveyors. The construction conveyors include a trunk and surface radial stacker conveyor. General specifications of the construction trunk conveyor include the following: • Conveyor length to suit mains development progress to a final length of approximately 1,250 m • A volumetric capacity of 1,800 t/h and a power capacity of 600 t/h, 1,200-mm belt width operating at 3.0 m/s, and solid woven 8,000 belting • Head end drive unit (1 × 500 kW) with transformer/starter unit • Electric auto winch take-up system General specifications of the construction radial stacker conveyor include the following: • Same volumetric capacity and belting width as the construction trunk • Nominal 11-m vertical height discharge rotating through 45°, providing ~5,000-t stockpile capacity On installation of the permanent conveyors, the temporary conveyors are decommissioned and removed.
Ramp conveyor	Conveying ROM coal to surface ROM stockpile. General specifications include the following: • Conveyor length of 710 m with a lift of 108 m from tail to head • Volumetric capacity of 5,900 t/h and a power capacity of 5,550 t/h • 1,800-mm belt width operating at 4.5 m/s, EP2500/5-ply belting • Head end drive station (3 × 750 kW) with transformer/starter unit • Nominal 35-m vertical height gantry structure for ROM stockpile discharge to provide ~100,000-t conical stockpile capacity • Gravity take-up system The complete conveyor is installed during the project phase construction.
Trunk T1A conveyor	Used for conveying ROM coal to the ramp conveyor tail end. General specifications include the following: • Conveyor length of 1,575 m with a lift of 151 m from tail to head • A volumetric capacity of 5,900 t/h and a power capacity of 5,000 t/h • 1,800-mm belt width operating at 4.5 m/s, EP2000/4-ply belting • Head end (2 × 750 kW), tripper 1 (2 × 750 kW), and tripper 2 (1 × 750 kW) drive stations with transformer/starter units as required • Electric auto-winch take-up system The complete length conveyor is installed during the project-phase construction, including both tripper stations. The tripper 2-drive unit and associated transformer/starter is not installed during the project phase because the additional power is not required until panel 2 longwall coal is being produced.
Trunk T1B conveyor	Used for conveying ROM coal to the Trunk T1A conveyor tail end. General specifications include the following: • Conveyor length of 2,000 m with a lift of 136 m from tail to head • A volumetric capacity of 5,900 t/h and a power capacity 4,600 t/h • 1,800-mm belt width operating at 4.5 m/s, EP2000/4-ply belting • Head end (2 × 750 kW) and tripper (2 × 750 kW) drive stations with transformer/starter units as required • Electric auto-winch take-up system A nominal 300-m conveyor length is installed during the project-phase construction, in line with expected mains progress at that time. Only one of the head end 750-kW-drive units is installed due to the low loading requirements, accepting coal from mains and gate road MG103 development units. The second head end 750-kW-drive unit and tripper drive stations, plus remaining structure and belting, is installed in line with mains development progress and conveyor duty requirements.
Trunk T2 conveyor	Used for conveying ROM coal to the trunk T1B tail end. General specifications include the following: • Conveyor length of 1,360 m with a lift of –38 m from tail to head • A volumetric capacity of 5,900 t/h and a power capacity of 5,050 t/h • 1,800-mm belt width operating at 4.5 m/s, EP2000/4-ply belting • A head end (1 × 500 kW) drive station with transformer/starter unit as required • A conveyor braking station located at the conveyor head end is required to ensure suitable conveyor stopping times • Electric auto-winch take-up system This conveyor is installed during the project phase construction.
Gate conveyor—longwall duty	Used for conveying ROM coal from the longwall system to the trunk conveyor system. General specifications include the following: • Conveyor lengths ranging from 1,620 to 6,200 m with lifts ranging from 77 to 103 m from tail to head • A volumetric capacity of 4,200 t/h and a power capacity of 4,000 t/h • 1,600-mm belt width operating at 4.0 m/s and solid woven 10,000 belting • Power requirements up to tripper 1 (2 × 500 kW), tripper 2 (2 × 500 kW), and head end (3 × 500 kW) drive stations, with transformer/starter units as required • A conveyor braking station located toward the tail end of the conveyor is required to ensure suitable conveyor stopping times • Electric auto-winch take-up system This arrangement of power and braking is for the worst-case longwall gate conveyor. Other conveyors with shorter length and less lift may require less drive units and no braking requirements.
Gate conveyor—development duty	Used for conveying ROM coal from the development system to the trunk conveyor system. General specifications include the following: • Conveyor lengths in accordance with "Gate Conveyor–Longwall Duty" section of this table • A volumetric capacity of 550 t/h and a power capacity of 200 t/h • 1,600-mm belt width operating at 2.0 m/s and solid woven 10,000 belting • The longwall duty head end and take-up equipment is used for development duties; however, belt speed is reduced to 2.0 m/s with a single 500-kW-drive unit and a transformer/starter being employed • A conveyor braking station located on the return belt is required to ensure suitable conveyor stopping times

Courtesy of MineCraft Consulting.



Courtesy of Sandvik.
Figure 12.3-22 Utility loader with a 10-t capacity



Courtesy of Bucyrus.
Figure 12.3-23 Heavy lift LHD

- Dewatering pump stations
- Area lighting

Table 12.3-9 provides an example of the various power supplies utilized in the underground electrical power distribution system for an Australian mine. For other countries, the voltage and frequency will differ (e.g., the United States uses 60 Hz and 950 V for the development face and 4.2-kV for the longwall face). The applicable standard may revert to international standards as opposed to Australian standards.

All underground power circuits operate using an impedance-earthed system (IT earthing system code in accordance with AS 3007.2-2004). The purpose of this is to limit the magnitude of an earth fault and, hence, the resultant touch-potential, thus reducing the risk of electric shock from indirect contact. Also, because most electrical faults occur as a result of cable damage, cables used underground are generally constructed with an earthed screening around each phase conductor, so that the initial type of fault is a low-energy earth fault rather than a high-energy short circuit. Controlling the energy dissipated during an electrical fault in this way reduces the risk of injury to personnel, fire, ignition of methane or dust, or damage to equipment as a result of an arc flash.

The performance of the underground electrical power-distribution system has a significant impact on the performance

Table 12.3-8 Mobile equipment—general requirements

Equipment	Requirement (Nominal Quantities)
Personnel carriers	• Three production vehicles (14 seat) for longwall and development crews • Three support vehicles (14 seat) for maintenance and outby support crews • Two spare vehicles (14 seat) that are used when the others are in maintenance • Two utility vehicles (6 seat) for maintenance crews
Material transporters	• Three-panel supply LHDs (7 to 10 t) for longwall and development • Three outby LHDs (7 to 10 t) • Two spare LHDs (7 to 10 t) • Two roadwork and outby LHDs (15 t)
Special-purpose vehicles	• One grader—roadwork • One mobile generator

Courtesy of MineCraft Consulting.

Table 12.3-9 Underground electrical power distribution*

Purpose	Voltage	Number of Phases and Frequency
Underground reticulation	11,000 V	3 Phase, 50 Hz
Development equipment	1,000 V	3 Phase, 50 Hz
Longwall equipment	3,300 V	3 Phase, 50 Hz
Underground conveyors	Drive type and size dependent	3 Phase, 50 Hz
Dewatering pumps	1,000 V	3 Phase, 50 Hz
Underground lighting	240 V	1 Phase, 50 Hz

Courtesy of MineCraft Consulting.
*IT earthing system code in accordance with AS 3007.2–2004.

and reliability of the electrically powered mining equipment. Consequently, the design for the underground power supply system needs to account for the following key power supply performance parameters:

- Fault level
- Thermal capacity
- Voltage regulation
- Transient motor starting capability (particularly with respect to the large longwall AFC motors)

Other factors that may influence the quality of the power supply and require detailed study include transient stability and harmonic distortion.

All electrical equipment must comply with the relevant parts of AS/NZS 4871:2002. In addition, equipment for use in an area designated as an explosion risk zone must be certified as being explosion-protected by complying with the relevant parts of AS/NZS 60079.10:2004.

Underground Reticulation

The underground reticulation system is typically comprised of the following components:

- Power supply (11 kV) from the surface to underground (via either cables in the mine entry roadway or via cased boreholes)
- Underground switchboard (11 kV)
- Underground reticulation cables (11 kV) (mounted along the roof of the development roadways)
- Section circuit breaker (isolators)

For a new mine, the power supply from the surface to underground typically occur in the following two stages: the initial development phase and the operational phase.

The power supply for the initial underground development is typically via the mine access portals and is achieved by means of an overhead power line installed overland from the surface main mine substation (e.g., 132/11 kV) to a site in close proximity to the portals (this could be to the top of a highwall if the entry is via an open cut or trench). A cable then connects to switchgear located at the portal entry switch room, which will control the power supply entering the mine and will be interlocked with the mine ventilation fans, to trip the underground power in the event of a ventilation failure.

This feeder will initially provide a power supply for the following loads:

- Initial underground power supply
- Development equipment
- Temporary drift conveyor
- Temporary ventilation fans
- Pumps and other ancillaries

In the longer term, this feeder will also provide the permanent power supply for the drift conveyor, pumps, and other loads located in the portal area.

Prior to the commencement of the longwall, the power supply to the underground operations will require upgrading, typically by using borehole feeder cables originating from the surface main mine substation (132/11 kV). Two 300-mm^2, 12.7/22-kV, cross-linked polyethylene (XLPE), single-point suspension borehole cables provide the power supply to an underground switchboard (11 kV) located in the main headings at the bottom of the boreholes. The underground switchboard is configured with two incomers, one bus tie, and four outlets for the control and distribution of the power supply to the following loads:

- Longwall feeder
- Development and conveyor feeder
- Pit bottom feeder
- One spare

The underground-cable selection process is a balance of the following considerations:

- Achieving acceptable performance parameters
- Minimizing the number of different types and sizes used underground
- Practical sizing for the purposes of handling the cables underground

All 11-kV reticulation cables must comply with Australia/New Zealand safety standards (AS/NZS 1972:2006), while cable couplers must comply with AS 1300-1989.

Load-flow modeling of the underground reticulation system is required to accurately specify the required cable types and sizes. A typical specification is listed in Table 12.3-10.

Although paper-insulated lead-covered (PILC) cable has traditionally been used for underground reticulation cabling, the use of XLPE cable is often recommended because it offers superior current-carrying capacity for the same size cable (approximately 50% increase in capacity over the PILC type). This is important to provide adequate current rating, while keeping the cable size as small as possible for cabling handling purposes.

Table 12.3-10 Underground 11-kV reticulation cable sizes

Application	Cable Type	Cable Size, mm^2
Development/conveyor feeder in the main headings	12.7/22 kV, XLPE	150
Development feeder in the gate roads	12.7/22 kV, XLPE	95
Longwall feeder in the mains headings	12.7/22 kV, XLPE	240
Longwall feeder in the gate roads	12.7/22 kV, XLPE	150
Longwall conveyor tripper drives in the gate roads	12.7/22 kV, XLPE	95

Courtesy of MineCraft Consulting.

Section circuit breakers are used to sectionalize the underground reticulation system (11 kV) and to control the power supply entering the various areas of the mine. Section circuit breakers are required at each gate road entry to control the power entering that development or longwall panel. The section circuit breakers incorporate the following features:

- Incoming and through-going supplies
- Two switched outlets
- Circuit breakers and protection relays for each outlet
- Isolation and earth switches for each outlet
- Programmable logic controller (PLC), network, and control equipment
- IP56-rated (minimum) enclosures (IP56 is an International Electrotechnical Commission weatherproof standard)
- Arc fault rated enclosures (using arc fault control methods)
- Base frame, wheels, draw bar and stabilizing/leveling legs

Development Panels

The electrical equipment for the development panels includes the following items: panel substations, panel distribution and control boxes, and panel trailing cables.

The power supply equipment up to the input connection of the development substation, which includes the 11-kV cables and section circuit breakers, forms part of the underground reticulation system.

The panel substations transform the incoming power supply (11 kV) to a 1,000-V (nominal) supply for distribution to the development machines and ancillary equipment. The current growing trend in the industry is for 2-MVA- (megavolt-ampere) rated substations to accommodate the increased power requirements of the development equipment. Generally, these substations are not flameproof, as they are located far enough outby of the working face to be in a nonhazardous zone. The panel substations typically incorporate the following features:

- Incoming and through-going high-voltage supplies
- High-voltage circuit breaker and protection relays
- Isolation and earth switches for the transformer section
- 2.0-MVA, 11/1.05-kV transformer
- Four-outlet low tension end
- 1,000-V switchgear and protection relays
- PLC, network, and control equipment
- Communication components for the CM
- IP56 (minimum) rated enclosures
- Arc-fault-rated enclosures (using arc-fault-control methods)
- Base frame, wheels, and draw bar

The panel DCB receives the 1,000-V power supply from the panel substation via a 150-mm^2 trailing cable and provides

Table 12.3-11 Panel trailing cable requirements

Application	Cable Type (AS/NZS 1802:2003)	Cable Size, mm^2	Cable Length, m
Panel 1,000-V feeder cable	241.1	150	110
Continuous miner cable	241.1	120	150
Shuttle car cable	275.1	35	275
Auxiliary cable (fan/feeder/pump)	241.1	35	110

Courtesy of MineCraft Consulting.

separate outlets for the distribution control and protection of the 1,000-V power supplies to the various development panel machines. The panel DCB incorporates the following features:

- Flameproof enclosures certified to the relevant parts of AS/NZS 60079.10:2004
- Incoming and through-going supplies
- Six to seven 1,000-V outlets with plug receptacles
- 1,000-V switchgear and protection relays
- Arc-fault-rated enclosures (using arc fault containment methods)
- PLC, network, and control equipment
- Communication components for the other development equipment
- Base frame with either wheels or fork tine boxes for transport

In Australia, all trailing cables must comply with AS/NZS 1802:2003, while the cable plugs must comply with AS 1299-1993. The plugs must be fitted and the cable assemblies tested in accordance with AS/NZS 1747:2003. Table 12.3-11 summarizes the recommended cable requirements for each development panel. All cables are normally supplied assembled complete with plugs, labels, and dust covers. Back-to-back couplers are required for the connection of the individual lengths of cable.

Longwall Panel

The electrical equipment for the longwall panel includes longwall substations; longwall DCB(s); monorail and face trailing cables; and control, monitoring, and signaling equipment. The power supply equipment up to the input connection of the longwall substation, which includes the 11-kV cables and section circuit breakers, forms part of the underground reticulation system. The longwall electrical equipment is normally included in the scope of supply for the longwall equipment package.

Underground Conveyors

The electrical equipment for the conveyor systems includes conveyor substations, conveyor starters, power and control cables, and control and signaling equipment. The power supply equipment up to the input connection of the conveyor substation forms part of the underground reticulation system (11 kV). The conveyor electrical equipment is normally included in the scope of supply for the respective conveyor system package.

Dewatering Pump Stations

The electrical equipment for the dewatering pump stations includes substations, pump station starters, power and control cables, and control and monitoring equipment. Generally, the dewatering pump stations will obtain their power supply from a substation conveniently located nearby, such as a conveyor substation or starter. However dedicated substations may be required for more distant pumping locations such as the pit bottom area to provide a power supply for pumping, an equipment tramming facility, and general power for the area.

Underground Lighting

Underground lighting is typically required in the following locations:

- Electrical equipment locations such as switchboards, section circuit breakers, substations, and DCBs
- Conveyor drives, loop take-ups, and transfer points, including the walkways in between
- Pump stations
- Transport access portal
- Other areas where a particular risk exists and which can be mitigated by lighting
- Initial 50 m (approximately) of the mine access drifts to assist the transport vehicle operators in the transition from daylight to darkness (and vice versa)

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CHAPTER 12.4

Underground Horizontal and Inclined Development Methods

Michael A. Tuck

INTRODUCTION

Two types of development exist in mining: primary and secondary. The main difference between these two types is the life expectancy of the development.

Primary implies long life and includes such items as shafts, declines, and other mine accesses; main tunnels, haulages, crosscuts, ore- and waste passes, crusher chambers, and other large chambers. Thus primary development is the development of the more permanent features of a mine. These do not have to be developed early in a mine life, but they need to have a relatively long life span.

Secondary development is more temporary in nature and tends to be associated with the needs of a particular production unit or units. In many cases this type of development is consumed as production proceeds. Thus a typical lifetime for this type of development is 1 or 2 years and is often less than this.

Development is a necessary part of mining as it provides the infrastructure with which production of ore can be undertaken. Development is essential but represents an additional cost. In most situations this cost is not recoverable until mining operations—such as preparing to excavate secondary development undertaken within the ore body itself—begin. However, in some situations (e.g., coal mine development of revenue-generating material and some payback is obtained during development. Primary development is normally almost entirely undertaken in waste external to the ore body; the exception to this is the mining of bedded deposits where some ore in primary development can sometimes be taken.

Development requirements are influenced by a number of parameters; the following list is far from exhaustive:

- Ore-body type
- Ore-body depth
- Mining method selected
- Economics
- Ore-body extent

The mining method selected is a significant influence, especially with regard to secondary development but also with respect to primary development, as it is important to the layout and size of the primary mine infrastructure to be able to handle the production of waste and ore over the lifetime of the mine. It is also important to account for possible variations in production over the mine life as well. Given the cost of development and the usual need for mining companies to generate income relatively rapidly, due to the long lead times between exploration and eventual production, the aim in most cases is to bring production on stream as quickly as possible. However, the long-term capabilities of a mine are dictated by the capabilities of the primary mine infrastructure, the need for accurate geological information, environmental considerations, and numerous other factors. During the mine planning stage, the primary infrastructure and secondary development are investigated. It is essential at this stage that a thorough economic, technical, and sensitivity analysis of all possible mine production systems be undertaken to evaluate the project on a logical basis.

Mine development serves a number of purposes, which include

- Accessing the mineral deposit;
- Provision of accurate geological information to aid the production process; and
- Providing accurate information on other technical needs such as geotechnical characteristics of the rock, water inflow rates, ventilation pollutant inflow rates, and so on.

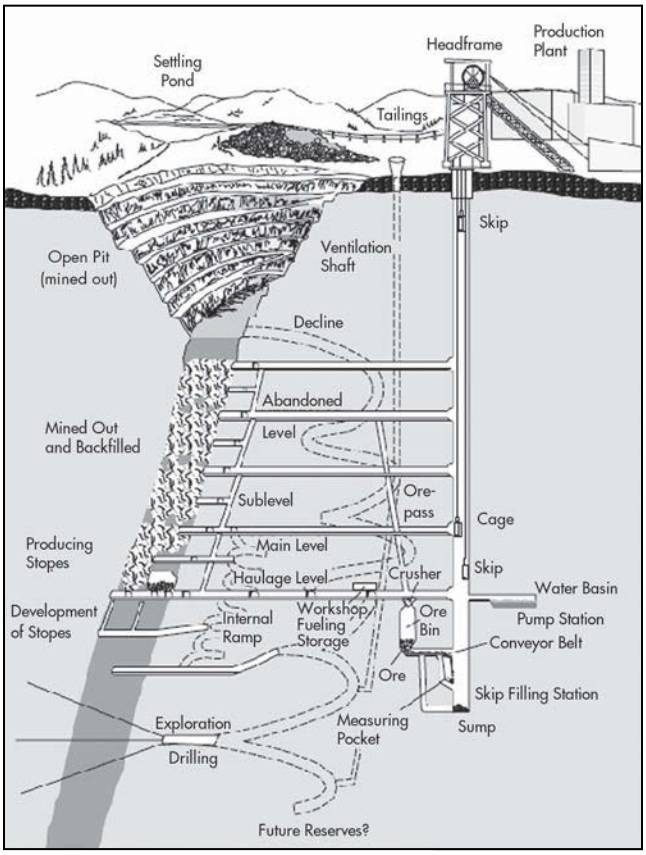
Figure 12.4-1 illustrates some of the openings whose construction will be discussed in this chapter.

EXCAVATION METHODS

Development openings in mines can be excavated in one of two ways, either by drilling and blasting or by mechanical excavation. Both techniques are used and are discussed in the following sections for various types of development openings. Both have their advantages and disadvantages; mechanized mining has the following advantages compared to drilling and blasting methods:

- Higher advance rates/development rates
- Smooth development profile and superior rock stability (tunnel boring machine and mobile miner circular profile in particular)

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Source: Atlas Copco 2010.

Figure 12.4-1 Basic infrastructure of an underground mine

- Minimized ground support
- Reduced overbreak
- Less surface disturbance
- Less noise
- Lower running costs
- Continuous haulage system, such as conveyor
- Automated haulage systems
- Less labor required
- Automation
- Precise direction control (laser guidance)
- Safety (no explosives)

However, there are disadvantages also, including

- Cost,
- Difficultly dealing with variable ground conditions, and
- In hard, abrasive rocks, high rates of wear on the cutters can lead to high operating costs and long periods of downtime for cutter replacement and other maintenance.

Shafts

According to *The Mining Glossary* (AusIMM 2007), a shaft is defined as "a primary vertical or non-vertical opening through mine strata used for ventilation or drainage and/or for hoisting of personnel or materials; connects the surface with underground workings." Although this is a good definition, shafts can also be internal within a mine connecting vertically separate workings as well as levels.

In some ways, a better definition of a shaft is a vertical or subvertical opening in a mine that is excavated from the top downward, as opposed to a raise, which will be discussed in a later section of this chapter.

Shafts are the most important capital openings for deep mines. Shafts provide all the services for underground operations including

- Fresh air,
- Transport of ore and supplies,
- Personnel transport,
- Power,
- Communications,
- Water supply, and
- Drainage.

The depth of the shaft has a direct influence on the total time to develop a mine prior to first production; in some cases, shaft sinking can account for up to 60% of the total development time. Consequently, proper selection of the shaft-sinking method to minimize sinking time and ensure uninterrupted operation is of great importance.

Determining the shaft diameter and hoisting depths requires consideration of future mining needs. It is generally better to overdesign the shaft in the initial stages of the project life span than prevent feasible increases in production later by the creation of a bottleneck.

Designing the excavation requires data to be gathered and solutions to elements within the design to be determined. In order to design the excavation the following need to be taken into account:

- Site geology, including a description of the geological column, usually in a table form identifying the rock strata, their geotechnical parameters, groundwater levels (including water heads, inflow rates, and chemical contamination, if any)
- Determination of the shaft diameter
- Choice of shaft-sinking technology
- Description of the shaft lining (should include a list of lining sections with their thicknesses detailed)
- Shaft foundations, including their location and dimensions
- Shaft collar depth, foundations, thickness, and materials to be used (the type and number of openings with a description of their function, dimensions, and elevations should also be detailed)
- Shaft sump, including depth, structural characteristics, pumping arrangements, and cleaning system
- Surveying data
- Calculations, including ground and water pressure acting on the shaft lining, the resulting lining thicknesses, shaft insets and their dimensions, and airflow capacity
- Timetable of construction
- Cost specifications
- Engineering drawings

Shaft Shape and Diameter

Numerous factors regarding the shape and diameter of a shaft need to be considered, such as

- Shaft duties and purpose, such as rock hoisting, ventilation, and worker and materials transport. Is the shaft to be used for a single purpose or for a variety of purposes? What are the quantities involved?
- The amount of water to be handled, number of pipes, and the diameter of the pipe are important parameters, as is the method of hanging these within the shaft.
- Ground conditions
- Size of mining equipment to be handled by the shaft. Is the equipment to be handled in single or multiple parts? For transportation of oversize elements is suspension underneath the shaft conveyance possible?
- The expected lifetime of the shaft

The majority of shafts in modern mining are circular in shape; however, rectangular cross sections were common in the past. Although rectangular shafts were more popular within metalliferous mines, most modern shafts tend to be circular. Other shapes are used such as square or elliptical. Circular shapes have inherent strength and have the advantage that for a particular cross-sectional area the minimum perimeter results. Elliptical shafts tend to be used in areas where a significant horizontal stress field acts in a particular direction; the major axis of the ellipse is aligned to this direction to enhance the shaft strength.

Shaft Linings

Shaft linings basically serve to support the shaft equipment and the walls of the excavation. The minimum lining usually required to support shaft equipment is about 200 mm. Thicknesses required for wall support are generally higher. In modern projects where round or elliptical shafts are employed, concrete lining is used almost exclusively. Such shapes reduce airflow resistance, make sinking easier and allowing the full advantages of concrete to be employed. The placement of concrete can be mechanized, allowing high sinking rates as well as lower costs. The strength of the concrete can be adjusted as required and watertightness can be ensured within aquifers of moderate head. Where heads exceed the strength of concrete, cast iron linings are employed.

In order to calculate the required thickness of the lining material, a number of factors need to be considered. First, the ground pressure acting on the circular shaft needs to be determined.

The vertical or hydrostatic stress, σ_v, is proportional to the weight of the overburden and can be calculated from

$$\sigma_v = \sum \gamma \times h$$

where

γ = density of the rock strata multiplied by the acceleration due to gravity
h = thickness of the rock stratum

The next stage is to evaluate the horizontal stresses. Horizontal stresses vary substantially from one location to another. The minimum value can be calculated using elastic theory employing Poisson's ratio to a maximum value derived from active or residual tectonic stresses. A good average ratio to employ at 500-m depth would show that the horizontal stress is two to three times the vertical stress.

For conventional shaft sinking, the technical limit for shaft water inflow is approximately 500 L/min. However, even the smallest inflow of water can cause problems and will require special arrangements to control it. Water seepage into a finished shaft may cause corrosion of steel installations such as buntons and reinforcing, and will lead to costly and troublesome replacement. Therefore, water inflow to the shaft should be controlled; this is especially true for the buildup of hydrostatic pressure behind a shaft lining. Numerous methods can be employed as described by Unrug (1992). Examples include the use of polyethylene or other shaft liners installed at the rock–lining interface.

The thickness of a shaft lining can be determined using

$$d_i = a\left(\sqrt{\frac{R_c}{R_c - np\sqrt{3}}} - 1\right)$$

where

d_i = shaft lining thickness (m)
a = shaft radius
R_c = allowable stress in the concrete (MPa) (see Figure 12.4-2)
n = coefficient of lining work conditions (see Figure 12.4-3)
p = calculated outside pressure acting on the lining (MPa)

Shaft Collars

The shaft collar is the upper portion of a shaft, which is anchored to the first footing as a minimum and to the bedrock (if required). Overall, the dimensions of the collar depend to a large degree on its depth, cross section, thickness, the function of the shaft, the overburden rocks, hydrologic conditions, the ground pressure (on a diurnal and annual basis), sinking method, and additional loading conditions. A simple collar design is illustrated in Figure 12.4-3.

The size of the shaft collar depends to a large extent on the loads in the immediate surrounding areas from the shaft and associated equipment. Examples of these loads include winding equipment and its associated dynamic loading, loading impacting on the shaft collar area from other surface facilities, and so on. The shape, thickness, and choice of shaft collars depend on numerous factors, but in essence depend on structural factors. Some typical collar shapes are illustrated in Figure 12.4-4.

Shaft Insets and Sumps

Shaft insets should be designed to minimize airflow resistance. Model and/or computational fluid dynamics models can be used to ensure this.

Sumps at the base of a shaft and at intermediate levels to minimize pumping heads must be of sufficient depth or size to cope with the water flow requirements. Obviously, suitable dewatering arrangements must be provided. In addition, periodic cleaning must be designed into the system to account for sedimentation, which will reduce the holding capacity of the sump.

Technology of Shaft Sinking

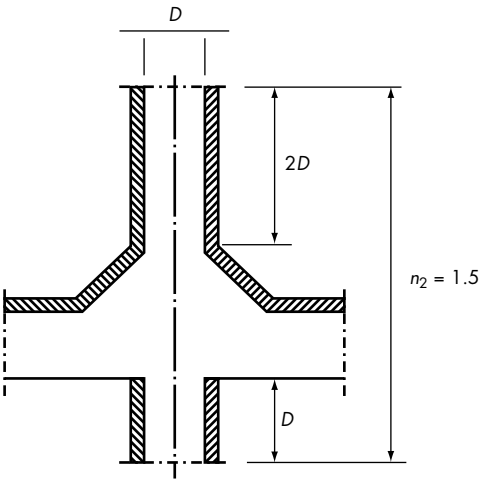
The main classification between traditional shaft sinking and other specialist methods is based on the engineering and hydrogeological conditions of the individual site.

The conventional method of sinking is summarized in Figure 12.4-5 and involves repeating cycles of face advance and lining erection without any previous ground stabilization. Such methods are applicable to good- to fair-quality rocks with limited water inflow into the shaft (<500 L/min).

Symbol	Type	Magnitude	Remarks
n_o	Influence of tectonic forces	In-situ measurements	n_o
n_1	Influence of shaft diameter D	$n_1 = \frac{1}{2}\sqrt[3]{D+1}$	
n_2	Influence of the shaft station	$n_2 = 1.5$	n_2
n_3	Influence of the strata dip x	$n_3 = 1.0$ if $x < 30°$ $n_3 = 1.25$ if $x > 30°$	n_3
n_4	Influence of the weak strata of thickness h	$n_4 = 1.0$ if $h > 1.3$ m $n_4 = 0.7$ if $0.8 < h < 1.3$ $n_4 = 0.3$ if $h < 0.8$	n_4
n_5	Influence of the water filtration through the lining	$n_5 = 1.0$ for watertight lining $n_5 = 0.1$ for formation grouting $n_5 = 0.1$ to 0.2 for water drainage n_5 = for preliminary lining	n_5

Remarks:

n_2



n_o—apply coefficient if $\sigma_h/\sigma_v > 1$

n_3 applies for cohesive soil and rock without water
n_4 applies for calculation of $H_{critical}$ in rocks
n_5 applies for calculation of water pressure

Source: Adapted from Roesner et al. 1984.

Figure 12.4-2 Determination of stress coefficient, *n*

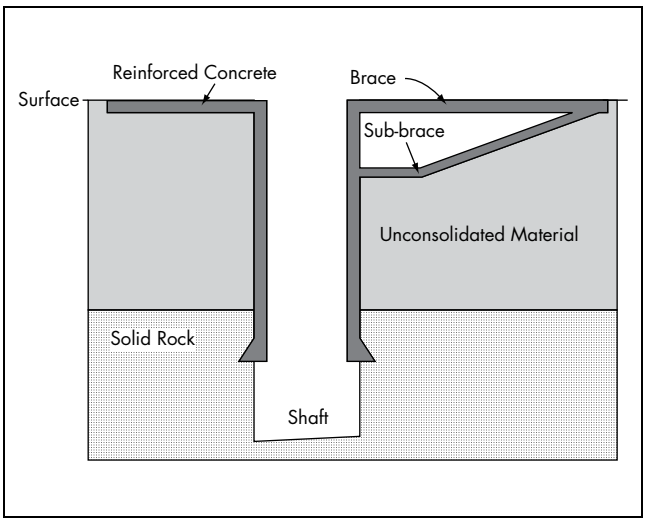
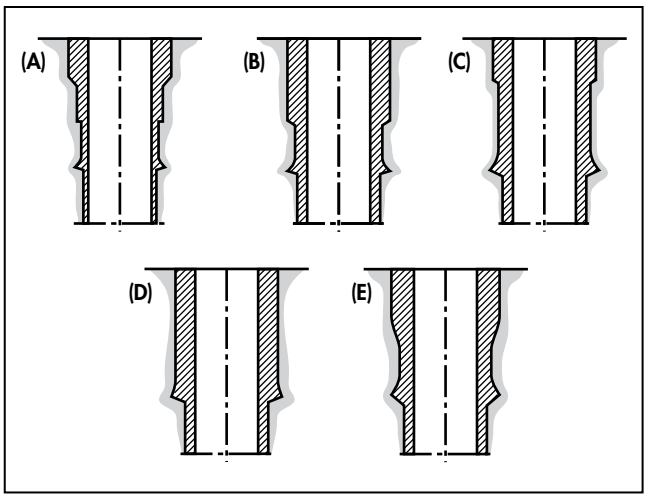


Figure 12.4-3 Typical shaft collar



Source: Unrug 1992.

Figure 12.4-4 Typical shapes of shaft collars

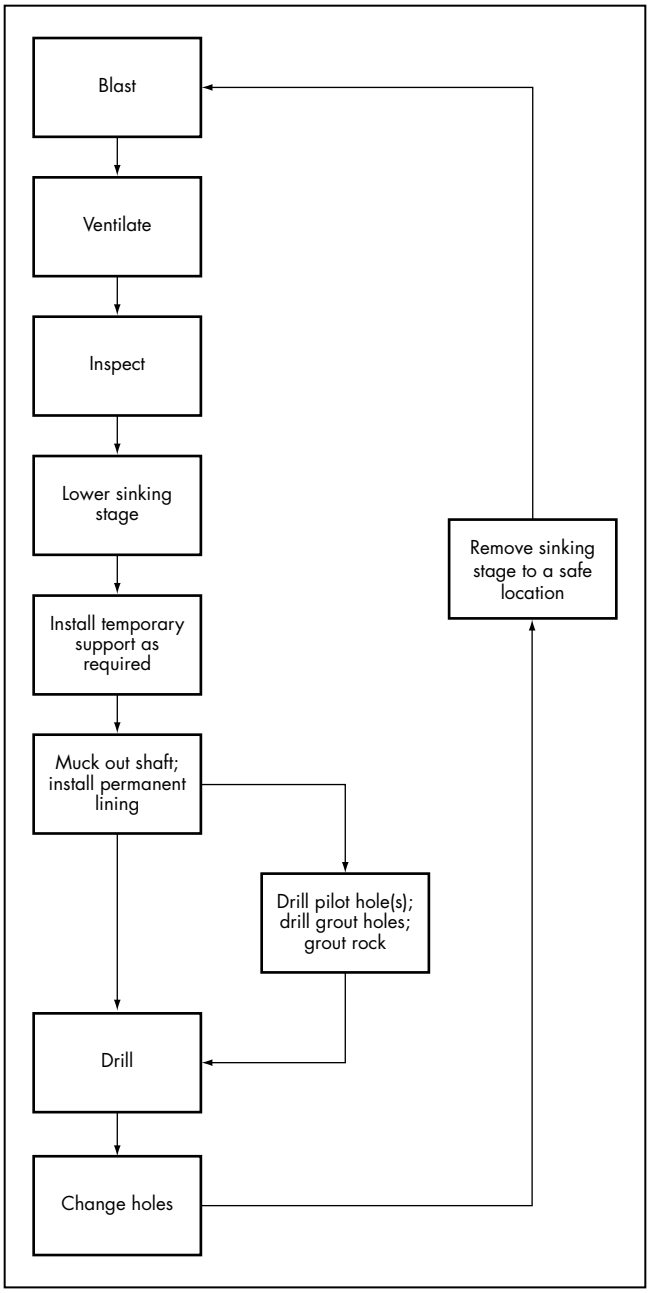


Figure 12.4-5 Typical drill–blast shaft-sinking cycle including a possible grouting sequence for rock strengthening

Specialized methods of shaft sinking include those applied to

- Strong rocks that have an extensive fracture pattern resulting in a strong water inflow (such as the South African goldfields);
- Coherent, weak, plastic flowing ground such as permafrost areas; and
- Loose water-bearing grounds.

Conventional Drill-and-Blast Shaft Sinking

The main components within a traditional drill-and-blast shaft-sinking operation are illustrated in Figure 12.4-5. Additionally, the following classifications of traditional shaft sinking can be made based on the order of the main components of the operation:

- *Series system*, in which sinking occurs with nonsimultaneous face advance and erection of the permanent lining. The shaft is sunk in sections. The length of the section depends on the rock quality. In a particular section of the shaft, the face is advanced first with a temporary lining. When sinking is suspended, the permanent lining is constructed. Fast sinking rates are not ensured; however, high capital expenditure is not required. This method is commonly applied to the sinking of small-diameter shafts at shallow to moderate depths.
- *Parallel system*, in which sinking with simultaneous face advance and lining erection at certain distances behind the face occurs. In this method, a temporary lining is required as with the series system. It is applied to the sinking of large-diameter and deep shafts where the high capital cost can be paid off over the life of the project.
- *Simultaneous system*, in which advancing the face and erection of the lining in the same shaft section occurs. This method can be further subdivided into series, simultaneous, and parallel simultaneous methods. The series simultaneous system involves face advance and lining completed in one cycle with each being done one after the other. The lining is erected either downward or upward and temporary lining is eliminated. The parallel simultaneous system involves simultaneous face advance and lining erection.

Shaft buntons, beams, guides, and so on are usually installed following the advance of the permanent lining; however, in some methods of shaft sinking (e.g., shaft boring) they are erected following the completion of the shaft lining.

Shafts can be sunk to the final depth or to depths required for certain stages of the mining process. Shaft extensions to subsequent levels need to be synchronized with reserve depletion in the upper levels to ensure uninterrupted production. Shaft extensions can be undertaken by either sinking or raising.

Before sinking a shaft, preparatory work needs to be done. This includes

- Determination of geological and hydrological conditions,
- Preparation of technical documentation, and
- Construction work associated with site preparation and construction of the shaft head.

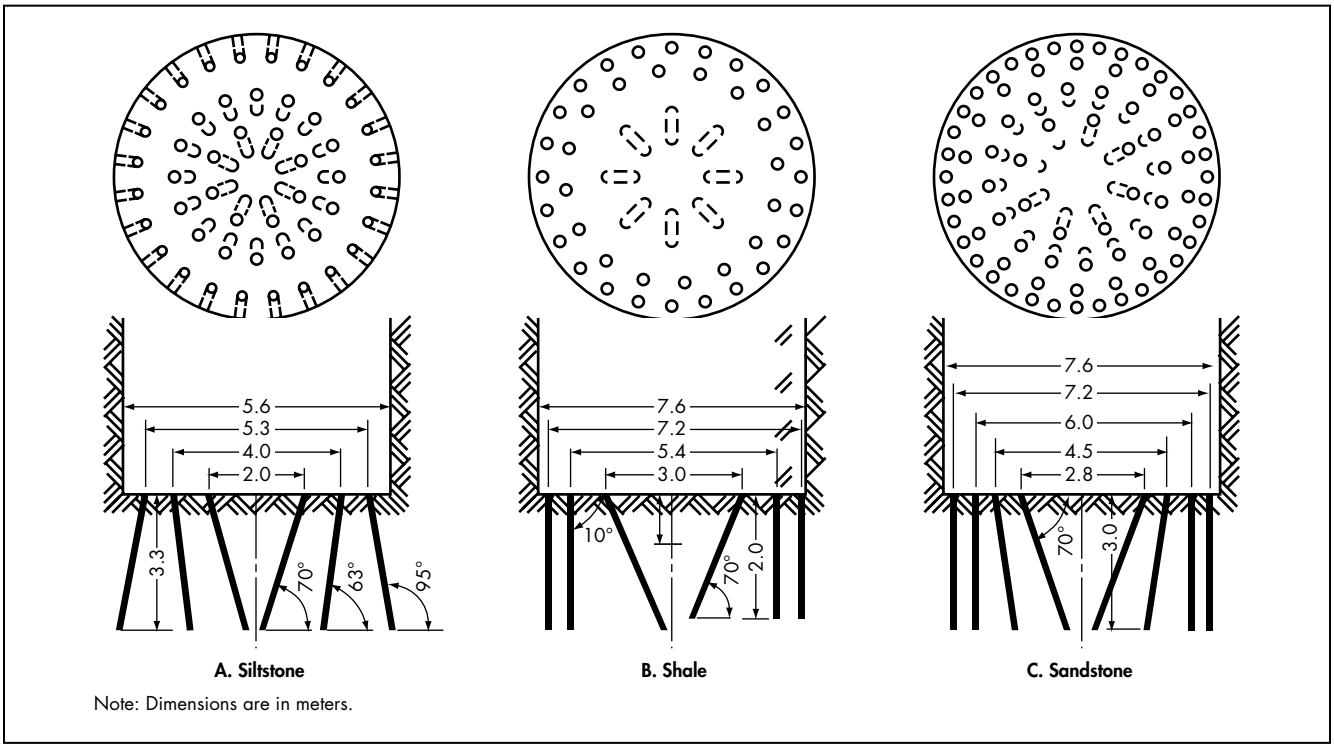
The shaft site should be secure from flooding. From a design perspective, this would be based on a 100-year flood level. Numerous activities associated with the preparation of a shaft site include but are not limited to

- Construction of access roads and materials storage areas,
- Grading,
- Provision of a water supply,
- Provision of electrical power and other power sources, and
- Temporary and permanent buildings.

Drilling and Blasting

When undertaken correctly, drilling and blasting will ensure

- Correct size and shape of the excavation,
- An even face surface for simpler mucking and drilling of the next round,



Source: Unrug 1992.

Figure 12.4-6 Typical circular shaft drilling patterns for different rock types

- Uniform sizing of the broken rock for effective mucking, and
- Safe and economic operation.

Shaft blasting uses a variety of high explosives. The correct product has to be used given site-specific conditions such as explosive gas emissions, reactive ground conditions, and the presence of water. Electric detonators with millisecond delays are also employed. The drilling pattern has a strong influence on sinking performance. It should ensure

- Accuracy in the shape of the excavation,
- Clean separation of rock from the bottom of the excavation,
- Even and sufficient comminution of rock,
- Avoiding cutoffs of neighboring active holes, and
- Avoiding damage to nearby shaft installations.

The drilling pattern depends on such factors as

- The shape of the shaft section,
- Rock strength,
- Cleavage,
- Dip of strata,
- Water inflow, and
- Hole-loading structure.

Typical drilling patterns for circular shafts are shown in Figure 12.4-6. With small and moderate strata dip, holes are placed on concentric circles drawn from the center of the shaft. The number of circles is three to five, depending on the shaft diameter. The central hole is vertical and shorter, about two-thirds the length of the other holes. This hole initiates conical excavation by cut holes and is especially recommended in strong rocks. Holes of the first (central) circle ("wedge holes") are inclined toward the shaft center and are fired first to open an additional free surface and to make the work of the remaining external holes easier and more efficient. They are drilled with an opposite (outward) inclination of 65°–75°. Four to ten holes are drilled for the first circle. The ends of four holes are intended to create a square with sides of 180–300 mm. For a larger number of holes, a 400–600-mm circle is used. In general, the harder the rock, the more first-cut holes are required. Holes of the next two, three, and sometimes four circles crush and lift the excavated material.

The inclination of holes varies from 65° to 85°; the farther they are from the center of the shaft, the more vertical they are. The distance between adjacent holes on a circle should not exceed 0.8 to 0.9 m for 32-mm and 36-mm cartridges; for a 45-mm cartridge, it should lie in the range of 1 to 1.2 m. The last circles of perimeter holes are supposed to leave the shaft wall smooth. They are holes drilled toward the shaft wall, 65°–85° (less inclination for weak rocks and more for stronger ones). In very weak rocks, they can be slightly inclined toward the center of the shaft. Perimeter holes are placed in a circle with a diameter of 0.6 m less than the shaft diameter in rocks with a coherence factor of $f = 2–8$ and 0.3 m for rocks with $f = 9–20$. The distance from the shaft wall has to be such that in rocks with $f < 8$, holes should not project from the design shaft section, and in rocks with $f > 8$, they should protrude about 100–200 mm outside the shaft contour. The distance between rib holes in moderate-strength rock is 0.9–1.2 m and in strong rocks 0.7–0.9 m.

Drilling of Shot Holes

Blastholes can be drilled with sinker percussive drills, hand-held or mounted on a special shaft jumbo. Sinkers are

Table 12.4-1 Factors influencing mucking efficiency

Depth, *m*	<150	150–300	300–500	>500
(factor)	(1.0)	(0.9)	(0.85)	(0.8)
Rock strength *f*	12–15	10–12 6–10	3–6 2–3	0.5–2
(mechanical mucking factor)	(0.85)	(0.89) (0.92)	(1.0) (1.1)	(1.4)
Shaft section area, m^2	<4.0	4–8 8–12	12–16 16–20	20
(factor)	(0.65)	(0.75) (0.85)	(0.9) (0.9)	(1.0)
Water inflow, L/min	<100	100–200	200–300	300
(factor)	(1.0)	(0.9)	(0.8)	(0.75)

Source: Unrug 1992.

self-rotating, driven by compressed air or hydraulically with air or water flushing.

Two systems exist:

1. *Series*, in which the drilling starts from the shaft center toward its perimeter after completion of mucking
2. *Parallel*, where mucking and drilling are done at the same time. Drilling starts from the perimeter toward the shaft center.

A shaft jumbo can be used only in the series system. A series system is used in smaller shafts when a Cryderman or similar type of mucker is used. This system ensures precision in blasthole drilling.

The parallel system is sometimes chosen in large cross-sectional shafts where cactus-grab cabin muckers start to clean the sides of the shaft moving toward its center. Drilling is done in the cleaned area. Owing to parallel drilling and mucking, some time saving can be achieved; however, it is at the expense of drilling accuracy. Blastholes can be drilled individually or by groups of drillers. In the first case, approximately 25% of the time is lost in changing bits. In the second, one shaft area is divided according to the number of drillers; each group drills in one sector holes of a certain depth only. Bits are changed only when they get dull. Then the drilling rod is changed (lengthened), and work is done in the next sector. Wooden sticks are used to prevent holes from being plugged by muck.

Drilling equipment is delivered in a special bucket (known as a kibble) or basket to the shaft bottom. An elastic hose with quick coupling is connected to the main steel-compressed air tubing. At the end of the elastic hose, there is a distributor with multiple outlets and valves for particular drills. Drilling is a labor-intensive operation that takes about 16% of the cycle time.

Shaft-drill jumbos have been developed to mechanize blasthole drilling. These permit the drilling of larger-diameter holes more precisely, faster, and deeper.

Mucking in Shafts

Mucking in shafts accounts for as much as 50%–60% of cycle time. Table 12.4-1 details the different factors influencing mucking efficiency. Figure 12.5-2 in Chapter 12.5 shows a schematic of the sinking phases with an Eimco 630 loader.

Mucking efficiency is also dependent on the degree of rock fragmentation and the height of the bucket. For mechanical mucking, medium-sized fragmentation, approximately 125 mm, is most efficient. Generally, in a muck pile after a shot, 70%–80% of the material is fine fragmentation, and the rest is composed of larger fragments.

Several types of muckers are in use depending on shaft shape, size, depth, and location:

- Scrapers, used in rectangular shafts and very large-diameter shafts (underground bunkers)
- Overshot loaders on caterpillars (e.g., Eimco 630)
- Cryderman muckers
- Cactus grabs (light)
- Cactus grab (heavy, on central pivot)

A variety of systems are used. Some are driven by an operator (e.g., Eimco), some are hand-guided at the face when hanging from a platform, and some are attached to the shaft wall (Cryderman) and operated from a control cabin. Heavy cactus grab units have a rotating arm on a central pivot below the stage. Muckers can be electric, pneumatic, or hydraulically driven.

To determine the capacity of the mucking unit, the following may be used. The theoretical mucking capacity is

$$P_t = 3{,}600 \times k \times \frac{V_b}{t}$$

where

P_t = mucking capacity (m^3/h)
k = fill factor (0.8–0.9 for sandstones and 1.1–1.2 for shales)
V_b = the bucket volume (m^3)
t = mucking time (seconds)

The mean actual efficiency is found by calculating the actual mucking time and all time losses for unproductive activity such as mucker lowering time to work position, time of temporary support erecting when mucking has to be suspended, and time for final cleaning.

Most shafts intersect one or more water-bearing horizons, so shaft dewatering is an essential part of sinking. The characteristic features of shaft dewatering consist of changing water inflow rates, changing the pump head, filtering water to eliminate rock and mud particles, and periodic changes in positioning dewatering devices.

Dewatering can be considered in two ways, either as limiting water inflow into the shaft or pumping water out of the shaft. Water inflow into the shaft can be diminished by initial drainage of the rock mass, sealing of the water flow routes, or by tapping the water. The following are used to dewater the face:

- Rock kibbles and special water kibbles
- Face pumps
- Combination of face and stationary pumps

- Combination of hanging and stationary pumps
- Submersible pumps
- Airlifts

Shaft Lining Construction

The type of lining is determined by the following factors:

- Hydrogeological conditions
- Shaft function
- Planned lifetime
- Shape of shaft section and its depth
- Availability of construction materials
- Construction cost

In choosing a shaft lining, the geotechnical properties of the rock and hydrologic conditions are decisive in most cases. The chemical activity (corrosiveness) of the water can also be an important consideration.

Since modern new shafts often have automatically operating hoisting gear (moisture sensitive), they should be dry. Main shafts are usually planned for the entire mine life span, so they should be constructed to minimize repairs and maintenance time.

Temporary support is placed to protect the crew and equipment against falling rocks from the exposed shaft wall before a permanent lining is placed. Numerous systems of temporary lining are in use. Forms of temporary lining include rock bolts and mesh and other traditional methods of temporary support used in other mining operations.

Depending on the shaft design and environmental conditions, several different permanent-lining systems can be used:

- Timber
- Brick or concrete blocks
- Monolithic concrete
- Reinforced concrete
- Tubing
- Shotcrete
- Anchor bolts

Shaft-Sinking Stage

Sinking stages are an important element of any shaft-sinking program; they provide a working platform from which to perform the sinking operations. They also ensure the safety of the personnel working at the shaft bottom. An example of a sinking stage can be found on the Andra Web site (2009). This Web site details the construction of test facilities in France for nuclear waste emplacement, with access by shaft. It shows the sinking stage being installed in the shaft following construction of the collar and gives technical details of the shaft-sinking process as well as the development of the test galleries underground. These can vary considerably in design from a simple single-deck working platform to complicated multideck arrangements with facilities for drilling, mucking, shaft concreting or timbering, and storage of hoses, pumps, and all other equipment associated with shaft sinking.

Sinking stages have been used since the earliest shaft sinkings and have developed to maximize efficiency of the operations while minimizing delays. The primary purposes of sinking stages are to enhance safety and maximize efficiency.

Stages are suspended in the shaft by wire ropes and winches that are independent of the main hoisting system. Stages are raised prior to blasting and then lowered to their new position at the end of each sinking cycle. Sinking cycles are usually based on a certain height of concrete pour or timbering.

Specialized Methods of Shaft Sinking

Special methods of shaft sinking are used when the following conditions are encountered:

- Incompetent, water-bearing rocks (e.g., quicksand)
- Weak, unstable, soil-type rocks
- Competent fractured rocks with high water inflows above 0.5 m^3/min

Numerous special methods exist that encompass many variations in method and application because of environmental conditions and range of depths. The characteristics of these are given in Table 12.4-2. The following methods will be described in subsequent paragraphs: depression of groundwater level, grouting methods, freezing method, and shaft drilling. An alternative to these methods can be applied in some cases (e.g., moving the shaft to another site).

Depression of groundwater level. Prior to designing a method for depressing the groundwater, the following decisions must be made:

- Choice of the most suitable method for depression of the water table under site conditions
- Determination of the number, spacing, diameter, and depth of the wells
- Calculation of the radius of the depression cone, amount of water to be pumped out, and the height of the remaining wet portion of the water-bearing strata
- Evaluation of equipment, hardware, and pumping facilities
- Choice of the auxiliary method to go through the wet remainder of the water-bearing strata
- Choice of a suitable water discharge plan

To lower the water table in the area of shaft sinking, the following systems can be applied:

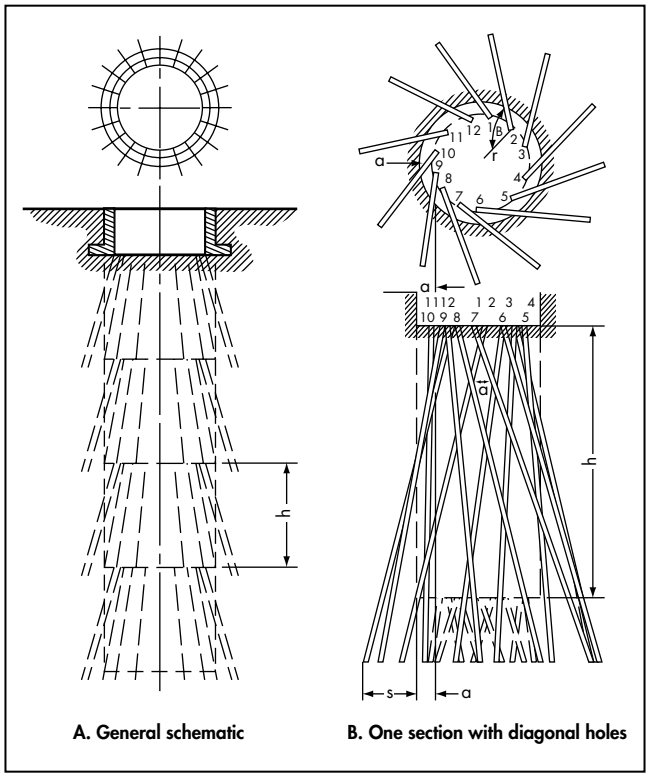
- Pumping water from three to six depression wells drilled around the planned shaft perimeter, placed on a circle about 4 m larger than the diameter of the shaft excavation (this system is used most often)
- Draining water through drill holes to existing openings in the mine below
- Draining water through drill holes to water-absorptive strata (e.g., dry sands under water-bearing strata)

Depression wells are drilled with 300–500-mm diameters, without drilling mud. They are spaced uniformly around the future excavation. Within the range of the water-bearing strata, they are equipped with filters perforated by round or elongated holes 5–10 mm in diameter, spaced in a checkered pattern, with an outside screen having eyes 0.4–0.9 mm in diameter, sized against entry of sand particles. Generally, the construction of the filter must be suitable for the character of strata to be drained. Submersible drainage pumps have a wide-capacity range, varying up to 350 m^3/h. Calculations for well efficiency are based on principles of flow through a porous media. The objective of the design is to determine a pumping regime to achieve a dry-out zone surrounding the shaft. This is created by the combined effect of the depression cones around particular wells. The movement of water through a porous media is given by the Darcy formula:

Table 12.4-2 Special methods of shaft sinking and their applicability

Method	Rock Mass	Thickness of Water-Bearing Strata, m	Water Inflow	Range of Depth, m	Approximate Rate of Advance, m/month	Comments
1. Pile wall • Wooden • Sheet pile wall (steel profiles or pipes)	Quicksands, soils (saturated with water without stones and rock fragments)	2–6 6–10	Limited	6–15 Down to 30	2–5 6–10	Required presence of impermeable strata of clay, into which the piles could be driven to lock water in above water-bearing strata.
2. Concrete pile wall drilled or driven piles	Quicksands, soils, loose rocks (sands, gravels), clays saturated with water (also with stones)	10–20	Any	25–30	8–10	Concrete pile wall can be used as a foundation for a shaft headgear.
3. Caisson method • Without pumping water in flow rocks • With steady pumping in weak and loose water-bearing rocks	Uniform flowrocks or loose rocks (sands, gravel), clays, without intrusion of hard rock or boulders	10–15	Not limited Up to 0.3 m^3/min	Down to 30; more in special cases when vibrators or compressed air can be applied	10–20	Flat-lying strata required or with little dip. Presence of the clay strata below water-bearing zone 2–3-m thick is necessary.
4. Caisson with compressed air	Loose or noncoherent water-bearing strata	8–30	Moderate	Down to 40	20–30	Maximum air pressure in the face chamber 3–3.5 atmospheres.
5. Depression of groundwater level	Mid and coarse solids and gravels with coefficient of filtration 10–100 m/d. Fractured competent rocks.	20–30	Great	Around 100–150	15–20	Not supposed to be applied in clay rocks, quicksands, and alternating packages of permeable and nonpermeable rock.
6. Chemical grouting	Sands with clay content not exceeding 10%–15%. Coefficient of filtration 2–80 m/d. Also competent rocks with small tissue.	A few	Moderate	Unlimited		Usually used as an additional interventional method.
7. Grouting (cement)	Competent rocks, with tissues and cracks filled with water (sandstones, shales, mudstones, limestones), and coarse sands and gravels.	Not limited	Mid and large, up to about 10 m^3/min	Any	10–15; 20–30 in good conditions	Not recommended in flow rocks or quicksands, with water flow velocity more than 80 m/d, or in presence of caverns and large voids.
8. Grouting with clay	Water-bearing rocks with large cracks and voids.	Not limited	Not limited	Not limited	10–15	Can be used in presence of aggressive waters. Usually is used combined with grouting (cement).
9. Bitumination	Competent rocks with large cracks and voids; also gravels	20–30	Mid and large	Down to about 40		Used as an auxiliary method for local sealing of the rocks with high flow velocity and aggressive waters.
10. Freezing of the rock mass	All types of water-bearing rocks	10–1,000	Not limited	Down to 1,000	Differs depending on depth, sinking frozen rocks up to 50 m overall less due to length of freezing method	Most reliable method, however, costly and time-consuming because of preparation work (drilling of the freezing holes and the time taken for an ice wall to form after freezing commences.)
11. Shaft drilling	Weak, water-bearing strata	Several hundred	Not limited	To 600		Thickness of competent hard rocks not supposed to exceed 10% of overall thickness of water-bearing strata.

Source: Unrug 1992.



Source: Unrug 1992.

Figure 12.4-7 Grout hole distribution when drilling from the face

$$V = K\frac{dh}{dl} = K_i = K\frac{h}{l}$$

where

V = mean velocity of flow or mean filtration velocity (m/s)
K = permeability or coefficient of filtration (m/s)
d = equivalent diameter of the flow routes (m)
$h = h_2 - h_1$ = difference of water levels in the ground or the depression created by the pumping of water
l = distance in which the head loss appears (m)

$$i = \frac{h}{l} = \frac{h_2 - h_1}{l} = \text{hydraulic gradient (dimensionless)}$$

Application of grouting. Based on geologic data, the following decisions have to be made prior to the start of the project:

- Choice of the grouting system, including the decision as to whether grouting will be done from the surface or from the bottom of the shaft; determination of the grouting depth; and determination of the length of the grouting sections
- Determination of the number of grouting holes and their respective distances, diameters, depth and angle of dip, and applicable drilling method
- Construction and calculation of the grouting plug
- Grout type to be used, its density and admixtures
- Injection system and required equipment
- Determination of working pressure to be applied in particular sections, and approximate calculations of grout consumption
- Number of control holes and their locations
- Set of drawings corresponding to this list of considerations

Necessary hydrogeological information should contain

- Number and characteristics of the water-bearing water horizons,
- Chemical analysis of water flows and determination of their degree of corrosiveness,
- Flow velocity, and
- Coefficient of filtration.

Properly performed grouting seals the rock mass in the area of the shaft, making sinking possible by a conventional method. Grouting is performed from the surface when the water-bearing strata are relatively thick and close to the surface, or from the shaft face when the strata to be grouted are located at such a depth that it is not feasible to drill long holes from the surface. Grouting is most successful in water-bearing, competent but fractured rocks. Grout fills the fractures and cracks, creating a sealed zone around the shaft to be sunk and protecting it against water inflow.

Grouting for sealing is often applied to improve tightness of the lining in existing shafts and to eliminate the voids behind the lining, as well as to seal rocks in the immediate vicinity of the lining (Petterson and Molin 1999).

When grouting from the surface, grouting holes with an initial diameter of 75–150 mm are drilled vertically around the future shaft on a circle with a radius 3–4 m larger than the excavation diameter. Holes should be straight, and with their greater depth, any deviation should be recorded.

Grouting from the shaft face is illustrated in Figure 12.4-7 and is recommended when water-bearing strata occur below 100 m and when their thickness does not exceed 70 m. The length of sections varies between 12 and 25 m.

Holes are drilled along the length of the grouting section, usually by means of heavy drill hammers or small rigs when space allows. Holes are directed at an angle in the vertical plane to get the hole bottoms 1–2 m away from the future excavation. To increase the probability of encountering fractures by the hole, they are also tilted in a tangential direction as shown in Figure 12.4-7. The distances between the hole entries are 0.8–1.5 m. Spacing between the circles of holes is 1.5–3 m. The number of holes drilled depends on the rock character, shaft diameter, and hydrogeological conditions. Practically, it varies from 10 to 30. The initial drill hole size is 50–75 mm; the end size is 30–50 mm. The sequence of drilling and grouting is as follows:

1. The first one-half of the total number of holes is drilled and grouted; during hardening of the cement, the second half is drilled and grouted.
2. All holes are drilled and grouted, one after another, starting with the hole with the largest water inflow.
3. A pair of holes lying on opposite sides is drilled and grouted; then the pair lying in the plane perpendicular to the first one is processed, and so on.

This system is the one most often used, being recognized as the best, because the initial maximum distance between the holes lowers the probability of grout penetration from one hole to another. Grouting plugs are put in the shaft bottom to prevent water and grout inflow into the shaft. Entrance pipes to the holes are secured firmly by grouting them in the plug. The plug can be either natural or artificial.

A natural plug is composed of a protective layer of grouted rock 3–5 m thick, depending on the strength of rock and expected pressure (water and grout).

Artificial plugs are used more often and are constructed of concrete or reinforced concrete. They are constructed on the shaft bottom, 2–4 m from the water-bearing strata. When grouting is completed, the plug is demolished with explosives, and sinking proceeds through the cemented section, but again is suspended 2–4 m above the end of the grouted zone. The shaft lining is constructed and the procedure with the plug is repeated until the shaft passes the whole water-bearing zone. Details of the use of grouting to sink a shaft through water-bearing ground at the Longos gold mine in the Philippines are described by Ayugat et al. (2002).

Freezing method. This is a method commonly applied where a relatively large thickness of water-bearing strata has to be sunk through. Some good examples are presented in Harris (1995). It is commonly applied when shaft sinking to the coal measures in the United Kingdom due to the thickness of the water-bearing Bunter sandstone.

For a freezing process, a suitable design procedure requires solution of the following problems:

- Calculation of the wall thickness of the frozen rock cylinder
- Calculation of the diameter of the circle where freezing holes are located, their number, depth and spacing, and the number and distribution of surveying and control holes
- Selection of the drilling system for freezing holes, specification of equipment, choice of freezing pipes and their diameter, and assessment of time required for drilling
- Design of a temporary shaft head with a freezing basement
- Planning of the freezing procedure in active and passive periods, with specifications of freezing temperature, heat balance, required efficiency of freezing installation, and freezing time
- Determination of the system and time of thawing and liquidation of freezing holes
- Initial planning of freezing technology, shaft sinking and lining, and complementary grouting

An artificially created cylinder of frozen rock or an ice wall around the shaft creates a shaft lining, thus shielding the inner space against water and unstable rock. The thickness of the frozen rock cylinder, e_c, in meters, for shallow and moderate depths is calculated for external pressure, p, in megapascals, from the Lamé formula (in metric units):

$$e_c = r\left(\sqrt{\frac{k_c}{k_c - 2p}} - 1\right)$$

where

r = shaft radius of the excavation (m)
k_c = allowable stress of the frozen rock (mPa)

The allowable stress of the rock is 50% of its compressive strength in the mean range of freezing temperatures. For calculation of the thickness of the cylinder of frozen rock, the mean temperature of the rocks should be taken. The temperature within the wall of frozen rock changes from a minimum value just beside the freezing hole to a maximum around 0°C on the outer wall limit.

The cylinder of frozen rock, when properly created, allows the sinking of a shaft of moderate depth even in long sections using temporary lining. This is desirable because there are fewer seams between lining sections.

The most important problem during the drilling of freezing holes is to ensure verticality of holes. Every hole is equipped with two columns of tubing as follows:

1. **Outside:** The freezing pipe itself has a diameter of 102–152 mm and is constructed of steel with inside connectors so it is smooth on the outside and has a welded bottom.
2. **Inside:** The inner tubing has a smaller diameter of 32–51 mm, is made of steel or polyethylene pipe, and has inside diameter and wall thickness of 75 × 43 mm, respectively. Polyethylene pipes are used because they have several advantages over steel. They are cheaper and easier to install (by unwinding from a reel), have lower thermal conductivity, and the specific gravity of the pipe is close to the specific gravity of the brine, resulting in less suspended weight. Freezing brine flows at high velocity down the inner pipe. It rises more slowly in the annulus, absorbing heat from the surrounding rocks.

The collars of freezing holes with pipes connecting them with the distributing ring, shut-off valves, and thermometers are installed all together in a so-called freezing cellar. A freezing cellar is designated either as (1) temporary, to speed up construction time (it will be demolished when the permanent shaft headframe is installed); or (2) permanent, considered a part of the permanent shaft headframe, requiring a longer construction time.

Freezing media used as refrigerants are

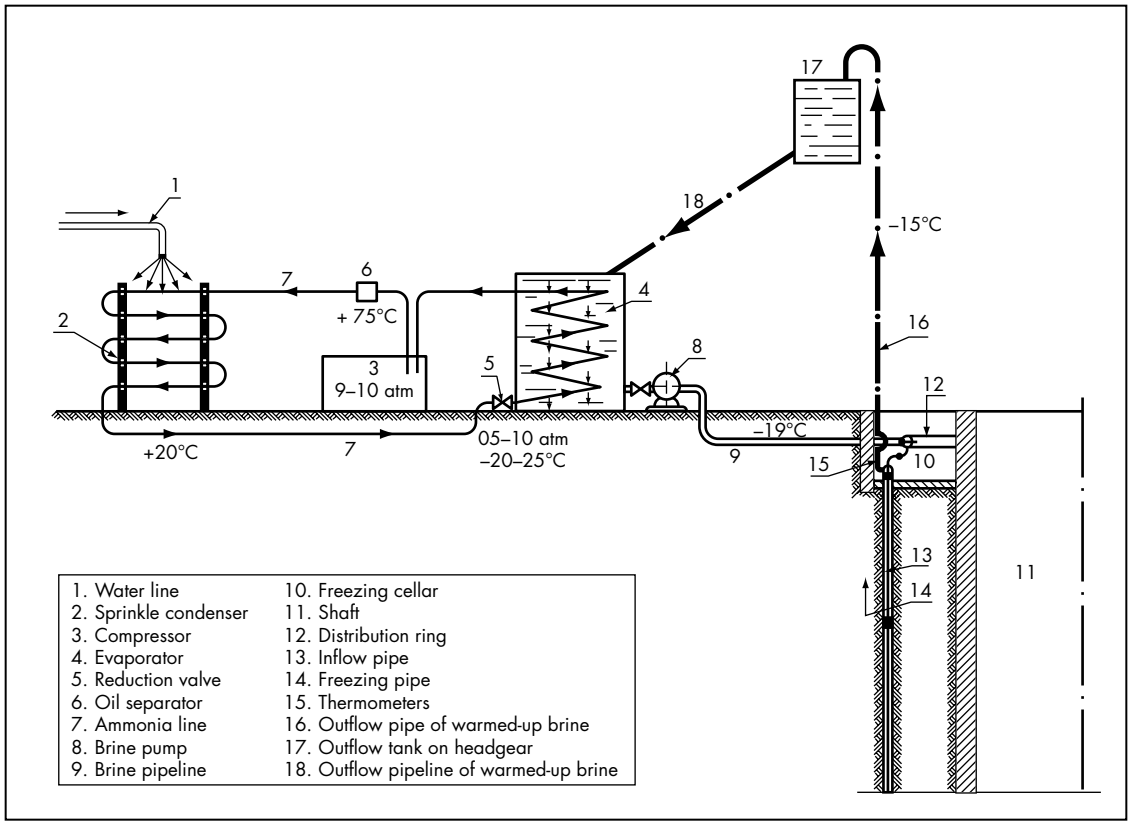
- Ammonia (NH_3): Ensures a freezing temperature of –20°C with one stage and –40°C with two stages of compression, and is used in most cases.
- Carbon dioxide (CO_2): Provides a freezing temperature of –50°C.

A schematic of an ammonia freezing station is shown in Figure 12.4-8. Two main periods of freezing can be identified: the active period and the passive or maintaining period.

During the active period, a frozen collar of a predetermined thickness is created around the future shaft. To achieve the designed capacity of the freezing station, two or three aggregates are installed together to provide the required heat efficiency. During an active period, all units work at full capacity. The passive period starts after the frozen collar is created and lasts through the sinking, lining, and assembling of the shaft installations. During the passive period, usually only one unit works, while the others are on standby as a reserve.

Methods of excavation in frozen ground are similar to those under normal conditions; however, when explosives are used, special care is required because of the proximity of the freezing holes and the potential danger of their destruction. When the core of a shaft is not frozen, a cactus grab or other mechanical mucker can dig the shaft bottom. When the rock is partly frozen or too hard to be excavated, handheld drill hammers with chisel picks are used; however, these working conditions are very unhealthy. In hard-frozen rock, the drill-and-blast method is used with limitations. The *Hard Rock Miners Handbook* (de la Vergne 2009) provides some good examples of freezing calculations and "rules of thumb" for ground freezing, as shown in Table 12.4-3.

Shaft drilling. Shaft drilling is a commonly applied technique for the development of small-diameter ventilation shafts in the United States. It has also been applied with moderate



Source: Unrug 1992.

Figure 12.4-8 Ammonia freezing station

success in Australia. Shaft boring uses a mechanical borer similar to a large-diameter drill bit to excavate the shaft. A typical 4.25-m cutterhead is illustrated in Figure 12.4-9 and further details of shaft drilling technology including technical details on equipment, methods of shaft drilling, and some case studies are provided on the Shaft Drillers International Web site (n.d.).

Zeni and Maloney (2006) identify the following hazards with a conventional drill-and-blast shaft-sinking operation:

- Falling of persons
- Falling objects
- Methane and other gases
- Hoisting and communications failures
- Equipment operating in a small work area
- Respirable dust
- Water inundation
- Back injuries
- Ignitions in the shaft
- Concrete burns
- Blasting
- Escape in an emergency
- Noise

Zeni and Maloney also state that many of these hazards can be eliminated using blind boring of shafts, because blind boring is probably safer than conventional sinking. Shafts in the range of 4.3–4.9 m in diameter are routinely bored using this technology, with larger diameters achievable. The blind-boring machine is rotated from the surface using a rotary table. In addition, the shaft is also completely filled with a fluid such as water; therefore, the boring element is completely submerged. The cuttings from the boring operation can be lifted from the bottom of the shaft in a number of ways; however, hydraulic and airlift systems are most common.

The most common problem associated with blind drilling is lining the shaft. The most common lining for shafts fewer than 4.6 m in diameter is the use of steel linings placed from the surface with concrete or grout used to bind the lining with the rock by injection in the annulus between the steel and rock. Other techniques do exist and are discussed by Zeni and Maloney (2006). These include remote slipforming concrete, lightweight composite linings made from extruded fiber-reinforced polymer, lightweight concrete combined with steel or polymer, and totally waterproof cast-in-place concrete. The future of shaft drilling depends on technological advances in fabricating and installing linings remotely, which would avoid the need for humans to work in a shaft environment.

Another good example of the method is presented by Bessinger and Palm (2007) for the installation of a ventilation shaft at the San Juan coal mine in New Mexico (United States). The paper particularly discusses the options available for construction of the shaft, the reasons why boring was chosen, and a number of the technical aspects of the project such as collar construction, pilot-hole drilling, shaft reaming, lining installation, and fitting of the ventilation and escape hoist system.

Table 12.4-3 Rules of thumb for shaft sinking using ground freezing

• For a ground-freezing project, the lateral flow of subsurface groundwater in the formation to be frozen should not exceed 1 m/d. (Source: Khakinkov and Sliepcevich)
• To determine the diameter of a proposed circle of freeze pipes around a shaft collar, 60% should be added to the diameter of the proposed excavation. (Source: Sanger and Sayles)
• When ground freezing is employed for a shaft collar, the area of the proposed collar excavation (plan view) should not be greater than the area to remain inside the circle of pipes (area that is not to be excavated). (Source: B. Hornemann)
• The minimum practical thickness for a freeze wall is 1.2 m. (Source: Derek Maishman)
• The maximum practical thickness for a freeze wall with a single freeze circle is 5 m. Concentric circles of freeze pipes should be employed when a thicker freeze wall is required. (Source: Derek Maishman)
• The radiation (heat transfer) capacity of a freeze pipe containing brine may be assumed to be 165 kilocalories/m^2 of pipe surface. However, if the brine velocity is too slow (laminar flow), this capacity will be reduced by 40%. (Source: Jack de la Vergne)
• The capacity of the freeze plant selected for a ground-freezing project should be 2–2.5 times the capacity calculated from the radiation capacity of the total length of freeze pipes installed in the ground. (Source: Berndt Braun)
• Groundwater movements over 3 to 4 ft/d are significant in a ground-freezing operation. (Source: U.S. National Research Council)
• If the drill casing is left in the ground after installing the freeze pipes, it will cost more but the freeze pipes will be protected from blast damage or ground movement and the heat transfer will be increased due to the greater surface area of the steel casing. (Source: Jim Tucker)
• The heat gain from circulating brine is equal to the sum of the friction losses in the pipes plus the heat generated due to the mechanical efficiency of the brine pump. The value calculated for the heat gain should not exceed 10% of the refrigeration plant capacity. (Source: Jack de la Vergne)
• The amount of liquid nitrogen (LN) required to freeze overburden at a shaft collar is 1,000 lb LN/yd^3 of material to be frozen. (Source: Weng Jiaje)
• Due to the heat of hydration, the long-term strength of concrete poured against frozen ground will not be affected if the thickness exceeds 0.45 m. Below this thickness, designers will sometimes allow a skin of about 70 mm. (Source: Derek Maishman)
• Ground freezing with liquid nitrogen should only be considered for temporary or emergency conditions (1–2 weeks). For the longer term, conventional freezing with brine is less expensive. (Source: George Martin)
• The freezing method may be simply applied to a small problem by laying bags of solid carbon dioxide (dry ice) against the problem area and covering it with insulation. In this manner, ground may be frozen to a depth of up to 1.5 m. (Source: C.L. Ritter)
• When making the calculations required for ground freezing, it is necessary to have a value for the natural temperature of the ground. If this temperature is not available, the average or mean annual surface temperature at the project may be used for this value without sacrificing accuracy. (Source: Jack de la Vergne)
• The brine circuit employed for a ground-freezing operation requires a surge chamber. The best one consists of an elevated tank (open to the atmosphere) in a short tower. It provides simple visual observance of the pumping head and, if equipped with a sight gauge, gives precise changes in the volume of brine in the system and early notice if a leak should occur. Slight losses in brine volume will occur as it becomes colder (coefficient of volumetric thermal expansion is 0.00280 per degree Fahrenheit). (Source: Leo Rutten)
• When freezing a shaft collar with a conventional brine system, one can tell when the freeze wall cylinder has built up to the point of closure by noting a sudden rise of the water level in the pressure relief hole drilled in the center. Excavation may normally commence soon afterward, since the ground pressure against the freeze wall is smallest near the surface. (Source: Derek Maishman)
• When freezing a shaft collar with a conventional brine system, the freeze pipe headers must be bled of any air accumulation once a day. (Source: Leo Rutten)
• When freezing a shaft collar with a conventional brine system, thermometer instruments should be recalibrated once a week by inserting a thermocouple into a container filled with water and ice (temperature exactly equal to the freeze point of water). (Source: John Shuster)
• If the freeze wall does not close when calculated, the problem may be due to a flow of groundwater. If the return temperature of the brine from holes on opposite sides of the freeze circle is slightly higher than the average, there is very likely a significant lateral flow of groundwater. The elevation of this flow in the ground strata may be determined by stopping the brine flow to one or more of the "problem" freeze holes, waiting 2 hours, and then measuring the standing brine temperature at vertical intervals (usually 0.6 m). The water is flowing where the brine temperature is slightly higher than the rest of the measurements. If the return temperature of the brine from only one hole is higher than average, it can mean that groundwater is flowing from the bedrock where it is penetrated by the freeze pipe. In either case, the default remedy is to grout from holes drilled to the vicinity of the problem area(s). If the return temperature from one hole is lower than average, it usually means there is a short circuit in that hole. (The inner freeze pipe, usually plastic, was not installed to the bottom of the hole, or it has come apart at a splice.) The proof is to pull the inner freeze pipe from the hole so it can be measured and examined. (Source: Jack de la Vergne)
• When a liability rider is incorporated in the contract documents for a ground-freezing project, the contractor will often extend the ground-freezing cycle. For a shaft collar, this often means that the excavation will be frozen solid inside the freeze circle before releasing it for excavation. Although this procedure provides added safety, it increases the costs and delays the project because excavation is much more difficult in frozen ground than it is in a soft core. (Source: Derek Maishman)
• An acoustic instrument has been developed to measure the thickness of a freeze wall under construction. (The speed of sound is different in frozen and unfrozen ground.) This instrument is not reliable (and its use may lead to great confusion) unless it is employed by a single experienced operator and is properly calibrated before the freezing process begins on a particular project. (Source: Jack de la Vergne)

(continues)

Table 12.4-3 Rules of thumb for shaft sinking using ground freezing (continued)

- If a brine freeze plant does not cool to the expected final temperature [at least –250°C], the most likely causes in order of greatest frequency are as follows:

Cause	Remedy
Low level of refrigerant	Top up
Clogged water spray holes	Remove and clean piping over condenser
Oil buildup in refrigerant	Bleed off at bottom
Air in refrigerant	Bleed off at top
Vacuum feed to compressor	Adjust to positive feed
Faulty valves in brine line	Replace
Faulty expansion valve	Replace
Undersize brine mains	Replace
Oversize brine pump	Replace

(Source: Jack de la Vergne.)

- If the freeze wall ruptures during excavation of a shaft collar, it will occur near the bottom of the excavation. Normal practice is to immediately clear personnel and equipment from the bottom and dump a truckload(s) of fine gravel or coarse sand in the excavation to prevent further inflow of soil through the freeze wall. It is then typical practice to drill from the surface and chemical grout in the vicinity of the breach. Then drill and install another freeze hole in the vicinity of the breach to accelerate and reinforce its closure, after which the excavation may resume. The new freeze hole may be frozen with liquid nitrogen to minimize the delay. (Source: Leo Rutten)

- If a freeze wall rupture occurs in a section of the excavation that is in very compact granular soil, undisturbed glacial till, or in the bedrock, normally only water will flow into the excavation. In this case, it is typical procedure to allow the excavation to flood. When the level of the water in the shaft collar has peaked, ready-mix concrete is tremied or pumped down through the water to form a concrete pad on the bottom. After the concrete has set and sufficiently cured, the shaft collar is pumped dry and grout is injected through the pad. When the grouting is completed and the freeze wall restored, the pad is broken up and the excavation can safely resume. (Source: Jack de la Vergne)

Source: Adapted from de la Vergne 2009.

WINZING

Winzing is the term for small-scale shaft sinking. Usually it is a form of internal mine development used, for example, to connect levels. Winzing has the following characteristics:

- Usually small diameter
- No room for mechanization, although in modern systems raise-boring machines can be used to develop winzes; however, most are still sunk by hand
- The possible range of sizes and depths are limited due to problems with muck removal
- Muck removal is usually by hand shoveling
- Use of pilot holes makes winzing easier as deviation can be avoided if levels are connected by a pilot hole
- Ventilation is highly important, but usually poorly done
- Secure ladders are essential for access
- Small winches are used for raising and lowering small kibbles
- Typically a bench cut is used
- Water can be a problem, but its impact can easily be overcome by using a pilot hole
- All services need to be removed prior to a blast (ventilation ducting, water pumps and hoses, compressed air lines)
- Hard physical work, especially in hot conditions
- Vacuum-lift devices have been used successfully for muck removal

Winzing is always a slower and more expensive undertaking than anticipated. In modern mining it is avoided wherever possible and raising is used in preference.

RAISING

A raise is a vertical or steeply inclined opening excavated from a lower level to a higher level (i.e., inverse winzing). The inclination is such that a ladder is required for passage. Raises are used to gain access to ore, as traveling and supply routes, and as ventilation routes. Raises are also used as orepasses, waste passes, or as slot raises to provide free faces for initial blasting of certain types of stopes.

Raises can be constructed through conventional raising, cage or gig raising, Alimak raising, longhole raising, and raise boring.

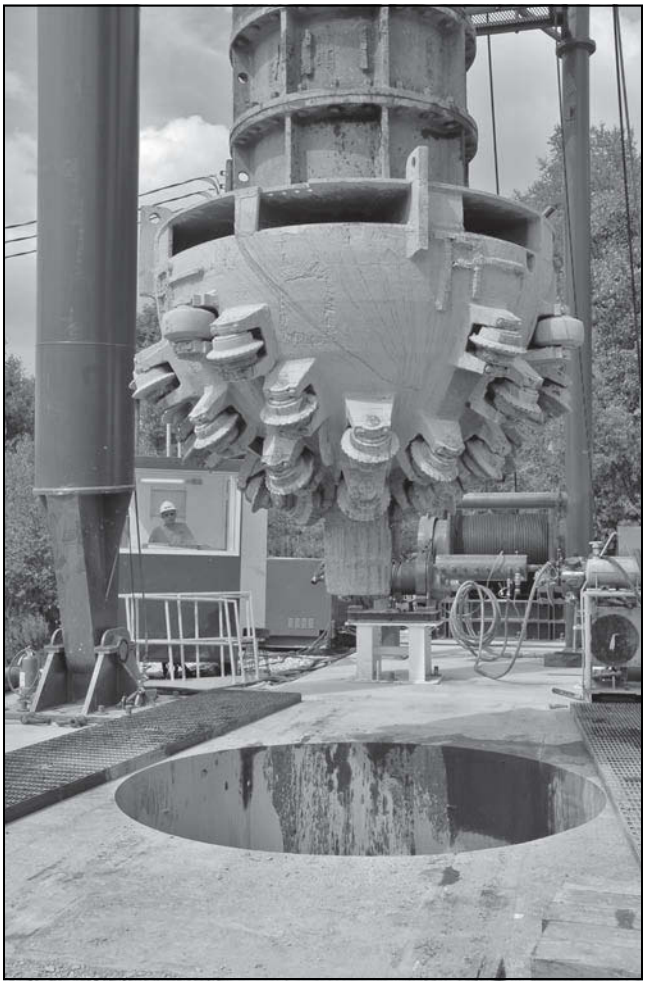
Conventional Raising

This method is restricted to shorter raises of up to 100 m in length and involves drilling and blasting using handheld drills. An important requirement in this type of raising is the provision of a working platform at the face to allow drilling and charging operations to occur. This platform is dismantled prior to blasting. Raising operations can be dangerous if correct procedures are not followed. The advantages of raising include the following:

- Raising is independent of mining production, so its input on current mine performance is minimal.
- Loading of muck into buckets is avoided.
- Blasting effectiveness is greater.
- No water pumping is required.

The disadvantages of raising include the following:

- The height of a raise is limited to between 100 and 120 m.
- Ladder climbing is an inconvenience for personnel and the transport of materials to the face.
- Potential hazards of rock falling from the face exist.
- Potential threats of muck jamming in the chute compartment exist.
- Surveying is difficult (deviation is possible).
- Time and cost implications are involved in preparing the opening for raising purposes.



Courtesy of Shaft Drillers International.
Figure 12.4-9 Blind boring cutterhead

Cage or Gig Raising

Cage or gig raising is a mechanized raising technique and is illustrated in Figure 12.4-10. A pilot hole is drilled from the upper level to the lower level. A hoist rope is passed down the pilot hole and attached to a cage (or work platform) that can be raised or lowered in the raise to the face, allowing work to proceed as normal by hand methods.

Alimak Raising

Alimak raising is a development of cage raising using a rigid steel guide rail attached to one wall of the raise. No pilot hole is required and all the controls are mounted on the cage or at the base of the raise. The design of the system incorporates provision for air and water hoses, drilling equipment, blasting requirements, and accessories. Alimak raising has the capability of driving longer raises at faster rates than conventional methods.

The machine runs on a guide rail that incorporates a pin rack. The segmented guide rail is fastened to the side of the excavation with rock bolts. The Alimak method has the following advantages over conventional systems:

- It allows the driving of very long raises, vertical or inclined, straight or curved.
- When traveling to the head of the raise, personnel are well protected in a cage under the platform.

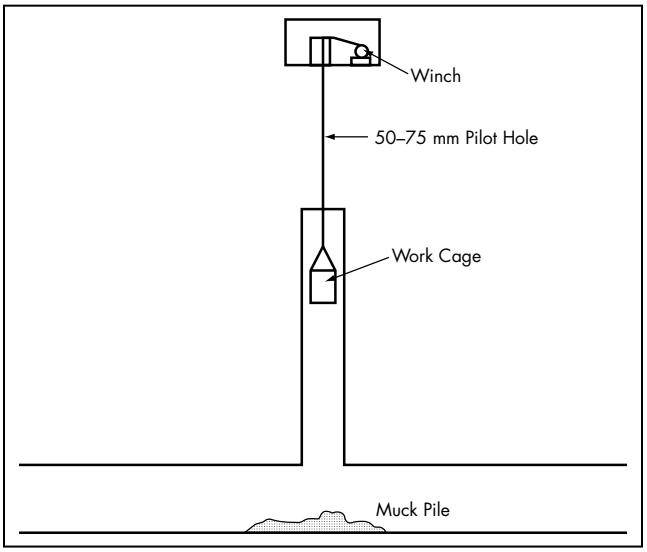


Figure 12.4-10 Cage or gig raising

- The miners work from a platform that is easily adjusted for convenient height and angle.
- Risks involving gases are reduced, as the raise is ventilated after blasting with an air–water mixture supplied in the raise climber.
- All equipment and material required can be taken to the head of the raise in the raise climber.
- Timbering is avoided, and if needed, rock bolts, with or without mesh, are used for support.
- The raise climber can be used for areas up to 8 m^2.

The disadvantage is the cost of the raise-climber system, which cannot be justified for short raises; therefore, the most frequent application is in mines where driving raises in hard rock is part of the mining method. The Australian Contract Mining Web site (2009) provides some excellent examples of the use of the Alimak climber as well as many images of Alimak climbers.

Longhole Raising

In this method all operations are undertaken from the top level. Long holes are drilled from the top level to the level at which the raise is to start. Holes need to be drilled accurately and can be reamed out. Charging is undertaken from the top level and blasting carried out in a similar manner to that used in vertical crater retreat mining. Figure 12.4-11 illustrates the longhole raise method.

Raise Boring

Raise boring is the most common method of raising in modern mining. It is a safe method for the driving of long raises, but it requires a high capital expenditure. Numerous methods of using raise borers are available such as blind-hole raising, pilot-hole methods, and so on.

Raise boring is one of the newest technologies in shafts and raises and has matured through improvements made during the decades since its initial application. Large-diameter holes or shafts can be drilled vertically or inclined. The costs of raise boring have been substantially reduced due to the improvement of cutters.

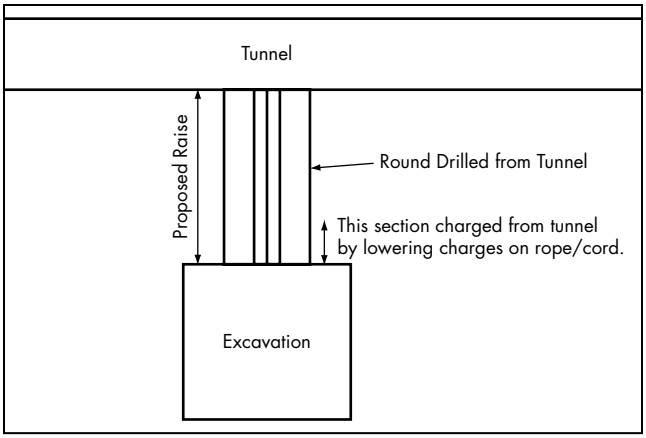


Figure 12.4-11 Longhole raising

Raises are used as a part of multilevel mining systems for orepasses and ventilation, as coal-mine ventilation shafts, and also as full-sized shafts, primarily in coal mining.

Excavating rock by a continuous boring system has several advantages over the conventional drill-and-blast methods, especially in vertical excavations such as shafts and raises, where it eliminates the hazards associated with the presence of workers at the face, similar to shaft boring. The continuous action of raise boring usually ensures faster completion of the project than conventional blast-and-drill methods. Boring does not disturb the surrounding rock, and as such, often there is no need for support or, alternatively, only minimum support is required for the finished excavation. A perfectly round shape makes the hole inherently stable, and its smooth walls reduce resistance for ventilation, as well as for the passage of ore with little prospect of hang-ups. Provided conditions are amenable, the cost advantage over other systems is clearly apparent for long raises and for shafts.

The disadvantages of a continuous boring system include a general requirement that rock conditions be similar along the length of the raise bore. Another disadvantage is that drilling in very hard rock is slow, and costs rapidly increase. Also, access to upper and lower levels is necessary. A raise-bored rock mass is preferably dry. The initial cost of the drilling rig is also high.

A large variety of machines are available that are designed for certain sets of rock conditions and nominal sizes and lengths of the finished hole. Typical characteristics of the machine specify the pilot-hole diameter, drill-pipe diameter, reaming torque, type of drive (AC, DC, or hydraulic), its rpm and power (hp or kW), and type of gear reducer. Additional specifications are the pilot-hole thrust and feed rates, reaming pull, size of drill pipe, base plate and derrick dimensions with characteristics (especially important for underground application), the type of transporter/erector, if applicable, and the water and air consumption.

Raise-boring machines operate on the principle of first drilling a small pilot hole and then reaming the hole, in one or more stages, to the desired size. The modes of operation are shown in Figure 12.4-12. The most frequently used is a conventional scheme with the pilot hole down and reaming upward. However, in some mining systems, box-hole drilling upward in one stage is chosen.

The raise-boring machine consists of several parts. The rig is composed of a rigid plate and a structure on which are assembled drilling, hydraulic, and electrical equipment. The rig has its own crawler driven by compressed air, or it is mounted on rail wheels for haulage. The rig can also be disassembled and transported. A pilot hole is drilled through the stem with stabilizers and a conventional drilling bit.

Stabilizers are used to ensure directional accuracy of the pilot hole. Several types are in common use. They are of a larger diameter than drill rods, with strips of tungsten carbide welded vertically or spirally on the outside, bringing the outer dimension close to the hole diameter. These stabilizers are used for reaming up. Other types, for pilot holes and reaming down, have more clearance, allowing cuttings to pass.

After the pilot hole penetrates to the target opening, the pilot bit and stabilizer are removed. The stabilizer is mounted on the drill stem and the reamer is put in the place of the small pilot-hole bit. Starting the hole (collaring) is a critical operation, and damage to the cutters or bit body can occur because of the uneven surface. The tension on the stem has to be adjusted to allow starting the hole without highly variable, dynamic loads acting on the drilling components. The stem is the longest at this stage of drilling and is, therefore, prone to maximum elongation and twist. The design of the reamer depends on the final size of the hole and the number of reaming stages.

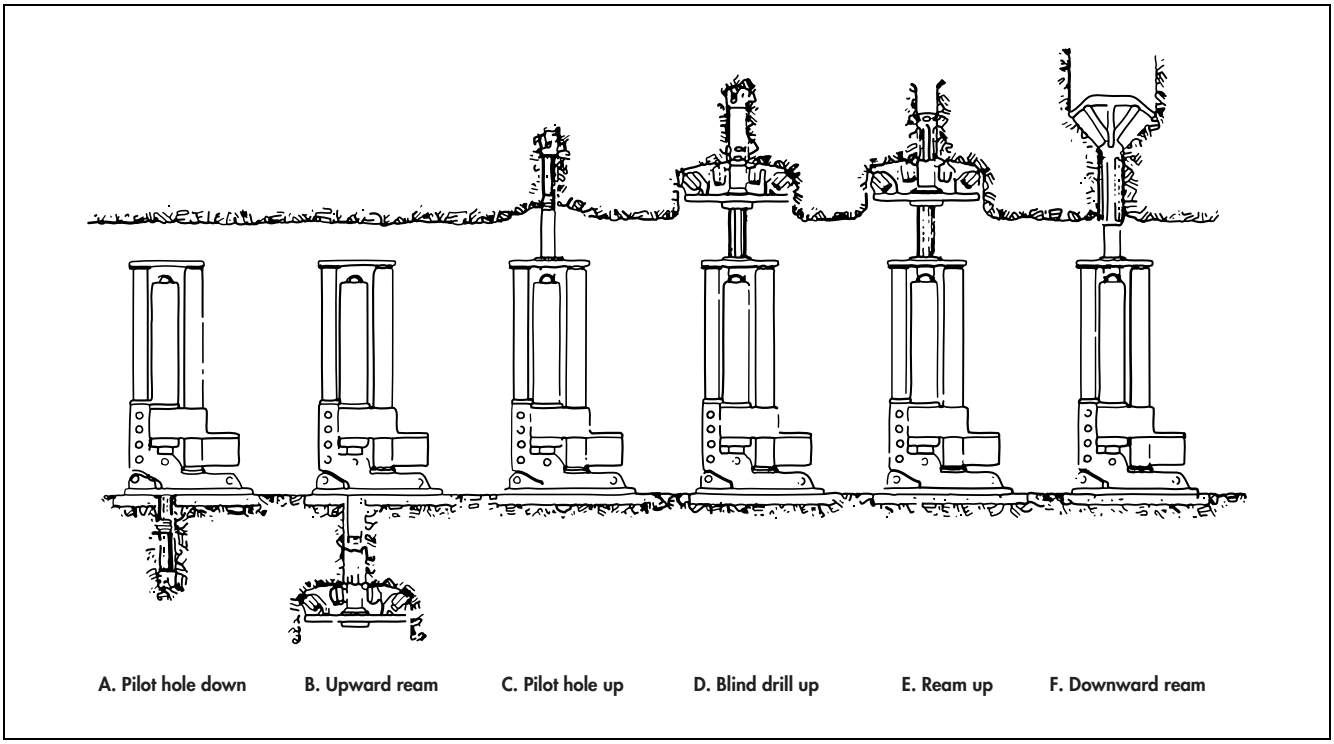
In the past, the primary body of the reamer was often supplemented with stages mounted below, resulting in a "Christmas tree"-shaped tool, as illustrated in Figure 12.4-13. Because of large torque requirements, this system is presently used only for shaft-sized holes, together with large nonrotating stabilizers. In another configuration, extensions are attached to the primary body. By doing this, an increase of the final diameter is achieved by adding more cutters in the same planes as the cutters on the primary body itself.

The accepted measure of drilling performance is the rate of penetration, which depends primarily on bit design. The cutting action has to match the rock properties to optimize the drilling rate. During the reaming operation, in addition to the cutters, the design of the reaming head also influences the raise-boring performance. If no other source of information is available, an analysis of the wear of cutters can help in the design of an improved cutting structure. Independent variables when drilling are rock properties, which are usually characterized by compressive, tensile (indirect), and shear-strength measurements.

One of the most important technological aspects of boring is the removal of cuttings. Standard drilling technology with compressed air, air and foam, or drilling fluid is used for cuttings removal during pilot holing. During the reaming operation, cuttings simply fall down the hole.

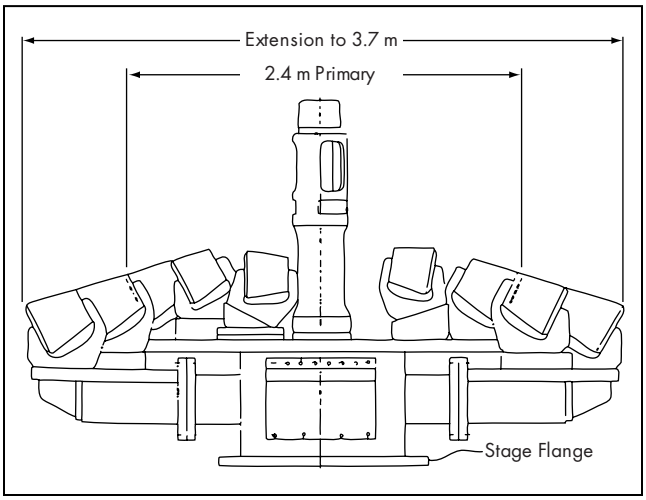
The access point usually determines whether to drill raises from the top or bottom. If only one end of the raise has adequate room, the machine must be put there. The extent to which raise boring will interfere with production must also be evaluated. Locating the borer where fewer disturbances will be caused is preferred. Cost is also of major concern. Generally, drilling down and reaming up is cheaper than other configurations. Rock quality and its predicted response to drilling can influence the choice of direction as well.

In strong rocks without fractures, usually no support or lining is needed. If support is required, shotcrete, rock bolts (with or without wire mesh), or steel lining can be applied. Shotcrete can be placed remotely or by workers from a cage.



Source: Unrug 1992.

Figure 12.4-12 Typical modes of operation for raise boring



Source: Unrug 1992.

Figure 12.4-13 Raise head with extensions

Rock bolts must be installed manually. Steel lining, required in unstable formations, is installed remotely, directly behind the reamer.

Table 12.4-4 compares the factors that influence the selection of vertical/subvertical construction methods used in mining.

DECLINES

A decline is an inclined tunnel from a higher level to a lower level, usually but not necessarily from the surface. The purpose of declines is to access ore bodies or coal seams and provide use at a later stage as haulage routes. In coal mining, declines are known as drifts. These either provide access from the surface or from seam to seam; in the latter case they are known as cross-measure drifts. Drifts in coal mining operations can be driven at steep grades, up to 1 in 2.5 (40%), as the predominant mode of haulage of coal is by belt conveyor, which can handle such steep grades. Steep grades shorten the length of the drift, saving costs and giving early access to the coal seam(s).

In metalliferous mining, declines also allow access and may be used for truck haulage. Grades of 1 in 7 (14%) are common; however, steeper grades are possible in modern mining as more powerful underground trucks continue to develop. Various cutoff depths for declines can be found in mining literature as shown in Table 12.4-5. Beyond these depths, shaft systems should be employed. Over the years, the economic cutoff depth for declines has increased; for individual mines, the cutoff depth of declines is a purely economic decision and should be treated as such (McCarthy and Livingstone 2009). Medhurst (2001) shows that other factors also need to be considered in the determination of a cutoff depth including advances in technology such as the development of larger underground haul trucks, production rate, proposed mine life, potential for mine life being increased, and ventilation flow. Tables 12.4-6 and 12.4-7 compare and contrast decline and shaft access.

Declines are excavated and supported in a similar manner to horizontal tunnels, but drainage may require more attention. Details of tunneling techniques applied are described in the following sections.

TUNNELING

After the site of the surface connection(s) has been selected, the sites of tunnels or drives are often limited by the position

Table 12.4-4 Comparison of factors influencing the selection of vertical and subvertical construction methods

Factor Influencing Selection	Blind Shaft Borer	Blind Drilling	Raise Drilling	Vertical V-Mole	Conventional Shaft Sinking
Design Considerations					
Safety	Uses underground operators working under cover and behind shields. No operation of large moving equipment required.	Does not require miners to work underground	Only requires underground labor setup and mucking. No labor required in the shaft prior to shaft lining.	System uses underground operators working in a controlled environment. Considered safer than conventional shaft sinking as operators are located remotely from the working face.	Requires equipment operation in confined environment. Considered to be the most dangerous of the five shaft construction methods.
Shaft size	Depth is limited by skip-hoist rope capacity.	Limited by required depth, available equipment, and cost. Drilled shaft diameters range from well size to 20+ ft.	Nominally limited by available machine torque. Short (300-ft) shafts have been raised at 20-ft diameter.	Limited by available equipment from 16 to 23 ft in diameter.	Required to be larger than 10–12 ft for most applications. Upper limit not controlled by method.
Shaft depth	Depth is limited by skip-hoist rope capacity.	Limited by required diameter, available equipment, and cost.	Raise depths up to 3,200 ft have been reported.	—	—
Shaft verticality	Verticality can be controlled within extremely tight tolerances. Equal to or better than conventional.	Deviation can be estimated from geological data. Difficult to maintain an absolutely vertical shaft in subvertical structure. Suggested tolerance for design purposes = 0.25°–0.5°.	Shaft verticality is controlled by pilot borehole. Pilot-hole accuracy controlled by directional survey and careful drilling practice. Design tolerance should be specified based on use requirements.	Shaft verticality is controlled by pilot hole. However, offset reaming can be controlled by the operator providing additional control when following a deviated pilot borehole.	Verticality can be controlled within extremely tight tolerances. The most accurate shaft construction method with regard to verticality.
Ground disturbance	Minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Provides minimal mechanical disturbance of shaft wallrock.	Conventional (drill-and-blast) excavation can result in deep-seated blast damage.
Timing/ schedule	Setup time is 1 month or more. Advance rate should be between 20 and 40 fpd. In general, muck haulage and lining limit advance.	Typically much faster than conventional shaft construction methods.	Reaming range from 15 fph (5-ft dia.), 3 fph (10-ft dia.), to 1 fph (15-ft dia.).	Reported rates between 30 fpd (16-ft dia.) and 60 fpd (23-ft dia.).	Generally restricted to one round per shift (e.g., 1 fph).
Operational Considerations					
Groundwater	Same techniques as conventional sinking.	Method provides superior control of groundwater during excavation.	Groundwater controlled by pretreatment where necessary.	Large projected inflows require pretreatment (e.g., using grouting or freezing).	Large projected inflows require pretreatment (e.g., using grouting or freezing).
Support during excavation	Temporary or final support can be installed a short distance behind advancing face.	Support provided by hydraulic pressure and impermeable polymer skin permitting excavation in very poor ground conditions.	Not possible.	Temporary/final support can be installed a short distance behind advancing face.	Temporary/final support can be installed a short distance behind advancing face.
Final lining	Installed during excavation.	Steel/concrete composite lining typically used in weak ground. May incorporate bitumen layer for groundwater control and be designed for full hydrostatic, bituastatic, or lithostatic loading conditions.	None is usually required. Many rapid lining systems available	Final lining may be installed during excavation.	Final lining may be installed during excavation.
Miscellaneous	—	—	Requires existing underground access.	Requires existing underground access.	—
Other Considerations					
Shaft outfitting	—	If a final steel lining is installed, considerable time can be saved through surface installation of guides, brackets, and pipes. All furnishings can be aligned prior to welding to the downhole liner assembly.	Pilot-hole deviation usually prevents raises from being used for worker or materials winding.	Outfitting in line with excavation and final lining if required.	Outfitting in line with excavation and final lining if required.
Initial costs	—	High	Medium	High	Low
Operating costs	—	Medium	Medium	Medium	High

Source: Breeds and Conway 1992.

Table 12.4-5 Cutoff depths for declines compared to shafts over time

Maximum Decline Depth, m	Comments	Source
350	Average value	Northgate and Barnes (1973)
500	Based on a maximum haul length of 3,500 with a 1 in 7 (14%) gradient	Thomas (1978)
1,000	Average value	Medhurst (2001)
1,400	Stawell gold mine, Victoria, Australia	Atlas Copco (2008)

Table 12.4-6 Shaft/decline impact on mining operations

Operation	Decline Truck Haulage	Shaft Hoisting
Truck haulage	• Direct to surface • Long haulage cycles from depth • Expanding fleet size • Rising insurance for fleet • Increasing fleet maintenance	• Underground haulage to shaft • Additional trucking or conveying at surface • Shorter underground haulage cycles
Ore fragmentation	• Popping of oversize at surface	• Stope firing to suit hoisting systems • Underground crusher and/or grizzlies required to limit ore size at shaft loading station
Decline	• Increasing decline congestion for haulage access • Increasing road maintenance in decline	• No decline haulage between the shaft production level and the surface
Production shaft and underground crusher	• Continued surface crusher operation	• Hoisting system operation and maintenance • Underground crushing station operation and maintenance (if applicable) • Fixed plant insurances required
Workshops	• All truck servicing at surface workshops • Surface refueling for trucks generally	• Routine truck servicing underground • Truck refueling underground • Major truck service/overhaul on the surface
Site crew	• Increasing crew size for drivers, service staff, grader operation, support staff, and management • Expanding fly in/fly out routine	• Crushing and hoisting system maintenance staff required • Overall smaller site crew
Change of shift	• Inefficient shift change with large truck fleet and long haulage cycle times (approximately 1-hour cycle from 800-m depth)	• Organized crew transport for shift change of underground truck and maintenance staff
Energy source and greenhouse effect	• Diesel fuel required for haulage (generally) • Increasing fuel usage with depth (30%–50% higher than fixed plant)	• Hoisting system may be supplied from the grid (if available) • Remote power plant may use diesel fuel or natural gas • Overall lower energy use
Operating costs	• Haulage costs increase steadily as depth increases • Higher cutoff grades required for economic mining	• Reduced total haulage/hoisting costs at depth • Lower grades may be mined economically, extending reserves and mine life • Deferral of costs associated with mine closure
Health and safety	• Increased risk of truck fires in decline	• Safety systems and procedures required to ensure safe operations and maintenance in production shaft

Source: Adapted from Medhurst 2001.

of the ore body in relation to the surface connection(s). If choice is available, the most competent ground should be chosen. As with surface connections, the drive size is determined by duties such as ventilation, access, haulage, and so on. Another prime consideration is the mining method, as this controls the size of mining equipment required.

In modern mining, drives smaller than 2×2 m are not common. A 2×2 m drive will accommodate a small loader and allows easy passage for people. Medium-sized drives have a cross-sectional area in the range 10–25 m^2 and represent the most common size range of mining drives. Such drives can accommodate locomotives, conveyors, and rope haulage systems as well as allowing ventilation flow and access.

From a mining viewpoint, large drives lie in the range 25–40 m^2. This drive size is becoming increasingly popular, especially in mines employing trackless equipment and mines employing decline access. This drive-size range allows for the use of off-highway trucks and other larger items of trackless equipment.

Drives have a limited range of sizes due to the need to maintain a level floor for haulage and access reasons. Common shapes are square, rectangular, and arched. Arched roofs have better strength characteristics than square or rectangular cross sections; they are used for drives where long-term stability is required. Square and rectangular section drives are very popular in mining, especially in areas of competent ground or in production areas where the drives have a limited life span.

Two methods of excavation exist: traditional drill-and-blast and mechanized methods (which are described in subsequent sections). Drill-and-blasting is a cyclical operation, whereas mechanical methods offer a more continuous system of excavation.

Table 12.4-7 Shaft/decline impact on mine infrastructure

Infrastructure	Decline Truck Haulage	Shaft Hoisting
Truck fleet	• Larger truck fleet for decline haulage at depth • Upgraded truck capacity (subject to decline limitation) • Surface haulage fleet (optional)	• Modest truck fleet for underground haulage to shaft • Surface fleet for haulage from shaft to plant (if required)
Mechanized ore handling	• Continued use of surface crushing and conveying facilities	• Introduction of hoisting system, underground crushing, ore bins, and associated plant • Local mine development to integrate with shaft
Mine ventilation	• Expansion of mine ventilation for decline haulage from depth	• Hoisting shaft acts as main airway from surface • Reduced ventilation demand
Decline	• Additional passing bays and/or duplication of the decline to address decline congestion • Upgrade to traffic control system in the decline • Upgrade decline fire protection and associated refuge bays	• No significant impact
Workshop	• Upgrade/expansion of surface workshops to suit larger truck fleet • Increasing inventory of truck spares	• Expansion of underground workshops • Upgrade electrical maintenance facilities
Power supply	• Upgrade for ventilation fans and surface facilities	• Upgrade for demand of winder drive, underground crusher, and associated conveyor drives
Fuel	• Upgrade diesel storage for truck fleet • Expand fleet refueling bays for larger truck fleet	• Upgrade fuel storage for power station demand (if applicable) • Provide underground fuel storage and refueling bays
Water supply	• Upgrade for larger crew and truck fleet, and reticulation for decline fire protection	• Upgrade underground water reticulation for maintenance, sprays, and deluge systems
Second means of egress	• Extend/upgrade emergency egress facilities at depth	• Hoisting shaft provides emergency egress above loading station
Surface facilities	• Upgrade/extend due to larger crew	• Minimal upgrade
Underground services: rising mains, raw water, potable water, power cable(s), fiber-optic cable, leaky feeder, blasting control cable	• Route mechanical and electrical services via the decline and/or boreholes • Establish rising mains in boreholes	• Most services can be routed directly via the hoisting shaft
Mine control	• Mine operations generally controlled from surface	• Main control center usually located at winder house with unstaffed satellite stations underground

Source: Adapted from Medhurst 2001.

Drilling and Blasting

The majority of mining drives are excavated by traditional drill-and-blast; however, mechanical mining is becoming much more popular outside its traditional area of coal and other soft-rock mining. Drilling can be done by hand using air legs. The number of air legs in use will depend on the drive size (e.g., in a 2×2 m drive, only one drilling machine can be accommodated). In larger drives, drilling is mostly undertaken using mechanized drill jumbos with twin or three booms common. The most common form of drill jumbo is electrohydraulic, but other forms of power are used. In dry holes, ANFO (ammonium nitrate and fuel oil) is the most commonly applied explosive, whereas in wet holes, explosive emulsions or slurries are increasingly being used. Figure 12.4-14 shows the basic cycle of operations for a typical drill-and-blast tunnel.

A good example of a typical cycle of operations is provided by Argyle Diamonds (2007). Argyle is developing a new block cave operation to mine diamonds in Australia. The cycle of operations during the mine development prior to establishing caving uses typical drill-and-blast tunneling, with one of the key design elements being that at no time does the work force work under an unsupported roof, a common element in Australian mining. For each heading under development, the cycle takes about 18 hours to complete for a nominal face advance of 4 m/cycle and an average heading advance rate of 32 m/week.

In small drives, mucking may be carried out by hand, shoveling into mine cars. This is really only a viable method when cheap labor is available. In the past, rail-mounted shovel loaders were used extensively. These were almost universally powered by compressed air and operated as a front-end loader, "slinging" the muck over the body of the loader into mine cars located behind. Generally, bucket capacity was limited to 0.5 m^3.

The real revolution in mucking operations came with the development of rubber-tired loaders, which lead to the development of load-haul-dump units (LHDs). The advantage of these units is their mobility and flexibility. Bucket sizes vary from 1 to 10 m^3; the size selected depends on drive dimensions and production rate required. Motive power is either by diesel engine or in some cases electrical power (either battery or trailing cable). LHDs have a limited economic hauling distance range; if haul distances become too large, a more economic intervening haulage system such as trucks needs to be added between the LHD and the transfer point to the main mine haulage system. Numerous other methods of mucking also exist such as front-end loaders and gathering-arm loaders.

Numerous methods of haulage exist. In small drives, hand tramming of mine cars may be used. Other commonly applied methods include the following:

- Locomotive (rail) haulage: Motive power can be provided in numerous ways such as diesel engine, battery locomotive, electric trolley wire systems, or rope haulage.
- LHD with or without truck.
- Shuttle cars with or without trucks/conveyor.
- Belt conveyor.

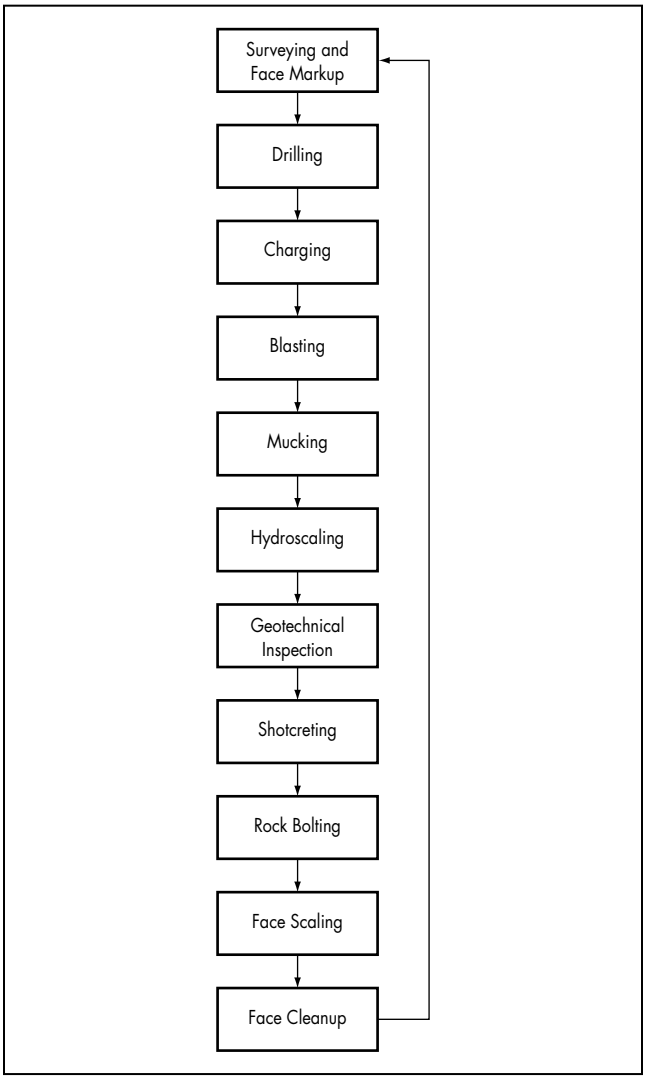


Figure 12.4-14 Basic tasks in drill-and-blast tunneling

Several methods of support are possible; the support required depends on local conditions. The main methods of support include

- Rock bolts,
- Cable bolts,
- Steel sets,
- Timber sets,
- Shotcrete,
- Concrete,
- Combinations of methods,
- Mechanical shields, and
- Natural support.

To design a drive blast, the method outlined by Persson et al. (1994) can be employed. The method can be broadly summarized as follows:

- Determination of the burden and spacing for the cut holes, to provide the relief space required such that the main blast does not freeze, for both angled and parallel hole cuts
- Determination of the burden and spacing for the main blastholes or production holes, also called easers, for the holes that blast downward and sideways into the cut area
- Determination of the burden and spacing for the main blastholes or production holes, also called easers, for the holes that blast upward into the cut area
- Spacing of the trimming or perimeter holes in the backs and sides of the tunnel for both conventional and/or perimeter blasting
- Spacing of the lifter or floor holes

Mechanized Tunneling

Three basic rapid mechanical tunneling systems exist: full-face tunnel boring, mobile miners, and roadheaders or boom-type machines.

Full-Face Tunnel Boring Systems

Full-face boring systems, or tunnel boring machines (TBMs), have been in common use in civil tunneling for many years but are used less frequently in mining projects. Nevertheless, TBMs are viable for mine development. Some good examples of TBM application and other mechanized tunneling techniques are shown in the video *The Burrowers* (Harris 1994).

Constant development and improved tooling have resulted in machines that are capable of advancing large-diameter openings in strong igneous and metamorphic formations at rates that compete favorably, and in many cases exceed, conventional drill-and-blast methods. In addition to high advance rates, TBMs leave a smooth profile, and usually minimal ground support is required. A disadvantage of TBMs is their wide turning circle, although a range of mini-full-face machines are available that have smaller turning radii. The high initial cost of these machines is balanced by low running costs compared to drill-and-blast excavation systems.

Full-face boring machines consist of a rotating cutting head fitted with disk cutters, drag bits, button bits, or various combinations of these. Advanced machines are available on which the tool type can be changed and tool spacing varied. The cutting head may be an open structure with spoke-like cutting arms, or it may completely conceal the face except for muck-removal openings and accessways for tool maintenance. The open-type head gives better access to the face and tools, and can be used with a forepoling arrangement. Cutting forces are provided by the head rotation, whereas normal forces are provided by the thrust of the machine against the tunnel face. Reaction to this thrust is provided by grippers mounted on the TBM body, which in turn react against the tunnel sidewalls. Mucking is performed by buckets mounted on the periphery of the cutterhead, and muck is removed via a central conveyor system.

Preparations for TBM excavation typically involve construction of a portal, placement of a concrete pad on which the TBM is assembled, and installation of support services and equipment as shown in Figure 12.4-15. Careful excavation, including the use of controlled blasting techniques, is usually required to mine the setup area to the tight tolerances required. Placement of a thin concrete layer against the start-up face is recommended to reduce out-of-balance loads.

TBM excavation is a continuous process, with cutting, mucking, and support installation proceeding concurrently. As the cutting head rotates, it moves forward, reacting against the grippers. The grippers are repositioned periodically when they reach the limit of their travel. The grippers are also used to steer the machine in some cases; in others, a floating main



Courtesy of The Robbins Company.
Figure 12.4-15 Tunnel boring machine

beam is used to steer while boring. Mucking in the immediate vicinity of the face is done by buckets located on the head periphery, and a central conveyor system moves the muck through the body of the machine to a bridge conveyor. When the rock is wet or where water inflow is a problem, earth pressure balance systems are used and the muck is removed in a coagulated form in a batch process. The bridge conveyor allows access for track laying and service installation without disrupting the mucking operation. A variety of mucking systems can be used to haul muck to the surface, but usually conveyors or shuttle trains are used. Adequate muck removal rates are critical to optimum face advance rates.

Rock support requirements for hard-rock TBM drives are generally minimal, and estimates have suggested that the savings in rock support costs (compared to drill-and-blast) can offset the cost of the machine in as little as 7 km of tunnel. Hard-rock TBMs are commonly equipped with a partial or slotted shield, and when support is required, conventional rock-support methods are used. Both soft-rock and hard-rock TBMs can be equipped with a full shield and segmental linings installed. This equipment enables hard-rock TBMs to cope with localized occurrences of soft ground.

Mobile Miners

Mobile miners are continuous hard-rock mining machines. A prototype Robbins mobile miner was used in 1984 for the development of a 1,150-m decline at Mt. Isa in Australia. Advance rates of 3.66 m/shift were attained mining a decline 3.66-m-high × 6.4-m-wide section through high-strength quartzite (Boyd 1987).

Roadheaders

Roadheaders have been used in mining and tunneling for a number of years. They are known by a number of names including boom headers, boom-type tunneling machines, and selective tunneling machines. Figure 12.4-16 shows commonly used terminology for the various machine components.

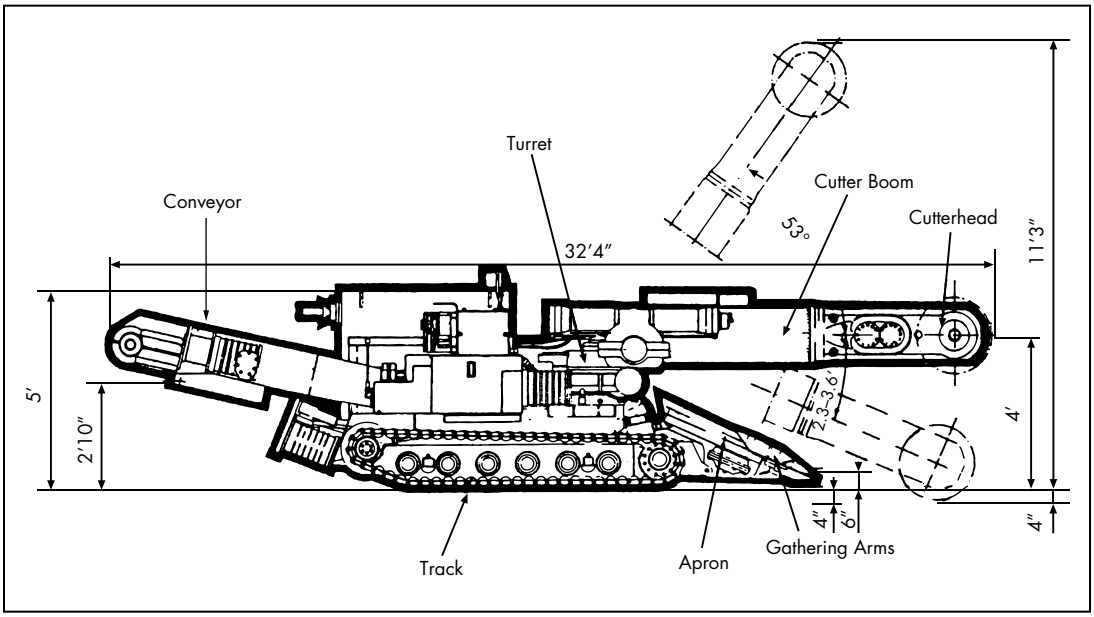
Originally developed as a means of advancing roadways in underground coal mines, early roadheaders were limited to cutting relatively low-strength strata. Development of these machines has greatly extended the range of applications, and they are now used in a wide variety of mining and civil tunneling work. Improvements in cutterhead design and the increasing use of water-jet-assisted cutting will result in a further extension of the range of roadheader applications in the coming years.

Roadheaders offer a number of advantages over TBMs, chiefly related to flexibility. They can cut a variety of cross sections, limited only by the basic dimensions of the machine, and are able to cut tight curves or corners.

These machines also offer advantages over conventional drill-and-blast methods. One of the most important advantages of roadheaders is the avoidance of blast damage to the rock and the consequent savings in ground-support costs. In addition, because mechanical excavation is a continuous process, shift time is more effectively used.

Roadheader cutting assemblies consist of a cutterhead on a movable, hydraulically powered boom, mounted on a rotatable turret attached to a track-driven chassis. In addition to the cutterhead, the machine also incorporates either a gathering arm or chain conveyor mucking system to remove broken rock from the face. The machine may be controlled by an operator seated on the machine or located some distance away, inside a shield, or beneath a supported roof section. To improve machine stability during cutting, many machines are equipped with hydraulically powered steering rams that are used to brace the machine off the excavation sidewall.

Two types of cutterheads are available from several manufacturers and are interchangeable on specific machines. These heads are termed *transverse* (or "ripping") heads and *in-line* (or



Source: Breeds and Conway 1992.

Figure 12.4-16 Main components of a roadheader

"milling") heads. In-line heads rotate coaxially with the boom. These heads are best suited to cutting rock with an unconfined compressive strength of 80 MPa or less. In-line heads require less thrust when sumping, and the head shape allows greater selectivity in cutting specific beds or bands. Transverse heads rotate at right angles to the boom axis. Advances in cutter booms have resulted in machines equipped with telescopic booms, or booms with extended length for cutting high backs or crowns. Telescopic booms are useful for cutting on steep gradients or on weak floor materials where the thrust from the machine's travel system would not be adequate.

Roadheader face-mucking systems generally consist of an apron with either a scraper chain or gathering arm system. Without adequate mucking capacity, the face will become muck bound and excavation will be delayed. In low-strength formations in which high cutting rates are possible, the mucking system may become the limiting factor in controlling advance rates. Roadheader cutting booms are generally mounted on a custom chassis with track propulsion. Track dimensions control ground pressures and should be carefully assessed in conditions in which the invert rocks are weak or prone to slurrying.

In the local area around working roadheaders, high concentrations of airborne dust, generated during both cutting and transport of the muck, and high ambient temperatures commonly occur. In addition to exceeding statutory respirable dust limits, excessive dust may completely obscure the face, resulting in inefficient excavation and increased overbreak. High temperatures and humidity result in labor inefficiency and overheating of electrical motors. The use of water-jet-assisted cutting leads to a reduction in dust levels, but does not entirely eliminate the problem, and may actually increase humidity at the face.

One of the primary advantages of using a roadheader excavation system over drill-and-blast methods is the elimination of blast damage and the consequent savings in rock-support costs. All types of rock support can be adopted for use in conjunction with roadheaders. Because it can be relatively difficult to reverse a roadheader away from the face, the machine must be covered prior to shotcrete application.

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CHAPTER 12.5

Subsurface Mine Development

Richard L. Bullock

INTRODUCTION

To prepare a deeply buried ore body for extraction by underground mining methods, the mine must be developed. This means driving all of the underground openings for entry into the mine as well as all of the openings needed for exploration, exploitation, and services for the mine. Many openings must be driven before the mine can begin production and many more ongoing openings must be progressed as the ore body is extracted. For some block caving mines, this means driving an average of as much as 25 km/yr (14 mpy). This chapter discusses the types of openings that must be developed, the purposes for developing them, what they are called, and how they relate to various mining methods.

A development in a hard-rock mine is any initial opening driven to provide an accessway anywhere in the mine. It may not necessarily be in the ore. Such accessways usually are driven as single openings and provide space for roads, ventilation, power, air and water lines, drainage, storage areas, shops, dump stations, offices, and so on. They also expose the ore so that stope exploitation can begin at the points of exposure. The term *development* implies only that there is an opening for some mine service or where production might begin.

In contrast, in coal mining the term *development* refers to that phase of the mining cycle during which multiple entries are advanced through a virgin seam, forming pillars by breaking crosscuts or breakthroughs between the entries. When the development has reached the extent of the advance, the coal-extraction process removes all or part of the pillar on the retreat, or a longwall panel is set up for the extraction. The only area of obvious similarity between the two meanings is where the development opens more extractable ore or coal.

All underground mining development openings can be classified into four categories:

1. To open production accessways
2. To obtain information
3. To construct service facilities
4. Combinations of the first three categories

For each development, decisions must be made pertaining to each of the five physical variables:

1. Length
2. Direction
3. Inclination
4. Size
5. Method of ground control

Each development must serve the purpose for which it is intended and usually named. Predevelopment studies should include a discounted cash-flow analysis and ensure that all elements of each alternative are included and are indeed comparable.

Spacing and Alignment of Development Excavations

There are other general rules for recommended or preferred location and direction of mine developments. The following is taken from the works of Spearing (1995) and has been distilled from many years working in South African deep mines. While it may be true that all of this information applies to rock under considerable stress, it applies equally to weaker ground under low to moderate stress.

These rules are based on the theory of stress concentrations around underground openings and the interaction of those stress concentrations. The usefulness of these guidelines has been proven by experience obtained underground. Stress interaction between excavations can obviously be controlled by an increase in the installed support, but costs will also increase significantly. If there is adequate available space, it is generally more cost-effective to limit stress interaction between excavations.

Flat and Vertical Development

In flat development, the spacing between two square cross-sectional openings is

- Spaced horizontally at three times the combined width of the excavations; and

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- Spaced vertically at three times the width of the smaller excavation, provided the area of the larger excavation is less than four times the area of the smaller.

The spacing between a rectangular cross section is

- Spaced horizontally at three times the combined maximum dimensions of the excavations; and
- Spaced vertically at three times the maximum dimension of the smaller excavation, provided the height-to-width ratio of either excavation does not exceed either 2:1 or 1:2.

The spacing between two circular cross-sectional openings is

- Spaced horizontally at three times the diameter of the larger excavation; and
- Spaced vertically at three times the diameter of the smaller excavation, provided the area of the excavation is less than four times the area of the smaller.

In vertical development (e.g., shafts and raises), the spacing of

- A square cross section is three times the combined widths of the excavation,
- A rectangular cross section is three times the combined diagonal dimensions of the excavations, and
- A circular cross section is three times the diameter of the larger excavation.

Successive Turnouts (Breakaways)

Multiple turnouts from a given intersection should be avoided to reduce dangerously large roof spans at intersections. The rule is that turnouts should be spaced at six times the width of the excavation between successive tangent points. (This rule does not apply to room-and-pillar stoping. In this case the roof spans are safely designed.) Acute turnouts (less than 45°) should be avoided because these result in pointed "bullnoses," which are unstable. Fracture and failure of a bullnose result in large, unsupported hanging wall spans. The breakaways for inclines are often brought too close to the connecting crosscut. The length of the connection between an incline and the flat drifts should be three times the diagonal dimension of the flat end.

Orientation of Adjacent Excavation Developments

The most highly stressed part of an excavation is the corner. Therefore, positioning a development such as a decline in an unfavorable orientation, too close to the corner of an adjacent opening, should be avoided.

Development Influenced by Geology

In all cases, the geology of the area should be taken into consideration in planning the development. Known weak geological horizons should be avoided even at the expense of longer developments. In permanent excavations (sumps, settlers, hoist chambers, etc.), the position of faults and the orientation of joint sets are critical to the stability of the development. Layouts must therefore cater to such geological features. All excavations should be kept away from high stress dikes where possible. When a dike is traversed, the most direct route should be used. Breakaways should not be sited in dikes, even at the expense of extra development. Haulages (drift opening) should not be positioned in fault zones, and development should not occur alongside a fault. A fault should always be intersected at an angle as near to 90° as possible.

Development Influenced by Stress Condition

When haulages are positioned beneath mined-out areas, consideration should be given to the 45° destressing guideline. An overstoping angle of 45° is generally required to destress a haulage. At an angle greater than 45°, stress concentrations are higher. This rule applies to reefs of 0° to 20°. Dips greater than this often call for computer modeling to show the extent of overstoping needed. Haulages should not be laid out too close to remnant pillars, which are very highly stressed. Another case of this application is where the production level for block caving is developed beneath and well ahead of the extraction level to avoid the high-stress conditions that will come once the extraction level is opened.

OPENING PRODUCTION ACCESSWAYS

Initial Entry into the Underground Mine

Figure 12.5-1 shows various types of entries into an underground mine. The initial entry may be for further exploration, a test mine, production openings, or eventual service openings. The opening may be a drift entry (adit) into the side of a hill, a decline (slope), a vertical shaft, an incline shaft, or a multiple-stage hoisting shaft. The type of opening will depend on many factors: the purpose of the opening; the elevation of the terrain at the surface of the mine in relation to the elevation of the ore body; the type of ore body or shape of the ore body; how material, workers, and ore/waste will be moved in and out of the mine; the depth needed to eventually develop openings; and the mining method that will be used to exploit the ore. Each of the types of initial openings is discussed in the following subsections.

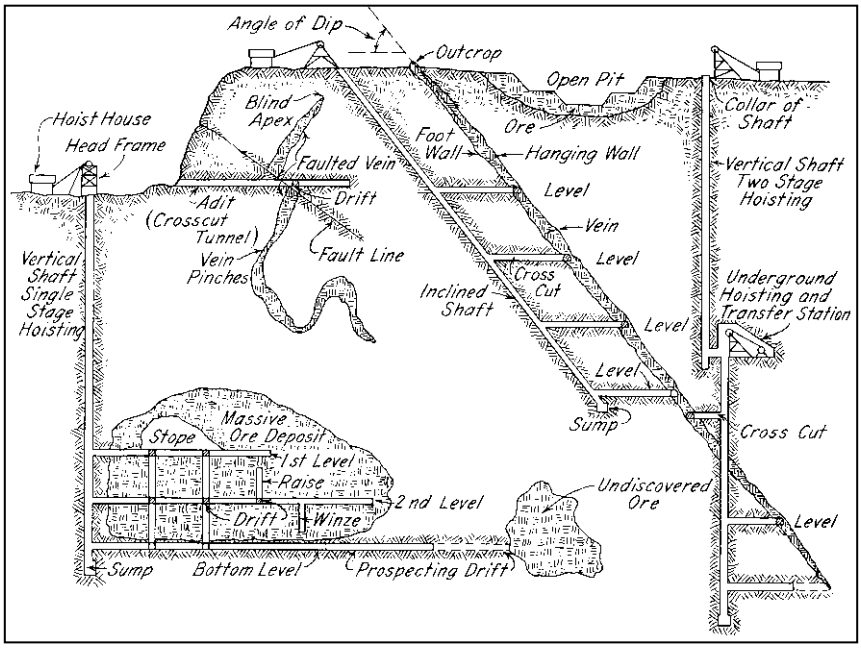
Drift Entry (Adit)

An adit, or drift entry, is the most economical approach when the extractable material is above the floor elevation of a nearby valley. Literally thousands of Appalachian room-and-pillar coal mines have been opened this way to mine coal seams exposed on the hillsides; these are known as "punch mines." Most of the early western-U.S. hard-rock mines used this type of development, as did most of the limestone and dolomite mines throughout the Missouri–Mississippi river basin. The exposed rock could be opened through horizontal adits into the sides of the mountain bluffs.

A few of the advantages of the drift entry are

- Mine drainage by gravity,
- Level or downhill haulage for mined material,
- Easy access for equipment in and out of the mine, and
- Rail haulage can be used if desired.

Adits are usually driven by drill-and-blast methods using trackless jumbos, rail-mounted jumbos, jackleg handheld drills, or with a mechanical excavation machine such as a tunnel boring machine (TBM) or roadheaders. The grade of the adit should be driven at a +2% to 3% grade for good drainage, but probably limited to +1% to 2% if rail haulage is to be used. Almost never should a drift be driven by drill-and-blast methods on a flat grade if the adit goes any significant length, because there is no way to drain it unless the water rises enough to top the roadbed. For nearly flat grades, laser-guided equipment should be used, and if there is a lot of water, it will likely be necessary to overexcavate the bottom, build a flume-type ditch (U-shaped) on the side, and build a roadbed up to the grade that is needed.



Source: Lewis and Clark 1964.

Figure 12.5-1 Underground mine showing names of mine openings

When the mineable material goes to a depth lower than the surrounding terrain, either a slope (decline) or a shaft must be developed. Many mines start out as adit mines and then a shaft is sunk or a decline is driven to exploit the lower portions of the ore body. The ore is then brought to the level of the adit, where it is hauled out of the mine. The same is true for ore lying above the adit. The shaft or ramp will be excavated above the adit and the material will be dropped down to adit level for haulage to the surface. Some refer to these as "upside down" mines. A few examples can be found at the Pine Creek tungsten mine (Bishop, California, United States), the El Soldado copper mine (Chile), and the Stillwater and East Boulder platinum mines (Montana, United States).

Decline (Slope) Entry

Again, there is a difference between what this development is called in hard-rock mining and coal mining: coal mines term such development as *slopes*, and these slopes usually contain a production conveyor belt; hard-rock miners call them *declines*, and unless it is a very high production mine, more than about 4,540 t/d (5,000 tpd), they probably will not contain a conveyor belt. Slopes/declines are usually limited to relatively shallow mines because for a given vertical depth, declines require approximately five to six times the linear distance compared to a shaft. Thus, to reach a depth of only 460 m (1,500 ft), an 18% slope would be approximately 2,540 m (8,333 ft) long.

The declines will normally be driven by using drill jumbos and trackless loaders and trucks. Roof bolting and shotcrete will normally be used for support. It is common practice to drive "muck bays" at regular intervals as the decline is being driven so the blasted round can be moved very quickly from the face to the muck bays, leaving the face available for another drill round sooner. The development muck can then be transferred to the surface while the next round is being drilled and charged. The usual distance between muck bays is about 120–180 m (400–600 ft). Declines can, in theory, be excavated by TBMs but rarely are.

There is an ongoing debate regarding what the proper decline grade for truck haulage should be. However, it depends on the length of the haul, power of the truck's engine, and drivetrain versus the size of the load and the surface condition of the decline. Normally, for good, efficient haulage, the truck should be able to maintain a speed of 12–13 km/h (7–8 mph) hauling a full load up the ramp. As an example, hauling a 48-t (53-ton) load up a straight, well-maintained road surface at 15% grade, a truck with a 485-W (650-hp) engine should be able to maintain 12 km/h (7 mph) for the full length of a 4-km (2.4-mi) decline. Many operators try to drive haulage declines at 18%–20% to save initial cost with a shorter decline, but this usually increases the operating cost by increasing the cycle time of the haul. This can be remedied by employing engines and drivetrains that are much more robust than those used for normal mine haulage. Many suppliers of haulage equipment today can supply the needed drivetrain for very steep grades, even for large-haul trucks.

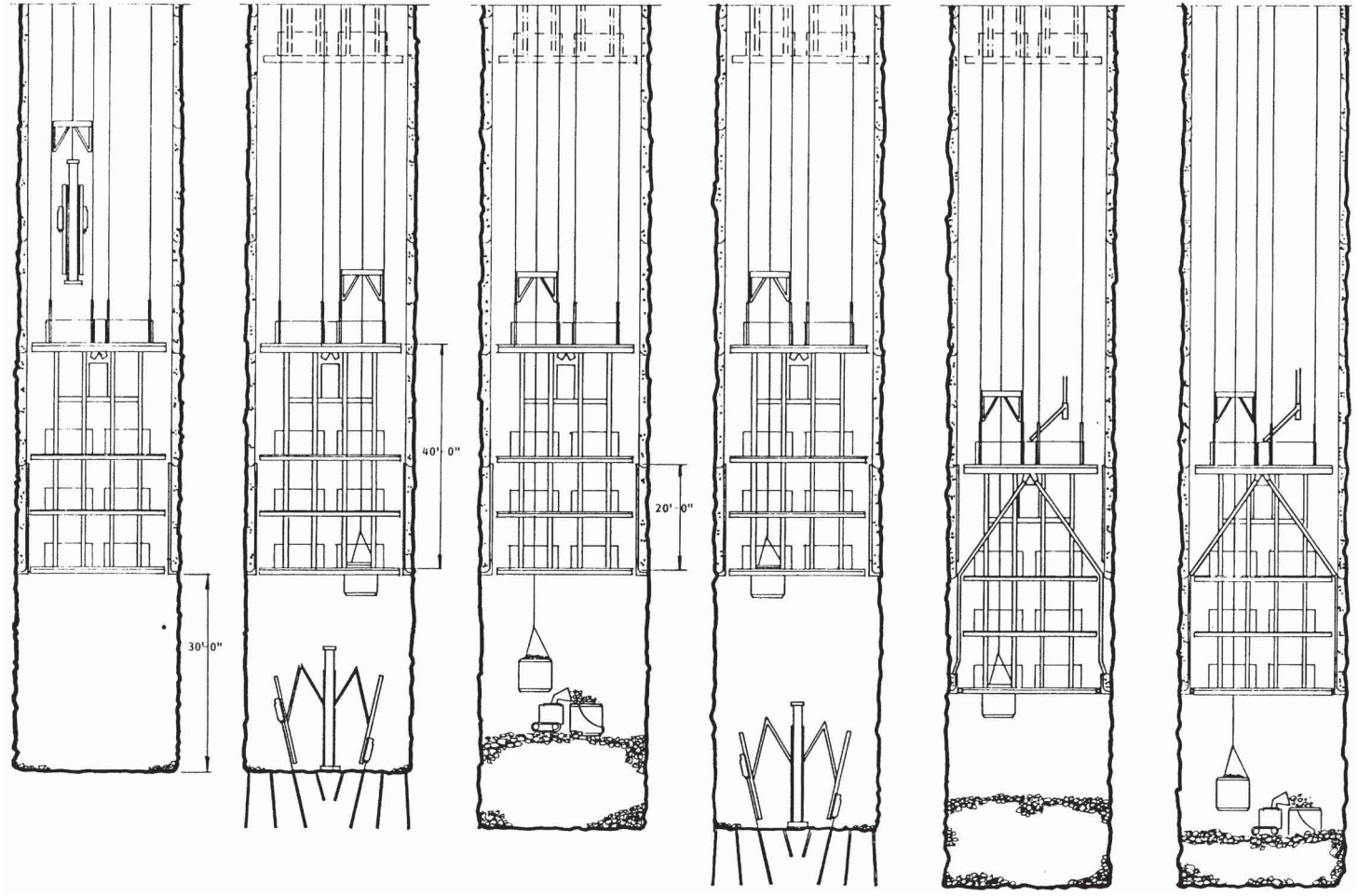
Gradients steeper than 15% are not practical for main production areas due to difficulties maintaining the road surface to an adequate standard. The road surface tends to undulate and creep, particularly when wet. There are also issues with accelerated transmission wear on trucks.

The slope driven for conveyor-belt haulage for most crushed ore and coal can be up to 30°; however, if rubber-tired mobile equipment must also operate on the decline, this may limit the grade to less than 20% for safe operation.

The advantages and disadvantages of declines are discussed later in this chapter (see the "Slopes Versus Shafts" section).

Vertical Shaft Entry

In North America, for most deep hard-rock mines over 450 m (1,500 ft) deep, vertical shafts are still the preferred method of opening the mine for production. The purposes of vertical shafts



① Begin cycle.
Lower drill jumbo.

② Drill out and load round.
Raise stage 200 ft.
Clear shaft and blast.

③ Blow smoke.
Lower stage, EIMCO 630, and buckets.
Muck out.

④ Lower drill jumbo.
Drill out and load round.
Raise stage 200 ft.
Clear shaft and blast.

⑤ Blow smoke.
Lower stage.
Break curb and A-ring.
Install dashpot.
Position curb and A-ring and pour.

⑥ Break forms.
Lower and position forms.
Lower EIMCO 630 and buckets.
Begin mucking, pour through doors, and top off.
Complete mucking. Repeat.

Figure 12.5-2 Cycles of the sinking operation

are distributed as: production, 22%; exploration and test mine development, 18%; service (including ventilation), 27%; and production and service combination, 33% (Dravo Corporation 1974). These proportions have probably not changed since the 1980s for North America. However, in Australia, where declines are used more than shafts for production, the primary use for vertical shafts is ventilation.

Basic design considerations. Like all underground openings, vertical shafts demand considerable study, planning, and design; indeed, this becomes even more important with shafts. The detailed engineering includes seven specific areas that must be considered:

1. **Size:** In the United States, shaft size varies from only 1.2 m (4 ft) to 8.5 m (28 ft), depending primarily on the purpose of the shaft.
2. **Configuration:** While most new shafts that are sunk from the surface today are circular, many older shafts and those sunk from underground are rectangular.
3. **Ground support:** For deepening old shafts, timber supports are still used. However, semimonolithic concrete lining is the most important shaft lining used today. Concrete linings have many advantages:
   - Good bond between the lining and the rock
   - Decreased labor intensity for permanent linings
   - High strength-resisting rock movement
   - In some cases, complete mechanization of lining placement

   The disadvantages of concrete linings are
   - The inability to take load immediately after placement,
   - Lower resistance to some corrosive waters, and
   - Higher sensitivity to mass rock movement and more difficult to repair.
4. **Internal arrangement of services:** Various compartments for hoisting ore and waste—which may or may not include timber or steel conveyance guides, workers, materials, and utility services—all have to be designed. Sometimes the utilities are incased within pipes embedded in the concrete lining.
5. **Depth (length of shaft):** For hoisting ore and waste rock, the shaft needs to be sunk to a depth where there is adequate room below the haulage level for
   - A large storage bin (orepass) between the rock dump and the feeder system that feeds the crusher,
   - The crusher system,
   - Feeding the crushed material into the skip loading system,
   - Accommodating the length of the skips, and
   - Accommodating some sort of shaft-spill rock cleanout equipment.

   The practical depth for shafts sunk for hoisting in one pass is about 1,220–1,500 m (4,000–5,000 ft); beyond that depth, an underground hoisting and ore transfer system should be considered (see Figure 12.5-1). The notable exception to this is what is being done in South Africa. The constraints on the unique deep gold mines in South Africa have led to improvements in the development of deep-shaft hoisting. These followed an industry research and development program undertaken in the last decade. Changes in legislation, winder control, and operating practice has allowed the depth limits of single-lift hoisting to be extended beyond a previous practical limit of about 2,400 m (7,900 ft). Several shafts have been commissioned where the winding depth is beyond 3,000 m (9,850 ft).
6. **Shaft linings:** Concrete lining strength is usually between 19 and 33 MPa (2,800 and 4,200 psi) but may be as high as 50 MPa (7,200 psi). Type III cement is usually used where the concrete is mixed on the surface and can be placed either by dropping batches down the shaft in a container and then distributing to the forms (shutterings), or by using slick lines and pots that distribute the concrete to the forms. In either case, the concrete must be compacted in place with pneumatic vibrators. Usually, the shutterings are collapsible steel forms that can be quickly erected and torn loose. Curb ring forms are placed and poured first, then the shuttering forms are placed on top of them.

   Shotcrete linings can be placed in a few variations depending on the shaft wall stability. The shotcrete may be applied directly on the wall by itself; it can be sprayed over woven wire fabric or the walls can be rock-bolted before or after the shotcrete is placed. Some operators prefer using steel fibers in the wet shotcrete for the added yield strength rather than shotcreting over woven wire fabric. Shotcreting of very small shafts is particularly desirable where steel forms are difficult to work with. Using shotcrete over wire mesh is also fairly common in European coal mines for internal shafts where the permanent lining is only roof bolts, wire mesh, and shotcrete.
7. **Shaft collars:** All hoisting shafts require a massive block of reinforced concrete designed to anchor the headframe and secure the top portion of the shaft in the unconsolidated material through which the shaft collar was excavated. They must be designed to allow all of the mine utilities to pass through the collar into the shaft.

Basic shaft-sinking equipment. Most deep shafts are sunk using a multistage platform that is installed in the shaft and stays in the shaft until it is completed. Some call this platform a Galloway stage. The drill jumbos can pass through the opening in the stage, the concrete forms can be stored and assembled on the stage, and mucking-out the bottom of the shaft can take place with the stage in place while buckets (kibbles) can pass through the opening.

Most shafts use one of the following mucking (lashing) systems:

- Cactus-type air-operated grab
- Pneumatic or hydraulic Cryderman mucker
- Backhoe-type machine
- Riddle clamshell-type machine
- Crawler-type, overshot loader

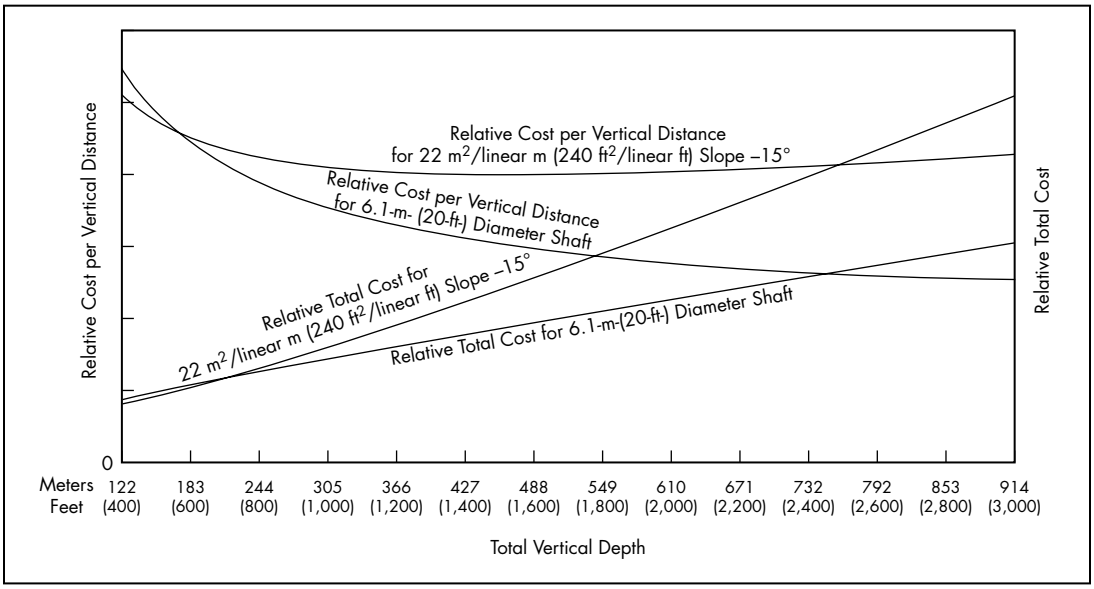
Also, the first portion of any shaft (the "presink") may be excavated with a clamshell or cactus grab bucket operated from a surface crane, before the Galloway stage is placed in the shaft.

The shaft-sinking schedule using a multistage platform is shown in Figure 12.5-2. Shaft sinking is normally performed by specialized contractors.

Another concept is blind boring or upreaming the shafts. Literally hundreds of blind-bored shafts have been completed in the United States and elsewhere around the world.

Slopes Versus Shafts

No general rule governs whether a shaft or a slope is the proper type of entry to be developed; each has advantages



Source: Adapted from Dravo Corporation 1974.

Figure 12.5-3 Relative comparison between shafts and slopes of cost in vertical distance versus total depth

and disadvantages. Every major development decision should include a complete trade-off study combined with an accurate discounted cash-flow analysis that considers net present value (NPV) and internal rate of return (ROR) on capital investment alternatives.

Although the cost per unit distance for a decline may be somewhat lower than for a vertical shaft, the overall cost usually is higher and the development time is usually longer as a result of the greater length required. Thus the cost per vertical meter (foot) for the development, except for very shallow depths (e.g., 150 m [500 ft]), is considerably higher for slopes than shafts. Figure 12.5-3 compares relative costs per vertical unit of distance with total depth for declines and shafts. This chart was taken from work done many years ago on U.S. mines that produced between 1.6 Mt/a and 6.6 Mt/a (1.8 million tpy and 7.3 million tpy), and while the actual cost is no longer accurate, the relative costs are still approximately comparable.

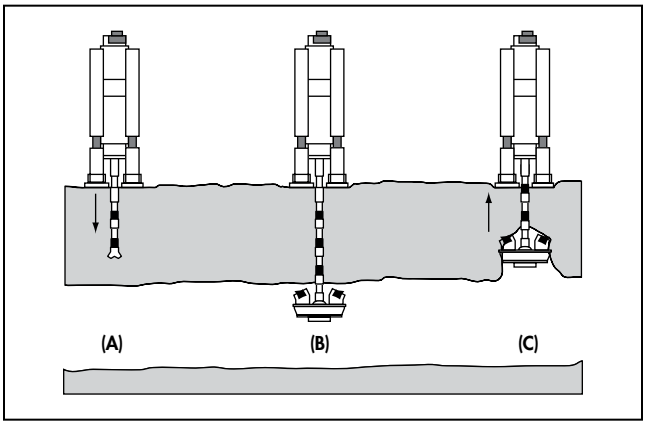
Declines (slopes) have specific advantages over shafts:

- The technique of driving a decline does not require as much specialized technique as sinking a shaft.
- Declines can be used for trackless, track, or conveyor-belt haulage. When used for conveyor-belt haulage, the tonnage can be very large.
- The ground support cost for roof bolts and shotcrete is usually less expensive than the concrete liner normally required in a shaft.
- Large equipment can be moved in and out of the mine without disassembly.
- Maintenance is easier and less costly.
- If the production is from a fairly shallow depth and greater than about 5,000 t/d (5,500 tpd), the decline with a conveyor is likely to be lower cost.
- No visible head gear is required (i.e., no visual pollution).
- No winding gear is required, which in a remote location may need to be transported considerable distances along an unsuitable road network.
- No surface winding facilities need to be constructed, maintained, or manned.
- For major wrecks within the opening (shaft or decline), there are fewer production delays in declines. For an inclined ore body, and if only the top portion is initially developed, declines incur a much-lower initial cost than a shaft that would go to the depth to mine the entire ore body.

The disadvantages of declines over shafts are

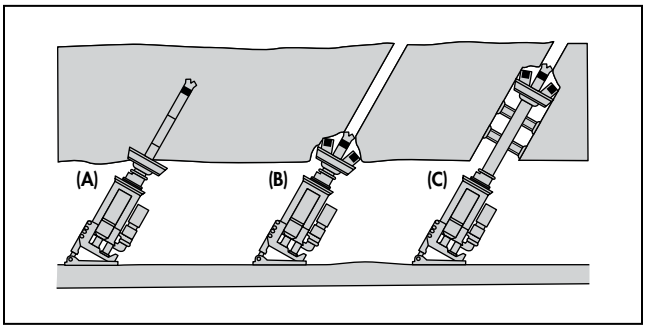
- For an equivalent depth, the slope will be four to six times longer. The total cost and the time to construct for very deep declines will be greater (but for inclined ore bodies, production may be able to start early in the upper ore body).
- If there is a thick, very weak zone of rock to penetrate, the shorter distance of the shaft to penetrate through this zone will probably be much easier and less costly than for a decline where substantial ground support may be required.
- If the development opening is to be used for ventilation, the length of the opening will be increased four to six times, and the wall rock will be rougher than the concrete-lined shaft. Both of these issues will require more power to overcome the increased resistance for the same size development opening.

From depths of about 250 to 350 m (800 to 1,100 ft), mines producing from 3.3 to 6.6 Mt/a (3.6 to 7.3 million tpy) have been developed with declines. At these tonnages, at depths of more than about 350 to 500 m (1,100 to 1,600 ft), a shaft is probably more economical than a decline, except for very high tonnage mines using conveyor belts. However, many factors must be taken into consideration before making this decision. For example, if developing from the bottom of a mined-out open pit at a depth of 300 to 400 m (980 to 1,300 ft), the decline down another 800 m (2,600 ft) would probably make more sense than a shaft from the surface down to 1,200 m (3,900 ft). Likewise, if there is only proven ore down to a depth of 400 to 500 m (1,300 to 1,650 ft), and the annual tonnage was to be between 1 and 2 Mt/a (1.1 and



Source: Bullock 1997, © The McGraw-Hill Companies, Inc.

Figure 12.5-4 Raise boring by upreaming. (A) The machine drills a pilot hole. (B) The drill bit is replaced by a reamer. (C) The raise drill lifts the reamer, and tailings fall down the shaft and are removed.



Source: Bullock 1997, © The McGraw-Hill Companies, Inc.

Figure 12.5-5 Raise boring using a blind "box hole" drill. (A) The machine drills a pilot hole. (B) The drill bit is replaced by a reamer. (C) The reamer bores into rock, and tailings fall down the shaft and are deflected by the drill.

2.2 million tpy), then in all probability the decline would have the advantage. However, there is a strong caveat even to this general statement: if the surface material is very much unconsolidated and/or is below the water table, the cost of developing through this material is much higher for a decline rather than a shaft. For example, if the ground must be frozen in order to penetrate safely, a shaft would be the obvious choice. This is one of the reasons that shafts in North America are sometime favored over declines, particularly in the plains area where large regional aquifers occur.

Incline Shaft Development

An incline shaft is an opening driven steeper than a decline but not vertical. They were often used in the past for maintaining a somewhat constant haul distance between the dipping ore body and the shaft. However, with today's trackless haulage equipment this advantage is minimized. Hoisting with incline shafts is less efficient and requires more maintenance than vertical shafts. Thus they are used very little today.

Multiple-Stage Hoisting

When the hoisting depth reaches approximately 2,400 m (8,000 ft) in depth, a second hoist may be placed underground and continued with multiple-stage hoisting. Usually these facilities are not placed one beneath the other as shown in Figure 12.5-1, but rather the lower shaft is placed wherever it makes more engineering sense to optimize both hauling to the shaft and hauling from the lower shaft to the loading facility of the upper shaft. Also, ground conditions could play a role in where to locate the lower shaft. Many times with today's powerful diesel or electric trucks, the choice may be to haul up an incline rather than sink a second deeper shaft.

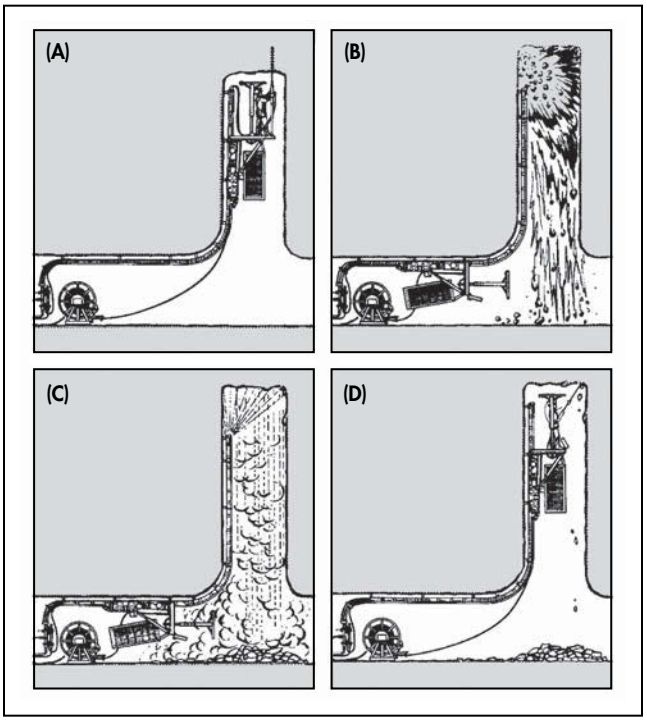
Types of Raise Developments

Raises are needed for many reasons, mainly for the transfer of ore, workers, and materials. They also provide for a void space for stope blasting, ventilation, escapeways, and combinations of these. There are many ways to excavate a raise:

- **Conventional raise drilling and blasting:** Here the miner enters a bald raise and drills and blasts the drill round. A work floor is created by drilling pin holes in the sides of the raise.
- **Raise boring by upreaming:** This system requires two levels, between which the raise is bored (Figure 12.5-4). First, a pilot hole is drilled from the top down, then a large reaming bit is put on the drill string and the hole is reamed upward.
- **Blind-hole raise drilling with a "box hole" drill:** In this case there is only one level, and the blind-hole boring machine usually drills a small pilot hole and then reams out the blind raise (Figure 12.5-5). This is sometimes called a "box hole."
- **Drop raise by predrilling and blasting the raise in stages from the bottom up:** The drillers use long-hole drill machines and very accurately drill small blastholes and larger, parallel-cut open holes. The holes are charged and blasted in sequence from the bottom up, making sure that each sequence blasts clear before the next round is blasted.
- **Raise climber:** The raise climber is a mechanical machine that climbs on a cog-wheel track installed in the raise (Figure 12.5-6). The miner drills out the raise round, charges the round, then moves the raise climber back out of the raise before he shoots the round. The miner then runs the machine back up the raise, scales the loose rock under the protection of a canopy, installs another section of track, and repeats the cycle.
- **Boliden raise cage:** As raise-boring technology has become increasingly efficient, the Boliden raise cage is not used as much as in the past. Still, it is a viable method. Where there are two levels, a pilot hole is first drilled between them, and a small hoist is set at the top of the hole. A hoist cable is dropped down to the lower level and a two-deck drill cage is hoisted to the top of the raise. The upper deck has a canopy, to protect the miners while scaling and drilling. After drilling and charging, the cage is lowered and moved from the bottom of the raise for blasting. The cycle is then repeated.

Other Types of Mine Developments

Many other types of developments shown on Figure 12.5-1 need some explanation. The terms *drift* and *crosscut* have unique meanings in a vein type of ore body. Drift usually means the opening is driven parallel to the strike of the vein, whereas a crosscut typically is an opening driven perpendicular to the



Source: Svensson 1982.

Figure 12.5-6 Raise boring using a raise climber. (A) The raise climber track is installed; the climber goes up and drills and charges a raise round. (B) The raise climber travels back down the track and the raise round is blasted. (C) The raise is ventilated. (D) The raise climber goes back up the raise with the miner under the protection of the steel cage. The miner, from the top of the raise climber, scales the loose material in the raise and extends the new track up the raise. Drilling can again commence.

strike of the ore body. However, crosscuts can also mean any opening driven between two other openings.

The term *raise* is commonly used for an opening driven at a high angle upward from the existing opening, and a *winze* is aan opening driven at a high angle downward from an existing opening. However, with today's technology, it is very common to drill and blast a drop raise from the opening in the downward direction and then blast it from the bottom up; this would still be called a raise.

Ramps are similar to declines in that they are driven at a steep grade that can still be navigated by trackless equipment. However, they are usually driven between mine levels, more for the purpose of traveling between production stopes and dumping points or to develop a new stoping entry. Ramps are better developed in an oval or figure-eight shape than in a true circular configuration. Ramps are more efficient when they are limited to a 15% –18% grade.

Stope Developments for the Various Mining Methods

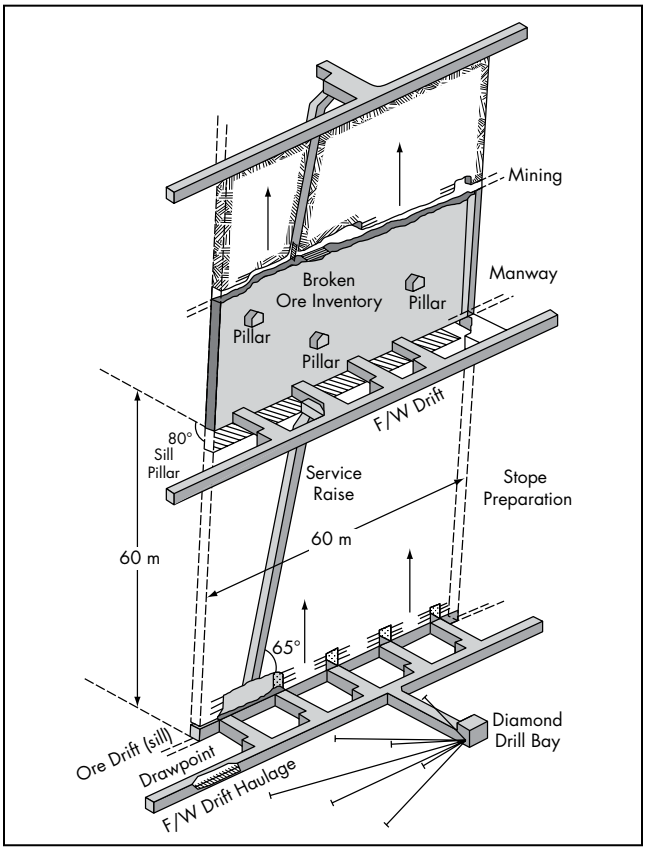
For those who are not familiar with the various underground mining methods, the following discussion should be studied along with the chapters on mining methods in Part 13 of this handbook. It is not the intent of this chapter to explain or describe how the mining methods work, but rather to describe the differences in the developments that set up the various mining methods.

Hard-Rock Room-and-Pillar Mine Development

The configuration of a room-and-pillar mine generally follows the shape of the mineralized ore body to be mined. Therefore, the basic shape of the mine should start by determining the path of the main haulageways (entries), branching from these to reach the lateral extent of the reserve. Wherever possible, a few basic principles should be followed in planning the production developments:

- The central gathering point for hoisting should be as near to the center of the extractable mass as practical. That keeps the haulage, personnel, and material movement distances to a minimum. Although this may mean a large shaft pillar is left in the center of the mineral reserve of a flat-lying deposit, it does make the mining operation more efficient. At the end of the mining project, most of the pillars can probably be extracted, providing that proper mine planning has been used in the design and proper pillar removal systems have been established.
- Normally, room-and-pillar mines without ground stability problems should be developed as symmetrically as possible to allow the maximum number of working areas to be opened with a minimum of development time. In trackless mines, it is advisable to cut the main haulage opening as much as 50% larger than the minimum size needed to operate the existing equipment. Large main haulageways improve productivity, ventilation, and road conditions, and they are much more efficient to drive. They also allow productivity to be maintained by permitting efficient operation of larger equipment. The haulage drifts in several Mississippi Valley–type room-and-pillar mines are driven 5.5 × 9.8 m (18 × 32 ft), and the mines use very large equipment such as 36-t (40-st) or larger trucks.
- Haulage roads and railroads should be kept as straight as possible, with steep grades other than conveyor slopes being kept as short as possible. The grade limits depend on the type of haulage equipment being used. For main haulageways, railroad grades should be less than 2%, and for rubber-tired diesel equipment it is advisable to keep the grade less than 8%. However, for very short distances where the equipment is allowed to gain mass momentum, 15%–18% grades can be negotiated quite efficiently. If a 20% decline is to be driven for a conveyor system or service slope, rubber-tired, diesel-powered loaders are the best equipment for loading and hauling the material from the development. As mentioned previously, for such steep declines, very high horsepower engines are required for efficient haulage.
- The radius of a curve that can be negotiated by trackless equipment is somewhat dependent on the size of the equipment in relation to the size of the drift and the turning radius of the equipment. Normally, a radius of 15 m (50 ft) is handled easily by any trackless mining equipment. A road grader usually requires the greatest turning radius. For rail-mounted haulage, a radius of 24 m (80 ft) is adequate for stope development, but a limited-service main line requires a minimum radius of 38 m (125 ft). However, a radius of 60 to 150 m (200 to 500 ft) would be appropriate for main-line haulage, depending on the speed of the train.

The preceding discussion has outlined the need for operating flexibility in relation to the size, shape, and thickness of the mineral reserve in the room-and-pillar mine plan. For



Source: Adapted from Marchand et al. 2001.

Figure 12.5-7 Typical shrinkage stope mine development

example, it is often advantageous to incorporate a lower level of uniform grade below the main ore horizon to create a ventilation circuit, providing mine drainage that removes water from the operating headings and roadways to make ore movement more efficient. The Fletcher, Brushy Creek, Buick, and Sweetwater mines—all located in the United States—of Doe Run Company incorporated multiple-level planning in the early years of operations, with all of the planning features described. These mines have been extremely efficient over their 45 years of operation.

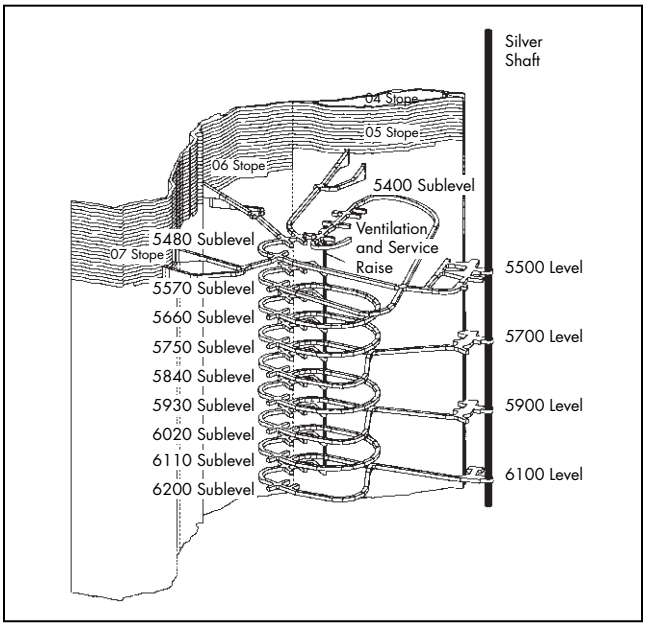
Shrinkage Stoping Development for Production

Shrinkage stopes for extraction can be developed in many ways. Several years ago, William Lyman (1998) wrote on the subject, describing thirteen ways to develop the shrinkage stope and variations of the ventilation and service facilities. In Chapter 13.3 Haptonstall describes several of these methods. What is described here is a very common method that can be adapted to developing a trackless or a rail haulage shrinkage stope as depicted in Figure 12.5-7 (Marchand et al. 2001). The mine described here is Cambior's Mouska mine in Quebec, Canada.

An ore drift (sill) is driven and kept in the ore at about 30 m (100 ft) ahead of a footwall haulage drift. It is important that the sill drift width stays as near the same width of the ore vein as possible. The sill drift and the haulage drift are kept about 10 m (30 ft) apart. In this case the vertical distance between levels is 60 m (200 ft) with a stope width of 60 m (200 ft). At 10-m (30-ft) intervals, crosscuts are driven between the haulage drift and the ore drift to serve as drawpoints to shrink the broken ore from the stopes. (*Shrink* refers to the process of shrinkage stoping where the amount of broken ore out of the stope is decreased; most of it is left temporarily as a working platform and ground support.) The crosscut size will depend on what type of equipment is used to remove the ore. In this case small overshot pneumatic muckers were used and trammed back to the railcars for dumping, and the crosscuts were only 2.7 × 2.9 m (8.9 × 9.5 ft). For load-haul-dump (LHD) haulage from a drawpoint to an orepass or dump pocket, the crosscuts would have to be sized at least 1.2 m (4 ft) larger than the dimensions of the LHD. From the lower sill drift, a service raise is driven to the haulage drift of the level above. This service raise is equipped with a ladderway, service pipes, and a slide (with air tugger) all connecting to the level above. The upper level also provides the ventilation circuit. Shown also is a service manway from the bottom of the stope, which is, in this case, timbered as the stope advances upward, thus completing the ventilation circuit and maintaining two access points to the stope at all times. Other mines sometimes leave pillars between the stopes and place the second access in these pillars, which would require small doghole drifts between the raises and the stope. (A *doghole* is a very small drift driven to serve as an access between a vertical manway and the stope. In some cases it may be only 1.2-m (4-ft) high, where one must crawl on all fours to access the stope.) These pillars can later be removed and capture the ore in the drawpoint below when the stope is completed.

Sublevel Open-Stope Development

Most sublevel open-stope mines are applicable to dipping-type deposits, but the thickness of the deposit between the hanging wall and footwall normally varies from a few meters to tens of meters wide. The development access from the main decline or hoisting shaft is gained by level developments, usually in the footwall, that vary from 46 to 122 m (150 to 400 ft) depending on the vertical extent of the ore body, the eventual stope heights, and the number of production stopes that may be excavated per level. These levels usually become the haulage drifts and approach the bottom of each stoping block running parallel to the block with connecting access crosscuts running over and into the ore. Between the main levels, ramps are usually driven for trackless haulage transport between levels. These ramps also give access to the sublevels, which are developed at intervals to remove blocks of ore. The sublevel drifts are located in the footwall for haulage, with crosscuts into the ore for connecting to the drifts in the ore, which first become undercut access drifts and later draw drives. These sublevel drifts in the footwall can also be used for accurate delineation of the vein (reef) by drilling prospect holes from the sublevel drifts. Up above at the next sublevel, these drifts driven in ore are used for drilling the long holes that blast the ore. For wide ore bodies, there will probably be a hanging wall drift and a footwall drift. Usually at the widest part of the ore for that block there will be a slot crosscut drive and a slot raise drive between the sublevels, which opens to the undercut of the lower level. This slot raise is then slabbed (slashed) out to the full width of the ore body, and when the broken slot ore falls to the undercut opening, the slot allows room for the vertical slabs of ore to be drilled and blasted from the upper sublevel drill drifts. The ore then falls down to the draw drifts, which are accessed by drawpoint crosscuts for loading out the ore. When the ore is completely drilled and blasted from the first sublevel block,



Source: Peppin et al. 2001.

Figure 12.5-8 Lucky Friday undercut-and-fill mine development infrastructure

the drilling and blasting moves up to the next sublevel and the cycle is repeated until the next main level is reached. An ore sill pillar is usually remaining at each main level.

Sometimes the ore body is narrow enough that a horizontal room can be opened and, instead of drilling fan holes from drill drives, vertical holes can be drilled for the long-hole blasting. This also applies when vertical crater retreat stoping is used. In the sublevel extraction method, termed "longitudinal mining" in Chapter 13.4 (Figure 13.4-21), the sublevel sill drifts serve as both the top cut for the downhole drilling and as a bottom cut for remote mucking. Ventilation raises can be placed in the ore or close to the stope accessway from the ramp. Raises are usually opened with raise boring machines, raise climbers, or drop raises. Rarely in today's modern mechanized mines are raises drilled and blasted by conventional methods.

Cut-and-Fill Stoping

Because cut-and-fill mining is so versatile and can be used to mine any shape of an ore body, it is impossible to generalize a specific development approach that fits all types of cut-and-fill systems. Normally the essential criteria for using cut-and-fill stoping is that the excavated stope needs support; the remaining rock is not competent enough to stand open to allow further mining of the area. However, where the ore is so high-grade that it is planned to extract nearly 100% of the ore and leave no pillars, there is a need for immediate backfill, even in competent rock. Consideration must be given to the general types of deposits that are to be mined by one of the many forms of cut-and-fill mining.

Steeply dipping, narrow ore bodies. Steeply dipping ore bodies are most commonly mined by cut-and-fill systems. Such mining occurs throughout the Silver Valley (in Idaho's Coeur d'Alene district), the Stillwater mines and the Cannon mine in Washington (all in the United States), to name but a few. The major infrastructure developments are normally developed in the footwall, but not necessarily, since nothing will be allowed to cave—even the hanging wall should remain structurally competent for developments. A classic example of infrastructure development is seen in Figure 12.5-8 of Idaho's Lucky Friday mine, illustrating the main shaft, the haulage levels coming off the shaft, the oval-shaped ramp contacting all of the sublevels, and the service and ventilation raise. Because this is an undercut-and-fill mine, there are 15% stope attack ramps to the top of each sublevel stope block and working down, slice by slice, to the bottom of the sublevel stope block. In an overhand cut-and-fill mine, the sublevel attack ramps would start at the bottom of the stope block and work up the ore block in slices, establishing a new attack ramp for each slice. Not shown on this sketch are orepasses from the sublevel ramp down to the next haulage level.

Flat, thick, and wide ore bodies. A few years ago only steep vein deposits were worked by cut-and-fill mining, but with today's technology wide and thick ore bodies are also mined in this manner. Almost all of the underground mines of the Nevada (United States) gold district use cut-and-fill mining, and more specifically, undercut, drift-and-fill, or ramp-and-fill. In this type of mining, the upper-level drifts should be driven and filled at a different direction azimuth than the drifts immediately below. This can be seen in Chapter 6.5, Figure 6.5-6.

Sublevel Caving

Sublevel caving is an underhand method where all of the blastholes are drilled upward. Gravity moves the ore down to the extraction and drilling drift. The main haulage drifts on the major levels will probably be longer from the hoisting facility to the production stoping than in noncaving mining systems, since this mining system allows for the overburden to cave. When the angle of the caving surface is taken into consideration, the hoisting facilities must be well removed from the zone of potential caving.

For transverse sublevel development, the haulage drift for a particular level is driven down the strike of the ore body, probably in the footwall waste, with production crosscuts turned off at regular intervals crossing the ore body. This divides the ore body into geometric blocks that will correspond to the geometric spacing of the sublevels. These drifts are driven to the hanging wall, and a slot raise up to the cave above is developed at the end. These slot raises must be slabbed out to the full dimension of block and will provide the opening for the initial blasting at the end of the stope. There are many development schemes to connect the sublevels, but all of them involve ramps that connect to the various sublevel haulage drifts. Also connecting to the sublevel haulage drifts are major orepasses that transfer the ore to a main haulage level.

Development for a sublevel cave stoping method is extensive. This is because so much of the ore must be mined by development to set up the drilling and blasting of the ore so that the surrounding waste rock will cave uniformly. In 1990, about 25% of the ore was extracted by drifting in the sublevels; now this can be as little as 6% (Bullock and Hustrulid 2001). The reason for the drastic decrease in development is because the better drills and drilling systems today are more efficient, drilling larger, longer, and much more accurate holes for blasting; thus the spacing between sublevels has more than doubled.

As indicated, ore is recovered both through drifting and through stoping. Because the cost per ton for drifting is several times that for stoping, it is desirable to maximize stoping and minimize drifting. This has meant that through the years, the

height of the sublevels has steadily increased until today they are up to 30 m (98 ft). The sublevel intervals have changed from 9 m up to nearly 30 m (30 to 98 ft). A number of factors determine the design. The sublevel drifts typically have dimensions (width × height) of 5 × 4 m, 6 × 5 m, or 7 × 5 m (16 × 13 ft, 20 × 16 ft, or 23 × 16 ft) to accommodate LHDs.

After the sublevel vertical interval has been decided, it is necessary to position horizontal dimensions of the sublevel drifts. As an example, the drifts might be placed so that the angle drawn from the upper corner of the extraction drift to the bottom center of drifts on the overlying sublevel is 70°. This is approximately the minimum angle at which the material in the ring would move to the drawpoint. The resulting center-to-center spacing is 22 m (72 ft).

Longitudinal sublevel development is commonly used in ore widths of 18 m (60 ft) or less but may also be used to advantage in wider ore bodies. For longitudinal sublevel caving developments, almost all of the stope development is in ore, but for irregular ore bodies, there is a greater chance of leaving ore undrilled and blasted.

Mine Development for Block Caving (Panel Caving)

In this section, the term *block caving* will be used to represent both block and panel caving, suggesting the mining of individual blocks, and panel caving, indicating a laterally expanding extraction. The time and cost of developing a block caving mine is very significant compared to other types of mining.

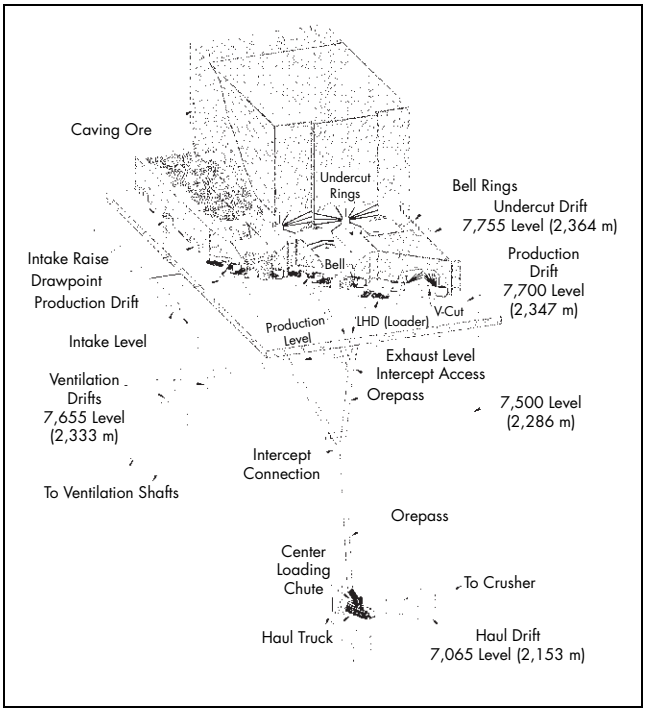
Since block caving methods are associated with extremely high production rates, the main haulage developments are likewise designed for large high-volume/high-speed haulage. These initial developments must also accommodate very large ventilation capacities to handle all of the equipment needs of a modern massive caving mine. At the same time, the major permanent development facilities, including shafts and permanent ventilation raises, must be located far enough from the caving material so as not to be engulfed within a zone of high stress or rock-mass deterioration. An example of this infrastructure development in relation to the caving ore body is shown in Chapter 6.1 (Figure 6.1-2).

The most important development elements of a caving system are the undercut level, which removes the support for the overlying ore column; the funnel, trough, or bell through which the ore is transported downward to the extraction level; and the extraction or production level. In some cases, the undercutting can take place on two levels rather than one (see Chapter 6.5, Figure 6.5-24C). There are three types of systems for drawing the ore from the caving ore body: the LHD system, the slusher system, and the gravity system (in order of importance). Much of the main development infrastructure is similar for the three methods, but the stope developments are somewhat different.

Stope development for LHD caving systems. In Figure 12.5-9 the undercut level is the upper level where the long drill holes of the undercut ring are shown. The production level is the level below where the LHD is loading ore from the bell draw holes. Note the intake and exhaust drifts and raises that carry the ventilation and the orepasses and load-out chutes on the mail haulage level drift.

For more extensive discussion and illustrations on LHD caving systems development, see Chapter 13.10.

Stope development for slusher caving systems. In the top level shown in Figure 12.5-10—the undercut level—longhole drilling will open the troughs that the ore will collapse into and then flow down into the multitude of finger raises, where it flows by gravity to the grizzly (haulage) level where it is slushed to the main haulage level. In some cases, instead of slushing directly into the ore trains, the ore may be slushed into another orepass leading to an even lower main haulage level. The ventilation drifts and raises are not shown in the diagram. This mining system is labor intensive and would only be used where labor costs are extremely low.



Source: Adapted from Rech 2001.

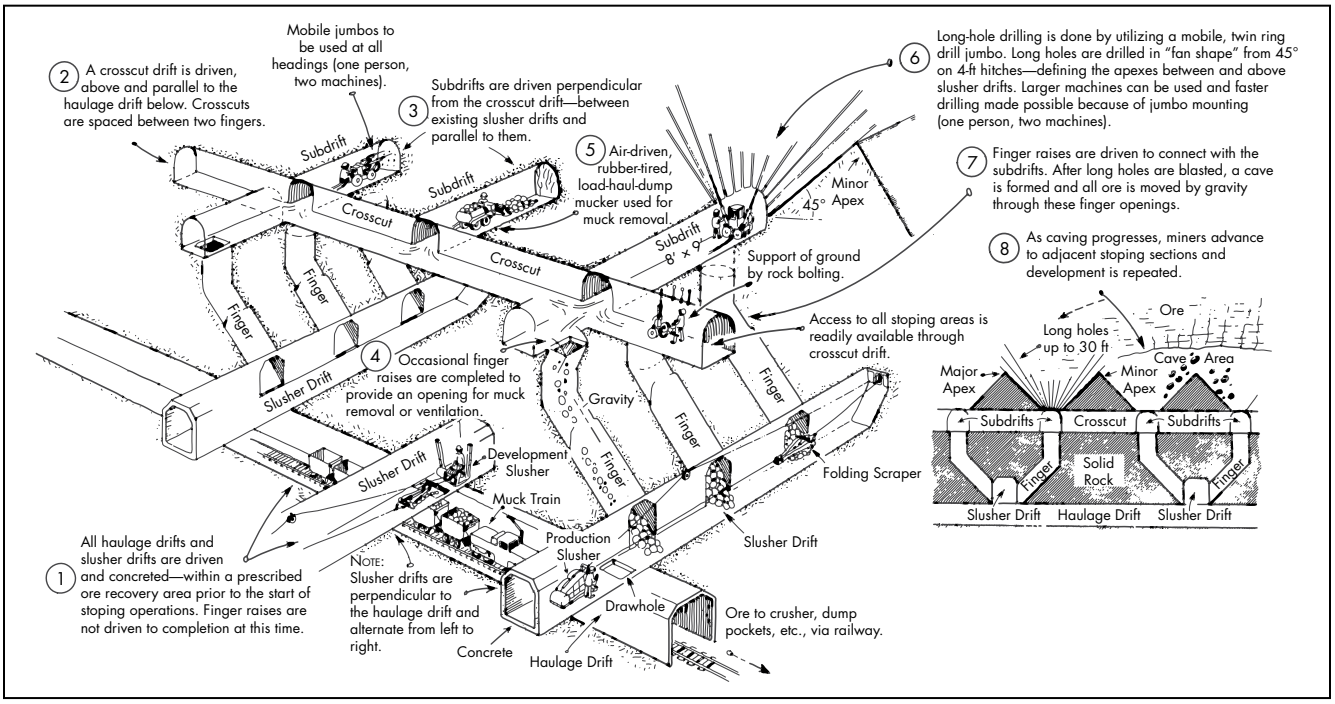
Figure 12.5-9 Underground development needed for LHD block caving

Stope development for gravity caving systems. Figure 12.5-11 illustrates how the development has much less horizontal development, but many more raises for orepasses and draw extraction bells, in a 14 × 7.6 m (76 × 25 ft) grid pattern. When the ore is being drawn from a specific area, a miner must stand and assist the ore flow with a mining bar or explosive bombs in each of those grizzly chambers, as the ore flow becomes blocked with boulders. Because this method of block caving is so labor intensive, it is seldom used in new mines being developed in countries where labor costs are high.

PLANNING FOR EXCAVATION

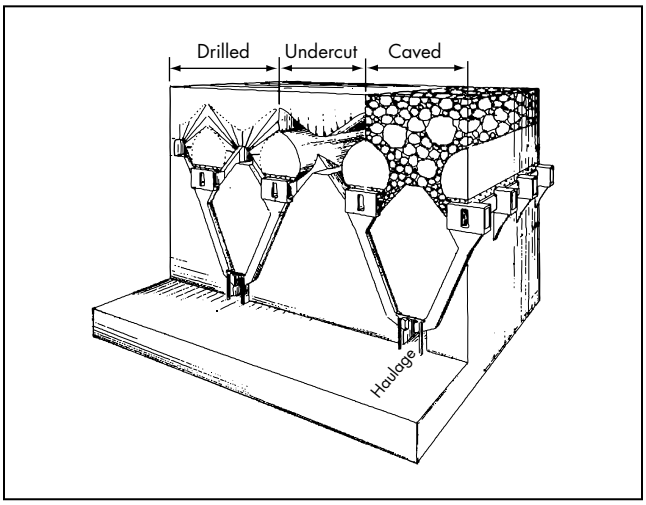
Rapid-Development Incentives

Very large initial capital expenditure is required to construct a block cave mine and bring it into production. The eventual NPV of the investment is dominated by the time taken to start production, typically 5–8 years. The rate of advance is a key factor, and cost per meter, although important, tends to be secondary. One very large, multinational mining company that is currently operating or developing four large block caving mines, plans to develop 1,000 km (600 mi) of development and 12,000 drawpoints over the next 11 years (Moss 2009). Over many years and projects, this mining company's advance rate has stayed at around 4 m/d (13 ft/d), and the company states this has been a critical factor for bringing any block cave projects in on schedule.



Source: Gould 1998.

Figure 12.5-10 Development needed for slusher-type block caving



Source: Lacasse and Legast 1981.

Figure 12.5-11 Stope development required for gravity draw block caving

While rock-drilling efficiency has increased by fivefold over the years between 1960 and 2000, the advance rate in meters per day of development has decreased by threefold; thus, the cost per meter has gone up by tenfold. The primary reason for this is the new emphasis on safety at the development face. These protocols have now evolved in the mines to a point where most development requires that all ground is at least fully bolted and in most cases has skin support in the form of shotcrete or wire mesh. This means that currently most of the ground control functions must be carried out within the development cycle. Yet a few mines and many tunneling operations get much higher advance rates, some more than 10–15 m/d (31–49 ft/d). How is this accomplished? It must be planned, equipped, and executed as if it were a civil tunnel contract.

What Must Be in Place?

From the very beginning, there cannot be any compromise of worker safety; management must develop procedures whereby no worker is exposed to ground that has been inadequately secured. By reviewing many studies of rapid excavation, it is apparent that the following steps must be set in place to achieve these advance rates:

- The mining operation must know the best drill round to achieve 95% of the drilled distance for the type of ground at the mine location.
  - A thorough geotechnical study must be ongoing to define rock types related to blast design and ground control problems, and detailed planning must precede each step of the mine development.
  - Drill rounds must be designed by qualified engineers, and the execution of drilling and blasting must be followed according to the design. In a recent survey by the National Institute for Occupational Safety and Health (NIOSH), 60% of mines allow drillers to decide what type of a drill pattern and how much explosive to use in the round.
  - Miners think in terms of blasting or bombarding the rock with as much force as possible, and every hole is the same; blast designers tend to think of cutting the rock, particularly in the cut area and the periphery of the drift.
  - Oxygen-balanced emulsions should be used to minimize ventilation time, and electronic blasting should probably be used to ensure perfect hole timing and minimize vibration and peak particle velocity where it may be required.

- Since 50% of the cost and about 25%–40% of the face time are now spent in ground control, controlled smooth wall blasting should be used. Damage to the remaining rock must be limited and overbreak minimized. NIOSH recently conducted a research study showing that the total cost of drifting using controlled blasting is about 4%, or $59/m ($18/ft) less than conventional drilling and blasting. No credit was given for less ground-control cost (Camm and Miller 2009). The ground-control savings could easily be $150/m ($45/ft).
- Driving the largest faces consistently with geotechnical issues of ground control and ground stress conditions allows for the most drifting efficiency, since larger equipment can be used and, in some cases, there is room for multitasking at the face. In one case the development advance went from 1.7 m/shift (6 ft/shift) to 3.4 m/shift (11.1 ft/shift), even though the size of the drift tripled (Bullock 1961).
- The most efficient and reliable equipment should be acquired for driving the developments, particularly the critical path developments, including computerized drilling controls and the use of improved explosives systems designed for controlled blasting. The initial, up-front cost may be millions of dollars, but many millions more will be saved in the NPV of the operation.
- The entire mine development operation must focus on getting the men and supply logistics and maintenance planned, so there will be little or no time lost at the face between cycles. In one Rio Tinto study (Moss 2009), 39% of the time was spent "waiting" for the next step in the 15-hour cycle.
- Procedures and metrics must be put into place where all parameters of the drilling, blasting, ground control, and loading portions of the drifting cycle can be defined in quantitative forms and are measured for development projects. If quality is not measured, it cannot be managed.
- To achieve the desired results, every project needs to establish a quality control group that has the responsibility and authority to enforce the drilling, blasting, and ground control procedures that have been put in place. This group should report to the mine manager. In one project observed by the author, where quality control was completely ignored during the "controlled" blasting of poor ground, the result was that the tunnel had a slab of more than 1.5 m (5 ft) of overbreak for at least 15 m (50 ft) of tunnel, which the contractor (on a cost-plus contract) was allowed to simply fill with layers of shotcrete until the correct tunnel dimension was achieved.

The following are a few recent examples of high-speed development.

Norwegian tunneling technology opened a new haulageway for a coal mine, which averaged 103 m/week (338 ft/week) driving a 38.5-m^2 (414-ft^2) tunnel 5,630-m (18,471-ft) long (Nilsen 2009). Much of the tunnel was in permafrost, making the advance more difficult. The key is to use the face to advance development. To achieve this, some Norwegian tunneling operations use

- Side-tipping loaders;
- Trucks sized to the tunnel, enabling drivers to perform a three-point turn in the tunnel;
- Drilling equipment with navigation that is fully instrumented to implement a plan of the drill layout, including the survey of the hole collar and hole toe;
- Use of 5.8-m- (19-ft-) long drill steels (where ground is appropriate), logging of the drilling performance, blasting using emulsion, and rapid-setting shotcrete (one-half hour in freezing conditions); and
- Most importantly, a dedicated, highly skilled work force regularly achieving more than 100 m (328 ft) per week of development in openings up to 38.5 m^2 (414.4 ft^2). The best week achieved 150.1 m (492.5 ft) of advance.

The world's best decline advance rate is that currently achieved at the Cadia East project in New South Wales, for Newcrest Mining Limited, which was 260 m/month (853 ft/month) for 9 months ending in May 2008 and included a single best month of 311 m (1,020 ft). This project also shows a cost of A$4,500/m (US$3,894/ft), which is 31% lower than the industry average for Australia (Lehany 2008).

In general, tunnel contractors do a better job of rapid advance rate excavations than mining companies do.

What Is the Payoff for Rapid Excavation?

Assume one is developing a 2% copper, block caving mine to produce 100,000 t/d (91,000 tpd). One can also assume a typical cost per ton of copper production, a copper sales price of about $0.90/kg ($2.00/lb), a $360 million capital cost ($45 million for 8 years versus $72 million for 5 years), a 10% discount rate, and a regular development rate of 4 m/d (13.1 ft/d) compared to 8 m/d (26.2 ft/d). Based on this data, the mine would come into full production in the sixth year rather than the ninth year. This would result in a difference in NPV in the eighth year of $260 million for the regular development compared to $86 million for the accelerated development, or a difference in NPV of $154 million, favoring the accelerated case. However, by the 13th year, the accelerated development case shows a positive discounted cash flow and NPV compared to the regular development case, which does not show a positive discounted cash flow until approximately the 35th year. This theoretical example illustrates that with a difference of $154 million, considerably more capital dollars might be spent to put the proper equipment and practices in place, resulting in the doubling of development advance per shift.

Developments Driven for Obtaining Information

Exploration

Often developments must be driven out ahead of the rest of the mine infrastructure to identify both the specific location and grade of the ore to be mined. There are no set rules as to how these developments must be driven, since so much depends on the type of deposit. In some cases, the development needs to be driven directly into the ore to assess its character, and in other geologic settings the developments need to be driven a specific distance from the deposit so a series of diamond drill holes can be used to sample those deposit characteristics. The primary objective should be to get these developments completed well ahead of the need to mine the area. This means getting the developments done many months before the information will be needed for mining operations.

Information for Mine Planning Obtained by Development

During the very early stages of evaluation and planning, a lot of information is needed to make the correct decisions concerning the property. Therefore, the initial opening might be driven to test a particular aspect of the mineral reserve that

is considered critical for successful exploitation. More often, this is termed a *test mine*. This could be from an adit, ramp, or shaft (see Chapter 4.6). In any case, permitting is usually required that may delay the project.

A large amount of the technical information required can be obtained from diamond-drill holes, particularly where the mineralized zone is fairly shallow, continuous, and flat. Unfortunately, the true economic potential of many mineral resources cannot be determined until after an exploration development or test mine is driven. From the development opening, underground prospecting can further delineate the materials present, a bulk sample can be taken for metallurgical testing, and/or true mining conditions can be revealed. This procedure involves considerable financial exposure and, unfortunately, the delineated resource may turn out not to be economically viable. Nevertheless, there may be little choice about driving a test development opening if the potential of the resource is to be proven and the risk to larger downstream capital expenditures is to be minimized. This is another case where the decision must be made between a decline or a shaft as well as the size of the opening.

Regardless of the reason for driving a preliminary development, three basic principles should be observed:

1. The development should be planned to obtain as much of the critical information as possible without driving the opening in a location that will interfere with later production. For most mining methods, it is usually possible to locate at least a portion of the exploration development where it can be used in the developed mine that may come later, even if it becomes necessary to enlarge openings to accommodate production equipment at a later date.
2. The preliminary development should be restricted to the minimum needed to obtain the necessary information.
3. The development should be completed as quickly as possible. Most exploration drifts for metal mines are only large enough to accommodate small loading and hauling units (from 2.4 × 3 m to 3 × 4.3 m [8 × 10 ft to 10 × 14 ft]). If geologic information is needed, it is often possible to locate the developments in proximity to several of the questionable structures without actually drifting to each one. With modern high-speed diamond drills having a range of 213 to 305 m (700 to 1,000 ft), these structures can usually be cored, assayed, and mapped at minimum cost.

After the preliminary development has been completed to satisfy a specific need, the development should be used to maximize all other physical and structural information to the extent that will not delay the project. As much geotechnical information on the rock mass quality should be obtained as possible, and should be just as important to the company as resource information.

No rules regulate the size of the shaft or the entry for exploration development. The size depends on many factors, including the probability of the mine being later developed to the production stage. If the probability favors follow-up with full mine development immediately after exploration development, and if the shaft depth is not excessive (e.g., 244–366 m [800–1,200 ft]), it would be logical to sink a shaft sized for later use in production development. This is particularly true of operations normally using small shafts for production. Using a small shaft of 3.7 to 4.0 m (12 to 13 ft) in diameter is not uncommon in hard-rock mining in the United States. For example, most of the Doe Run Company's shafts in the Viburnum Trend area of Missouri are of this size, as well as many of the shafts in the Tennessee zinc district—the Elmwood mine, originally opened by New Jersey Zinc, is a good example of this practice. Initially a 3.7-m- (12-ft-) diameter conventional shaft was sunk for exploration. After the continuity and quality of the ore were proven by drifting and underground prospecting, the shaft was used for further development of the mine and was eventually used as the production shaft.

The Savage Zinc Company's mine at Carthage,Tennessee (United States) is another example of a 3.8-m shaft originally sunk as a test mine and then later opened as an operating shaft. In contrast, some companies choose smaller-drilled shafts to speed shaft sinking and minimize early expenses. Such shafts are usually completed considerably faster than conventional shafts, but not as fast as a raise-bored shaft (i.e., where an opening already exists, and the hole is reamed upward, and the cuttings fall back into the mine). Because this section is directed toward initial mine openings, removal of cuttings must be upward through the hole and out from the shaft collar. Various methods involve the use of air or mud, with either direct or reverse circulation. A large number of blind-bored shafts exist in the industry. Although bored shafts may stand up very well without support, most of these shafts contain either a steel or concrete lining.

Developing Service Facilities

Shop and Storehouse Developments

The efficiency of a modern mining system depends heavily on the productivity and availability of the equipment used to extract the material. Most of the energy to move the material horizontally must be provided. Since the material may be very heavy and/or abrasive, and because the environment of an underground mine imposes adverse operating conditions, underground mining equipment requires a great deal of preventive maintenance and repair. One of the most serious and most prevalent errors made in designing a mine is the failure to provide adequate space and equipment for necessary maintenance and repair work. The amount of service, if any, provided underground depends on several factors:

- The degree of difficulty in moving equipment into or out of the mine is a major consideration. If the property is a limestone mine with adit entrances and a good shop on the surface, it would probably not be advantageous to duplicate the facilities and personnel underground. As a generalization, most decline mines do not need an underground workshop; they require only service and refueling bays. Maintenance is done on the surface.
- Underground shops should provide a safe and clean working environment. In mines that are gassy or carry a gassy classification, building underground shop facilities may not be practical.
- If the active working area will be totally abandoned after a fairly short life, it would not pay to invest in an extensive underground shop area. Mines such as the punch mines in Appalachia (United States) are typical examples of this situation.

Most underground mines with shaft entries should develop good underground shop facilities. In the Viburnum Trend mines, which are developed with shafts 152–396 m (500–1,300 ft) deep, all of the mines rely on large mining

equipment and have well-developed underground shop areas. The total area usually used as a shop area can range from 465 to 1,394 m^2 (5,000 to 15,000 ft^2). The important criteria for shop design are to provide

- One or two large bridge cranes over the motor pits;
- Easy access from several different directions for access to and around the cranes when they are in service for extended periods of time;
- A motor pit and service area for scheduled lubrications performed as part of the preventive maintenance program;
- A separate area for welding operations;
- A separate area and equipment for tire mounting and repair;
- An enclosed separate area for recharging batteries;
- Various work areas equipped with steel worktables or benches;
- Close proximity to the main supply house;
- An area for washing and steam-cleaning the equipment;
- An office for the shop foreman, records, manuals, catalogs, drawings, and possibly a drafting table. The main pit areas should be visible from the office windows; and
- If rail haulage is used, a separate locomotive and car repair shop must be developed, with provisions for at least one motor pit.

The size of the underground supply room also depends on several factors. Some mines have no supply room, whereas others have very large rooms providing as much as 1,115 m^2 (12,000 ft^2). Factors influencing the inventory policy include the frequency and ease with which the mine receives supplies, whether a supply system is installed at the collar or portal of the mine, whether there is a separate access to obtaining repair parts without disrupting the mine production, and the dependability of the parts suppliers for the equipment being used.

Sump Area and Pump Station

In mines below surface drainage, areas must be provided to store the water before it is pumped to the surface and discharged. No general rule exists to determine the capacity for water storage. The water flow into mines in the United States has varied from 0 to 2,208 L/s (0 to 35,000 gpm). Adequate pumping capacity is the only permanent solution to removing water from a mine. However, fluctuations in the water inflow and/or the time periods when some or all of the pumps are inoperable must be handled by having an adequate sump capacity.

Many mines use "dirty water" positive displacement pumps. The challenge is to keep the mud suspended until it enters the pumps, screening out the big lumps. This is economical up to about 80 L/s (1,320 gpm). Beyond that, settlers with centrifugal pumps will be necessary. Sumps are also needed in wet trackless room-and-pillar mines; the rubber-tired vehicles traveling on the roadways create fine material that eventually collects in the water ditches if the roadways have drips or continuous streams of water on them. Although this situation should be corrected to keep water off the roadways, it invariably occurs or recurs. A place must be provided for the fines to settle out of the water so they do not damage the clear-water high-head pump impellers. This can sometimes be done effectively in a small catch basin that is frequently cleaned out. However, if the water flows are substantial, the main sump receives the bulk of this fine material. As a result, provisions must be made to clean the sump with conventional or special equipment, divert the water into another sump while one sump is being cleaned, provide a place to put the fines ("soup") removed from the sump, and transport the fines to the surface.

One of the most common methods of cleaning a sump is to use conventional front-end loaders to remove the material. However, a ramp must be provided down to the sump-level floor. If deep-well impeller sections are suspended from a pump station down into the sump, considerable care must be taken to avoid bumping the impellers with the loader. Other systems using slurry pumps, scrapers, or diverting the material into skips have been tried, but there still is no single, efficient, low-cost way to clean a sump.

When cleaning a sump, diverting the water flow into another sump can also be a problem unless at least twice the normally required pumping capacity is provided. Half of the pumps must handle all of the inflowing water while one sump is being pumped down and cleaned. Mines that make large amounts of water also produce large amounts of fine material that fills the sumps much faster than anticipated. The sump of a new mine in the development stage cannot be cleaned too soon; if the mine is not producing much water at a particular stage of development, the sump should still be kept clean. The next shift may bring in more water than can be handled with half the pumps, and when that point is reached, there will be no way to reach the muck in the bottom of the sump for cleaning.

Two methods are often used to divide the sumps for cleaning. One is to locate a concrete wall between the two areas and provide a means of closing off one side and making the water flow to the other side. With this method, the wall should be anchored properly on the top and bottom with reinforcing steel into the rock. This provides a safety factor in preventing the wall from collapsing into the side being cleaned. The other system is to develop two physically separate sumps. Both sumps must be provided with pumps, a method of stopping the water inflow and diverting it to the other sump, and a means of access for cleaning.

Many mines locate the pump station below the sump chambers, using horizontal centrifugal pumps instead of deep-well turbine pumps. These mines avoid the installation of a vacuum priming system because the overflow feeds the main pump. Although this system provides easier access to the sumps and makes cleaning easier, the risk of losing the pump station by flooding is increased because of its lower elevation.

If the power for the pumps originates from a source not maintained by the mine (i.e., public power with or without "tie lines"), the mine may be without power for extended periods of time. Therefore, every protective measure should be provided. Using deep-well pumps on one of the upper or middle levels helps protect the facility for the maximum length of time. If the pumps are in a room or chamber with the power lines coming in and the water pipes going out, it may be impossible to access the pumps for replacement without a small overhead crane on a rail above the pumps. If deep-well pumps are used, sufficient headroom is needed to remove the multistage impellers.

Storage Pocket (Skip Pocket) Size

For mines that extend over a large vertical distance, there is usually enough storage capacity in the orepasses to accommodate any mismatch between production schedule and hoisting schedule. But for mines that are on a single level, there is often

a need to build a large storage capacity between the production of the mine and the hoisting system.

The correct size of the skip pocket depends on what it is intended to accomplish. For example, if management decided to try to hoist for 20 shifts per week but stoped two shifts per day in a mine producing 4,536 t/d (5,000 tpd) between midnight Friday and 7:00 AM Monday, the minimum pocket size would be found from

$$C = [S_m/S_h] \times S_{md} \times T_{ms} \quad (12.5\text{-}1)$$

where C is the capacity needed in metric tons or short tons, S_m is the number of shifts per week the mine operates, S_h is the number of shifts per week the hoist operates, S_{md} is the number of shifts the mine is down, and T_{ms} is the number of metric tons produced per mine shift.

For this example, the capacity would be calculated as

$$C = [10/20] \times 6 \times 2{,}268 \text{ t/shift} = 6{,}804 \text{ t} \quad (12.5\text{-}2)$$

or in English units

$$C = [10/20] \times 6 \times 2{,}500 \text{ st/shift} = 7{,}500 \text{ st} \quad (12.5\text{-}3)$$

It is invariably found that a much larger pocket or alternative surge capacity will improve the performance of the mine. The loading station and shaft hoist are often a bottleneck on production.

Other Service Facility Developments

A variety of other rooms or drifts must also be developed. A lunch area (or room) is essential around an underground shop. In many underground mines the lunchroom doubles as a mine rescue chamber. Office space should be provided for the privacy of the underground supervisor and the security of records and equipment.

Another underground facility that must be developed is a fuel storage area. For supplying all of the diesel-powered equipment used in the mines, it is usually advisable that a system be developed for centralized fuel storage. Where the mines are shallow, it is common to put the fuel oil underground through boreholes. However, where the mines are deep and have multiple levels, palletized storage containers are common. In either case, a retaining wall must be maintained that will contain the fluids in case a tank ruptures.

A new consideration is that of storing biodiesel fuel underground. Such fuel has the same requirements as diesel fuel in that it is constantly needed in the refueling of the trackless equipment, but there is also the added advantage that biodiesel is low-temperature sensitive and storing it underground will keep it fluid.

There must also be a space for explosive storage. Of course this should be located in a safe location, away from main haulageways and high-voltage electrical lines. Equally important is to construct the explosive magazine with complete security and management control. In addition, a separate storage and secure location must be built for the detonator systems.

MAKING MINE DEVELOPMENT DECISIONS

Many ways exist to develop a mine. However, there is only one "least-cost" method for any specific mine, and it is up to the mine engineering staff to develop trade-off studies to determine the optimum method to develop the mine. A classic example was recently completed for Stillwater Mining Company (SMC) for the development of mining of ore below the 3,200 Level down to the 2,000 Level. The mine production is at a nominal capacity of 4,200 t/d (4,600 tpd) of combined ore and waste, and is projected to increase to 4,900 t/d (5,400 tpd) over the next 10 years.

The 3200 Haulage System Trade-Off Study (C. Jacobs and B. LaMoure, personal communication) at the Stillwater mine was initiated to evaluate several life-of-mine muck-handling systems, each having the potential to provide for long-term muck-handling requirements for the operation while meeting common operating criteria and long-range goals. Each system evaluated was unique with respect to mine development, timing, required infrastructure, and expenditure profile.

The objective of the study was to evaluate alternate long-term production from the deeper levels of the mine for ore and waste handling systems of various hauling options in lieu of the proposed belt conveyor system, and to make a recommendation for the best haulage option. The proposed conveyor system was listed as an option to provide a "base case" comparison with other options. The primary options in this study included

- A decline with a conveyor option, with diesel trucks (base case);
- Deepening of the existing shaft with rail haulage;
- An internal shaft or winze with rail haulage and secondary hoisting;
- A decline with trucks using only diesel fuel; and
- A decline using electric trucks.

All options consider a common 20-year mining plan, with a maximum total throughput of 4,900 t/d (5,400 tpd).

Economic evaluation of the trade-off study options considered incremental differences in capital and operating expenditures within SMC's gross mining model (GMM). This precluded a pure incremental analysis of the different options but allowed SMC Technical Services to construct 20-year cash flows for each different haulage option, which consider all gross mining costs with respect to each plan, the revenue stream generated for each plan based on metals price assumptions, and downstream processing and commercial cost combined with applicable credits.

It is important to evaluate both annual cash flow and cumulative cash flow. The option with the highest cumulative cash would logically be the best option but may not be the desired option if annual cash flow is negative for 1 or more years.

Net cash flow (NCF) from each option was taken directly from option-specific cash-flow models and compared to perform an economic analysis considering mutually exclusive projects. The NPV for each cash flow is considered. Incremental analysis is then applied to derive incremental NPV values and ROR. Based on the analysis, the following results are relevant:

- NCF in all options are positive values; there are no negative years. This is due to full consideration of revenues and costs via the GMM, applied to what are relatively small capital costs and incremental capital costs of the haulage options.
- Using a discount rate of 10%, the electric trucks option is the best economic choice based on NPV. The incremental NPV for the cash-flow comparison confirms this with a

cash-positive incremental NPV, and a positive incremental ROR. Present value ratio is also favorable.
- In the absence of the electric trucks option, the shaft-deepening option becomes economically favorable. Increasing the discount rate in the analysis to 20% indicates that the base-case option is the economic choice based on NPV, where the shaft-deepening option has more cost in forward years. The diesel truck option was eliminated because of ventilation constraints and the logistics of operating eight trucks in the ramp system.

MECHANICAL EXCAVATION METHODS OF DEVELOPMENT

Because of the many advantages of mechanical mining when the physical properties of the rock and the tools are right for cost-competitive excavation, hard-rock mining companies keep trying to push the edge of technology so they, too, can benefit from mechanical mining. While it has been a very slow process of change, there have been a few notable successes. The principal advantages to mechanical excavation are

- Improved personal safety;
- Minimal ground disturbance;
- Reduced ground support needed;
- Minimum overbreak, thus less material to move;
- Minimal ground vibration or air blast;
- Uniform muck size;
- Reduced ventilation requirements;
- Continuous operations;
- Conducive to automation of system;
- Where the system is applicable, higher production rates; and
- Where the system is applicable, reduced tunneling costs.

The disadvantages to mechanical excavation are

- In hard rock, the system may have a poor penetration rate;
- In hard or abrasive rock, the system may have a high cutter cost;
- Some mechanical systems are not easily adaptable to changing ground conditions;
- It may lack flexibility of operations (TBMs can't cut sharp curves); and
- It often has a very high initial cost for the mechanical excavating equipment.

For designing a TBM for mine developments as compared to civil applications, Friant (2001) makes the following design recommendations:

- Multiple-speed cutterhead—An infinitely variable drive with high/low-speed torque is needed for broken ground changing to solid ground. A variable frequency drive is available.
- Closed or shielded-face cutterhead—Recessed cutters combine with radial bucket openings that extend toward the center of the head. Cuttings should not have to drop all the way to the invert through the slot between the rock face and cutterhead shield to be picked up. Back-loading cutters should be mandatory. There is no excuse for a fatality due to rock falls while changing cutters.
- Adequate shielding for rock support—Shielding, sometimes covering a 360° area, should extend up nearly to the gauge cutters. Adequate space must be provided near the cutterhead to install bolts and ring beams. Shields should not have wide gaps that allow small rocks to rain in. Shields should be active (i.e., with hydraulic adjustments) so if the machine gets squeezed, it can release itself.
- Soft-touch grippers—Grippers, if applied with too much pressure, can break up the tunnel wall and cause massive cleanup jobs after each push. Putting pressure on the crown should be avoided.
- A small length/diameter ratio—Either the whole machine length or its segment lengths dictate the radius curve through which it can bore. Often a 30-m (100-ft) radius is desirable on a midsize machine. A short stroke is an acceptable way to achieve the short turn radius desired. Continuous boring systems are desirable (i.e., with two sets of grippers), but only if they do not compromise other requirements of shielding and grippers.
- Mobility—A machine needs the ability to back out of its own tunnel, break down into manageable size subassemblies, and come apart and go together quickly. This may be achieved by the machine itself or through accessories for the machine. It must be kept in mind that in a mine there will often be many short tunnels as opposed to one continuous drive.
- Access for drills—In addition to rock-bolt drilling, the machine must supply access for drilling for grout curtains, spiling, soil stabilization, probe holes, and drain holes.
- Design simplicity—The simplest methods of thrust, steering, and control must be used, not those that are the most clever. Human-engineered controls and safety systems should be incorporated.

In soft material, such as coal, borax, potash, and trona, the applications of mechanical excavation are too numerous to mention. However, the successful hard-rock applications in mining are covered by a few examples:

- Stillwater Mining Company—Three different TBMs have been used to drive very long lateral drifts that are used first for ore-body delineation drilling, and then for main-line haulage. The rock has an unconfined compressive strength (UCS) ranging from 87 to 165 MPa (12,500 to 24,000 psi). The first machine used had trouble following the curvature of the ore strike, but the latter two machines were built for underground mine applications and were more adaptable. The biggest benefit for using the TBMs was time saved in the development schedule (Bullock 1994).
- Magma Copper Company—To expand the mining from the San Manuel (Arizona, United States) mine, a 4.5-m- (15-ft-) diameter TBM was used to drive 3,156 m (10,353 ft) of development during preparation for panel caving of the Kalamazoo ore body. The UCS of the rock ranged from 150 to 180 MPa (21,755 to 26,105 psi). The tunneling involved some 150-m- (500-ft-) radius curves and a decline of 5.5% (Bullock 1994).
- Heinrich Robert mine—At this coal mine in Herringen, Germany, a Voest-Alpine AM 105 roadheader was used to drive a drift 38.5 m^2 × 3.9 km (421 ft^2 × 2.3 mi). The rock is a massive quartzite sandstone with a UCS of 119 to 167 MPa (17,259 to 24,235 psi). During a period of 1 year, the machine averaged 5.6 m/d (18.5 ft/d), which included arch support on 1-m (3.3-ft) centers (Bullock 1994).

Table 12.5-1 Typical underground development estimated costs in 2009 U.S. dollars*

Development Item	Dimensions in Metric Units	$/m (Accuracy ±25%)	Dimensions in English Units	$/ft (Accuracy ±25%)
Drifts and crosscuts	3.7-m w × 4.0-m h, bolted	2,790–3,100	12-ft w × 13-ft h, bolted	850–950
Stope crosscuts	3.4-m w × 4.0-m h, bolted	2,550–2,900	11-ft w × 13-ft h, bolted	780–880
Stope attack ramps	3.0-m w × 3.0-m h, bolted	1,950–2,050	10-ft w × 10-ft h, bolted	600–630
Decline (contracted)	4.9-m w × 5.5-m h, bolted, and 152-mm shotcrete	6,900	16-ft w × 18-ft h, bolted, and 6-inch shotcrete	2,100
Ventilation drift	4-m w × 4.6-m h, bolted, and 76-mm shotcrete	3,850–4,050	13-ft w × 15-ft h, bolted, and 3-inch shotcrete	1,175–1,230
Ventilation raise	2.4-m diameter, raisebored and lined	6,250	8-ft diameter, raisebored and lined	1,900
Pump station, shops, stores, etc.	10.7-m w × 6.1-m h × length needed for m^3, bolted, and shotcrete	19.0–20.70/m^3	35-ft w × 20-ft h × length needed for ft^3, bolted, and shotcrete	5.80–6.30/ft^3

*Fairly fractured and weathered ground located in central Nevada requiring regular bolting with split sets on 1.2-m (4-ft) centers.

Other Types of Mechanical Excavation

Several other types of tools have been tried in mechanical excavation (Bullock 1994):

- Impact breakers (tunnels in Italy)
- Machines with disks on rotating drums (mobile miners)
- Machines with disks on rotating, programmable arms (continuous mobile miner)
- Various microboring machines

DEVELOPMENT COST ESTIMATES BY DRILL-AND-BLAST

It is not extremely useful to present cost estimates for developments, because the many variables of the rock types, rock-mass conditions and strengths, water condition, size, inclination, and use will affect cost. However, a few examples are presented in Table 12.5-1.

In another example, the Stillwater mine contracted two Alimak 3.35-m- (11-ft-) diameter raises, driven an average of 521 m (1,705 ft) for a cost of $5,612/m ($1,700/ft) in 2005 and 2006. When escalated to 2009 costs, this approximates to $6,600/m ($2,000/ft). There were two reasons why the raise climber was used for these ventilation raises, rather than using upreaming with a raise-boring machine: (1) there was no access to the top of the mountain in this isolated area and everything would have had to have been transported by helicopter; and (2) permitting to build a heliport and raise-boring site would have been difficult to obtain in this pristine area.

In the end, designers and consultants must remember that in planning a mine project, unlike a civil project, the product is not the hole: the hole is only a means to get at what you came for, the ore. Versatility, capability, flexibility, and at times raw power are the features that count.

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CHAPTER 12.6

Construction of Underground Openings and Related Infrastructure

Joe Cline

INTRODUCTION

The extraction of deep mineral deposits usually requires the development of significant underground infrastructure in the host rock. Construction of these openings constitutes a major aspect in the development of an underground mine. Depending on their use, development openings will assume various sizes, shapes, and orientations. The openings must be large enough to provide ample space and perform the intended functions to support the underground mining process with allowances for the geologic and geotechnical characteristics of the rock mass.

Mine excavations are categorized into two distinct groups: excavations that are used primarily for rock handling and excavations that are used for service. Development openings that fall in the latter category are considered permanent or semipermanent, usually with an operating life equal to or exceeding that of the operating mine. Thus the locations of these excavations are particularly critical in the overall mine infrastructure plan; they must be designed and constructed to meet rigorous performance standards while maintaining their operating role.

In the development of underground mine openings, the design and construction aspects of the excavations must be considered concurrently. The design aspect will address the size, location, and ground support requirements of the openings with respect to the geotechnical characterization of the host rock to ensure the associated equipment can be installed and any unfavorable geologic/geotechnical conditions will be accommodated. The construction aspect will address the excavation methods and equipment, excavation sequencing, and integration of the development with the associated construction activities.

A significant portion of the underground development schedule is normally dedicated to the construction of major mine service facilities such as pump rooms, crusher rooms, underground maintenance shops, ore bins, orepasses, loading pockets, ventilation and haulage drifts, and so on. In hard-rock mines, the openings for most of these facilities are normally excavated using drill-blast-muck cycles. Vertical and inclined openings, including raises and orepasses, can be excavated using either drill-and-blast or semicontinuous methods (such as raise boring). Accessibility requirements are critical as well in order for mobile equipment and workers to gain access and construct the associated facilities.

Each underground facility has a significant role in the continuous operation of the underground mine. The decisions that are made from the development of the design (such as operational redundancy) to the operation and maintenance phase (such as implementation of control systems and periodic maintenance) are critical in ensuring constant, uninterrupted mine operations. For instance, the pumping system for an underground mine facility with high water inflows would be designed and implemented with enough redundancy to ensure that in the event of a system failure, auxiliary equipment is present that can be commissioned immediately to augment the primary system until such time that repairs are completed and the primary pumping system can be put back into operation. The ultimate reasoning behind designing and implementing redundancy in most underground mine service facilities is to ensure (with economic considerations for capital expenditure) a safe working environment and eliminate or significantly reduce the potential for catastrophes that could result in the eventual loss of revenue.

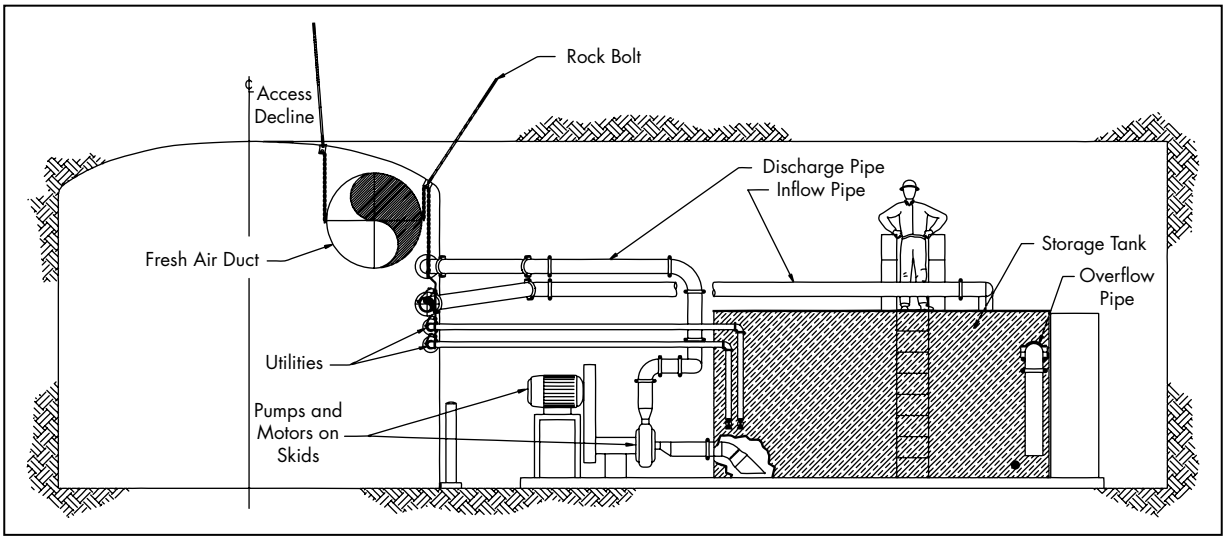
UNDERGROUND OPENINGS AND RELATED INFRASTRUCTURE

A number of underground openings and related infrastructure are common to most underground mine operations. They include developments such as access drifts, ventilation drifts, sumps, refuge stations, battery rooms, and so on. Major infrastructure such as conveyor galleries, underground crusher chambers, pump rooms, sumps, and underground maintenance shops normally involve significant capital expenditure but are necessary facilities in underground mining operations.

Sumps and Pump Rooms

Sumps and pump rooms are used in underground operations primarily to collect and manage water in and around the mine

Joe Cline, Lead Discipline Engineer, Cementation USA, Inc., Sandy, Utah, USA



Courtesy of Kennecott Eagle Minerals Co., Miller Engineering, and Stantec – Mining.
Figure 12.6-1 Unitized storage tank and pump arrangement

workings. Water inflow in a mine can occur from the following primary sources:

- Drilling and other associated activities during underground development and production (service water)
- Groundwater inflows

In either case, the water must be effectively handled by locating, designing, selecting, constructing, and maintaining a dependable storage and pumping system. The size and configuration of the pumping system depends largely on the predicted amount of water inflow to the mine. Depending on the predicted water inflows, the water handling system arrangement may consist of

- A series of drainage ditches, channels, and pipes at different working levels that direct water to sumps; the water is then stored and ultimately pumped to the surface either with a single high-efficiency pump or by a series of pumps; or
- Drainage galleries or drifts used to transfer water to a central sump or storage facility and then pump it to the surface.

The first arrangement is best suited for small to medium water inflows (up to several hundred cubic meters per hour) where large inflows not anticipated in the hydrologic study could overwhelm the pumping system. The second arrangement is suited for underground operations with large water inflows (several thousand cubic meters per hour) where hydrologic studies indicate there is a potential for flows to peak beyond those predicted.

Common characteristics of underground sumps and pump rooms include the following:

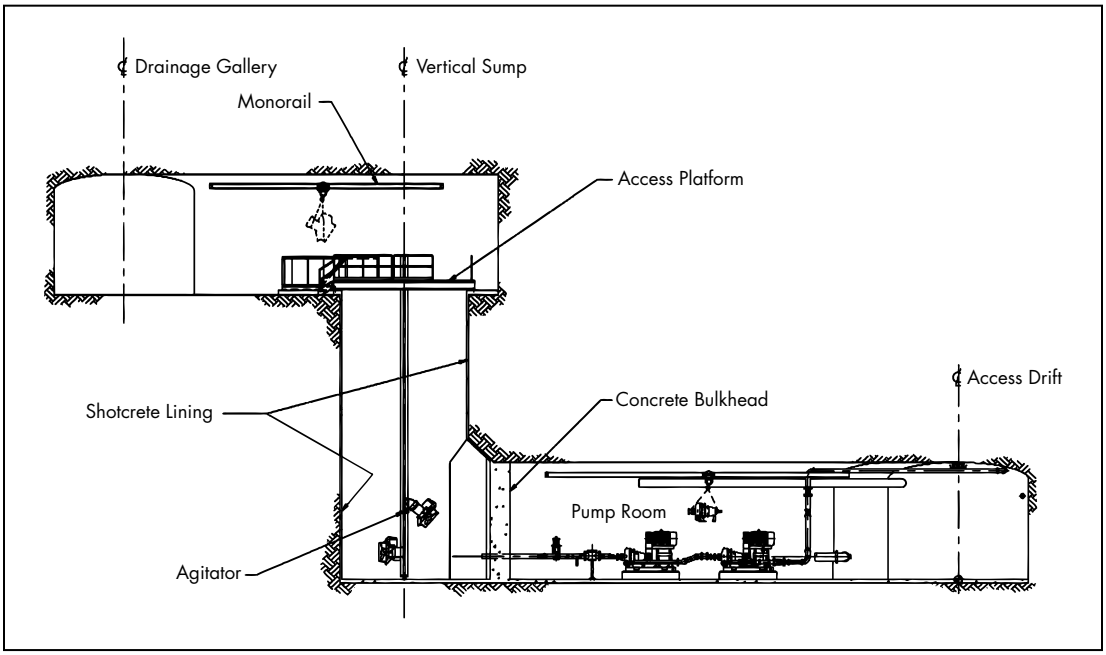
- Excavation and concrete structures for the sump capacity and pump room size
- Concrete bulkhead(s) for the sump containment, equipment foundations, and pipe support structures
- Access platforms and walkways to provide for equipment inspection and maintenance
- Mechanical equipment (pumps, motors, piping, and associated appurtenances)
- Power, electrical, instrumentation, and control equipment (switchgear, mine load center, drives, etc.)
- Lifting devices (monorails, cranes) for removing major equipment and reassembling after service

Figures 12.6-1 through 12.6-3 illustrate three varieties of underground pumping system arrangements that are common in underground mine operations.

In the majority of cases, underground mine pumping systems are designed to handle suspended solids that can be abrasive (depending on the size and distribution of the particles in suspension) on the pump components that are in direct contact with the carrier liquid. Loss of pump operating efficiency due to wear of pump components is a major concern in underground mine operations. The pumps specifically designed for this type of application are referred to as slurry pumps.

Centrifugal pumps and positive displacement pumps are the two most common types of pumps used in underground mine applications. Other pumping arrangements, such as those employing multiple-stage vertical or horizontal pumps, have been employed in underground mining for pumping clear water. These do not handle suspended solids well and thus require the excavation of settling sumps. Vertical deep-well pumps are unpopular in conventional mine dewatering because the entire pump column needs to be removed to access the pump for repair or maintenance; however, they are typically used when dewatering a flooded mine.

The pumping unit illustrated in Figure 12.6-1 is one of several skid-mounted packaged units that comprise the entire pumping system. This type of pumping system is commonly used during the early development of a mine. The pumping arrangement can consist of multiple units of similar design connected in series and located at specific elevations within the mine to collect and transfer water upstream from the lowest extremities of the mine workings. Each unit can incorporate single or multiple centrifugal pumps with motors driving the pumps integrally or via V-belt drive systems. Typically, each unit discharges water upstream into the storage tank of the successive unit in the pumping circuit. The setup in Figure 12.6-1 can potentially handle flows up to several



Courtesy of Barrick Gold and Stantec – Mining.

Figure 12.6-2 Combined vertical sump and pump-room arrangement

hundred cubic meters per hour; the actual performance is contingent on the total head to overcome, the number and size of pumps per unit, and the total number of units within the pumping circuit.

Figure 12.6-2 (also shown in plan view later in Figure 12.6-21) incorporates a vertical sump that is located directly upstream of a pump room. A concrete bulkhead separates the sump compartment from the pump room. In this type of arrangement, particular attention is required to locate, design, and construct the bulkhead to ensure stability and provide a watertight seal around the rock–concrete interface and at penetrations through the concrete bulkhead. The sump walls are typically lined with shotcrete or concrete to limit and/or eliminate any seepage into the surrounding rock and to maintain the integrity of the rock wall.

Pumps are usually erected on a concrete pump base that is raised several centimeters above the pump room floor. This arrangement prevents equipment from coming into contact with water in the event of a pipe leak or other malfunction. An overhead crane or monorail is usually provided to aid initial installation and for removing and replacing pump components. The pumping arrangement shown in Figures 12.6-2 and later in 12.6-21 consists of multiple banks of pumps, each having multiple centrifugal pumps connected in series. The number of pump stations that are required to make up the complete pumping system depends on the pressure rating of the pumps.

Contingent on the quantity and specific gravity of suspended solids in the carrier liquid, agitators may be installed inside the sump to keep solid particles in suspension at all times. This type of layout provides the following advantages:

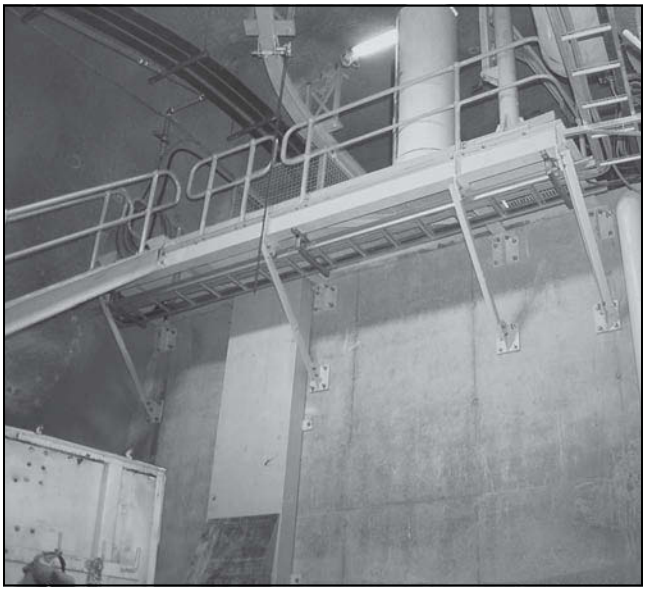
- Can be used to provide redundancy in a mine's pumping system where only one bank of pumps operates at any time while the other is on standby
- Allows for the possibility of installing additional banks of pumps should there be a future demand for more pumping capacity
- Supports the ability to perform equipment maintenance or repair without a complete system shutdown



Courtesy of Northparkes Mines.

Figure 12.6-3 Remote sump and pump-room arrangement

The primary pumping equipment shown in Figure 12.6-3 comprises multiple piston diaphragm positive displacement pumps, complete with electronics and other accessory features. Each unit is capable of operating at up to 1,000 m of head at low flow rates, and up to about 90 m^3/h at lower heads. They are efficient in handling high concentrations of solids. The figure also illustrates an arrangement where the pump room is located



Courtesy of Northparkes Mines.
Figure 12.6-4 Remote sump at an underground pump station

in a separate excavation that is completely independent of the storage reservoir/sump cutout (see Figure 12.6-4 for sump illustration). Water is fed to the pumps by pipes running from the sump to the main pump-room intake. Unlike Figure 12.6-2, allowance is made immediately above the concrete bulkhead to access the agitator (the agitator keeps solids in suspension) and other equipment in the sump. Pumping arrangements of this type offer several operating advantages, including

- Pumping of dirty water,
- Operating under high static heads,
- Pumping in a single lift without the need for multiple intermediate pump stations,
- Operating at a high efficiency, and
- Eliminating the need for large settling sump cutouts.

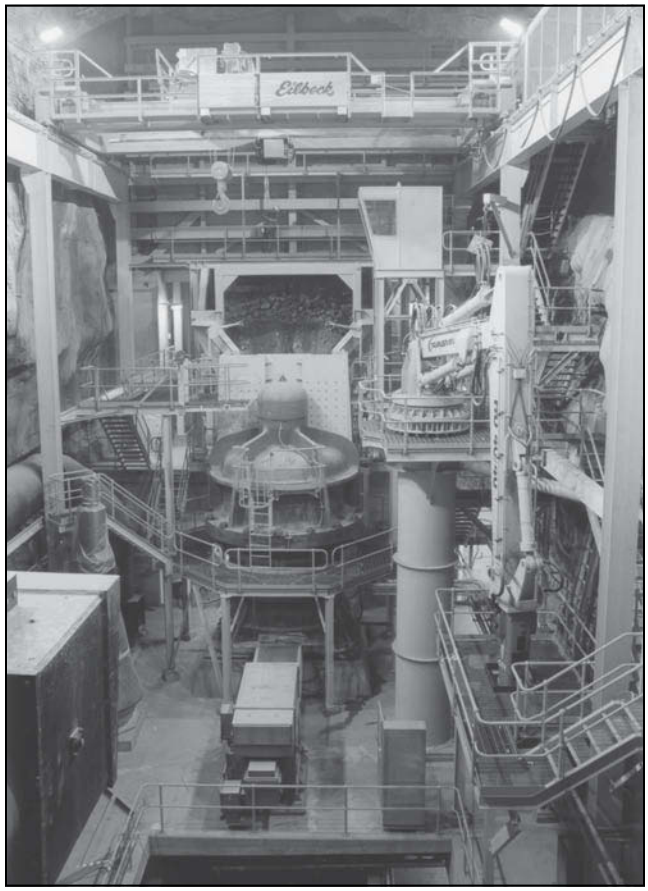
Crusher Rooms

Underground crushers are used to reduce run-of-mine ore or waste to a size that is suitable for the materials-handling system to transport to the surface. Crusher chambers can be the largest single excavation and infrastructure development in an underground mine. Substantial capital expenditure normally goes toward the planning, design, procurement, and construction of these underground facilities. An underground gyratory crushing station is illustrated in Figure 12.6-5.

Critical information necessary for making a decision regarding plant suitability and final underground crushing arrangement includes

- Location,
- Access,
- Mining method,
- Ground conditions,
- Ore properties, and
- Required support facilities.

The underground ore transport network must be integrated with the crushing plant location and design, as the mined ore must be transported and fed into the plant for crushing before being transported to the surface.



Courtesy of Northparkes Mines.
Figure 12.6-5 Gyratory crusher

Underground crushing operations can be classified as single-stage or multistage (i.e., primary crushing or primary–secondary crushing). The size and complexity of the facility depend on the number of crushing stages. Because of advancements in belt conveyor equipment and other associated materials-handling systems (to handle larger and more abrasive materials), most modern underground mines only need to install a primary crushing system. There is also an added capital and operating cost–savings benefit that can be realized through elimination of a stage in the overall crushing circuit. Some underground mines in the past have implemented multistaged crushing operations. In the 1970s, ore produced from underground operations at the Climax mine in Colorado, United States (currently abandoned) was crushed by primary and secondary crushing at their No. 4 crusher station.

In hard-rock mines, gyratory and jaw crushers are most commonly used because of their proven capability in breaking hard rock. Sizer technology is advancing to the point where they are now suitable for size reduction of hard rock. Feeder breakers are more successful in softer ores. In coal mines where rock fragmentation activities are not as aggressive, other crushing plant alternatives (such as roll and impact crushers) are more predominant. When developing underground crusher rooms, the following must be considered:

- Location (remote or central) with respect to ore delivery and conveyance after crushing



Courtesy of Northparkes Mines.

Figure 12.6-6 Underground shop service bay with overhead crane

- Crusher chamber access (primarily for maintenance and supply of spare parts)
- Excavation size, stability, ground control, geotechnical instrumentation, and monitoring of the rock mass around the chamber
- Overhead cranes of suitable capacity and other localized lifting devices within the chamber
- Dump pocket (truck or rail dump) and coarse ore surge bin
- Oversize handling arrangement at truck and rail dumps (grizzly, rock breaker, etc.)
- Crusher, drives, mountings, control room
- Concrete structures supporting the crusher, drives, and motors
- Crushed ore loading pocket beneath the crusher or remote surge bin
- Pan feeders, impact feeder belts, screens, and belt conveyors
- Access platforms and walkways
- Dust collectors, scrubbers, and crusher room ventilation
- Power, electrical, instrumentation, and control equipment (substation, switchgear, drives, etc.)

Underground Maintenance Shops

When no direct ramp access exists between the underground operation and the surface, or when the access decline is too long to tram equipment to the surface in a reasonable time frame, a shop and maintenance facility may be constructed underground to service and repair mobile and stationary equipment. In mines with only shaft access, major equipment is normally transported, assembled, and maintained underground.

After commissioning and operating, major equipment often only leaves the underground mine for replacement or major rebuilding in a surface or off-site shop. An underground shop is a major lifeline for the mobile fleets in an underground mine. The main advantage of locating a shop facility underground is for close proximity to the operations so scheduled maintenance and emergency repairs can be completed promptly with less equipment downtime.

Whether fleet maintenance is contracted out to a separate entity or performed by the mine's own personnel, it is essential to establish an ergonomic layout for the shop plan that is safe and maximizes service to equipment with easy access to supplies, spares, and consumables so that equipment times in the shop are limited. For this reason, multiple entry and exit points in the shop must be created to ensure that immobilized equipment undergoing repairs will not hinder the exit or entry of other equipment. When an underground mobile fleet is comprised of both rubber-tired and track equipment, underground maintenance of both types of equipment must be available. The maintenance requirements for track equipment (such as work on engines, braking systems, and other critical locomotive components) are fundamentally different from those of rubber-tired equipment. For these reasons, it is essential to have a dedicated repair shop for track equipment that is independent of the primary maintenance shop. Underground shops are often equipped to replace components such as engines, transmissions, axles, buckets, and so forth.

An essential component of any underground shop is the overhead crane and other associated lifting devices (see Figure 12.6-6) used to move heavy components around the shop floor. Other items of prime importance in an underground shop include

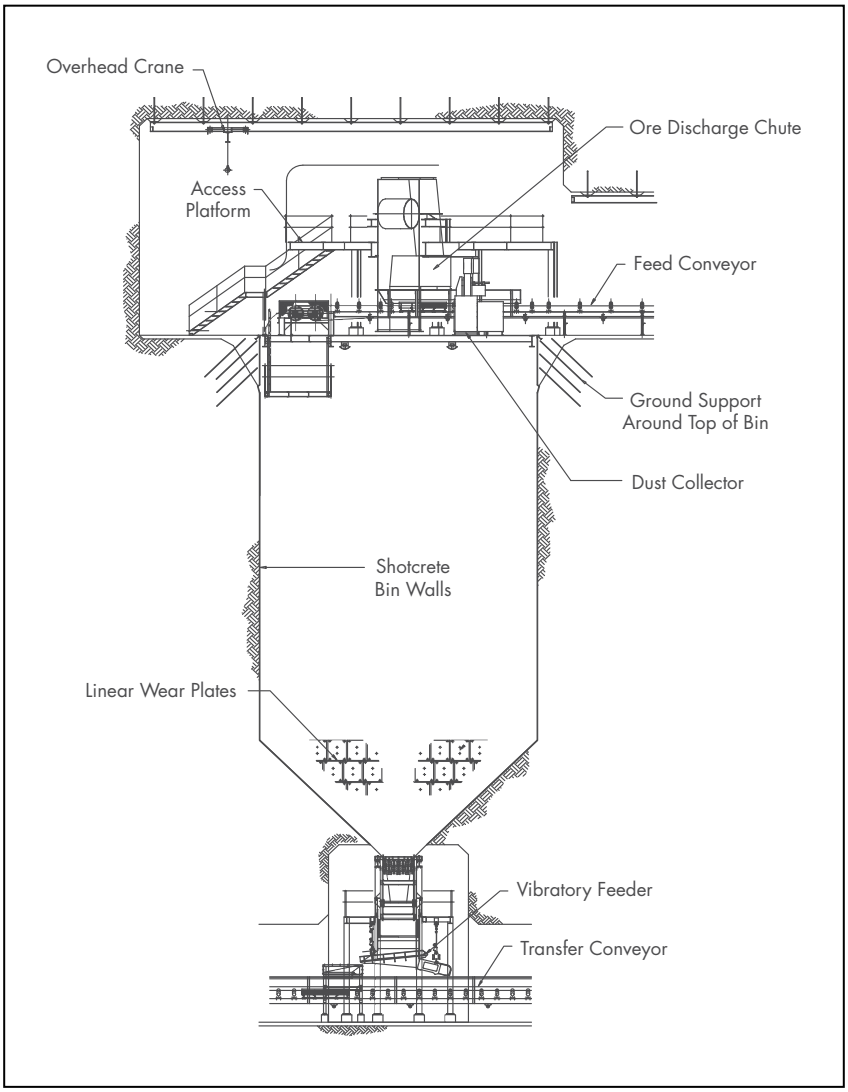
- Ventilation inside the shop;
- Fire protection;
- Access in and around the shop;
- Number of bays and their function;
- Types and level of services to be performed;
- Supply and management of consumables;
- Management of spent hydrocarbons, lube, and gray water;
- Shop lighting and illumination;
- Compressed air supply;
- Storage of tires, spare parts, and tools; and
- Equipment cleaning.

Ore Bins

Ore bins are used primarily for temporary storage of ore underground before final transport to the surface. Depending on the location in the ore circuit and the materials handling methodology, ore bins are generally referred to as run-of-mine or coarse-ore bins when used to store ore that has been sized with a grizzly or sizer, or that discharges from the primary crusher.

Because of the cyclical nature of ore extraction from the mining face, an underground ore bin provides the surge capacity necessary to maintain a constant flow of the ore stream. The material to be stored is normally fed into the bin via trucks, rail, belt conveyor system (typical for remote bins), directly from a system of orepasses, or directly from the crushing plant (if located directly beneath the crusher). The content of the bin is later discharged from the bin bottom through a hopper-chute arrangement for further ore transport by a belt conveyor, rail, or truck system.

Ore bin design and construction depend on a variety of conditions, but, most importantly, on the type and condition of the host rock. In massive hard rock, ore bins are normally excavated in and integrated with the host rock (Figure 12.6-7). In other rock environments (such as medium, soft, or sedimentary rocks), ore bins are predominantly constructed as independent structures in order to limit the effect of any



Courtesy of Stantec – Mining.

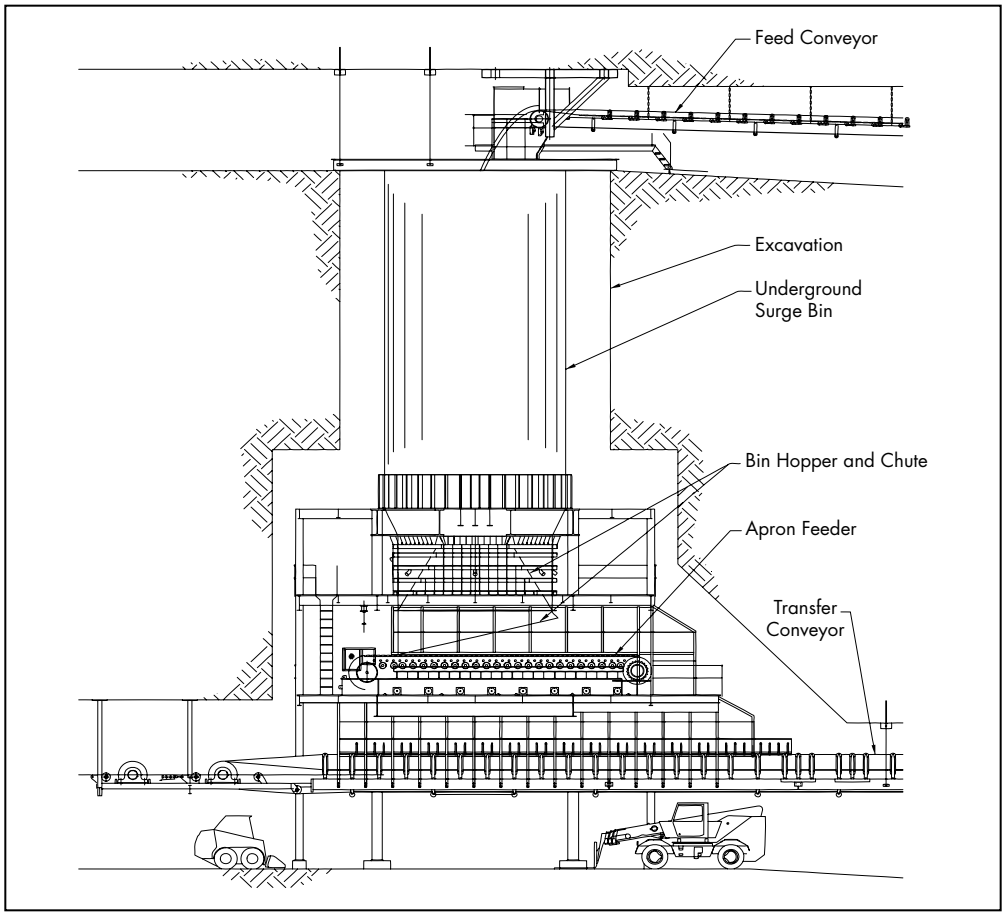
Figure 12.6-7 Ore bin and associated infrastructure in a hard-rock environment

potential ground movement on their ongoing performance (see Figure 12.6-8). Following are other important decision factors to consider with underground ore bins:

- Permanent access
- Access during bin development and construction
- Surrounding rock type and ground support requirements
- Location (above or beneath crusher, remote, close to hoisting plant, etc.)
- Size and capacity (based on production rate, scheduled, and/or nonscheduled shutdowns). Refer to the "Storage Pocket (Skip Pocket) Size" section in Chapter 12.5.
- Shape (circular shape most common, can also be rectangular)
- Associated materials handling infrastructure (feeders, chutes, conveyors, other materials handling equipment) to aid in transporting the material effectively
- Bin wall lining and other materials of construction (depending on abrasiveness of the stored material and characteristics of the host rock)

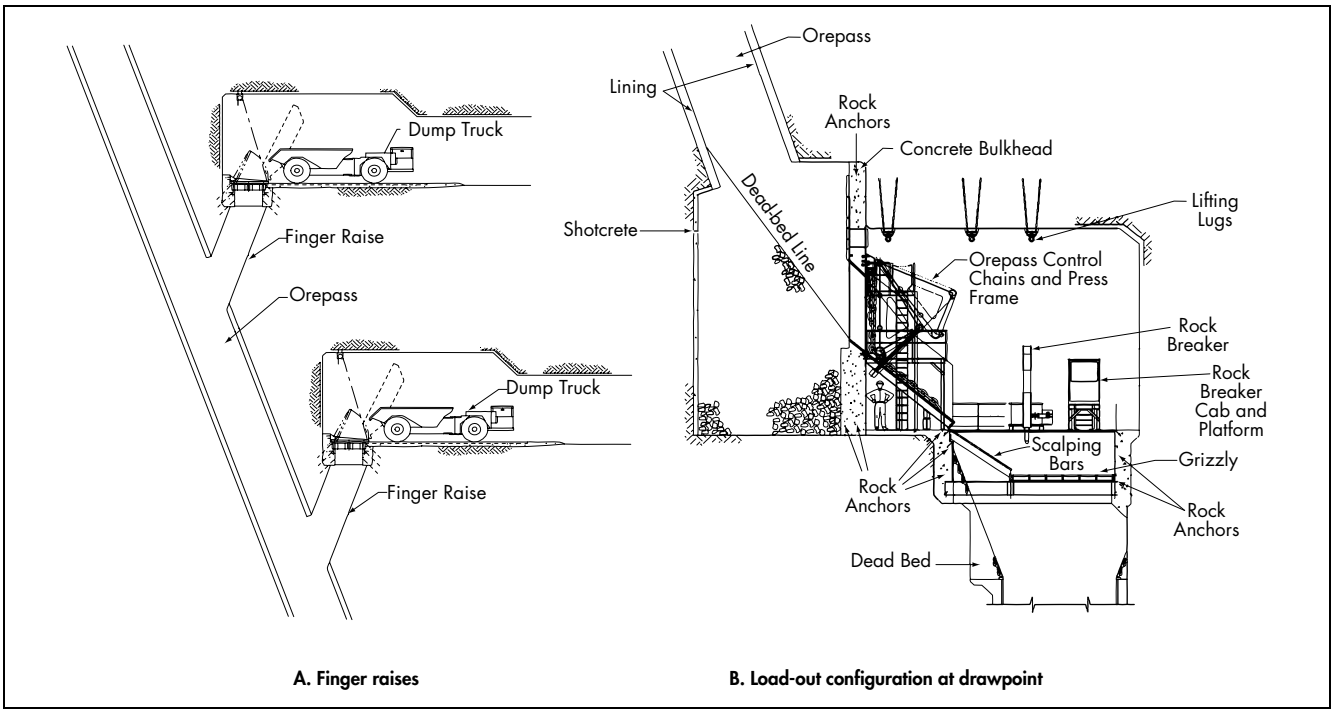
Ore- and Waste Passes

Passes are vertical or subvertical raises that are excavated in the host rock for transporting ore or waste by gravity flow from the upper levels in a mine to a predetermined lower level. Orepasses can be used to improve underground equipment productivity because they minimize overall travel distances from the ore loading point to the final ore dump point. The location and design of the orepasses are therefore critical in mines that primarily haul waste and ore with trackless equipment. Orepasses commonly run subvertically and may connect production levels by means of finger raises, as shown in Figure 12.6-9A. The top of the pass and finger raises, are usually equipped with a rock-sizing mechanism (such as a grizzly) to ensure that oversize materials which can potentially cause hang-ups in the pass are reduced in size. A load-out control mechanism (such as a discharge chute, chain controls, and press frame) is usually installed at the lowest point of a drawpoint, as illustrated in Figure 12.6-9B. Following are some major items to consider in underground orepasses:



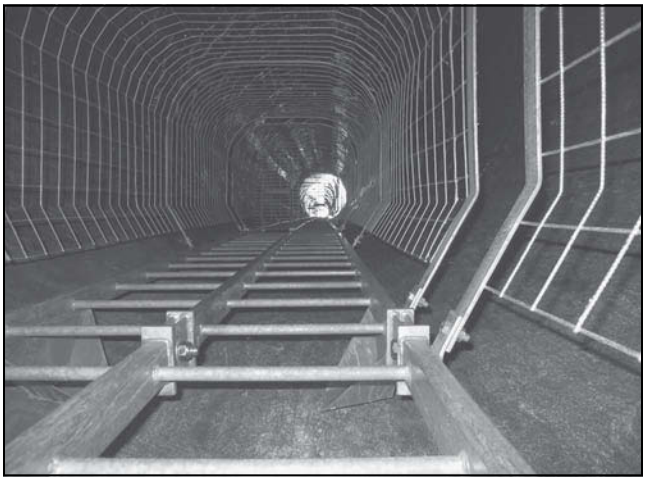
Courtesy of Potash Corporation of Saskatchewan–Allan Division.

Figure 12.6-8 Ore bin and associated infrastructure in an underground potash mine

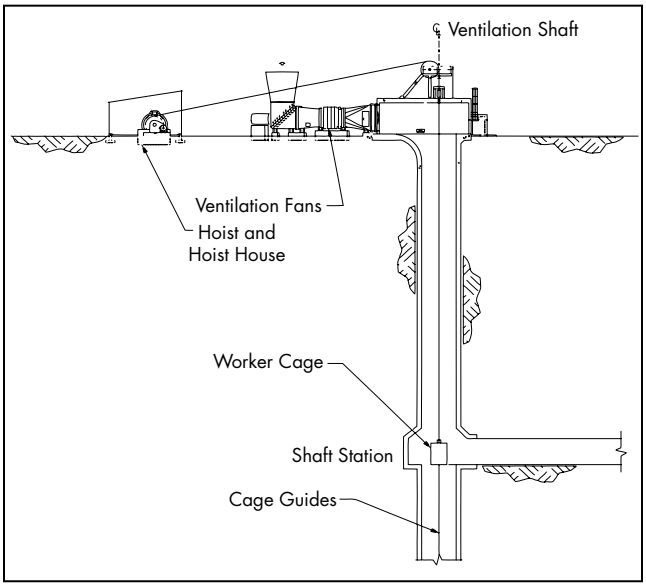


Courtesy of Stantec – Mining.

Figure 12.6-9 Orepass and finger raises



Courtesy of Newcrest Mining Ltd.
Figure 12.6-10 Access ladderway looking from top to bottom



Courtesy of Stantec – Mining.
Figure 12.6-11 Emergency hoist and escape pod

- Shape and size of the orepass, enough to prevent material hang-ups
- Stability of the orepass
- Maximum interval spacing between successive passes
- Orepass inclination
- Daily and total tonnages anticipated
- Type and orientation of ore body
- Material characteristics (fines, moisture content, particle size, etc.)
- Load-out configuration at bottom
- Static pressures on load-out mechanism
- Effect of impact on load-out mechanism (from release of material hang-up or material falling down an empty pass)

Emergency Access

A secondary means of egress is required in all underground mine operations and normally governed by local mining regulations. An emergency access (worker access) is a secondary means of egress that is provided primarily for miner exit from the underground workings in the event of a fire, temporary loss of electric power, or any other occurrence warranting worker evacuation. It is important that emergency exits are located in an area where the ground is stable and the serviceability of the facility can be guaranteed and will not be hampered if the primary means of egress is blocked. A worker access is a vertical or subvertical shaft that is constructed by drill-and-blast or raise bore and is normally located within the mine's fresh-air intake network to ensure the presence of fresh air in the shaft at all times when in use. During the development phase of the mine, rather than install a separate worker access shaft, one of the compartments of a hoisting or ventilation shaft could be dedicated to emergency access. (After a U.S. mine is in the operating phase, the Mine Safety and Health Administration [MSHA]requires a separate entry as an escapeway.) A worker access is usually equipped with one of two evacuation systems, illustrated in Figures 12.6-10 and 12.6-11: an access ladderway, or an emergency hoist and escape pod.

If an emergency hoist and escape pod system is implemented, provision must be made for standby electric power (provided the emergency hoist is not powered by an internal combustion engine) to operate the hoist in the event of an electric power failure. Following are some major items to consider in designing and constructing a worker access (emergency exit):

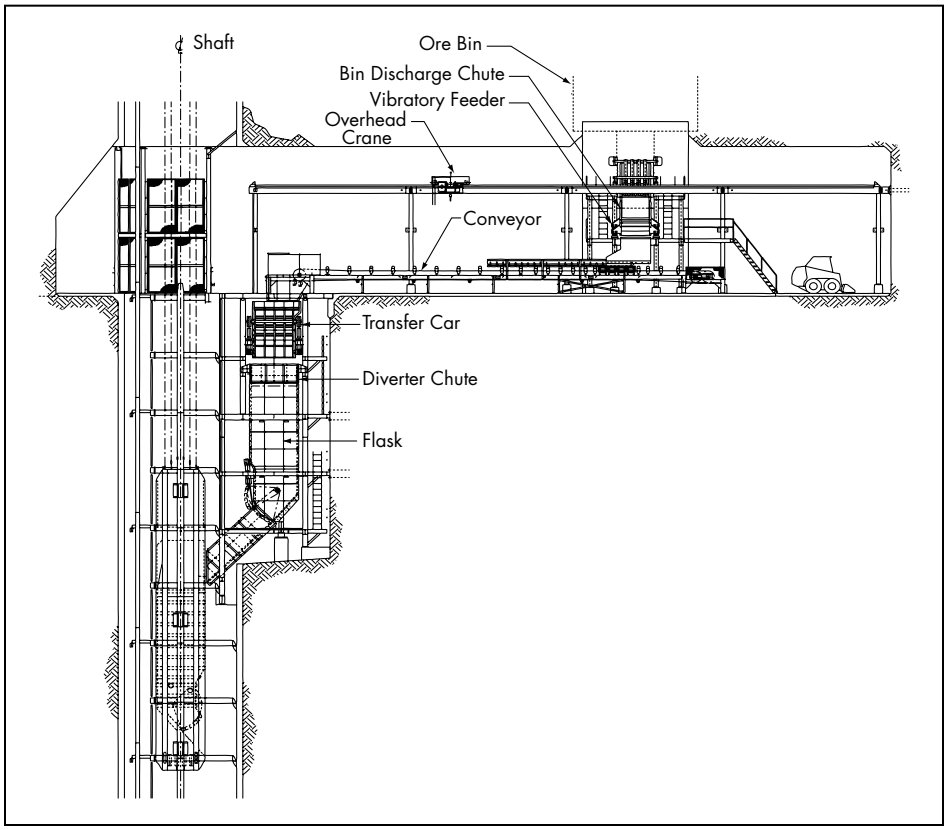
- Depth of mine operations from the surface
- Type of primary access (shaft or decline)
- Type of evacuation system
- Mine environment (corrosive, may require galvanized steel components)
- Cost

Loading Pockets

Loading pockets are used to feed ore or waste into skips in a production shaft system. Crushed rock or run-of-mine ore is conveyed and discharged into flasks at the loading pocket by rail, belt conveyors, or mobile equipment. The flasks at the loading pocket serve as a temporary underground repository for rock waiting to be fed into production skips. Several factors, including the following, govern the design and construction of loading pockets:

- Shaft production rate
- Shaft layout and design
- Type and capacity of the skips
- Method of ore (or waste) delivery to the loading pocket
- Construction methodology
- Ground support requirements

Loading pockets consist primarily of steel structures and usually must be integrated with the shaft steelwork at the skip's loading level. It is important that the whole infrastructure around the loading pocket be designed to effectively sustain impact loads from falling rock without inducing undue stresses and movement that could hinder its performance. The design of the loading pocket excavation is therefore important to ensure the structure's long-term stability and must incorporate adequate ground support. Ore delivery to the loading pocket can be semiautomated (with a belt conveyor system feeding from an ore bin or crushing facility) or by means



Courtesy of Stantec – Mining.

Figure 12.6-12 Loading pocket with belt conveyor feed

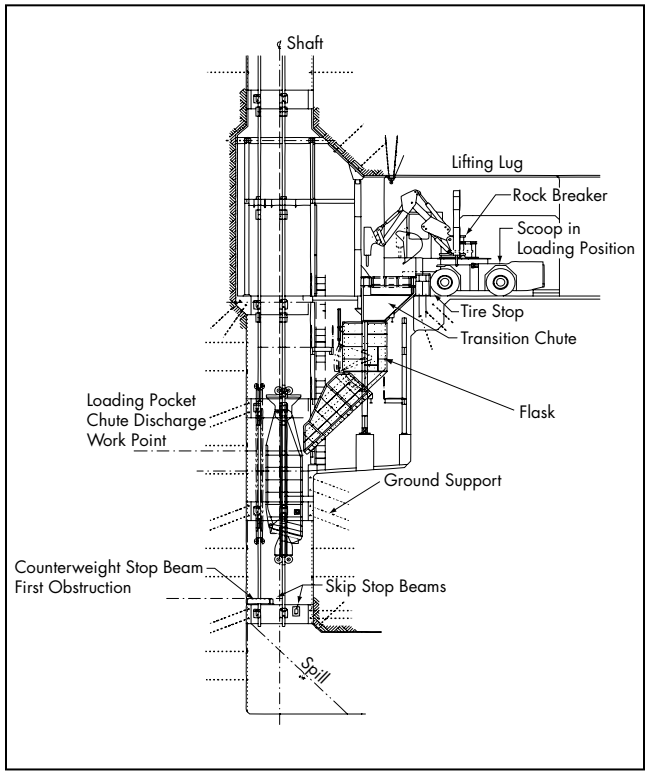
of mobile equipment (such as trucks and load-haul-dumps [LHDs]). Figures 12.6-12 and 12.6-13 illustrate typical loading pocket arrangements for both methods of ore handling.

Special Purpose Drifts

Conveyor, ventilation, and access drifts are three primary types of major lateral development that belong to the category of special purpose drifts. Common characteristics among all three underground openings include size (normally constructed to the smallest practical cross section to meet the service requirement), stability of the final excavations (usually over the life of the mine), and slope of the drift (to maximize equipment productivity in tramming).

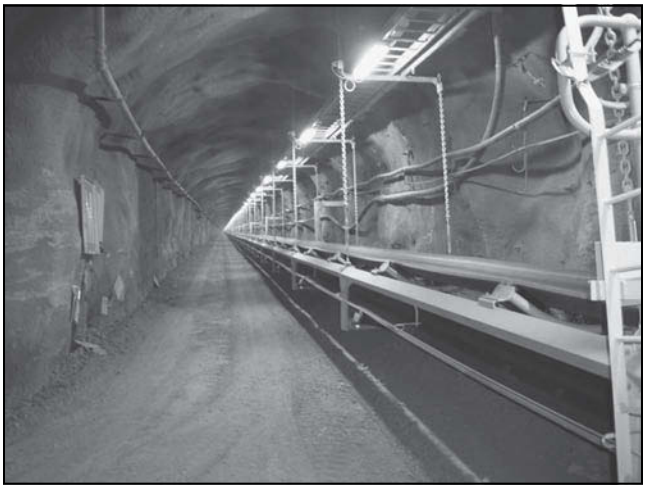
Conveyor drifts are normally constructed with a minimum cross section to accommodate the associated equipment while allowing for access, maintenance, and material spill cleanup. They may or may not be inclined, depending on the vertical distance the conveyor has to travel. The belt conveyor equipment is normally supported from the back of the drift, as shown in Figure 12.6-14, usually with rock-bolted chain supports or on the sill of the drift with steel supports on concrete pedestals as shown in Figure 12.6-15. The former method of support is more cost-effective and can be installed faster. Depending on the size, capacity, and layout of the conveyor equipment, the drift may be overexcavated at the head and tail ends or transfer points to accommodate additional equipment such as drives, motors, take-ups, transfer tower structure, chutes, and access platforms (see Figure 12.6-16).

Access drifts have similar characteristics to conveyor drifts except that they are sized and constructed to accommodate



Courtesy of Stantec – Mining.

Figure 12.6-13 Loading pocket fed by rubber-tired equipment



Courtesy of Northparkes Mines.
Figure 12.6-14 Belt conveyor supported from back of drift



Courtesy of Northparkes Mines.
Figure 12.6-15 Belt conveyor supported on sill of drift

mobile equipment and utility lines where applicable. Passing bays (to accommodate two-way vehicular traffic), muck bays, and safety bays (where personnel and vehicular traffic interact) may be incorporated into the construction depending on the use of the drift, as in Figure 12.6-17. The layout and construction of an access drift must take the following into consideration: type and size of haulage equipment to be used (trucks and LHDs or rails); clearance requirements around equipment for passage of mine personnel; routing of ventilation ducts, pipes, electric cables, and other utilities as shown in Figure 12.6-18; vertical distances between successive mining levels; presence of spiral ramps; maximum slope of invert to minimize tramming effort and maintain equipment productivity; type of sill construction; and drainage requirements.

Ventilation drifts are essential for moving air into and from the underground mine workings. These drifts are the primary link between the ventilation shafts or ventilation decline and the underground mine environment, and are classified as follows:

- Intake drift—for taking fresh airflow from a fresh-air intake facility to the mine workings
- Exhaust (return) drift—for taking the mine's return air from the underground workings to an air-exhaust facility

Most mine operations normally use the primary and secondary accesses for ventilation. However, large underground operations with large ventilation airflow requirements may require dedicated ventilation drifts, shafts, and raises. These ventilation routes are usually not accessible to personnel and equipment traffic because of the high velocity of air being moved.

Primary considerations in ventilation drift design and construction include the ventilation fan size and airflow velocity, total length of the ventilation circuit, excavation cross-sectional dimensions, roughness of the excavation wall (to limit friction losses in airflow), and stability of the excavation. The fan and associated equipment can be installed underground in the drift, at the ventilation shaft collar, or at the portal, depending on the system design and operating requirements. Figures 12.6-19 and 12.6-20) illustrate a ventilation decline and an underground ventilation drift, respectively. The setup in Figure 12.6-19 represents a ventilation system used

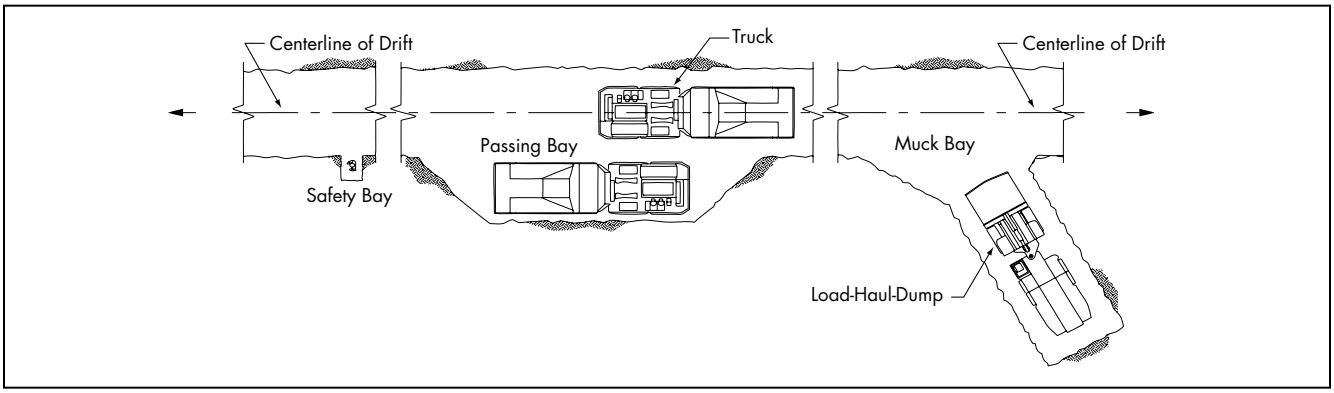


Courtesy of Northparkes Mines.
Figure 12.6-16 Underground conveyor drift and equipment at transfer point

in the interim (i.e., during initial mine development) until the required lateral development and permanent ventilation raise is completed. Fresh air is heated and, with the aid of booster fans located downstream of the heaters, drawn through heating plants and discharged through vent ducts that extend down the decline to the working face at all times. Return air is exhausted up the decline and at the portal.

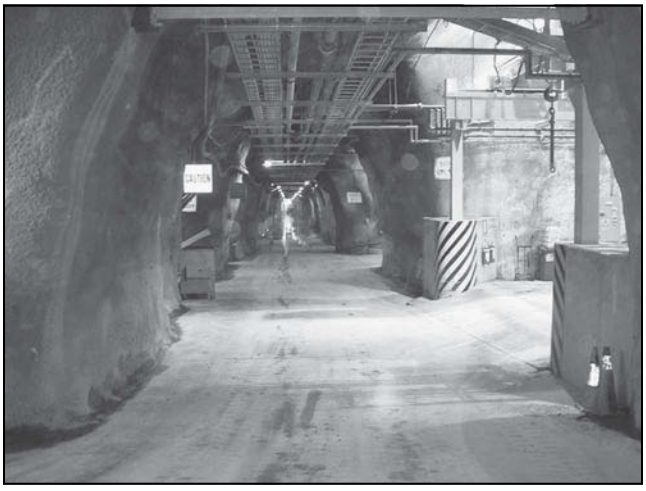
DESIGN, CONSTRUCTION, AND MAINTENANCE

Development openings and their associated infrastructure are unique in terms of design approach and design methodologies. The interrelationship between various components of the infrastructure usually requires multiple design iterations. This can be further complicated by the intrinsic variation of the mechanical properties of the host rock in which the development occurs. Design methods are established on sound engineering principles combined with empirical concepts, research, and extensive practical experience. Because the design process can sometimes be quite involved, only the salient points concerning design criteria and design methodologies are addressed in this section.



Courtesy of Northparkes Mines.

Figure 12.6-17 Mine access drift with passing bay, muck bay, and personnel safety bay



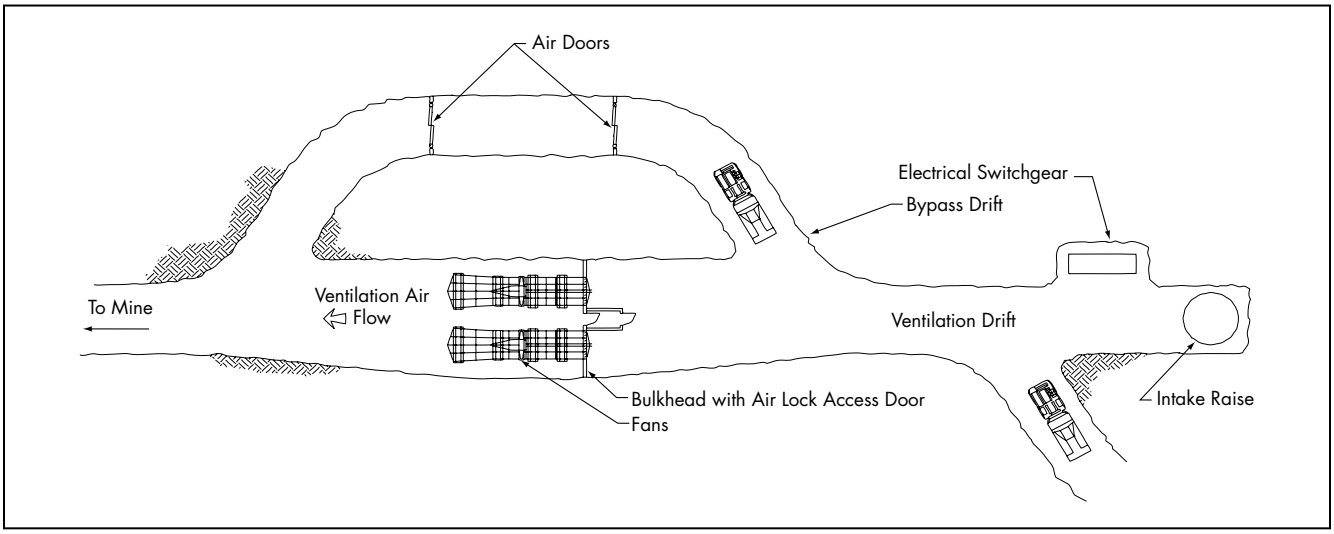
Courtesy of Northparkes Mines.

Figure 12.6-18 Mine access drift showing utility lines with ample clearance for mobile equipment



Courtesy of Rio Tinto.

Figure 12.6-19 Ventilation decline with surface heating plant fan and duct



Courtesy of Stantec – Mining.

Figure 12.6-20 Underground ventilation drift with fan

Underground mine infrastructures are normally designed and constructed with four primary materials of construction: steel, concrete/shotcrete, wood, and rock. In most modern mines, wooden structures have been completely replaced by steel and concrete components. Other types of construction materials (such as polymeric materials and carbon-fiber reinforced polymer) have recently been introduced in the underground mine environment but have not gained widespread acceptance because of limited performance data. Steel, concrete, and the host rock will continue to be the primary materials of construction in underground mine development for the foreseeable future because of their proven durability and the ability to accurately predict their behavior under service.

Design Aspects

The following key aspects must be addressed at various stages of the design of underground openings and related infrastructure:

- Scope of work—Including, but not limited to, the extent of work to be performed, the different engineering disciplines that will be involved, and battery limits
- Functional requirements—Including the intended purpose and performance requirements, end results to be achieved, effect of the underground environment on performance, size of excavation necessary, and location within the global infrastructure plan
- Design criteria—Including the key factors and information that govern the design development, size, shape, capacity, and ultimate performance of the completed facility
- Design development—The initial and final design of the facility must consider the following:
  - Duty requirement of the excavation
  - Maximum size of excavation needed to satisfy duty requirement
  - Geotechnical characteristics of the surrounding rock
  - Access limitations
  - Constructability
  - Materials of construction to be used
  - Impact of the facility's construction and operation on the mine environment
  - The underground environment's impact on the facility's serviceability
  - Possible methods of assembly of the various equipment
  - Supporting structures underground
  - System operating costs
- Planning and engineering—Including the potential impact of the new facility on existing operations, resource requirements (e.g., can engineering be completed in-house or must it be contracted out to a consultant?), and schedule for completion of engineering

Construction Aspects

The techniques, installation methods, and equipment normally employed during construction and installation depend on the type and size of the overall facility. The construction phase normally starts with planning of the various stages of construction and installation of the complete facility. The primary stages during construction and installation are

- Project preplanning,
- Procurement,
- Construction and installation,
- Schedule and resource management, and
- Closeout and commissioning.

Construction and installation aspects of the underground development must address the following:

- Safety—Implementation of a comprehensive and practical safety program. There are fundamental differences between the protocols of construction project safety and the safety protocols implemented by the mine operation group.
- Resource requirements—Ascertaining whether the work should be outsourced to an independent development contractor. (The mine may or may not have the equipment and/or skilled personnel available to self-perform construction.)
- Contractual—Preferred method of outsourcing (competitive bidding, single source, alliance, etc.).
- Construction resources—Materials, equipment, and services that can be supplied by the mine during construction (i.e., electric power, compressed air, water, concrete, shotcrete, etc.).
- Planning—Emphasis on minimizing interference with other ongoing mine operations.
- Infrastructure—Underground and surface infrastructure necessary to support the construction and installation work.
- Safeguards—Adequate lighting around stationary operating equipment, guards around rotating equipment, and so on.
- Access requirements and interface with existing mine operations.
- Construction equipment—Various pieces of equipment needed for each phase of the development, size, and capacities of equipment with respect to underground access.
- Waste handling—Removal of rock generated from excavation.
- Ventilation—Ensuring adequate ventilation of the construction area.
- Constructability—Construction methods and approach for each phase of the development.
- Installation sequencing—Consideration of construction equipment limitations.
- Construction management—Determination of whether the mine has the resources to dedicate toward management and procurement of supplies and equipment during construction and installation.
- Concrete—A versatile component in underground mine construction. It is used in almost all facets of underground development, from ground support to foundations for structures.

Blasting is a routine activity in underground hard-rock mines, and the resulting blast-induced vibrations, if left uncontrolled, can have detrimental effects on concrete structures. The level of impact of blast vibrations on a concrete structure is dependent on variables such as the distance of the structure from the blast initiation, the age of the structure, size of the structure, and type and amount of explosives used. Because it is common practice for concrete construction to occur in close proximity or adjacent to underground excavation development, blasting around newly constructed concrete structures should be considered during the design and installation

phases of underground mine development. The hydration and crystallization process in concrete normally occurs within 24 hours of placement. This period is very critical in all concrete structures because it is when the hardening and strength-gain process starts to occur. Normal concrete usually attains about 30%–40% of its compressive strength within 3 days and about 60% within 7 days of placement. Thus, blasting in close proximity to major new concrete structures is not advisable before 3 days following placement. Between 3 to 7 days following the concrete installation, blast vibrations adjacent to major concrete structures should be limited to that which produces a peak particle velocity of not more than 25 mm/s. After 7 days following concrete placement, a peak particle velocity of 50 mm/s or less should be used as a guide to limit blast vibrations until the concrete reaches its maximum strength around 28 days.

Project Preplanning

Tasks that are normally accomplished during preplanning include the following:

- After the contract has been awarded, become familiar with and understand the contract documents and general conditions of the contract. Arrange for independent services such as inspections and nondestructive testing, training, and so forth. This is particularly applicable to underground mine development contracts that are outsourced to an independent contractor.
- Determine and secure the various construction equipment (drill jumbo, LHD, haul truck, access platforms, mobile cranes, etc.) required for completing the work.
- Understand the ground conditions and plan the excavation, muck handling, and ground support sequence in accordance with the excavation designs.
- Establish procurement protocols and determine the long-lead items. Review their manufacture and delivery schedules and verify when they should be procured in order to avoid schedule disruptions.
- Establish availability of services and supply of all construction materials.
- Implement subcontracts for various fabrications (such as for miscellaneous steel structures, pipe spools, etc.).
- Plan the site set-up and administrative aspects of the site office, office equipment requirements, required project personnel, and so on (depending on the contractual arrangement).
- Arrange for transportation and schedule mobilization of resources to site.

Procurement

Procurement involves soliciting of suppliers and services; leasing of equipment; and purchasing and marshaling of permanent equipment such as pumps, electrical gear, overhead cranes, crushing equipment, lifting devices, and any other special item associated with the permanent installations. One aspect of the procurement process that must be addressed in the initial stage of planning is the on-site management of materials and equipment that are delivered for the construction of mine infrastructure projects. Planning for such undertakings (involving either a single major project or multiple small- to medium-size construction activities) must consider the logistics of transportation, sequencing of material delivery, and the need for adequate lay-down space and warehousing. It is advisable to manage construction project–related procurement separate from that of mine operations. A list of critical spare parts should be developed during the procurement stage. The procurement process can become lengthy when dealing with multiple vendor selection, whether prequalification-based or otherwise. Mine operations normally establish service agreements with suppliers and contractors that can be capitalized on in order to fast-track the procurement process.

Construction and Installation

Excavation methods. Raises, orepasses, and other vertical or inclined openings can be excavated by either drilling and blasting or semicontinuous methods such as raise boring, raise climbing, or drop raising. Ground conditions and rock type ultimately influence the excavation method chosen.

The most commonly employed excavation method for underground service facilities in hard-rock mines is drilling and blasting. Each drill-and-blast cycle follows a repetitive pattern consisting of blasthole drilling, charging and blasting, ventilation and dust removal, scaling, muck handling, and ground support, as described in the following paragraphs.

Work at the face is normally started by the mine survey crew that sets controls for the excavation and demarcates the excavation limits and layout of the blasthole drill pattern. The blasthole drilling sequence is normally arranged in a manner that maximizes drilling productivity.

Blasthole lengths are selected to complement the length of each round, which in turn are dictated by ground conditions and the maximum unsupported roof span allowed. Short rounds require short drilling depths, accomplished with a single-boom feed during fully mechanized drilling. All subsequent activities (face advancement, etc.) can be completed quickly because there are no equipment-related delays for handling drill rod extensions. In contrast, longer rounds require longer drilling depths, accomplished with multiple-boom feeds. Longer rounds can result in blasthole drilling inaccuracies, which can lead to increased drilling times, increased charging and blasting times, longer muck handling times (due to larger muck volumes), and larger roof spans to be supported.

After drilling is complete the holes are flushed, loaded with explosives, and prepared for the blast initiation.

Upon completion of charging and blasting, the blastholes are flushed with water (or other flushing agent, depending on the underground requirements). Out-of-reach blastholes are normally accessed with a work platform or basket. All equipment at the face is receded to a safe location prior to initiating the blast.

After the charging and blasting phase is complete, the face is ventilated to remove dust and fumes. Depending on the distance to the new working face, the vent duct and fan system may need to be extended to ensure that adequate fresh air reaches the working face.

Any loose rock on the excavation walls are removed by scaling. This process can be completed manually (by a worker in a basket with a scaling bar) or mechanically with a mechanized scaler. Manual scaling can be time-consuming and difficult. Mechanized scaling, on the other hand, is quicker and does not expose mine personnel to difficult working positions.

The muck handling phase commences after it has been determined that no potential rockfall hazards are imminent. Muck removal can be accomplished by one of the pieces of muck handling equipment listed in Table 12.6-1, or a combination thereof. For short rounds, high mucking rates may

be attained with only LHDs, depending on the muck volume and haul distance. For long rounds or where a large volume of muck is involved, other equipment combinations (such as trucks and LHDs) may be necessary in order to attain the required mucking speeds.

Hauling the muck away as soon as possible frees up the face for advancement and ultimately improves work cycles. As the distance to the face increases, the haulage distance and time increase as well. To free up the face for subsequent rounds, muck can be temporarily moved into muck bays excavated at regular intervals from the working face. The muck can later be hauled from the muck bay to the final dump location (as a separate operation off the critical path of other face-development activities) while the face is being further developed.

Ground support normally commences after the muck has been hauled away from the face. The type of required ground support depends on the excavation size and ground conditions. In poor ground conditions, it may be necessary to perform multiple ground support activities (initial and final) instead of a single activity at the end of each round. If the ground is fractured and roof stability becomes critical, initial ground support (such as with shotcrete and mesh) can be installed immediately after scaling to stabilize the excavation before mucking out completely. Final ground support (bolting, final shotcrete, etc.) can be installed after the muck has been completely removed and the excavation fully exposed for the next round.

Consideration should be given to the benefits of using the same drill jumbo for drifting and ground support installation; however, it may prove to be more cost-effective overall to obtain a separate jumbo for the ground support exercise. With a single jumbo, the bolt holes could be drilled while the muck is being hauled away. In large excavations and where the excavation completion schedule is critical, separate jumbos for blasthole drilling and ground support installation may be warranted.

Excavation sequence. Ideally, underground openings are excavated full face, depending on the excavation layout, ground conditions, and operating range of available equipment. In openings where the cross-sectional dimensions exceed the operating reach of available equipment (particularly in fractured rock), the face may be advanced by multiple headings in order for the following to occur:

- Limit the unsupported roof span of the excavations.
- Prevent collapse.
- Maximize equipment operating range per round.
- Reach the back and promptly install ground support.

Large excavations for pump rooms, crushers, and similar facilities can be developed by driving a top heading followed by a single bench or multiple benches several rounds behind the top-heading development. Other excavation techniques for large, regular-shaped openings include driving a pilot drift and slashing the back of the excavation on retreat. Factors that influence the excavation sequence include rock properties, ground conditions, equipment operating range limits, and imposed limits on blast vibrations.

Major construction equipment. Several types of major equipment are used for the various work phases. Examples of the major equipment used in the construction of underground

Table 12.6-1 Work task, major equipment, and selection criteria

Excavation Development
1. Blasthole Drilling • Equipment—Drill jumbo (with a two-feed boom plus one with a work basket), air-leg drill • Criteria for selection—Access drift dimensions; equipment operating range or excavation dimensions (face area, maximum height); excavation layouts and turning radii requirements; length of each round, number, length and diameter of drill holes per face; maximum single drill rod length that can be handled; construction schedule; and number of working shifts per day
2. Charging and Blasting • Equipment—Examples include ammonium nitrate and fuel oil (ANFO) loader or ANFO cassette on cassette carrier truck, and emulsion loading units. • Criteria for selection—Fixed-frame or articulated-frame vehicles must meet underground access requirements; integrated units that can be used with available mine utility vehicles; type and size of mining operations
3. Ventilation and Dust Removal • Equipment—Ventilation fans and ducts • Criteria for selection—Volume of fresh air required at the face; system resistance to deliver air to the face • Comments—Flexible vent ducts (vent bags) can be installed quicker and are best suited for temporary applications; rigid vent lines have less resistance and allow the system to operate on negative pressure.
4. Scaling • Equipment—Work basket with scaling bar or mechanized scaler • Criteria for selection—Size of excavation; equipment reach
5. Muck Handling • Equipment—LHD, LHD-haul truck, or excavator-haul truck combinations • Criteria for selection—Clearance and access requirements; desired mucking productivity; haul distance; tramming surface; and haul road condition and grade • Comments—Hydraulic excavators may be used in medium to large excavations where ample room exists for maneuverability.
6. Ground Support • Equipment—Drill jumbo, handheld drills, work platform, work basket, scissor lift, rock-bolting jumbo, shotcrete equipment/truck or shotcrete cassette on cassette carrier truck • Criteria for selection—Similar to that of blasthole drilling • Comments—Ground support may be staged (initial and final) depending on ground conditions.
Fit-out and Installation of Permanent Equipment
1. General Lifting and Erecting • Equipment—Mobile rough terrain crane(s) with telescopic boom, work platform (telescopic boom), scissor lifts, heavy-duty forklift, portable light towers, portable welders, portable generators, portable air compressors • Criteria for selection—Available working room, equipment reach, maximum single load to be picked, etc.
2. Concrete Work (including bulkheads, dams, equipment foundations, slabs, etc.) • Equipment—Concrete truck(s), concrete pumps, vibrators • Criteria for selection—Equipment productivity; available access to the work area; distance from concrete source to the final placement location
3. Steel Structures (all major steelwork, access platforms, ladders, etc.) • Equipment—Similar to General Lifting and Erecting • Criteria for selection—Similar to General Lifting and Erecting
4. Mechanical and Electrical Equipment (pumps, piping, valves and fittings, switchgear, cable trays, and cable runs, etc.) • Equipment—Similar to General Lifting and Erecting • Criteria for selection—Similar to General Lifting and Erecting

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CHAPTER 12.7

Tunnel Boring Machines in Mining

Lok Home and Odd G. Askilsrud

A tunnel boring machine (TBM) is a device used to excavate tunnels in a circular cross section through a variety of soil and rock strata. TBMs can bore through hard rock, sand, and almost anything in between. Tunnel diameters can range from ~1 to about 16 m (3.3 to 52.5 ft); tunnels smaller than that are typically created by horizontal directional drilling rather than by TBM.

This chapter describes mechanical excavations in soft/moderate to very hard rock and in rock-mass conditions ranging from marginally self-supporting to massive. TBM types discussed include state-of-the-art open, shielded, and other types of machines that incorporate specific features to suit particular applications, all using disc cutters. Mechanical tunnel excavators that create noncircular cross sections, such as mobile miners, mobile tunneling machines, roadheaders, and drum miners, are not discussed in this chapter, although they also serve important roles in civil and mining applications. These machines are discussed elsewhere in this handbook.

TBM TYPES

TBMs are normally classified into three groups:

1. Open-type machines: main beam, Kelly drive, and open gripper
2. Shielded-type machines: single shield and double shield
3. Soft-ground machines: hybrid, earth pressure balance, mixed shield, and slurry

A feature common to all TBM types is a rotating cutterhead that does the following: (1) presses forward and against the rock face under high pressure; (2) uses individual disc cutters placed strategically on the cutterhead; (3) performs muck removal by means of buckets; and (4) is supported by a main bearing with the structure being moved forward by a thrust system composed of cylinders. All TBMs have a backup system of power units and muck-removal systems.

Shielded-type machines are seldom used in mine applications because of the difficulty in removing the shields. A third shielded-type machine, the gripper shield, was designed to actively support the ground from the cutterhead to behind the TBM where rock support can be installed and has been used in a wide range of geological conditions. However, it is no longer available because of limited acceptance in the industry.

Because most mining applications take place in moderate to hard rock, soft-ground machines are not discussed in this chapter.

GENERAL TBM APPLICATIONS

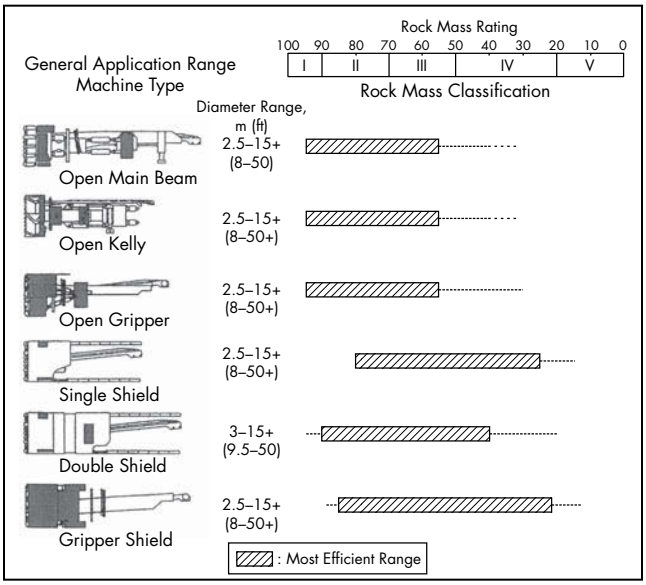
Compared to their use in civil-engineering applications, TBM use in mining has been relatively modest since the first successful hard-rock TBM was introduced by James S. Robbins in 1957.

TBMs in mining have ranged in diameter from ~1.7 to 7.6 m (7 to 25 ft). They have been used predominantly in soft to moderately hard formations. Mine applications at times place severe restrictions on component size and weight so that equipment can be lowered down existing shafts, transported to a designated assembly and launching area, and, after tunnel completion, removed. TBM systems in some mines must include explosion-proof equipment. Such equipment has been used for access to coal mining in Germany, England, and Canada.

Detailed study is required to select the proper TBM system for a project. Such study involves reviewing geological design reports, geological profile drawings, and baseline design or equivalent reports prepared by the mine owner or a consultant depicting anticipated conditions along the drive and required design parameters. Important factors to consider include the following:

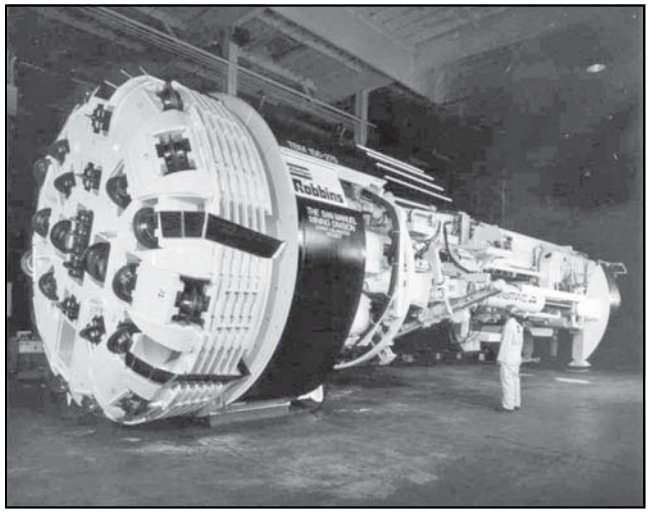
- Rock types
- Rock strength parameters from a testing laboratory
- Rock mass characteristics, including stand-up time at the actual tunnel depth, profile, and alignment of the proposed tunnel
- Tunnel diameter
- Temporary and permanent tunnel-support requirements
- Water and gas
- Geological profile
- Mucking requirements

Lok Home, President, The Robbins Company, Solon, Ohio, USA
Odd G. Askilsrud, President, Tunnel Engineering and Applications, Inc., Renton, Washington, USA



Source: Askilsrud and Moulton 2000.

Figure 12.7-1 General application ranges for various TBM types



Courtesy of The Robbins Company.

Figure 12.7-2 Main-beam TBM

Figure 12.7-1 can be a useful tool for discussion purposes in the planning stage. The chart shows typical TBM types and their most efficient application ranges based on rock mass rating.

The estimated TBM advance rate is based on rock strength and fracturing characteristics, in-situ condition of the rock mass, machine characteristics, and operating condition in the tunnel. The actual TBM advance rate is equal to the product of the average instantaneous penetration rate in millimeters per cutterhead revolution (inches per revolution), cutterhead speed (rpm), shift hours per day, and machine utilization (percent).

TBM DESIGN FEATURES

Important design features to consider for achieving efficient rock chipping include effective thrust load on the cutterhead and cutters, layout and spacing of cutters, and diameter and tip width of the cutter rings.

The nominal load capacity of a single-disc cutter has evolved from 89 kN (20,000 lb) for the 300-mm (12-in.) cutter of the 1950s to >312 kN (70,000 lb) for the 500-mm (20-in.) largest-size cutter of today. Today's state-of-the-art hard-rock TBM typically has a flat-face cutterhead with single tracking center, face and gauge cutters, each of which provides concentric kerfs on the rock face during excavation. Kerf spacing (center-to-center spacing of cutter tracks) is nominally ~90 mm (3.5 in.) for the face cutters, decreasing into continuously tighter spacing for the gauge-cutter positions. A flat-type cutterhead normally consists of nine cutters in the curved gauge area. A typical 4.6-m (15-ft) TBM hard-rock cutterhead with 432 mm (17 in.) back-loading cutters is shown in Figure 12.7-2. Single-disc cutters (483 mm or 19 in.) in wedge-lock housings are shown in Figure 7.1-4 in Chapter 7.1.

Selection of cutter diameter and cutter characteristics is determined by the application study and/or specific requirements of the application. State-of-the-art cutter rings for hard-rock applications are made of tool steel, specifically heat-treated so as to reduce the number of cutter changes and downtime for the anticipated ground conditions during the drive. The cutter hub assembly includes high-capacity tapered roller bearings and seals capable of supporting multiple cutter-ring changes.

Early during TBM design, it must be decided whether the cutterhead should have front- or back-loading cutters. Front-loading cutters are faster to change but require miners to enter and work beneath unsupported ground in front of the machine. Back-loading cutters can be changed from behind the cutterhead and under the protection of a shielded structure. The largest individual cutters 500 mm (20 in.) weigh up to 190 kg (425 lb). Cutters are moved and installed by means of a cutter-handling system normally designed for the specific TBM.

Other design features vary by TBM type. For further information on TBM cutters, see Chapter 7.1.

Main-Beam TBM

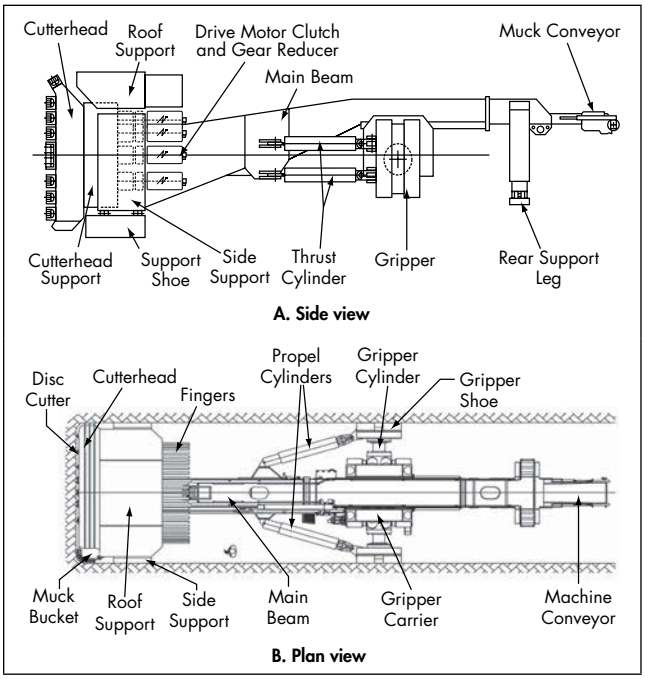
The main-beam TBM (Figure 12.7-3) is suited for high-strength rock and short faulted zones. Ground support methods can include ring beams, bolting, and shotcreting.

A main-beam TBM consists of four main elements: (1) cutterhead, (2) cutterhead support and main beam, (3) gripper and thrust assembly, and (4) conveyor assembly.

The gripper assembly with one set of grippers forms the stationary anchoring section of the TBM. It transmits thrust and torque to the tunnel wall during boring and carries part of the machine weight. The anchoring force is approximately 2½ to 3 times the total forward-thrust force.

The machine is forced forward a stroke at a time by hydraulic cylinders connected to the anchoring section and main beam as the cutterhead rotates against the rock face. At the end of a stroke and after the cutterhead stops rotating, the rear support is lowered to the invert, the grippers are retracted from the tunnel wall and moved forward equal to the stroke of the thrust cylinders, the grippers are again energized against the tunnel wall, and rear support is retracted so that boring can start again.

The front supports, support shoe, and extendable side and roof supports provide ground contact immediately behind the rotating cutterhead to stabilize the TBM during tunneling. The



Courtesy of The Robbins Company.

Figure 12.7-3 Main-beam TBM

cutterhead is typically driven by water-cooled electric motors located on the cutterhead support, each directly mounted to a gear reducer with output pinions. Drive pinions engage with the large ring gear, which is connected to the cutterhead. The cutterhead itself is supported by the main bearing.

The main-beam TBM has essentially become the industry standard open TBM mainly because of its ease of operation and ample room around the cutterhead area for ground support.

Kelly-Drive TBM

The Kelly-drive TBM (Figure 12.7-4) is suited for high-strength rock and short fault zones. Ground support methods can include ring beams, bolting, and shotcreting.

The Kelly-style TBM consists of two main elements: the anchoring section and the working or moving section.

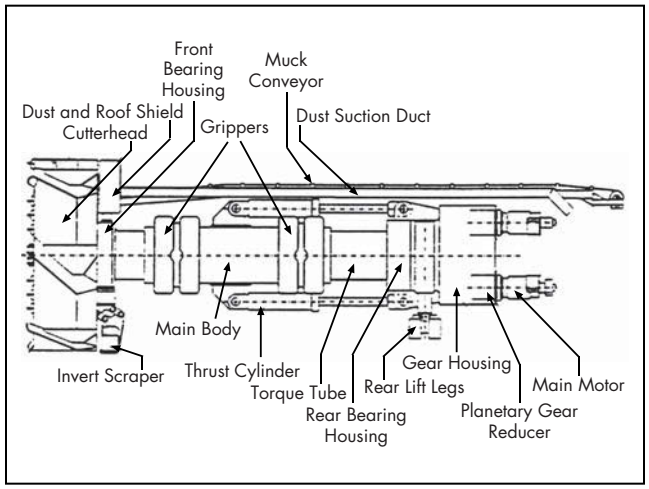
The gripper assembly with two sets of grippers forms the stationary anchoring section. It transmits thrust and torque to the rock during boring and supports the weight of the machine. Four grippers in the radial direction are arranged horizontally in pairs and controlled individually.

The working section includes the cutterhead, front bearing housing, torque tube, rear bearing housing, and drive train. Cutterhead rotation is controlled by motors at the rear of the machine coupled to gear reducers and pinions. A common ring gear turns the drive shaft through the center of the torque tube to transmit rotational power to the cutterhead. Hydraulic pulling cylinders develop thrust that is transmitted from the main body through the rear bearing housing and drive shaft.

Use of this type of TBM has declined, mostly because of the limited access around the TBM for installing ground support.

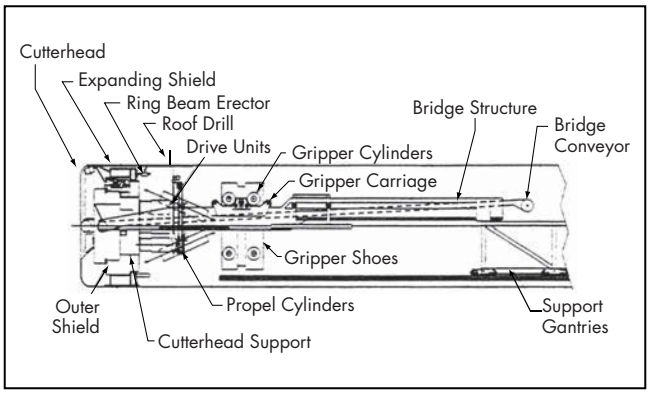
Open-Gripper TBM

The open-gripper TBM (Figure 12.7-5) is suited for high-strength rock and short fault zones. Ground support methods can include ring beams, bolting, and shotcreting.



Courtesy of The Robbins Company.

Figure 12.7-4 Kelly-drive TBM



Courtesy of Construction and Tunneling Services, Inc.

Figure 12.7-5 Open-gripper TBM

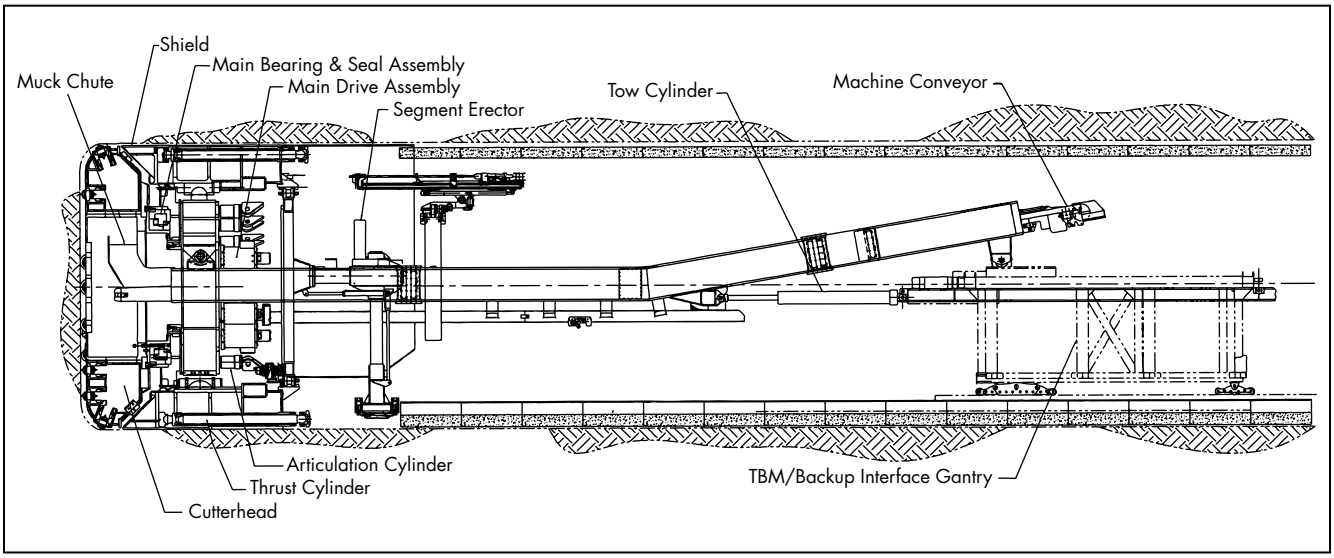
The TBM has expanding supports close to the excavation face and a simpler gripper design (no main-beam or Kelly-drive structures) than do the other types of open-type TBMs described. It typically uses an arrangement of lattice-type (diagonal pattern) thrust cylinders to react to cutterhead torque and correct roll. Advantages can include low cost, short overall length, and tight turning radius.

The machine consists of a cutterhead, forward expanding shield, and gripper assembly. The cutterhead and cutterhead drive are the same as typically seen on main-beam and gripper-shield TBMs. This configuration provides cyclic advance of the machine in the following sequence: bore, regrip, bore.

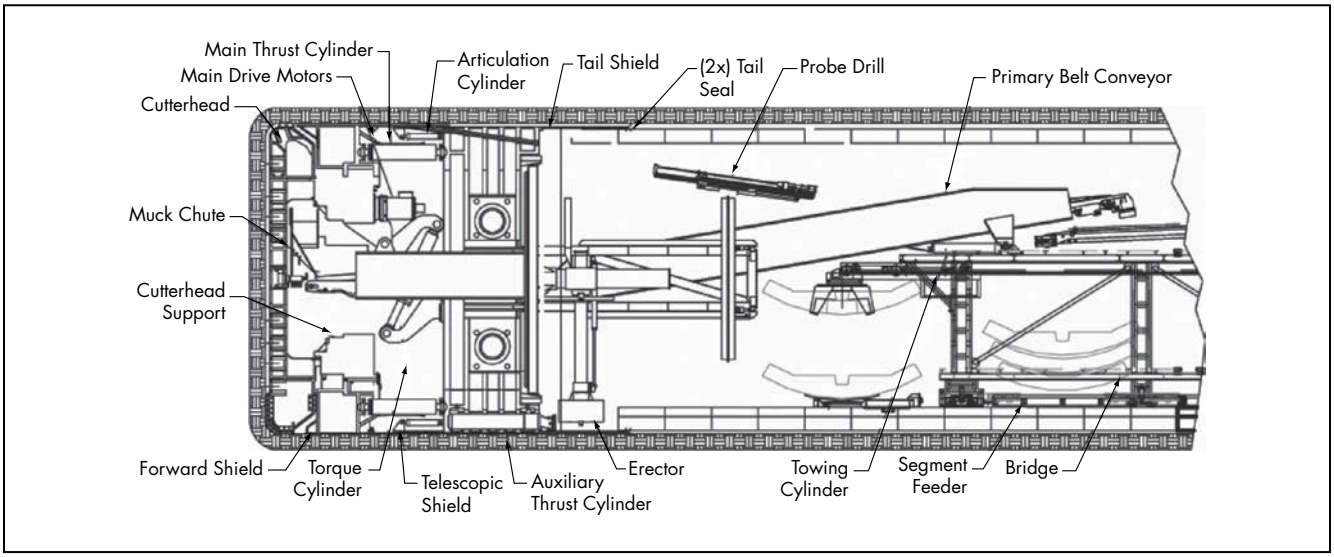
A bridge structure spans from the TBM cutterhead support to the first platform car. The bridge carries the grippers and also provides space to install muck-haulage rail and a mounting for the forward conveyor, material-handling equipment, exhaust duct, and tunnel-support installation equipment.

Single-Shield TBM

A single-shield TBM (Figure 12.7-6) performs boring and lining installation in sequence. Lining, typically concrete segments, is used as an anchoring station to propel the machine during boring. The machine can be used in hard as well as soft



Courtesy of The Robbins Company.
Figure 12.7-6 Single-shield TBM



Courtesy of The Robbins Company.
Figure 12.7-7 Double-shield TBM

ground, and is well suited for tunnels in mixed ground and loose formations with low stand-up time.

The main section consists of a cutterhead, cutterhead support with integrated ring gear, main bearing, and drive units. Thrust cylinders are located in the rear part of the machine and act against the tunnel lining. Complete lining rings are installed inside the shield by a segment erector.

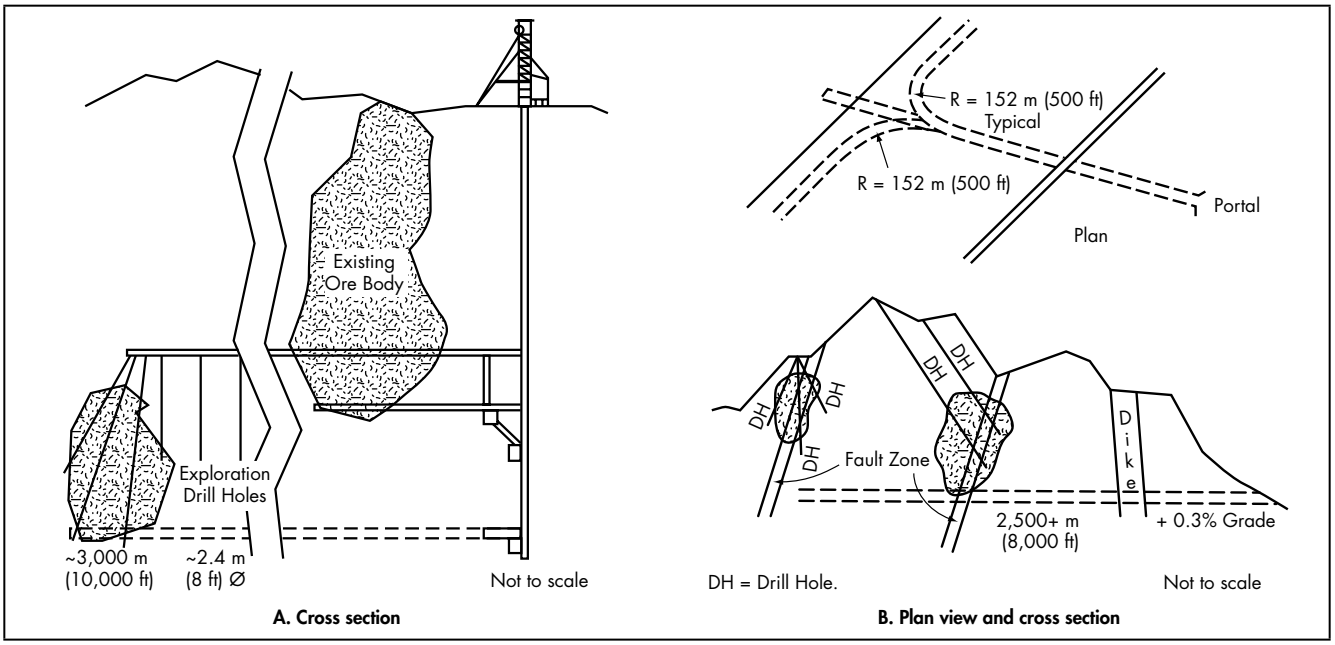
Steering is done by individual control of the oil volume supplied to separate groups of thrust cylinders. Steering can be enhanced by the articulation cylinders, which bias cutterhead support in relation to the shield.

Double-Shield TBM

The double-shield TBM, also called a telescopic-shield TBM (Figure 12.7-7), can be used in hard as well as soft formations. It is used when there is a risk of unstable ground conditions. Normally, the tunnel walls are used to anchor the machine, which allows installation of segmental lining during boring. If the walls cannot take the gripper pressure, the machine can push off the lining instead.

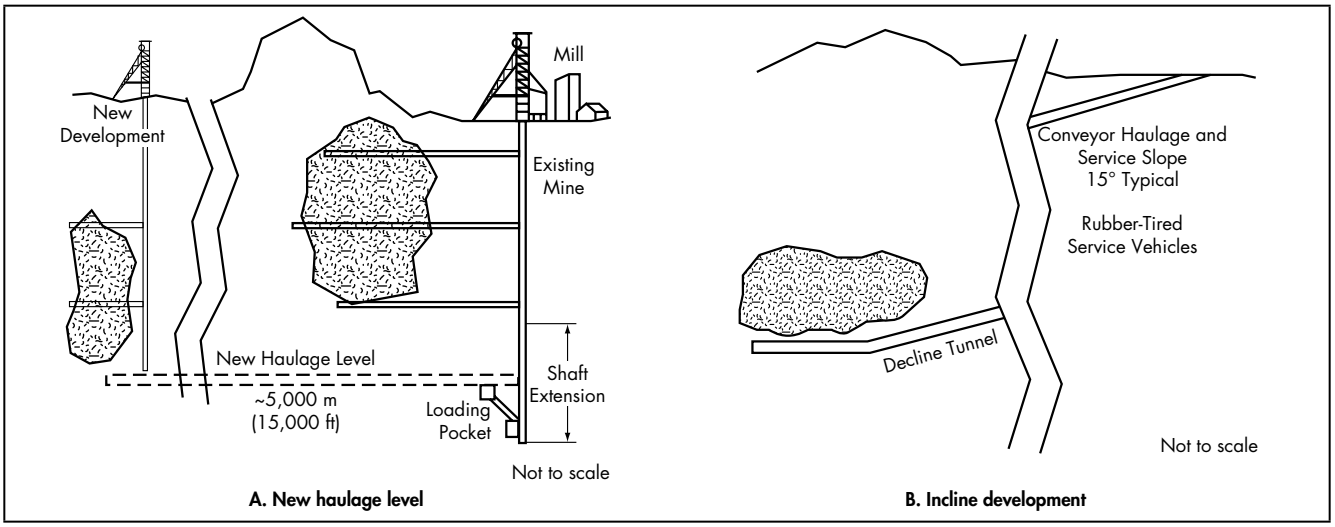
The working section at the front of the machine has a cutterhead, cutterhead support with integrated ring gear, main bearing, and drive units. A telescoping section between the anchoring and working sections includes hydraulic cylinders for thrust and steering. The anchoring section at the rear of the machine has two horizontal grippers that transmit thrust and torque. Auxiliary cylinders transmit these reactions to and propel the machine from the lining. A segment erector is installed inside the tail shield.

Steering is done by individual control of the oil volume supplied to separate groups of thrust cylinders. Steering can be enhanced by the articulation cylinders, which can off-angle



Source: Moulton 1997.

Figure 12.7-8 Mine openings for exploration



Source: Moulton 1997.

Figure 12.7-9 Mine openings for haulage by rail, rubber tire, or conveyor

the working section in relation to the anchoring section to bias the cutterhead in the proper direction.

If segment lining is being placed, an electronic guidance system is mandatory to ensure correct line and grade for installation of the segment ring within the tail shield.

TBM MINING APPLICATIONS

Underground openings in mines can be associated with exploration, access to and development of ore bodies, haulage (by train, conveyor, or rubber tire), ventilation, drainage/dewatering, movement of supplies and labor, safety, or a combination of these. Figures 12.7-8 through 12.7-13 show examples of some underground openings in mines.

A key consideration for TBM mining applications is the type of muck-removal system. Tunnels under consideration for TBM application are generally close to mine boundaries, because only long-distance tunnels could justify capital investments in TBM equipment. Tunnel cross sections are generally relatively small, because muck must be transported and hoisted without interrupting full-time mining.

Haulage of muck has historically been done by rail. Rail haulage is often the lowest option in terms of capital and operating costs and still reasonably flexible, but conveyor haulage is also common today, especially for longer tunnel drives. However, conveyor systems are not always suitable for TBM applications, because their minimum turning radius is ~200 m

(660 ft), while a custom-designed TBM may be able to bore a curve radius of ~100 m (330 ft).

Haulage by rubber-tire vehicles may be considered for larger tunnels with diameters of ~7 m (23 ft) or more. The invert is then typically backfilled with concrete or good-quality rock materials to provide a roadbed. Vehicle passing stations may be required at intervals to maintain a reasonable TBM utilization.

Another consideration for TBM mining applications is incline and decline. Declines have been limited to a maximum grade of ~15°, however, because a typical muck-removal system uses conveyor belts. Inclines can be quite steep since mucking can take place as safely contained gravity flow along the invert. In civil works, inclines have exceeded 45°.

Alignments are further determined by the end use of the excavation, geologic conditions, and other management or owner constraints such as property boundaries.

TBM VERSUS DRILL-AND-BLAST EXCAVATION METHODS

TBM excavation methods have both advantages and disadvantages compared with conventional drill-and-blast methods.

TBM Advantages

High Rate of Advance

Instantaneous excavation rates of 1–6 m/machine hour (3–20 ft/machine hour) are regularly achieved by TBMs with diameters of 3.5–5.5 m (11.5–18 ft). Overall advance rates per day, week, and month depend on the overall TBM system utilization and work schedules. Faster ore-body development shortens the time period of capital use and capital cost.

With TBM use, fewer headings are operated at one time, making supervision and planning easier and management effort more concentrated and focused.

Some types of geological environment can be more effectively supported if support is installed relatively quickly after excavation.

No Blast Vibration or Blast Fumes

TBMs enable work to be carried out continuously in the heading without delays for clearing gases from blasting from the heading or ventilation system. No workers need to be removed from other headings or work places to safety refuges for a blasting cycle. However, other drill-and-blast work in the mine can cause interruption in the TBM heading.

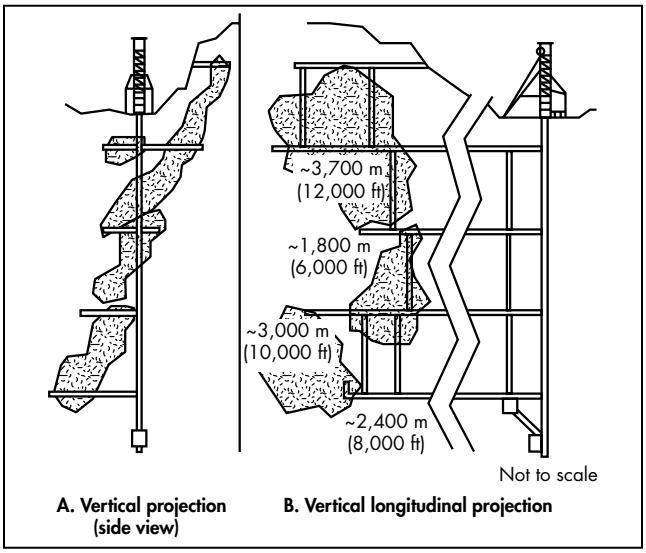
Reduced Ground Disturbance and Support

Mechanical excavation is well known to be less destructive to the structural integrity of material surrounding the excavated opening than is drill-and-blast excavation. Less ground support is required with significant reduction in material costs and installation time.

Roof bolts and steel ribs are the two predominant support systems used with TBM excavation in mines. Yieldable steel sets have commonly been employed in deep German coal mines for TBM drifts. Installation of ground support in larger tunnel cross sections can largely be automated.

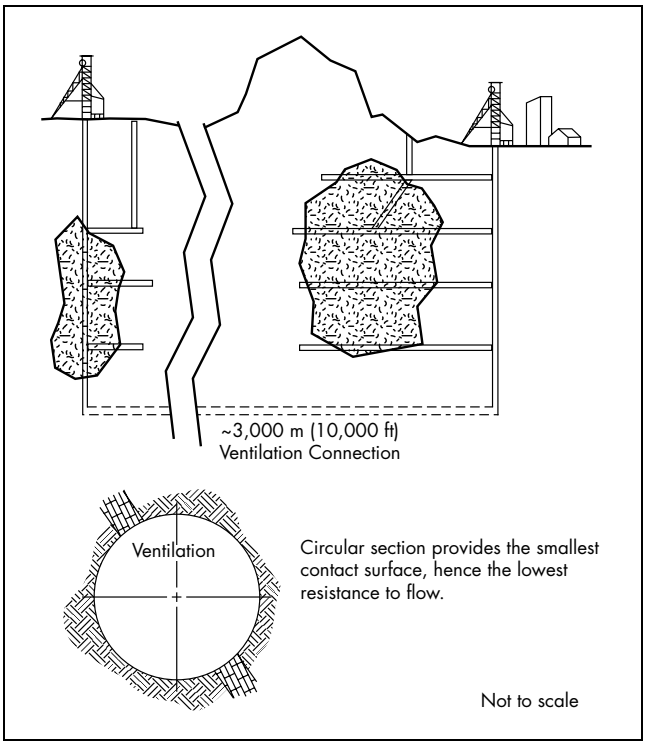
Reduced Lining Costs

Where a completed excavation must be lined with shotcrete or concrete, the smooth uniform surface created by TBM excavation reduces costs of the lining material and raises the possibility that lining installation can be automated.



Source: Moulton 1997.

Figure 12.7-10 Mine opening drifts for development access and haulage



Source: Moulton 1997.

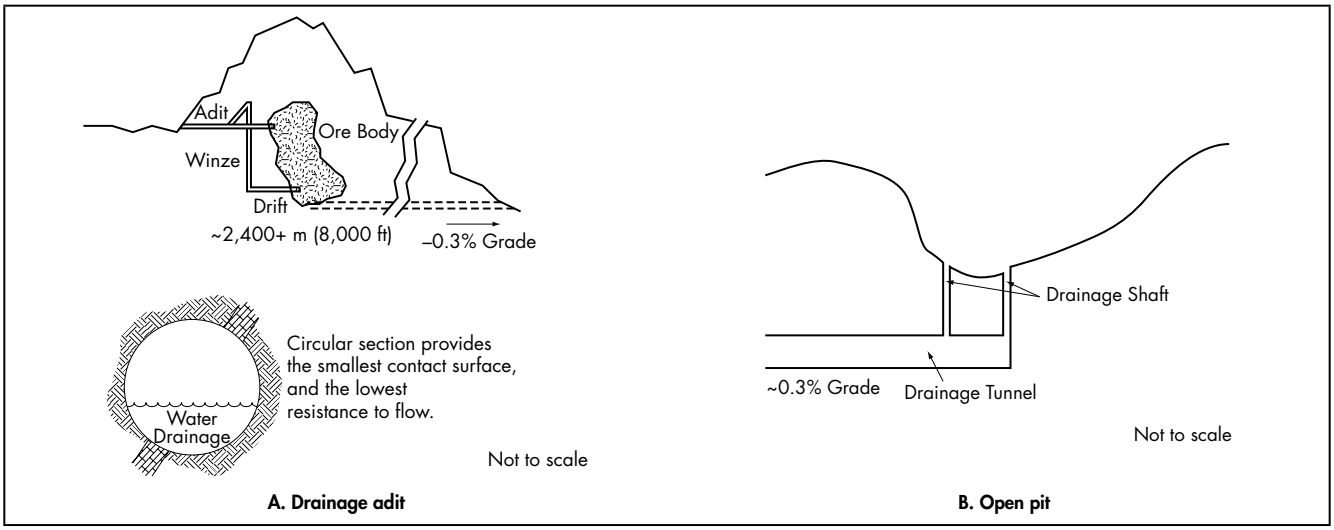
Figure 12.7-11 Mine openings for ventilation

Uniform Muck Size

Mechanically excavated materials essentially all pass through a 100-mm (4-in.) grizzly used for coarse screening.

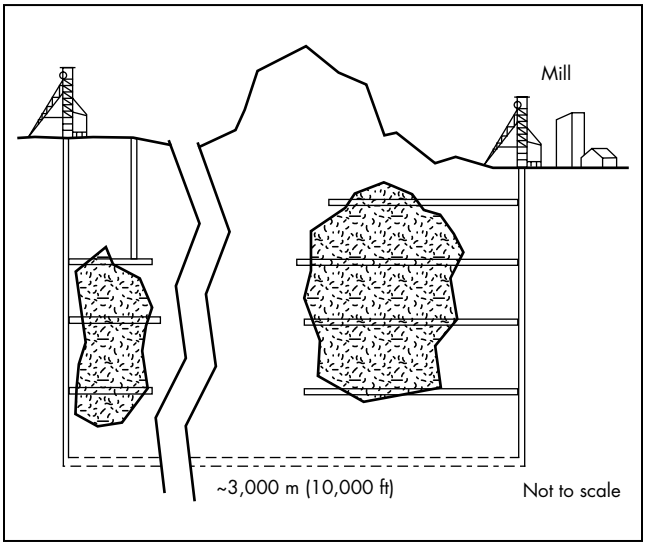
Suitable for Automation

The uniform size of mechanically excavated materials makes material handling ideal for automation. Less wear and tear occurs on transport equipment such as rail cars, trucks, or conveyors in the tunnel.



Source: (A) Moulton 1997; (B) Courtesy of The Robbins Company.

Figure 12.7-12 Mine openings for water drainage



Source: Moulton 1997.

Figure 12.7-13 Multiple-use mine openings for ventilation, safety, drainage, haulage, and supplies

Vertical transport by means of loading pockets and hoist conveyances can be automated, minimizing maintenance costs. Ore that is encountered in a heading can be partially segregated and sent to the concentrator, eliminating primary crushing costs.

Reduced Labor Costs

A rapid rate of advance reduces the unit cost of excavation. In most western countries, both crew size and rate of advance are larger for TBM than for drill-and-blast operation. Consider the following typical average progress for a tunnel of small cross section, ~10 m^2 (108 ft^2), each with two 10-hour work shifts per day:

- TBM operation:
  - Crew size: 4 to 7 (assume 6 for the following calculation)
  - Rate of advance: ~140 m (460 ft) per week
  - Productivity: 140 m/week ÷ 72 shifts/week = 1.94 m (6.4 ft) per shift
- Drill-and-blast operation:
  - Crew size: 2 to 3 (assume 3 for the following calculation)
  - Rate of advance: ~38 m (125 ft) per week
  - Productivity: 38 m/week ÷ 36 shifts/week = 1.06 m (3.5 ft) per shift

Good Safety Record

TBM excavations are considered relatively safe compared with drill-and-blast excavations because the latter incur greater risk to workers from falls or ground-related accidents, even in supported ground. Most TBM systems have protective canopies for workers in ground support installation work zones.

Both excavation methods involve some heavy lifting, which, if not properly performed, can lead to back injuries. A typical 432-mm (17-in.) single-disc-cutter assembly weighs about 136 kg (300 lb). Back-loading cutters normally required in mining applications are best changed with slings and hoists using proven cutter-handling systems. Back-loading cutters can normally be used for TBMs with diameters of ≥4 m (13 ft).

Lower Manning's Factor

TBM excavation in hard rock leaves a smooth circular profile with a roughness coefficient that is lower than that for a typical drill-and-blast operation. The cross section of an unlined TBM tunnel therefore needs to be only about two-thirds of the area of an unlined drill-and-blast tunnel to carry the same water flow.

Circular Cross Section

A circular profile typically gives the most stable cross section. TBM excavation requires substantially less ground support in fairly massive to massive hard-rock conditions.

Improved Ventilation Characteristics

The smoother rock surface of a TBM excavation means reduced friction and improved possibility of excavating longer drives from a single heading.

Less Concrete When Lining Is Required

A TBM tunnel has substantially less overbreak than does a blasted tunnel. Large savings in concrete and concrete placement may be realized.

No Muck Bays or Passing Bays

No muck bays or passing bays are needed for TBM tunnels with diameters of >3.5 m (11.5 ft) because all advance is centerline advance.

TBM Disadvantages

High Capital Cost

TBM costs vary widely with size, supply, and power requirements. The cost of a typical open-type TBM excluding roof drills, ring erectors, and other ground support or probing equipment is about US$1.7 million per meter of diameter (2009 price levels). A simple backup system in a heading with a diameter of 3–4.8 m (10–16 ft) varies from US$1.2 to $2 million, depending on facilities, storage, muck-transport systems, and so on. A system with a diameter of 3.6 m (12 ft), with spares and cutters, thus costs about US$8 million. A used, fully reconditioned, technically comparable system costs about 65%–75% of this amount.

Repair and maintenance work are necessary during the life of a machine. The life of a TBM system has been suggested at times to be ~15 km (50,000 ft) of tunnel excavation, although this varies widely depending on technical obsolescence, operating and maintenance practices, and application. Conditions in the marketplace external to the mine can have a large impact on equipment life and salvage value. A number of TBMs have bored more than 25 km (80,000 ft) of tunnel. Generally at least 2.5 km (~8,000 ft) of tunnel are necessary to justify the use of a TBM system; this includes equipment salvage value.

Large Value of Spares and Cutter Inventory

The recommended inventory of major spare parts, including main bearing, ring gear, drive unit (motor, clutch, gear reducer, and pinion), propel cylinder, and gripper cylinder are in the range US$600,000 to $800,000 (2009 price levels). Inventory of minor spare parts adds another US$60,000 to $100,000 of capital cost. US$300,000 can be spent for initial cutter mounting, cutter spares, and cutter repair tools and fixtures.

Long Lead Time for Delivery

Design and manufacturing time for a new TBM with a diameter of 3.6–5.5 m (12–18 ft) is typically a minimum of 10 months. The time for a smaller TBM, with a diameter of ~2.6–3.0 m (8–10 ft), may be as low as 8 months. Delivery time ex-works varies with actual supply, clarity of product definition, and subsuppliers' workloads.

Reconditioning and modification times for existing equipment is ~65%–95% of these values, depending on the particular equipment, its location, and the scope of work required.

Large Components That Are Difficult to Transport

To reach the high thrust and torque levels required for TBM excavation, components must be large and heavy, and are therefore difficult to move through existing mine shafts or mine drifts. TBMs are also often difficult to remove from completed tunnels.

Boring of TBM drifts from shafts can be challenging. Transport from the surface and assembly underground, as well as removal from the mine, require careful planning and preparation. At times transport and assembly must take place without interrupting full-time mining. Plans and methods for slinging and lowering heavy components down the shaft must be devised early during machine design. Machine components and support systems must be designed to be small and light enough for accommodation in the shaft and by the hoist. The existing hoist configuration, controls, or brakes may also require modification, and special transport or handling devices may be required to lower large components down the shaft or move them along the level.

Limited Range of Application in Varying Ground Conditions

TBM design is determined by the requirements of the particular application—the more uniform or consistent the geologic environment, the easier and more reliable the design.

Competent moderate hard rock, requiring moderate ground support and with no major water inflows, is ideal. However, mines are normally associated with geologic nonconformities that result in mineral deposits, and it is to be expected that all or part of an underground excavation will require some type of rock support. Machine design can, within limits, accommodate hard rock to caving or squeezing ground.

For mining applications, a high degree of flexibility is generally desirable. Today's machines can operate in varying ground conditions because of their variable-frequency drive motors for controlling cutterhead rpm and torque. They can be equipped with effective systems for probing, pre-excavation grouting, shotcreting near the cutterhead, rock bolting, ring beam installation, and application of mesh and lagging. In addition, they can be furnished with the McNally system, a longitudinal roof-support system for operation in bad ground conditions (McNally 2010).

Consumable Costs Can Be High in Hard, Abrasive Rock

Cutter costs can be a very significant part of the overall TBM costs. In soft to medium-hard sedimentary rocks such as are associated with limestone and other nonabrasive rock types, cutter costs are typically as low as US$1.00/$m^3$ ($0.75/$yd^3$) or less. However, in hard, massive quartzites, they can reach US$20/$m^3$ ($15/$yd^3$) or more due to the high abrasiveness of the rock combined with low instantaneous penetration.

When cutter costs are high, machine utilization drops because of downtime for cutter changes and for cutterhead maintenance and repairs.

Several laboratories can test rock samples needed for TBM application studies. Most TBM manufacturers and some TBM consultants can provide estimates on instantaneous penetration and cutter consumption based on these lab test data, anticipated rock-mass characteristics, and proposed machine and cutter specifications.

Limited to Circular Cross Section

When a heading is intended for haulage, the circular cross section produced by a TBM may limit the choice of haulage method and equipment. This is particularly true when rubber-tired haulage trucks or load-haul-dump units are to be used. The diameter of the excavation must be increased and the invert may require a concrete cast floor as a roadbed.

In some rock burst conditions, a TBM-produced circular cross section, although usually advantageous for ground support, can also be a detriment. In highly stressed and brittle rocks, the surface stress of the circular opening can be quite

high compared to that in rock that has been deeply fractured and stress-relieved by use of drill-and-blast methods.

Limited Turning Radius

A typical TBM has a minimum turning radius of ~200 m (660 ft), although custom-designed TBMs may be able to bore a radius of ~100 m (330 ft). Backup equipment, conveyors belts, mining cars, mucking solutions, and so on must all be able to negotiate a curve so created. In contrast, a drill-and-blast operation has basically no limitations regarding curve radius.

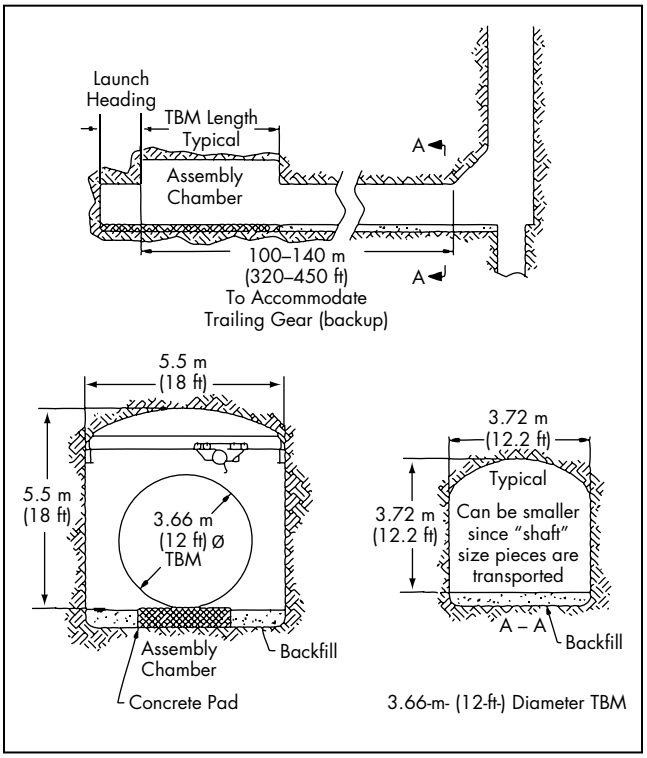
Substantial Preparation at Work Site

Site preparation for most mine projects that involve adits is not unusually difficult. However, for a TBM application, there must be ample room at the portal site, cranes for lifting large pieces, and adequate supply of power. Other requirements are basically similar to those for drill-and-blast headings.

When an underground work station is planned, adequate hoist capacity and ample length for assembling the trailing gear are needed. A typical overall length for a TBM with a diameter of 3–5 m (10–16 ft) can be 100–140 m (320–450 ft). The underground assembly chamber for a 3.66-m (12-ft) TBM is usually sized as follows (Figure 12.7-14):

- Width and height: ≥1.5 times the TBM diameter, depending on the type of hoist
- Length: Length of the TBM, ~20 m (65 ft)

For safety and cost reasons, it is desirable that the assembly chamber be constructed in good ground, close to the launch point for the drive.



Source: Moulton 1997.

Figure 12.7-14 Example of an underground TBM assembly chamber

Requires Substantial Crew Training and High Mechanical and Electrical Skill Levels

A TBM system is essentially a crude moving processing plant or factory. Material is removed and sized in a cutting process from its in-situ state. The process from that point on involves several electrical, mechanical, hydraulic, lubrication, compressed-air, cooling-water, wastewater, ventilation, material-handling, and storage systems along with monitoring and control of these systems. These systems must be maintained in a hostile environment of water, dust, vibration, and varying temperature. Some of these systems may not be in common use with other machines or systems in the mine operation.

Mine operating personnel must be trained in the use and maintenance of these systems and how they relate to the overall TBM production process. Training is largely accomplished in the TBM manufacturer's shop during floor testing and machine disassembly, and at the work site during final assembly and full-load testing. TBM operators receive training by visiting or temporarily working at other sites and by working with the TBM supplier's field service personnel.

Instead of training mining crews, a better solution can be to use contractors who specialize in TBM work. A TBM contractor must be carefully selected and must understand mine priorities and operations requirements.

Requires Large Power Supply

Most TBM systems today use total power in the range 1,000–2,600 kVA or more for TBMs with diameters of 3–5 m (10–16 ft). Total power includes power for cutterhead, hydraulics, ventilation, conveyor, lighting, pumping, cooling, and other systems.

The power required to operate a TBM system may not have been planned in the mine's initial design stage, as many mines are more than 25 years old. Even today, mine infrastructure planning does not usually include provision for the use of mechanical excavation equipment in major headings. The large power requirement usually necessitates addition of high-voltage distribution cables in the shaft and addition of underground substations.

Vary Excavation Diameter

Normally, it is not practical to change tunnel diameter during a drive, because of costly downtime. The TBM design itself may allow a change of ~1 m (3.3 ft) or more from a minimum established diameter. Possibilities with regard to diameter increases should be based on the original TBM design criteria and the upcoming project application, including desired diameter, length of drive, and rock-mass and rock-strength characteristics.

DESIGN AND OPERATING PROBLEMS AND SOLUTIONS

Historically, design and operating problems have arisen with TBM applications in the mining industry. Samples of past problems and their solutions are given in Table 12.7-1.

Since 2000, several important improvements have been implemented in TBMs and auxiliary equipment that decrease downtime and increase TBM utilization:

- Water-cooled motors and hydraulics
- Larger-diameter cutters
- Higher-capacity cutter bearings and cutter ring with improved abrasive resistance characteristics

Table 12.7-1 TBM design and operating problems and their solutions

Problem	Solutions
Ground fall on personnel between cutterhead and face while changing cutters	• Developed back-loading cutters that can be changed from inside the cutterhead • Replaced bucket lips from inside the cutterhead
Undercutting of an unstable face with dome cutterheads, causing collapse of the tunnel crown	• Developed flat-face cutterheads with smooth surfaces
Front-mounted cutter housings rip blocks of material from the face, blocking cutterhead rotation	• Developed flat-face cutterheads with smooth surfaces
Large distance from initial cutting of face to provision of adequate ground support, causing fallout and caving. Massive ground support effort required	• Developed flat-face cutterheads with short standoff distance and narrow rim bar to reduce the distance to the first adequate support of the excavation
Methane fire	• Implemented early detection of gas and shutdown of equipment • Increased ventilation • Implemented better sensing of hot materials, particularly cutters • Interlocked water sprays with cutterhead rotation
Probe drilling in advance of boring limits advance	• Implemented state-of-the-art high-capacity hydraulic drills that enable probe drilling during cutter changes and/or TBM system maintenance
Additional machine waste heat in deep mines detrimental to work-crew efficiency	• Upgraded to water-cooled electric motors, gearboxes, and cutterheads
Low utilization in hard abrasive rock in deep mines	• This is not a good application for TBMs. More research and development is needed, particularly for cutters
Entrapment by squeezing ground	• Implemented adjustable shield designs • Avoided stoppages or delays for weekends or holidays in zones of squeezing
Inadequate support for gripper pads	• Increased gripper shoe area to reduce ground load • Implemented timber cribbing or shotcreting in gripper-pad areas as necessary
Running ground cave-in	• Used higher torque at low speed on cutterhead with ground treatment • Applied shotcrete early to stabilize ground
High repair and maintenance of open-type cutter mounting	• Added breasting plates to smooth rock-contact surface • Designed and built new cutterhead
Equipment not suitable for geology	• Modified TBM to meet conditions
Small size of TBM limits workspace for ground support installation	• Installed shield for temporary support and placed permanent support behind TBM
Cannot hold curve-alignment radius	• Implemented modern laser guidance systems • Designed a short TBM • Reduced TBM stroke • Implemented steering by means of "crabbing"
Large blocks of rock have damaged TBM muck chute and conveyor system	• Installed grill bars • Decreased size of bucket openings • Replaced bucket lips from behind • Lowered speed and increased torque
Plugging of muck-flow passages in the cutterhead	• Removed grill bars • Increased size of bucket openings • Smoothed the contact surface • Used foam injection to lubricate muck
Ground caving above cutterhead	• Stabilized the ground by grouting and filled the void with quick-setting grout • Installed spiling or combinations of the above • Installed a McNally roof-support system

Source: Adapted from Moulton 1997.

- High-efficiency variable-frequency electric drives
- Improved hydraulic probe-drill schemes and pre-excavation grouting solutions
- Faster-setting grout

OPERATIONAL AND MANAGEMENT PROBLEMS AND SOLUTIONS

The following typical challenges can arise in a TBM mining application:

- Failure on the part of mine managers, due to categorization of the TBM as a "mine research project," to commit fully to the decision to use a TBM
- Failure on the part of direct supervisors and miners to commit fully to the success of the TBM system
- Failure to obtain acceptance of new technology due to unrealistic expectations
- Failure on the part of machine suppliers to fully understand the unique mine operating conditions and priorities related to design challenges, operating costs, performance, and the high skill levels required of personnel
- Lack of jobsite preparation and coordination with other mine activities
- Lack of training of operating and maintenance personnel

Several TBM projects have been considered failures when original schedules, costs, and full desired end benefits were not achieved. However, most had acceptable results when compared to realistic alternates. The solution to these

Table 12.7-2 Summary of TBM projects, listed by original machine manufacturer

Mine	Year	Model	Bore Diameter	Length, m (ft)	Geology
TBM Manufacturer: Robbins*					
Thetford mine asbestos, Quebec, Canada	1957	103-105	2.74 m (9 ft 0 in.)	30 (100)	Greenstone
Step Rock iron mine, Quebec, Canada	1957	103-105	2.74 m (9 ft 0 in.)	300 (1,000)	Limonite
Homer-Wauseca mine, Iron River, MI, USA	1963	71A-109	2.13 m (7 ft 0 in.)	355 (1,163)	Limestone & iron
White Pine mine, White Pine, MI, USA	1968	181-122	5.48 m (18 ft 0 in.)	2,590 (8,500)	Shale
Milchbuck mine vent, Switzerland	1969	104-120-1	3.22 m (9 ft 11 in.)	91 (300)	Greenstone
Chingola, Zambia	1970	125-135	3.65 m (12 ft 0 in.)	3,200 (10,500)	Granite
Minster Stein Coal, Germany	1971	163-136	5.4/5.1 m (16 ft 9 in. / 16 ft 0 in.)	6,797 (22,300)	Shale & sandstone
Schelklingen mine Addition, Germany	1971	111-117	3.5 m (11 ft 6 in.)	1,240 (4,068)	Coal measures
Libanon gold mine, Gold Fields, South Africa	1974	114-163	4.5 m (15 ft 2 in.)	Not applicable	Quartzite
Mt. Lyell mine, Australia	1975	133-146-1	3.93 m (12 ft 10 in.)	3,048 (10,000)	Andesite & schist
Monopol Coal, Kamen/Westfalen, Germany	1976	163-136	5.4/5.1 m (16 ft 9 in. / 16 ft 0 in.)	12,802 (42,000)	Schist & sandstone
Blyvoor mine, South Africa	1977	61-176	1.84 m (6 ft 6 in.)	305 (1,000 multiples)	Quartzite
East Driefontain mine, South Africa	1977	61-177	1.84 m (6 ft 6 in.)	305 (1,000 multiples)	Quartzite
Westfalen Coal, Germany	1979	202-201	6.1 m (20 ft 0 in.)	12,701 (41,670)	Sandstone, shale
Blumenthal mine, Germany	1979	214-202	6.5 m (21 ft 4 in.)	10,600 (34,780)	Sandstone, shale
Selby mine road, York, UK (decline)	1981	193-214	5.8 m (19 ft 0 in.)	13,600 (44,620)	Sandstone, mudstone, coal seams
Sotiel mine, Huelva, Spain (decline)	1981	122-126-1	3.81 m (12 ft 6 in.)	1,774 (5,820)	Shale & phyrites
Stillwater mine, Nye, MT, USA	1987	146-193-1	4.1 m (13 ft 6 in.)	9,673 (31,737)	Gabbro and Norites
San Manuel copper mine, AZ, USA	1995	156-275	4.62 m (15 ft 2 in.)	9,754 (32,000)	Granodiorite, quartzmonzonite dykes
Stillwater mine, Nye, MT, USA (East Boulder Project)	1998	156-275	4.62 m (15 ft 2 in.)	5,650 (18,550)	Gabbro, anorthosite, norite
Da Tong coal mine, Shanxi, China	2007	154-273-1	4.90 m (16 ft 1 in.)	2,800 (9,186)	Sandstone, siltstone
OTM mine, Papua, New Guinea (dewatering tunnel)	2008	1410-251-3	5.56 m (18 ft 3 in.)	5,444 (17,861)	Metamorphic, sandstone, siltstone
TBM Manufacturer: Wirth†					
Sophia Jacoba, Aachen, Germany	1972	TB V-540	5.30 m (17 ft 5 in.)	137 (450)	Sandstone
Consolidation, Ruhr, Germany	1972	TBII-300HT TBE 300/530	3.00 m (9 ft 10 in.) 5.30 m (17 ft 5 in.)	1,600 (5,250) 1,600 (5,250)	Sandstone
Anglo American Free State Geduld, Republic of South Africa	1976/1979 1982/1984	TBS II-340H	3.40 m (11 ft 2 in.)	1,150 (3,772) 2,280 (7,480)	Quartzite/granite Quartzite/sericite
Lohberg I, Osterfeld Dinslaken, Germany	1982 1983/1985 1986	TBS V-650	6.50 m (21 ft 4 in.)	5,455 (17,897) in total	Coal measures
Lohberg II, Osterfeld Dinslaken, Germany	1988/1990		6.50 m (21 ft 4 in.)	3,770 (12,369)	Coal measures
Lohberg III (used the same TBM as Loberg II at a different diameter), Germany	1992	TBS V-650	6.80 m (22 ft 4 in.)	2,630 (8,629)	Coal measures
TBM Manufacturer: Società Esecuzione Lavori Idraulici SpA (SELI)‡					
Los Broncos copper mine, Chile	2009–2010	Universal Compact Double Shield	4.5 m (14 ft 9 in.)	8,500 (27,887)	Quartzmonzonite, andesite, ryodacitic porphyry, breccias

(continues)

Table 12.7-2 Summary of TBM projects, listed by original machine manufacturer (continued)

Mine	Year	Model	Bore Diameter	Length, m (ft)	Geology
TBM Manufacturer: Jarva*					
Adirondac mine, 27% incline, Mineville, NY, USA	1967	MK 11-4	3.05 m (10 ft 0 in.)	234 (768)	Magnetite, granitic gneiss
Mather "B" mine, MI, USA	1967	MK 14-1	4.03 m (13 ft 3 in.)	325 (1,065)	Graywacke, shale
Hecla Mining Co. development drift, Wallace, ID, USA	1968	MK 8-3	2.74 m (9 ft 0 in.)	152 (500)	Quartzite
Oak Park mine, incline, Cadiz, OH, USA	1970	MK 14-1	4.27 m (14 ft 0 in.)	549 (1,802)	Sandstone shale
R&P Emilie No. 4 Shelochta, Indiana, PA, USA	1976	MK 12-13	4.27 m (14 ft 0 in.)	227 (746)	Shale, coal, sandstone, limestone
Bethlehem Steel, Grace mine, Morgantown, PA, USA	1976	MK 8-2	2.49 m (8 ft 2 in.)	183 (600)	Chlorite magnetite
R&P, Urling No. 3, Shelochta, PA, USA	1976	MK 12-13	9.27 m (14 ft 0 in.)	745 (2,443)	Shale, sandstone, limestone
Mine development drift, Pachuca, Mexico, USA	1978	MK 11-4	3.66 m (12 ft 0 in.)	1,524 (5,000)	Shale
Westmoreland Coal 30% Decline, Eccles, WV, USA	1979	MK 8-2	3.05 m 10 ft 0 in.)	319 (1,045)	Shale sandstone
R&P coal conveyor haulage tunnel, Indiana, PA, USA	1979	MK 12-13	8.27 m 14 ft 6 in.)	427 (1,400)	Shale, sandstone
United Pocahontas Coal, Thurman, WV, USA	1980	MK 8	3.05 m (10 ft 0 in.)	945 (3,070)	Coal, sandstone
R&P coal trans tunnel, Shelochta, PA, USA	1980	MK 12-13	4.27 m (14 ft 0 in.)	335 (1,100)	Coal, sandstone
Kiena mine, North Bay, Ontario	1986	MK 6-19	1.98 m (6 in.)	1,375 (4,511)	Basalt, peridiotite diorite
TBM Manufacturer: Demag§					
Niederrhein, Germany	1973–1979	TVM 54	6.0 m (19 ft 8 in.)	21,085 (69,176) in total	Schist, sandstone, shale
Victoria, Germany	1977	TVM 54	6.0 m (19 ft 8 in.)	1,700 (5,577)	Schist, sandstone
Petrosani, Germany	1979	TVM 45H	4.8 m (15 ft 9 in.)	13,000 (42650)	Sandstone, sandy shale, schist
Saarbergwerke, Germany	1979	TVM 55H	6.0 m (19 ft 8 in.)	16,000 (52,493)	Schist, sandstone, sandy shale
Franz Haniel, Germany	1980	TVM 54-58	6.0 m (19 ft 8 in.)	8,000 (26,247)	Schist, sandstone, shale
Hans Aden, Germany	1981	TVM 65	6.5 m (21 ft 4 in.)	19,000 (62,336)	Schist, sandstone
TBM Manufacturer: Calweld†					
Climax mine, Climax, CO, USA	1969	TBM 40	3.96 m (13 ft 0 in.)	607 (2,000)	Silver plume, granitic gneiss
TBM Manufacturer: Ingersoll-Rand†					
Magma Copper, Superior, USA	1974	008	4.27 m (14 ft 0 in.)	Not applicable	Not applicable
TBM Manufacturer: Boretec Equipment*					
Falconbridge Fraser mine, Sudbury, Ontario, Canada	1988	Cub 3000	2.29 m (7 ft 6 in.)	300 (984)	Ultra-hard granite
TBM Manufacturer: Construction & Tunneling Services (CTS)**					
Stillwater mine, Nye, MT, USA (East Boulder Project)	1997	460-12	4.57 m (15 ft 0 in.)	5,650 (18,550)	Igneous & sedimentary
TBM Manufacturer: Lovat Tunnel Equipment†					
Donkin-Morien, Cape Breton, Nova Scotia, Canada	1983	RM300RL	7.6 m (24 ft 11 in.)	2x4,000 (2x13,123) Maximum declines at 20%	Sandstone with coal seams
Port Headland, North Western Australia	1999	RME 200SE	5.08 m (16 ft 8 in.)	1,037 (3,402)	Soft soil, limestone

*Data courtesy of The Robbins Company.
†Data from Moulton 1997.
‡Data courtesy of SELI.
§Data courtesy of Mannesmann Demag.
**Data from Moulton 1997 and Alexander 1999.

problems is to keep expectations realistic, fully identify mine requirements, and possess adequate design experience.

A summary of mine projects in the Western world is presented in Table 12.7-2 by date and original TBM manufacturer. The summary may not be complete, but the large number of projects listed should indicate that there is enough experience in the industry to know what to do and what not to do when applying TBMs to mining applications.

Many market studies, in attempting to identify the worldwide need for mines, have estimated the total annual drive lengths of horizontal openings. The total number is very large. Of this total, much is not really suitable for TBM excavation. However, when drive lengths are sufficiently long and when mine owners, operators, equipment suppliers, and contractors focus on a common result that benefits each of them, they can commit to their mutual success and achieve it. A good example of such commitment is the case history that follows.

TBM MINING CASE STUDY

The Magma Copper Company's San Manuel mine in Arizona and the Stillwater Mining Company's platinum mine in Montana, both in the United States, are examples of successful TBM use in mining. Many comprehensive reports concerning these projects are in the public domain. TBM use in the San Manuel mine is discussed here in detail.

Mine Location and Layout

The San Manuel mine is located about 72 km (45 mi) north of Tucson, Arizona. In order to secure future production, it became necessary to develop the Lower Kalamazoo ore body 1,050 m (3,450 ft) below surface for a modern block-caving operation. The base of the mass to be caved would be accessed by drifts on two levels from the existing No. 5 shaft.

Based on a feasibility study, it was decided to use a TBM rather than traditional drill-and-blast methods because of the very tight schedule for starting production in the new ore body. Because of a lack of qualified labor and the short construction time, it was determined that an experienced TBM contractor was needed.

The development layout was designed specifically for TBM use (Figure 12.7-15). The TBM tunnel route consisted of 9,826 m (32,235 ft) of tunnel circling the ore body on three levels: (1) the 3,440 grizzly level, (2) the 3,570 haulage level, and (3) the 3,600 conveyor-gathering drift level.

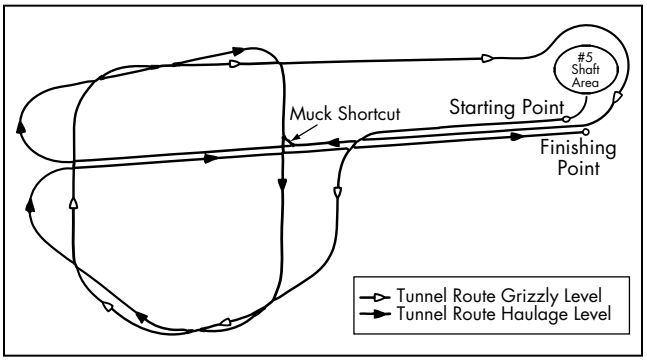
The grizzly level consisted of 3,963 m (13,000 ft) of TBM tunnel with slight grade changes to provide two permanent dewatering sumps. The level has five curves, with radii in the range 152–107 m (500–350 ft) and a total curve length of 1,031 m (3,383 ft).

The haulage and conveyor-gathering drift levels are accessed from the grizzly level by means of two parallel decline tunnels with grades of ≤5.7%, which accounts for the remaining 5,863 m (19,235 ft) of the TBM tunnel route. These decline tunnels have two curves, with radii in the same range as for the grizzly level and a total curve length of 1,255 m (4,119 ft).

Mine Geology

The geology of the San Manuel mine is quite complex, with rock structure ranging from very weak to reasonably strong. The area geology is described briefly as follows (Atlas Copco Robbins 1996).

The ore bodies resulted from porphyritic intrusion of granodiorite at the end of the Cretaceous period into pre-



Source: Janzon 1995.

Figure 12.7-15 TBM tunneling scheme in the San Manuel mine

Cambrian quartz monzonite. Mineralization along the contact zone was shaped originally like a hollow ellipse, which later was split and displaced by faults and intersected by numerous dikes.

The TBM's excavation route included the stable quartz monzonite of ~150–180 MPa (22,000–26,000 psi) unconfined compressive strength, two mineralized haloes, and a mineralized core. The bore path also crossed the San Manuel fault six times and the Virgin fault five times. The San Manuel fault was flat, dipping in an ~1-m- (3-ft-) wide clay zone that followed the bore some 30 m (100 ft) at a time, but it influenced the drive for a total of about 190 m (600 ft). The Virgin fault dipped steeply but a series of related minor faults produced poor rock conditions for about 500 m (1,650 ft) of the tunnel.

In addition, where dacitic, andesitic, and rhyolitic dikes contacted the granodiorite and quartz monzonite, weak zones 0.2–0.6 m (0.5–2 ft) wide affected the bore for approximately 180 m (600 ft). Along most of the TBM's path, hydrothermal metamorphosis had weakened the rock further by veining, fracturing, and jointing. Unsupported wall stability ranged from <30 minutes to months, and the critical zone with the shortest stand-up time was anticipated to be ~170 m (560 ft) in length.

Although none of the rock masses could be considered an "excellent" TBM environment, comprehensive studies and the 1992 geotechnical report (Janzon 1995) concluded that the rock conditions would permit mechanical excavation and that stability, even in the weakest sections, would be adequate for TBM passage and ground support installation. The report actually favored the TBM approach because the TBM machine's low-level vibrations would destabilize hydrothermally altered rock far less than would shock waves from blasting. The report concluded that TBM tunneling would be substantially faster than drill-and-blast excavation.

Mine Equipment Selection

TBM equipment design for the project was based on three parameters: geology, logistics, and tunnel geometry. It was decided to use a hard-rock main-beam TBM.

The TBM was ordered in 1993, before the TBM contractor was selected, in order to meet the construction schedule. Mine management conducted detail studies and held discussions with outside consultants, machine manufacturers, and all potential contractors, then placed the TBM order with The Robbins Company.

The machine (shown assembled at the factory in Figure 12.7-2) is relatively short to allow boring of tight

Table 12.7-3 Specifications for Robbins TBM

Parameter	Value
Boring diameter	4.62 m (15 ft 2 in.)
Cutterhead	
Installed power	1,260 kW (1,680 hp)
Two speed	12.2/4.1 rpm
Maximum recommended thrust	7,350 kN (1,650,000 lb)
Cutters	
Number	33 back-loading/front-loading type
Diameter	432 mm (17 in.)
Maximum recommended load	222.7 kN/cutter (50,000 lb/cutter)
Boring stroke	1.5 m (5 ft)
Minimum turning radius	105 m (345 ft)
Electrical system	
Primary power	12.870 V/60 Hz
Secondary power	600 V for main motors 120 V for control circuit/lighting 480 V for backup equipment
Transformer power	2 x 1300 kVA
Weight	225 t (248 st)

Source: Data from Van Der Pas and Allum 1995.

curves. It has back-loading cutters, roof drill fixtures, a ring beam erector for installing W6×20 ring beams, invert thrust system, finger shield, soundproof operator station on the backup, including a closed-circuit TV for monitoring selected areas along the TBM and backup. Specifications for the TBM are listed in Table 12.7-3.

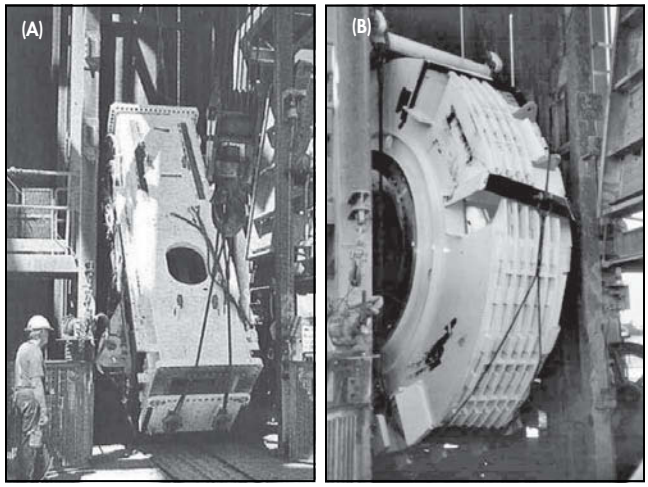
The TBM for the San Manuel mine was custom designed to be lowered down the mine's access shaft piece by piece and assembled underground. The cutterhead and main beam were built in two pieces to meet the space and weight limitations of the access shaft and its 24-t (26.5-st) hoist (Figure 12.7-16). Components were limited in physical dimension to $2 \times 4 \times 12$ m ($6.75 \times 13 \times 39$ ft). More than 65 components were lowered down the shaft. Large components were suspended beneath the service cage; other parts fit inside the cage.

A further requirement was that all transport and assembly operations had to take place without interrupting full-time mining. Assembly of the machine was essentially complete in 114 shift hours, which was proof of excellent planning and a dedicated work crew.

The TBM contract was awarded to Frontier Kemper Constructors, Inc./Daniel Haniel GmbH (FK/DH). The contractor designed and manufactured the trailing gear and support equipment to fit the same logistical criteria and requirements as specified for the TBM. The 143-m- (470-ft-) long backup equipment consisted of 16 frame cars to house all electrical, hydraulic, ventilation, cooling, and muck-handling equipment. Because of the many short-radius curves, the contractor selected rail haulage using Mühlhäuser 10-m^3 (13-yd^3) muck cars and Brookville 27.2-t (30-st) locomotives. A California switch was selected and moved; permanent switches were installed, two on the grizzly level and one on the 3570 haulage level.

Mining Setbacks and TBM Modifications

TBM progress in the initial 1,830-m (6,000-ft) tunnel was not as expected. A number of problems plagued the operation. For example, clay plugged the cutterhead; a large cavity ($h \times w \times l = 9 \times 6 \times 5$ m [$30 \times 20 \times 16$ ft]) developed above the tunnel profile; loose material fell through the finger shield; steering problems developed because of failure to maintain proper alignment in the curves; and some machine problems developed. In addition, stand-up times in highly fractured areas were not as predicted, because rock often turned to rubble immediately or within minutes in the crown or along the tunnel sidewalls.



Source: (A) Atlas Copco Robbins 1996; (B) Courtesy of Gomez International.

Figure 12.7-16 Two TBM components being lowered down the mine's No. 5 access shaft: (A) part of main beam; (B) split cutterhead

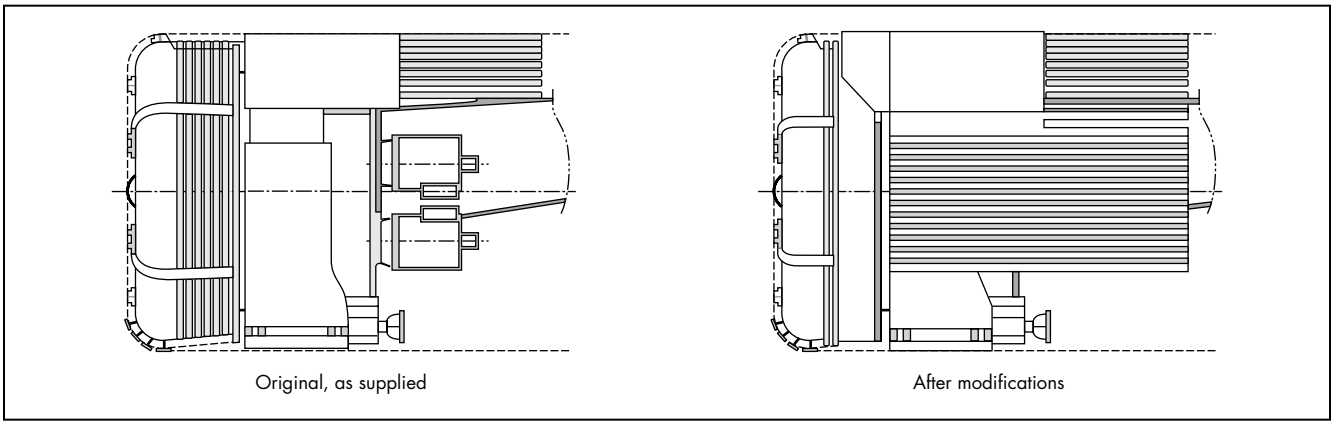
To achieve the planned rate of advance, the TBM needed to be modified quickly. An evaluation team consisting of Magma Copper, the TBM manufacturer, the contractor, and two outside consultants recommended the following modifications:

- Replace two of the gauge cutters with two additional muck buckets and close the peripheral side buckets off to reduce ingestion of rock behind the gauge cutters.
- Extend the roof support forward by a canopy to limit cutterhead exposure.
- Add fingers to the side supports to increase the surface area of the support.
- Improve starting and breakout torque by reducing cutterhead rpm from 12.2 to 9.3.
- Replace the two 1,300-kVA transformers with two 1,600-kVA transformers.

Some of these modifications are shown in Figure 12.7-17.

Early on, because of the highly variable ground conditions, the use of rock bolting had been abandoned, roof drill fixtures had been removed, and the operation had moved to exclusive use of ring beams to support the rock. Doing so not only saved labor but also provided more room in the congested work area immediately behind the cutterhead support.

The modifications greatly improved TBM performance in all areas. Cutterhead starting problems and ingestion of muck from above the cutterhead were eliminated, fallout of material from the sidewall was reduced to acceptable levels, and bucket plugging occurred only in areas of high clay content. A curve with a radius of 109 m (350 ft) was accomplished by means of "crab steering" without plowing the sidewalls. A beneficial side-effect of lowering the cutterhead speed was more efficient transfer of muck from the cutterhead buckets to the muck chute and machine conveyor belt. The results of the



Source: Van Der Pas and Allum 1995.
Figure 12.7-17 Modifications to the main-beam TBM to improve ground control and steering

Table 12.7-4 Measures of advance before and after TBM modifications

	Length of Advance, m (ft)	
Measure of Advance	**Before Modifications**	**After Modifications**
Average daily advance	6.46 (21.19)	22.6 (74.15)
Best shift	19.2 (63)	21.6 (71)
Best day	37.5 (123)	44.5 (146)
Best week	141.7 (465)	263 (863)
Best month	333.5 (1,094)	831.2 (2,727)
Best quarter	882.4 (2,895)	2,142.2 (7,028)

Source: Van Der Pas and Allum 1995.

modifications, measured by comparing the lengths of advance before and after modifications, are shown in Table 12.7-4.

The Magma Copper Company concluded the following (Van Der Pas and Allum 1995):

> Advance rates achieved after completion of the modifications have proven that the decision to utilize a TBM for the initial development of the Lower Kalamazoo was the correct one. Even though the TBM was designed to the best interpretation of conditions likely to be encountered, as so often happens in underground construction work, the conditions encountered did not exactly match the assumptions that were the initial basis for the project. The correct analysis of the problems by TBM operations personnel, the TBM Evaluation Team, The Robbins Company and the consultants resulted in the successful modifications. Careful planning, scheduling and teamwork by operations personnel allowed the modifications to be completed on schedule and as planned....
>
> Many individuals and individual organizations spent a great deal of time and effort to determine what was required to make the project a success. It was the ability of those groups of people to focus on solutions to the problems instead of who to blame that built the foundation to success. The formation of a joint management team enabled both Magma and FK/DH to function side by side while solving the daily coordination problems as created by both organizations. Commitment to a successful project by the owner, the contractor, and the equipment manufacturer created a true working team. That team collectively was able to generate the results needed to prove that TBM technology in a deep underground copper mine can be a success.

CONCLUSIONS AND RECOMMENDATIONS

Excavation of strategic mine openings by TBM methods is safe, fast, and economically feasible when

- Geological conditions are suitable for the intended mechanical excavation method;
- Length and alignment of the heading is amenable to the equipment selected;
- Operating and maintenance personnel are properly trained and committed to the success of the method;
- Mine management is committed to the priority and success of the work;
- Proper equipment, methods, and support supplies are selected; and
- The scope of work justifies the capital expenditure.

TBM manufacturers are prepared to make the following modifications to standard civil-engineering TBMs in order to expand successful use in mines:

- Lower machine capacity to enable consistent daily advances of 10 to 15 m (33 to 50 ft). This results in smaller, lighter TBM components and more room around the cutterhead for ground support.
- Tighten the turning radius to <100 m (330 ft).
- Create a total system that enables continuous concrete invert installation and access by standard mine vehicles, and muck removal by conveyor.
- Facilitate modern ground support techniques that allow loose rock to be held in place and rock support to be installed while boring.

With a good working relationship between the mine and suppliers, as took place at the San Manuel mine, beneficial use of TBMs in mines can be achieved.

ACKNOWLEDGMENTS

This chapter is based largely on work by Bruce G. Moulton, a retired mining engineer and former president of Construction

and Tunneling Services, Inc., who has kindly allowed use of his paper, "Tunnel Boring Machine Applications In Mining" (Moulton 1997). The information herein reflects technical information, project summaries, and price levels as of October 2009.

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CHAPTER 12.8

Underground Ore Movement

Martyn Bloss, Paul Harvey, Don Grant, and Cameron Routley

INTRODUCTION

This chapter provides an overview of the common ore-handling system components available to underground mining operations, including loaders, trucks, orepasses, crushers, conveyors, rail haulage, hydraulic haulage, and other systems. Shafts are described in more detail in Chapter 12.4.

Context Within the Underground Mine

The ore-handling strategy has considerable impact on the value of any underground mining operation. The definition of *ore handling* is any process that moves or transforms ore from one state in the overall value chain to the next more valuable state. In this context, every major process in the mine can be considered ore handling, including those that do not physically move ore. These include resource drilling, mine planning, mine development, production drilling, production blasting, and backfilling. As such, although this chapter mainly considers the physical movement of ore, it is only one component of the total ore movement. It is, therefore, important to consider a more holistic and integrated view when developing the physical ore-handling strategy, because this strategy is most likely to be influenced by (and complementary to) all value-adding processes within the mine, whether physical or not.

Poor ore-handling decisions can lead to underperformance of the mine, high production variability, declining performance over time, and unexpected costs. In addition, opportunities could be lost if little consideration has been given to future changes in mining strategy or expansion options. Value of the business could also be reduced if the ore-handling system is not flexible enough to respond to external influences such as changes in metal prices, technology, and license-to-operate conditions. A successful ore-handling system will demonstrate

- Planned variability within limits over the expected life of the operation,
- Practical and effective maintenance programs to maintain high equipment availability without creating long-term throughput bottlenecks,
- Good cost management, and
- Optionality to respond to changes in business strategy and external influences.

Because an ore-handling system can be the most inflexible component of the overall mining system, a mine's ability to adapt to future business conditions is linked directly to its ore-handling system's ability to respond to such changes. Planning for the future should therefore be a significant consideration in any ore-handling system design process.

Waste handling also needs to be considered as an integrated part of the ore-handling strategy. Because business value impacts (e.g., costs, destination, downstream processing) associated with handling ore and waste will be different, the two material streams cannot be considered in isolation. For example, if the financial cost of disposing waste separately exceeds the cost of including it in the ore stream, then the waste should be included in the ore stream.

Strategy

With reference to Figure 12.8-1, each step in a mining system value chain consists of an upstream buffer (stockpile) of ore, an ore-handling process that upgrades the ore to another value state, and a downstream buffer. The downstream buffer then becomes the upstream buffer to the next ore-handling process. A real system may be more complex in some parts than the simple series value chain shown in Figure 12.8-1, but the linkage between buffers and ore-handling processes remains unchanged.

Bottlenecks

Based on the theory of constraints (Goldratt and Cox 1992), any manufacturing process contains a single bottleneck, or a

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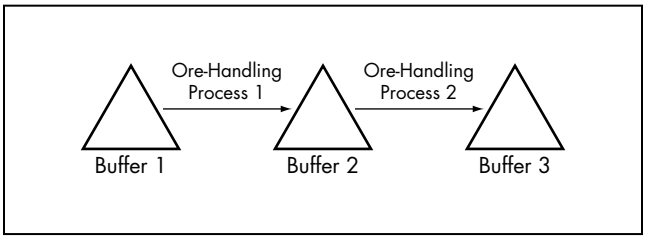


Figure 12.8-1 Linkage between ore-handling processes and supporting buffers

process that controls the output rate. By definition, the bottleneck has the lowest rate of production. Controlled systems have a clearly identified bottleneck that is stable, meaning it does not migrate from one process to another over time. Because of their predictability of performance, controlled systems are more amenable to cost efficiency and continuous improvement, whereas uncontrolled systems have high throughput variability and significant unplanned interruptions.

The process of developing a controlled ore-handling system is to

- Identify where the bottleneck is (or should be); and
- Establish capacities and rates for upstream and downstream buffers and ore-handling processes, respectively, in order to stabilize the bottleneck at the designed location in the value chain.

When this is achieved, the overall performance of the underground mine can be managed to achieve a desired outcome. If throughput is fixed, then cost efficiencies can be achieved by ensuring that only the minimum work required to maintain the buffers at the desired levels is undertaken. If throughput is to be increased, efforts should be focused on maximizing the rate of ore movement through the bottleneck, because this is the rate-limiting step in the overall value chain. Resources allocated elsewhere in the mine will not impact overall throughput. As the rate through the bottleneck increases, its location may migrate to another ore-handling process. At this stage the exercise is repeated at this new location, with the establishment of new buffer capacities and so forth, and an ongoing drive to increase the rate through the new bottleneck. This is continued until a new throughput level is achieved.

For an operation to run effectively there must be a single, well-defined bottleneck. Attempting to match all rates of the process will only lead to poor control of the overall operation, as the bottleneck will shift in response to short-term variability in the performance of each individual process.

The choice of the most appropriate bottleneck location depends on many factors. Foremost is whether the mining operation is currently in an operating state or the design phase. For a new operation, there is an opportunity to optimally locate the bottleneck from the start of the project. Ultimately, the constraint on mine throughput is the resource footprint and the rate at which this resource can be exploited. However, from a valuation perspective (e.g., net present value), sensitivity analyses for various operational configurations should be undertaken to establish the best location for the bottleneck. As a guide, the process in which the capital cost per ton of incremental ore is the greatest should be considered as optimal for locating the bottleneck. This is because over time it is likely that the bottleneck will naturally migrate to this location through continuous improvement of lower capital cost processes, so the sooner this is established, the greater the opportunity to bring forward (in time) positive cash flows to the business. This will not always be the case, however, because a good business case may be made for installing latent capacity initially in high capital items if expansion is planned in the short to medium term and the cost (in present value terms) of expanding these capital items is far greater at a later date. Essentially, the appropriate choice of bottleneck needs to factor in both initial and future plans for the operation.

If the operation is currently in production, an initial choice for the bottleneck is the most practical location to stabilize production in the short term. This allows control to be established and an effective process of debottlenecking to commence. Without this initial control, frequent shifts in the bottleneck will make any attempt at continuous improvement difficult to establish and progress. At the same time, a vision of the bottleneck's long-term location should be established, together with a plan for gradually migrating the bottleneck to this location. Once again, sensitivity analysis of various configurations should be undertaken to establish this optimal position, with a likely outcome being the process that has the highest capital cost per ton of incremental ore.

By definition, because the bottleneck is the only process that is 100% utilized at any given time, nonbottleneck processes cannot be 100% utilized, unless buffer capacity is unlimited and high-cost stock buildup was not being effectively managed. Therefore, nonbottleneck processes must cycle through periods of high and low utilization, according to the buffers' capacities. This is a difficult concept to manage in a mining operation where high utilization is demanded for all equipment and people. One option is to manage buffer levels between high and low trigger values, by deliberately under-resourcing with permanent equipment and people, and employing contractors for fixed periods of time when required. In that way, utilization of the mobilized resources is always high. This concept is described in Figure 12.8-2.

Bloss (2009) describes a practical example where the theory of constraints was applied to the underground mine at Olympic Dam in South Australia to increase throughput by 18%.

Buffers

As described in Figure 12.8-1, an effective ore-handling strategy needs to include the application of buffers between all processes in the ore-handling stream. Buffers exist in a number of forms; for example, as stockpiles, storage bins, orepasses, broken ore stocks within stopes, drilled and developed ore stocks, ore reserve, and mineral resource. They exist between two processes in the value chain in order to provide surge capacity so that variability in performance of either process can exist with minimal impact on the other process. Buffers are particularly important in order to protect the bottleneck, as, by definition, lost production through the bottleneck is lost to the system and cannot be recovered. As a default starting point, it is good practice to assume that a buffer is required between every pair of processes and that any decision not to provide a buffer is justified explicitly as part of the mining strategy. In that way, inadvertent omissions of critical buffers are minimized.

Some key aspects that need to be addressed during designing any buffer include the following:

- What is the size of buffer required?
- What is the required life?

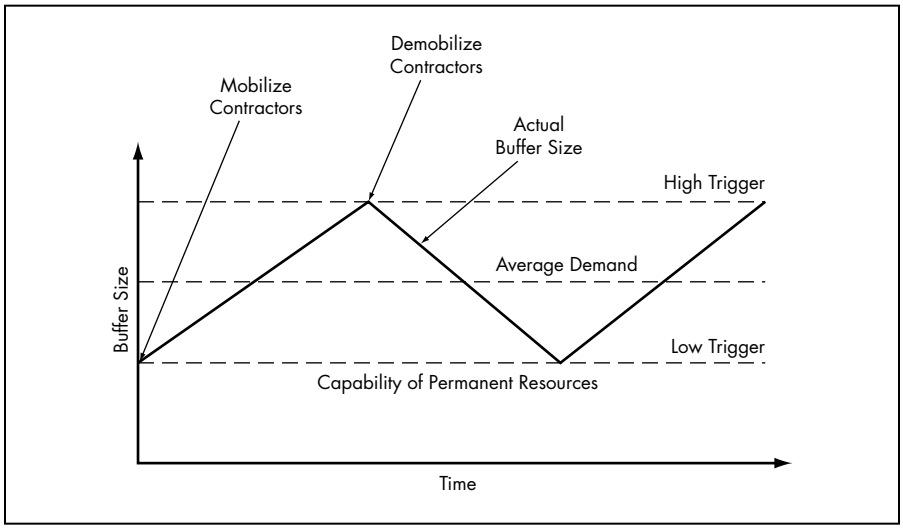


Figure 12.8-2 Use of contractor resources to manage buffers in nonbottleneck processes

- What options are available to provide a buffer of the required size and life span?
- What are the capital and ongoing operating costs of the first three options?
- How can it be designed to minimize operational downtime (e.g., blockages)?
- How can it be effectively maintained in a serviceable state (including replacement options) without adversely impacting future mine throughput?
- How effectively does the buffer integrate into upstream and downstream processes?
- What optionality exists for expanding the capacity of the buffer (enlarging, duplicating) to meet future strategic scenarios?
- Are there requirements for separating the ore into high/low/waste grade or according to other specifications?

Maintenance, Contingency, and Optionality Plans

In order to plan for future uncertainties, management strategies for each buffer and ore-handling process should include maintenance, contingency, and optionality plans. Maintenance plans protect throughput during ongoing operations and discrete and planned periods of major maintenance work. Contingency plans manage the risk of reduced performance as a result of unforeseen events. Optionality plans consider the potential for either a change in mining strategy or future capacity expansions. In many cases, these management plans are complementary. For example, provision of a redundant orepass can be an effective strategy to manage both maintenance and operational issues but can also be built into an optionality strategy in which the additional pass can be used, for example, to increase throughput in the mine or to segregate ores of different grades.

Table 12.8-1 describes some examples of maintenance, contingency, and optionality plans that could be employed to various components of an ore-handling system.

Selection Process

Materials handling is generally a big investment and can be the largest single investment in any new mine. The selection process needs to balance the knowns (capital cost, current operating conditions) against the unknowns (future operating conditions). This is not a trivial exercise, as the future operating condition has many dimensions, most of which are difficult to quantify with certainty. These include the following.

- **Safety:** Is the ore-handling system capable of adopting ongoing improvements in safety systems, processes, and technology (e.g., automation)?
- **Revenue:** Will changing metal prices affect business decisions that could impact the choice of ore handling?
- **Cost:** Will the cost of handling ore increase excessively?
- **Throughput options:** Does the ore-handling system provide the flexibility to increase throughput if required, without significant financial hurdles (including business interruptions)?
- **Geographic footprint:** Will the changing mining footprint over time adversely impact production cost and/or throughput?
- **Ground conditions:** What will be the impact of deteriorating ground conditions (directly relating to ore handling) on production cost and/or throughput?
- **License to operate (company strategy and regulations, government regulations, etc.):** Will mandated changes to operating practices (e.g., health, safety, environmental) impact the ore-handling system such that production costs and/or throughput are adversely affected?
- **Availability and reliability:** Will maintenance performance decline excessively and will the cost of sustaining performance be excessive? Can major maintenance works be carried out without adversely affecting mine performance?
- **Maintenance support:** Will the maintenance support be adequate? This can be an issue if new and/or specialized technology has been employed, where maintenance support can be problematic in terms of either speed of response or quality of service. A risk–reward evaluation is recommended to determine whether the benefits of the technology are outweighed by the reduction in equipment availability as a result of substandard maintenance support.
- **Replacement and/or major maintenance:** Is there scope for replacing/ maintaining key infrastructure, if required, without significant business interruptions?

Table 12.8-1 Examples of maintenance, contingency, and optionality plans

Component of Ore-Handling System	Maintenance Planning	Contingency Planning	Optionality Planning
Loaders/trucks	• Good availability track record • Fast response of spare parts • Duty/standby equipment • Major component change-out plan • Flexible fleet configuration • Good maintenance support—large skills base developed and maintained	• Capability to increase/decrease resources on demand • Flexible fleet configuration • Remote and automation capability	• Capability to increase resources on demand • Haulage routes capable of being expanded to account for changing mine's geographic footprint or mining strategy • Capability to adopt latest technology into fleet • Highly flexible to respond to different mine configurations
Orepasses	• Designed in best possible ground conditions, based on rock quality and both virgin and induced stresses over time • Good access for regular inspection and remedial work • Good monitoring and reconciliation systems (tons in versus tons out) implemented • Good understanding of long-term deterioration due to stress conditions and planned maintenance program • Artificial support options • Provision of backup pass • Simple design • Management of rock size going into pass (sizing)	• Provision of backup passes • Provision to duplicate passes if required due to failure of existing passes—passes have limited life, and, if small in number, premature failure could be a major issue • Good access for regular inspection and remedial work • Good access to manage hang-ups	• Provision for additional passes • Limited optionality for each individual pass—long lead time to design and install
Chutes	• Simple, robust, and reliable design • Integrated planned maintenance schedule	• Effective plan for unblocking chutes	• Fully automatable
Trains	• Duty/standby equipment • Good maintenance support—specialist skills required • Integrated planned maintenance schedule	• Latent instantaneous capacity • Critical spare parts inventory • Good maintenance support • Fully automatable	• Provision for additional capacity • Provision to expand rail system • Designed and constructed for long life and optionality—no option for long shutdown periods • Control of systems to optimize performance
Storage facilities (bins, stockpiles)	• Provision of backup storage facility—shutdowns are long and costly	• Adequate capacity storage (buffer) between all ore-handling processes • High-capacity buffers supporting bottleneck	• Provision for additional storage facilities • Optionality to split into different material streams (ore, waste, etc.)
Conveyors	• Latent instantaneous capacity • Good maintenance support • Integrated planned maintenance schedule	• Latent instantaneous capacity • Critical spare parts inventory • Good maintenance support	• Provision to increase belt speed • Provision to duplicate belt system • Provision to extend system to new mining areas
Crushers	• Latent capacity • Good maintenance support • Integrated planned maintenance schedule	• Latent capacity • Critical spare parts inventory • Good maintenance support	• Latent capacity • Provision to duplicate crusher
Shafts	• Latent instantaneous capacity • Good maintenance support (noting the extreme skills shortage in this field) • Integrated planned maintenance schedule	• Latent instantaneous capacity • Critical spare parts inventory • Good maintenance support	• Provision to increase skip weight/speed • Provision to duplicate shaft • Automation

- **New technologies:** Can the system take advantage of new technologies to add value (e.g., reduce costs, increase throughput)?
- **System robustness:** Is the system capable of effectively responding to unplanned variability in key input characteristics and maintain planned throughput? Some examples are
  - Particle size changes;
  - Presence of undesirable material (e.g., rock bolts, cables);
  - Presence of backfill material, clay, or other sticky waste material (e.g., fine, high moisture content potentially causing significant blockages in chutes and transfer points);
  - Water content changes;
  - Major unplanned failures of internal, upstream, or downstream processes; and
  - Changes in spatial distribution of ore supply.
- **Change in mining method:** Will potential future changes in the mining method be adequately serviced by the ore-handling system?
- **Change in ore-handling system:** Can possible additions, changes, or enhancements to the ore-handling system over time be adequately integrated into the existing system and implemented without adversely affecting system availability?
- **Continuous improvement:** Is the ore-handling system capable of facilitating ongoing improvement in order to increase operational efficiency?

- **Short-term operational variability:** Can the system adequately manage short-term operational variability through effective use of upstream and downstream buffers? This is particularly important if the ore-handling system is the business bottleneck.

A common selection process, described in Table 12.8-2, is based on the development of a matrix in which the various ore-handling options are assessed and ranked against key design and operational criteria. Each criterion is assigned a weighting based on the importance of that criterion to the selection. For each ore-handling option, a performance score is then assigned to each criterion. The weighted scores are then totalled, and the highest score is used to determine the preferred option.

Although this process is relatively simple and efficient, and can achieve good stakeholder buy-in, it has a number of potential shortcomings:

- The ore-handling options will be based on experience and may not include a broad enough choice.
- All relevant selection criteria, particularly those that relate to future uncertainty, may not be considered.
- The weighting assigned to each selection criterion may be subjective.
- The score assigned to some selection criterion for each ore-handling option may also be subjective.
- The summing of weighted scores for each ore-handling option to determine an overall score may not reflect the true relative impact of each selection criterion.

There is a risk that the selection outcome will be biased toward those options that prioritize currently known aspects of the design (e.g., capital and operating costs) over future unknown aspects. If, for example, two options have similar appeal in the current time frame, but the less attractive has greater optionality in the future, this option should be the preferred option. For this reason, it is recommended that this process be used as only one component of an overall selection study.

Table 12.8-2 Example of a common ore-handling selection process*

Ore-Handling Options	Selection Criterion				Total Weighted Score
	1	2	3	4	
Weighting	40	30	20	10	
Option 1	2.0	6.0	1.0	8.0	3.6
Option 2	7.0	7.0	3.0	1.0	5.6
Option 3	8.0	2.0	9.0	5.0	6.1
Option 4	6.0	6.0	7.0	7.0	6.3

*All scores are from a total of 10.

Comparison Between Ore-Handling-System Options

The following discussion compares the three most common ore-handling systems: shaft, conveyor, and truck. These systems usually contain other supplementary ore-handling components, including loaders, orepasses, and crushers. In this comparison, it is assumed that a significant vertical lift is required to bring the ore to the surface, therefore limiting the application of rail as a primary ore-handling system.

In assessing the relativity of these systems, notably, other issues not related to materials handling may often affect the choice of system. For example, if a decline system is required for access and provision of routine services to the underground, the incremental cost to also utilize this access for trucking, for example, may be much lower than the total cost. Hence the decline option may be cost-effective relative to other ore-handling options.

Cost

Shaft systems usually attract the highest capital cost; however, as depth increases, the alternatives become progressively more expensive, and the capital cost difference relative to shaft systems decreases. Once installed, however, shafts attract a relatively low operating cost.

Conveyor systems have recently been demonstrated to be cost-effective relative to shaft systems. Pratt and Ellen (2005) demonstrated negligible capital and operating cost differential between the two systems during a selection study at the Telfer operation in northern Western Australia. The shaft option was preferred due to other factors such as physical impact on the orebody, footprint required, and flexibility to service future economic ore zones.

Trucks are usually lower in capital cost but have higher operating costs. Recent improvements in trucking technology have increased the depth to which trucking costs remain economically attractive. Pratt and Ellen (2005) suggest that trucking is not economically viable over a range of mining rates from 1.5 Mt/a (million metric tons per annum) to 6 Mt/a below a depth of 650 m.

Figure 12.8-3 describes the capability of established underground mine haulage systems at the Telfer and Ridgeway gold mines in Australia. The choice of system is largely based on the cost efficiency as a function of depth and throughput.

Safety

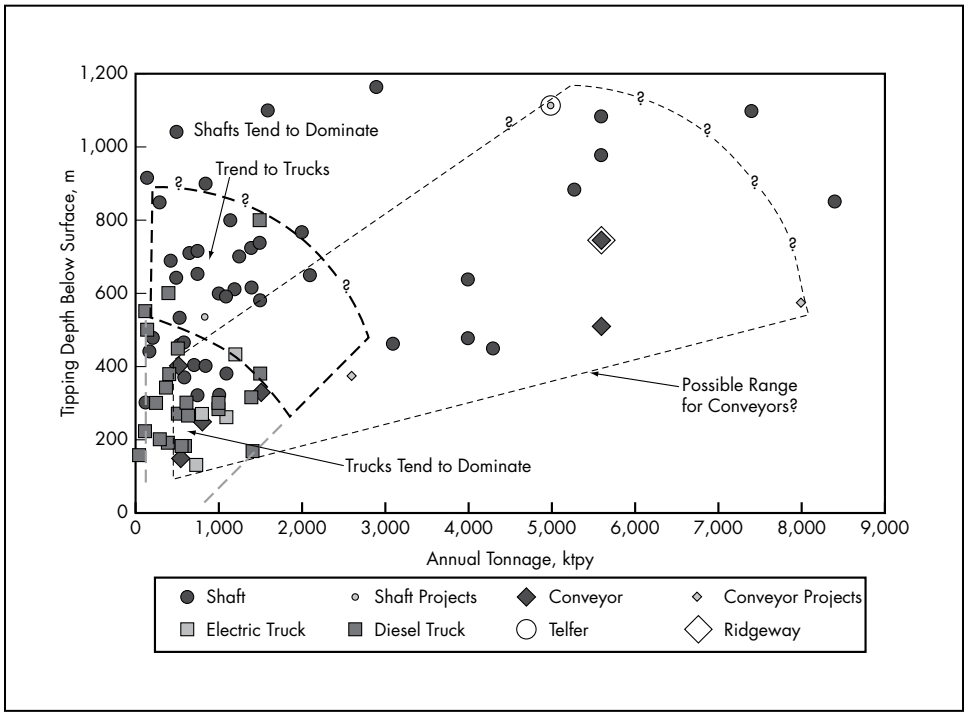
All options have safety risks that need to be managed. For shafts, risks are associated with working at heights during inspection and maintenance work, and interactions with personnel during operations. For conveyors, fires and personnel interaction with the moving belts are of particular concern. For trucks, fires and vehicle-to-vehicle and vehicle-to-pedestrian interactions are the highest risk issues, all of which can be managed effectively through the application of adequate design and appropriate procedures. Some examples are presented in more detail later in the chapter.

Timing

Truck haulage via a decline offers a particular advantage in the ability to access ore progressively as the decline is sunk. In this way, positive cash flow can be generated earlier in the mine life compared to alternative systems where installation must be achieved to full depth prior to ore production. In cases in which a surface access decline is being established in a mine where the ore will ultimately be hoisted using a shaft, it may be advantageous to utilize truck haulage up the decline early in the mine life, prior to commissioning of the shaft. Examples of this application include the BHP Billiton Cannington mine in northwest Queensland, Australia, and Newmont Mining's Leeville mine in northern Nevada, United States.

Optionality

Because mines typically evolve over time, the ability of a mine to vary its operating strategy, in order to optimize production



Source: Pratt 2005. Reprinted with permission from the Australasian Institute of Mining and Metallurgy.
Figure 12.8-3 Capability of established underground mine haulage systems

in response to changes, can often be a function of the optionality embedded within the ore-handling system. Although shaft and conveyor systems can be inflexible because of the limited number of fixed feed points, trucking systems are flexible because generally trucks can travel to most locations in the underground mine. Where fixed infrastructure ore-handling systems are employed, the tendency is to introduce hybrid systems (e.g., trucks feeding shaft) as the mine evolves in order to respond to changes.

In situations where the underground mine capacity needs to expand to increase business value, the potential is often related to the configuration and current utilization of the ore-handling system. If the existing system is based on trucks, the expansion can typically be achieved incrementally by adding trucks as required until the capacity (e.g., ventilation, traffic congestion) of the decline system is reached. Following this, additional access would be required at significant cost. Increasing throughput in fixed systems such as shaft hoisting and conveyors is relatively cheap up to a point where the systems' utilization is at the industry benchmark. Following this, higher utilization will probably require duplication of the existing system at significant cost. This is likely to be justified financially only if there is a step increase in throughput or the geologic footprint has evolved substantially enough that a duplicate system is justified to maintain or create a step change in expected efficiencies and associated operating costs.

LOADERS

Loaders move ore and waste from the development face, stope, pass, or stockpile to the next step in the ore- or waste-handling system. The types of loaders include

- Rubber tired,
- Rail mounted,
- Track mounted, and
- Shuttle loaders.

Loaders are most commonly a single-function unit but in certain applications can be part of a multifunction unit. The most common form of the latter is a continuous miner, which can cut coal from a coal face, load the coal into a transport vehicle, and also install roof support (when equipped to do so).

Rubber-tired loaders in metalliferous mines are referred to by many different names around the world, according to local preference. These include loader, load-haul-dump (LHD), scooptram, scoop, bogger, digger, mucker, and underground loader. An underground loader is shown in Figure 12.8-4.

Underground loaders are typically the first component of the ore-handling system. Loaders extract ore from the stope and either dump (tip) directly into an orepass or load into a truck that hauls the ore to an orepass or surface stockpile. The size of the loader fleet will be determined according to the following factors:

- Mining method
- Geometry of the stope/ore working face
- Size of underground excavation
- Geometry of the ore-handling system
- Production rate based on planned strategic objectives
- Productive operating time or availability and utilization per shift
- Loader productivity
- Geometry and condition of the tram/haul route
- Capital and operating costs

The mining method will determine the relative proportions of stope ore, development ore, and waste. The geometry of the stope or ore working face may limit the size of loader that can be used. Ground conditions will determine underground



Source: Caterpillar 2009.
Figure 12.8-4 2900G Elphinstone loader

excavation dimensions, which in turn will impact fleet size and equipment selection. The geometry of the ore-handling system will determine the proportion of material required to be rehandled. Sweigard (1992) provides a good overview of the equipment selection and the fleet size estimation process. This information allows mapping of loader tram segments to determine quantities of ore movement required to achieve the nominated production rate. Typical units of measurement are metric tons per operating hour (t/h) and ton-kilometer per operating hour (t·km/h), which is based on the one-way length of the tram. Preliminary estimates of total operating hours per year or total ton-kilometers per year for various sizes of loaders allow for an initial estimate of loader fleet size.

Further details on loader productivity are available in Sweigard (1992) and the various underground equipment manufacturers' specifications (e.g., available on the Caterpillar, Sandvik, and Atlas Copco Web sites). Gradeability/speed/rimpull charts similar to Figure 12.8-5 and equipment dimensions (Caterpillar 2004) provide useful specifications for determining loader performance in differing conditions. The calculation for estimating truck fleet performance is performed using a similar methodology.

Loaders tend to be as large as development allows, and tram distances are minimized to maximize productivity per loader. Loader trams greater than 250 m become less effective. To allow production to continue when the stope brow becomes open, the loader requires line-of-sight remote or tele-remote capacity. The loader bucket may also use different tooth configurations to assist digging in the stope drawpoint.

Safety Considerations

Safety issues that need to be considered during the design phase and implemented during the operations phase include the following.

- Light vehicle, heavy vehicle, and pedestrian interaction: Some strategies that can be employed to manage this issue include separating heavy vehicles from light vehicles and pedestrians, detection and collision avoidance systems, traffic rules, and visual and audible signalling systems.
- Remote loading at the stope drawpoint (operator interaction): The most effective method is to isolate the operator from the machine by using tele-remoting systems.
- Loading at a stope brow or orepass and initiating a mud rush or air blast: To manage this situation, a good understanding of the stope's material characteristics is required, together with strict controls and procedures regarding the methods of extraction from the drawpoint to ensure predictable behavior of material flow from the drawpoint.
- Fire on the loader: Minimizing risks and consequences of loader fires requires quality equipment design, effective preventive maintenance programs, well-trained operators, reliable automated onboard fire prevention and firefighting systems, and sound procedures for managing fires.
- Equipment failure during operation: Minimizing risks and consequences of equipment failure requires quality equipment design, an effective preventive maintenance system, well-trained operators, and controls that minimize personnel exposure to loader operations.
- Open holes (including tipples): Effective hard and soft controls are required to minimize risks associated with operating around open holes, including solid barriers, good lighting and signage, and procedures.
- Operator ergonomics: The well-being of personnel during loader operation and maintenance is paramount to sustainable operations. High-quality, fit-for-purpose design, both within the operator's cabin and wherever personnel work on the loader, is critical to achieving this objective.

Design and Construction Issues

Underground access development needs to be purposely designed to accommodate selected loaders and trucks. As a general rule, all rubber-tired equipment should operate in development with gradients of at least 1:50 to minimize water ponding and to provide good road conditions. Key development components are

- Drawpoints,
- Tramming drifts (drives),
- Tipple accesses,
- Stockpile bays, and
- Truck-loading bays.

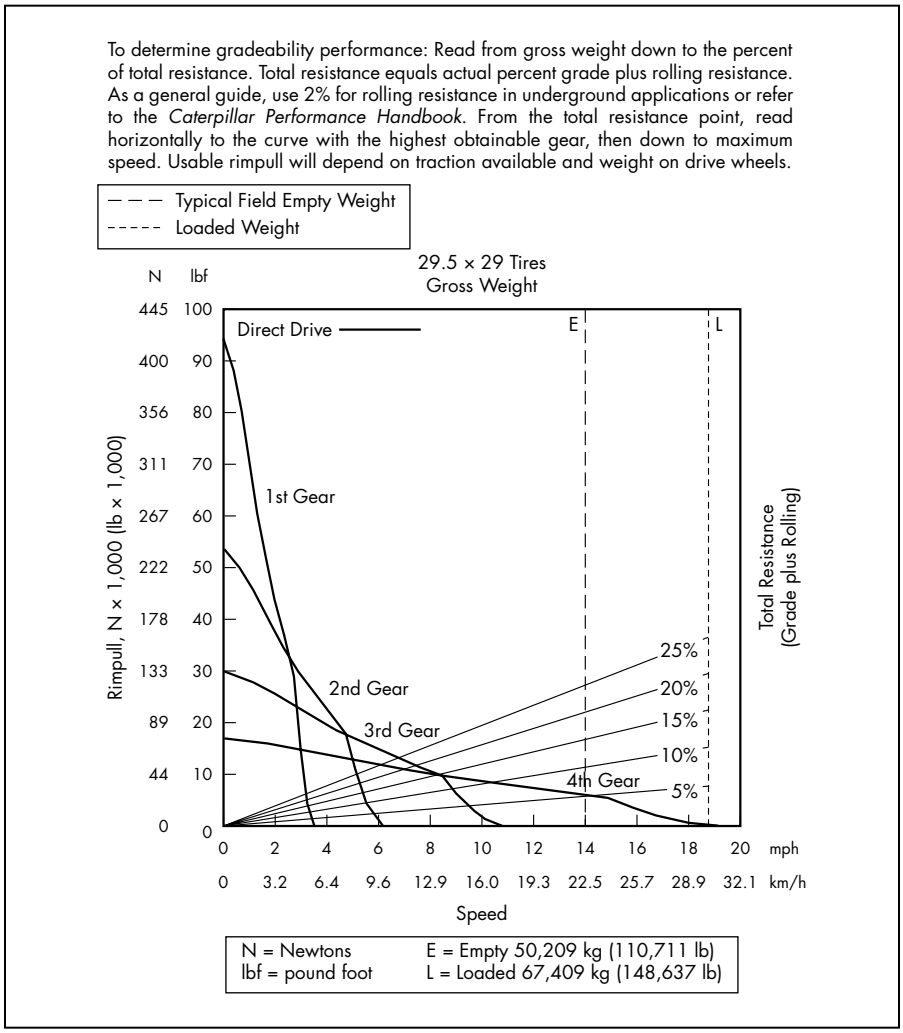
These components are described in detail in the following sections.

Drawpoints

Drawpoints provide access to the stope for loaders to extract ore from the stope. The design of a drawpoint needs to consider horizontal and vertical clearances relative to the access walls and back, and gradient of the access. Ideally, the drawpoint should be angled relative to the tramming drive (e.g., 60°) to facilitate ease of entering and exiting. It also needs to be of sufficient length to accommodate the loader while digging the muckpile, without being too long to trigger poor ventilation conditions. At mines where remote loading from the stope is required, a safe operating position needs to be designed into the drawpoint area to protect the loader operator. Figure 12.8-6 shows a tele-remote setup in a Canadian mining operation.

Tramming Drifts (Drives)

Similar to drawpoints, the main design considerations for tramming accesses are clearance, gradient, and orientation geometry.



Source: Caterpillar 2004.

Figure 12.8-5 Caterpillar gradeability/speed/rimpull chart for Elphinstone 2900G

Tipple Accesses

Once again, clearance, gradient, and orientation of tipple accesses are the main design considerations. Back heights above the orepass and in the approach to the orepass are critical to facilitate the dumping (tipping) of the loader bucket and truck tray. If back height becomes problematic, trucks equipped with hydraulic ejector beds and plates should be considered. A steep upward gradient into the orepass reduces the potential for mine water to accumulate and flow into the orepass. Orientation of the tipple access to the main tramming drift (drive) can be perpendicular or angled. A perpendicular access allows loaders to approach from both directions without the requirement for a turnaround bay.

Stockpile Bays

Stockpile bays provide surge capacity and can improve overall ore-handling system utilization. They allow loaders to keep producing while waiting for trucks or full orepasses to clear. Stockpile bays need enough capacity to minimize delays to other parts of the development or production cycle.



Courtesy of Atlas Copco.

Figure 12.8-6 Tele-remote station at a stope drawpoint in Canada

Truck-Loading Bays

Truck-loading bays need to consider back height for loading and localized gradient and geometry constraints. Detailed access development design criteria can be obtained from the different manufacturers' equipment handbooks. Examples of the information available in the equipment handbooks are illustrated in Figure 12.8-5. Generally, maximum efficiency and minimum cost are achieved if the loader is sized to load the truck on the same level, rather than from an elevated loading ramp.

Operational Aspects

Stope Mucking

The productivity of a loader is dependent on a number of factors that facilitate the removal of ore from the drawpoint. Ore size distribution should be relatively homogeneous with minimal oversize or high percentage of fines. Oversized rocks slow the production rate because they must be resized either in the drawpoint or in nearby bomb bays. Processing oversized material in the drawpoint also introduces safety issues, which need to be managed. A high percentage of fine material (e.g., from the failure of a backfill mass) can create difficult operating conditions throughout the ore-handling system, particularly if water is present. Both oversize and fines issues can be better controlled through drill-and-blast design, geotechnical design, and the placement of quality, engineered backfill.

Good physical operating conditions include good ventilation for visibility and water sprays to help suppress dust. Drawpoint brows need to be supported to minimize wear over their anticipated lives. Brow wear can lead to a reduction in effective drawpoint length, making loading difficult, and increases the tonnage of ore that needs to be mucked by remote control.

The ventilation system needs to be monitored, because changes in airflow can occur when the stope brow opens. Additional consideration is required in uranium mines where air transit time becomes a factor in limiting radon residence time. A ventilation quantity of 0.06 $m^3/s/kW$ of engine power is a good standard to meet acceptable environmental conditions within typical mining operations.

Tramming

Loader operators generally prefer to tram with the loader bucket at the rear when loaded and the front when empty. This reduces the occurrence of loaders driving over rocks that fall from the bucket while tramming.

The following key aspects need to be considered and implemented in a practical manner to achieve the best cycle time results:

- **Road maintenance:** Well-maintained roadways are critical for achieving maximum loader speeds and reducing equipment maintenance issues associated with continuous vibration of critical loader components.
- **Drainage:** The establishment of well-drained roadways will also increase achievable tramming speeds, reduce road and vehicle maintenance, and reduce collision with submerged objects that are not visible.
- **Ventilation:** A well-ventilated roadway will provide good visibility of the tramming route, allowing higher loader speeds and reducing the potential for vehicle collision.
- **Lighting:** The installation of lighting on the tramming route can allow higher loader speeds; however, lights should only be considered on a tramming route that is in use for a significant time period.
- **Tram geometry:** This includes the drift (drive) size or available clearance, the gradient (uphill or downhill loaded) and length of tram, and the number of corners or bends in the tram.

Loader Costs

Budget capital cost estimates can be provided from equipment manufacturers. The maintenance change-out program will determine ongoing sustaining capital requirements, depending on the frequency of rebuilds and major component change-outs. Loader operating costs are dependent on labor, fuel and lubricants, tires, bucket wear, maintenance consumables, and equipment damage.

A remote loading operation typically has a higher operating cost resulting from reduced productivity. The ability to estimate the cost of maintenance depends on the relative proportion of predictive and preventive maintenance to breakdown maintenance. Equipment suppliers can provide budget estimates for the majority of consumable operating costs. Equipment damage is difficult to estimate because it is related to a number of factors, including road conditions, visibility, operator experience, equipment clearance dimensions, mining method, tram geometry, and stope drawpoint conditions. A loader performing remote operation into an open stope will have a higher exposure to rockfall damage from unsupported areas of the stope.

Loader productivity and utilization will depend on the activity; for example,

- Development loaders: <4,000 h/yr;
- Sublevel open stoping production loaders: ~4,500 h/yr; and
- Block cave and sublevel cave production loaders: >5,000 h/yr.

Technology

Production Management Systems

A number of commercially available production management systems are used in underground mining operations, including Micromine's PITRAM and Modular Mining's Dispatch. Numerous operations have also developed their own in-house systems, which capture data and allow managers to monitor performance to determine whether operational targets are being achieved. The data are collected using radio communication or tag readers located throughout the mine.

Automation

Automation and remote control technology have been available for underground loaders for more than 20 years. Numerous automation systems are being developed by equipment manufacturers, mining companies, research organizations, technology companies, and various partnerships between these groups. The level of advancement with these automation systems and their ability to operate practically and cost-effectively in the challenging underground environment are still under debate. Substantial progress has been made, and cost-effective, practical automation solutions will be an important part of the future for improved loader productivity and operator safety.

Maintenance

All maintenance programs can be broken down into preventive and breakdown components. Both are required to minimize the risk of fatalities, injuries, and incidents arising from the use of mobile equipment underground and to maximize equipment productivity.

An effective predictive and preventive maintenance program that is prioritized over breakdown maintenance will reduce the potential for personnel injury and increase equipment utilization as a result of the following aspects:

- Replacement of components before they fail and while the equipment is not operating prevents potentially serious consequences, including injury to personnel and damage to associated components.
- More predictable scheduling of maintenance activities allows for better matching between supply of, and demand for, maintenance resources. Breakdown maintenance tends to create boom-and-bust work cycles that cannot be effectively managed using a fixed level of resources.
- More predictable availability of equipment allows for more efficient utilization, as a result of better matching between availability of equipment and supply of operators.

Best practice equipment availabilities are generally achieved by reducing the frequency of breakdown maintenance and increasing mean time between equipment failures. To maintain high availability throughout the useful life of equipment, a well-planned component change-out program is essential. The operating life and timing of major equipment rebuilds will have an impact on capital and operating cost profiles for the mine. Costs of preventive maintenance and major component change-outs need to be consolidated with other operating costs to determine an overall cost per ton-kilometer. This measure can be used to determine the effectiveness of the maintenance program.

TRUCKS

Trucks are used for long or inclined hauls to orepasses or directly to the surface. Commonly available trucking options include

- Diesel or electric articulated dump trucks, or ADTs (Figure 12.8-7);
- Diesel or electric rigid dump trucks;
- Diesel road trains (Figure 12.8-8); and
- Diesel, electric, and battery shuttle cars (Figure 12.8-9).

Trucks can be side loaded near the ore source (e.g., stope drawpoint), or side loaded or chute loaded at the bottom of an orepass.

Safety Considerations

Safety issues that need to be considered during the design phase and implemented during the operations phase include

- Light vehicle, heavy vehicle, and pedestrian interaction;
- Chute loading (the key to minimizing risks is to ensure that the design of the chute fully accounts for the material flow characteristics and the loads that the chute will be subjected to, so there are no unexpected failures in the chute);
- Equipment failure;
- Fire on the truck;



Courtesy of Atlas Copco.

Figure 12.8-7 ST1520 loader side-loading an MT5020 truck



Source: Findlater and Robertson 2004.

Figure 12.8-8 Ninety-ton road train exiting Sirius mine portal (Western Australia)

- Tipping into open holes; and
- Operator ergonomics.

Selection Criteria

Selection of the appropriate size and type of truck fleet depends on the following key factors:

- Ore reserve tonnage
- Production rate
- Haulage distance
- Depth below surface
- Lateral extent of the mine
- Mining method used
- Integration with upstream and downstream materials-handling processes

Truck haulage is generally the preferred option where the ore reserve is not large enough to justify the capital for a shaft or conveyor hoisting system. For hoisting shaft or inclined conveyor systems, the lateral extent of the mine, distribution of ore, and throughput requirements determine the most effective solution for upstream delivery to a centralized hoist, from a choice of trucking, rail haulage, or conveyor.

ADTs have traditionally been used because of their flexibility and ability to maneuver throughout the ore body in variable conditions. Road trains, electric articulated, and rigid-body trucks have been utilized with varying degrees of success depending on road conditions (grade, vehicle clearance, and road maintenance) and environmental conditions (temperature and visibility).



Source: Joy Mining Machinery 2008.
Figure 12.8-9 Shuttle car in operation in a potash mine



Courtesy of Joy Mining Machinery.
Figure 12.8-10 Coal shuttle car being loaded

Underground coal and potash mines have a unique application for trucks, given the typically narrow seams that are mined. Specialized shuttle cars are used to transport the coal or potash. These are low-profile AC-electric trucks designed to interact effectively with high-capacity continuous miners or roadheaders. They remove material that has been cut by the mining machine and transfer it directly to a main conveyor, which creates an efficient ore-handling system. Shuttle cars can operate in seam heights down to 1.2 m with a capacity of 8 t (e.g., Figure 12.8-10).

Design and Construction Issues

Access design considerations discussed in the "Loaders" section can also be applied to trucks (equipment clearance, gradient, and ground conditions). Additional considerations include

- Side-loading bays,
- Orepass dump point (tipple) configuration,
- Chute-loading systems, and
- Decline haulage access.

Side-Loading Bays

Side-loading bays facilitate the efficient loading of a truck. Side loading of trucks can be performed on level ground (Figure 12.8-7) or with the loader elevated above the truck, utilizing a perpendicularly configured side-loading bay. The latter configuration can be used when the loading point is fixed over an extended time frame and truck spotting times have a significant impact on truck productivity. To facilitate efficient truck loading and reduced spillage, the loader can be elevated 1.5 m above the truck.

Orepass Dump Points (Tipples)

Orepass tipple configuration for trucks requires additional height and length of back stripping compared with a loader configuration. It is also necessary to be able to reverse into the tipple. Additional development may be required to achieve efficient truck tipping.

Road trains dump (tip) to the side and therefore require a significantly different layout for an underground tipple. To accommodate side dumping (tipping), a truck loop is generally required. Consideration for a turning radius and a straight section at the dumping point can add a significant amount of underground development as compared with other truck dumping layouts.

Chute-Loading Systems

Truck loops and truck-reversing bays are two commonly used configurations for loading trucks from chutes. When sufficient tonnage is available (e.g., 3 to 4 million t), a truck loop configuration can achieve higher productivity. For lower tonnages, a chute that requires trucks to reverse provides an option with less capital. To further reduce capital costs in low-tonnage, short-term situations, ore can be side loaded into trucks at the bottom of orepasses. To facilitate chute loading of multiple trailers, road trains require a truck loop configuration.

Decline Haulage Access

Decline (or ramp) haulage provides access for trucks to haul ore from underground to a surface stockpile. Design considerations include clearance, gradient, geometry, and frequency and geometry of passing bays. Declines can be designed as a series of long straight sections or as a circular spiral. Passing bays can be designed parallel with or perpendicular to the decline, depending on ground conditions. In most Australian decline trucking operations, stockpile bays are developed every 150 m during construction and then utilized for passing bays. Block light systems are also commonly used to control traffic flow.

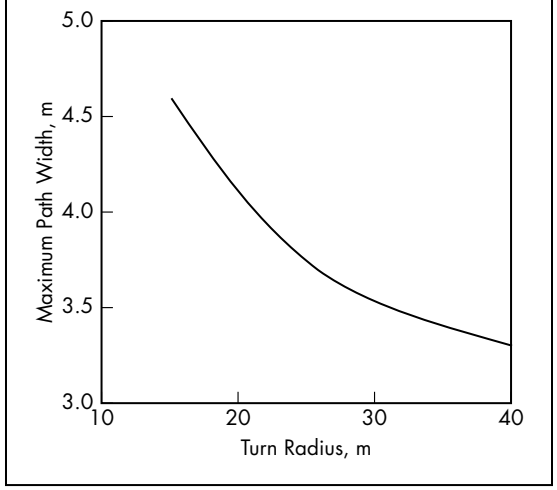
ADTs, large rigid-chassis trucks, and road trains with power trailer are effectively used to haul ore long distances to surface stockpiles. Figure 12.8-8 shows a road train exiting a decline portal, and Figure 12.8-11 shows design information for drift (drive) size and road train turning requirements. Robertson et al. (2005) provides a good overview of the future direction of trucking. Because decline haulage trucks usually have to share the decline with other vehicles, numerous traffic control systems are available to reduce the hazard of vehicle interaction.

Trucking Costs

Many factors that influence trucking costs are similar to those for loaders (see the "Loaders" section). The differences are primarily due to the specific functions of the equipment. Loaders are generally more influenced by loading conditions, whereas trucks are more influenced by haulage conditions (road conditions and haul distances).

The two technologies described under "Technology" in the "Loaders" section have similar application to underground trucks and should be considered if justified cost-effectively.

Turn Radius, m	Path Width at 30°, m	Path Width Maximum, m
15	4.2	4.6
20	3.9	4.1
25	3.7	3.8
30	3.5	3.5
40	3.3	3.3



Source: Findlater and Robertson 2004.

Figure 12.8-11 Underground road train vehicle sweep paths for Powertrans underground truck and Powertrailer

OREPASSES

Orepasses provide a low-cost method to move ore and waste downward between operational horizons in a mine. Initial capital is the dominant cost associated with an orepass system, with lower ongoing sustaining capital and operating costs.

Orepasses can be constructed as one long section or as multiple shorter sections depending on the geometry and/or timing of access to the top and bottom of the orepass and the relative operational and safety aspects during construction.

Flow of material in the orepass is critical to its success. A number of factors need to be considered to maintain continuous flow conditions. Early origins of material flow were analyzed by Janssen (1895), who determined that the weight of the material in the bin or orepass was actually transferred to the walls rather than the bottom of the orepass. This produces the potential for arching, where broken material stabilizes in the pass and prevents material flow. Strategies for avoiding this are discussed in later sections, for example, when considering orepass width.

Safety Considerations

The safety issues that need to be considered during the design phase and implemented during construction and operation include the following.

- **Risks during construction:** The main risks are people working below vertical openings from which material can fall and people falling into open holes. Thorough risk assessments need to be undertaken to ensure that key risks are identified and managed.
- **Clearing of blockages:** Clearance in orepasses involves drilling hang-up holes into the immediate vicinity of the blockage and feeding explosives into the pass to free the blockage.
- **Fall prevention:** Stop (bumper) blocks, orepass covers, signs, barricades, and flashing lights are used to prevent people from falling into or driving equipment into orepass open holes.
- **Mud rushes:** These hazards occur when material containing a high proportion of fines is dumped (tipped) into an orepass and the orepass becomes blocked. When mine water or natural groundwater mixes with the fines in the orepass, there is the potential for a mud rush. The severity of a mud rush depends on the amount of fines in the orepass and the proximity of personnel and equipment to the bottom (discharge point) of the orepass during a mud rush. Management of mud rushes includes understanding material flow properties and developing procedures for managing undesirable combinations of materials that report to the drawpoint. Typically, pedestrians are isolated from any orepass loading activity, and, in cases where mud rushes are likely, remote loader operations are common.

Design and Construction Issues

Orepass design is largely dependent on the geometry of the ore body and the strength of the rock mass in the mine. Hadjigeorgiou et al. (2005) provide a good review of orepass practice in Canadian mines.

Inclination Angle

Orepass inclination angles are generally greater than 60° and should have at least a small amount of inclination from vertical to allow the dumped material's energy to be dissipated on the orepass footwall and therefore prevent free fall of the material to the orepass bottom. Orepass fingers that connect to chutes can be as low as 50° in order to reduce the amount of energy transfer from the ore in the orepass onto the chute.

Length

The length of an orepass is dependent on the vertical distance between stope extraction level(s) and the common ore-handling level. Orepasses are sometimes extended upward to allow more cost-effective dumping (tipping) of development material from upper levels of the stope. The orepass system can be constructed using one or two longer sections or a larger number of shorter sections, depending on access available for construction. The length of each orepass section generally determines the excavation method. Typically, Alimak raising and raise borers can construct longer orepasses, whereas drilled and blasted drop raises and conventional raises are used for shorter-length holes.

Orientation

The orientation of an orepass can be a critical design consideration in rock masses with dominant structures. The most favorable orientation is as close to perpendicular to the dominant structure as practical. In situations where the orepass needs to be oriented parallel with the dominant structure, additional support, liners, or a redundant orepass should be considered.

Shape

The shape of an orepass is dependent on the excavation method. Raise boring will ream a circular shape, while Alimak, drop

raising, and conventional raising typically excavate a square or rectangular shape. If high stress is believed to be a stability issue for the orepass, circular shapes can be excavated with drill-and-blast methods.

Width

The width of an orepass is dependent on size distribution of the broken material, characteristics of the wall rock, and capacity requirements of the orepass. It generally needs to be no greater than five times the dimension of the largest particle that enters the orepass, as described in Table 12.8-3.

Recent work by Hustrulid and Changshou (2004) indicates that the ratios in Table 12.8-3 are conservative. Their work recommends considering size distribution and orepass wall conditions in the design, which will usually lower D/d ratios being practically achieved. Rock mass strength of the orepass wall relative to the predicted induced stress levels and orientation also needs to be considered when sizing the orepass.

Quality of the rock mass in orepass walls will partially determine what orepass size is achievable. Empirical design methods can be used to determine stable spans, and, if high stress fields exist in the mine, numerical modeling should be considered to understand possible future states of stress (magnitude and orientation) in the orepass walls.

Surge capacity requirements of the ore-handling system need to be considered when designing each individual orepass. When the ore-handling system downstream of the orepasses is taken out for maintenance, the loader and truck operation will stop when the orepasses are filled. Effective surge capacity in the ore-handling system allows the components of the system to be less dependent on each other. Surge capacity is generated by expanding the dimensions of the orepass and/or developing strategically located stockpile bays. Enough surge capacity needs to be designed into the orepasses to increase operational flexibility of the total ore-handling system. How much surge capacity can be installed has practical and cost limitations, and the amount of surge required will depend on the benefit realized by smoothing out interruptions in ore flow due to downstream process variability. Surge capacity from several hours to a week should benefit any mining operation if this surge supports the operational bottleneck. Where downstream process variability is significant, consideration should be given to installing the maximum possible surge capacity.

Material may become captured in certain areas of the orepass, and this reduces the effective capacity of the orepass. Sections of the orepass that have changes in direction or angle are more likely to create areas for captured ore and are more susceptible to blockages.

Construction

Orepasses can be constructed using a raise boring machine (Figure 12.8-12), drop raising (long-hole winzing), Alimak raising (Figure 12.8-13), or conventional handheld raising. The most common methods for longer passes are a raise bore machine or Alimak raising. The preferred construction method depends on the specific requirements of the orepass, the geometrical layout of the ore-handling system, and local preference and experience. In the past, Alimak raises were more popular in North America but are becoming less common because the Alimak platform operators are exposed to vertical openings. Alimak raises have the advantage of providing access to install ground support during construction, which can provide long-term orepass stability. The Alimak raise climber shown in Figure 12.8-13 runs on a special guide rail using the Alimak rack-and-pinion system. The guide rail contains three pipes providing compressed air and water for ventilation, scaling, and drilling. Drop raises are commonly used for short orepasses (60 m or less) as blasthole deviation becomes greater at 60 m. Conventional handheld raises are generally limited to short finger raises that connect chutes to the orepass.

Table 12.8-3 Orepass design to prevent interlocking arches

Ratio of Orepass Dimension (D) to Maximum Particle Dimension (d)	Relative Frequency of Interlocking
$D/d > 5$	Very low; flow almost certain
$5 > D/d > 3$	Often; very uncertain flow
$D/d < 3$	Very high; no flow almost certain

Source: Hambley et al. 1983.

Support Systems

Ground support requirements for orepasses depend on rock mass and induced stress conditions, orientation of the orepass relative to geological structures, size and inclination of the orepass, planned mode of operation (choke fed or open), and planned life span of the orepass.

The type and method of ground support depends on the excavation method. For example, if an Alimak raise is used, it will need to be supported with rock bolts and mesh to protect operators during the construction phase. Ground support systems include metal cans or rings, concrete liners, specialized wear-resistant sprayed-on shotcrete, mesh, and rock reinforcement systems such as rock bolts and cable bolts. In some cases the cost of ground support can exceed excavation, and therefore the decision can be made to leave the orepass unsupported. This option is justified if life expectancy of the orepass in an unsupported state is sufficient. A replacement plan and monitoring program needs to be well defined and communicated to management. Rehabilitation of an orepass, sometimes considered in order to extend its life, can be a time-consuming and costly exercise, which can be avoided by installing ground support at the construction stage or by planning to construct replacement orepasses. Hadjigeorgiou et al. (2004) provides additional details of the different orepass support systems.

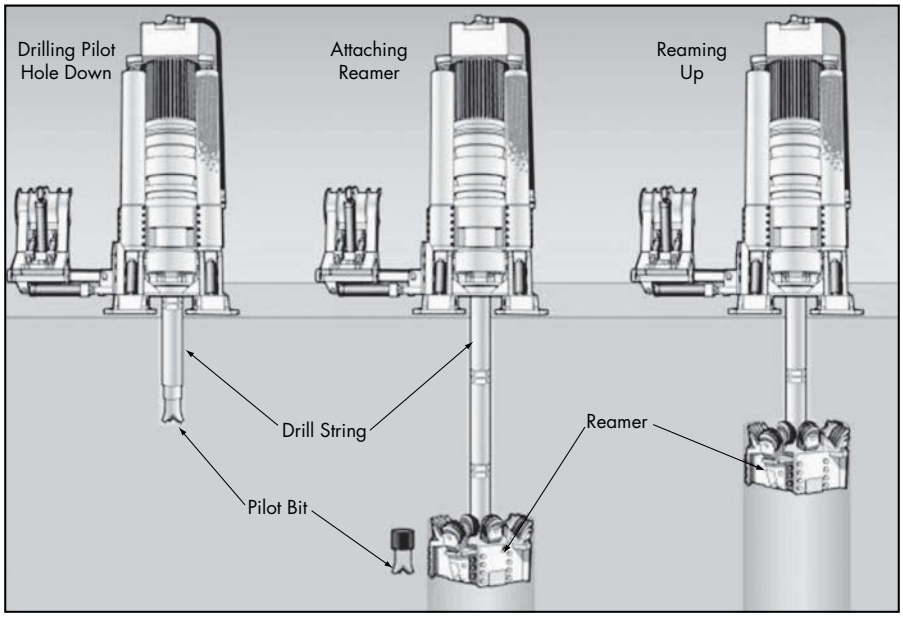
Ventilation

Good ventilation at orepass tipples and the extraction horizon is essential to maintaining a productive ore-handling system. Exhaust air from the orepass needs to be removed quickly from operating areas. Good ventilation can be provided by designing and constructing a parallel return air system, which provides effective exhaust capacity at each tipple and the extraction horizon. Spray bars can also be used to suppress dust. A number of ventilation design systems for orepasses are reviewed in Calizaya and Mutama (2004).

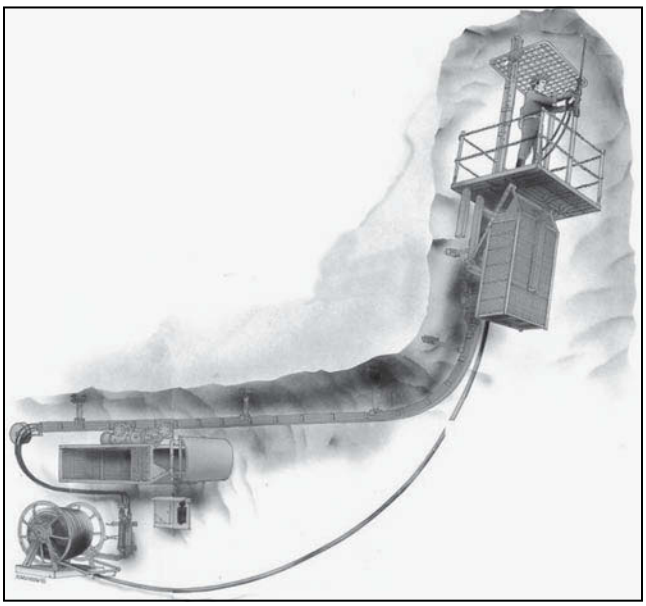
Operational Aspects

Key considerations for safe and efficient operation of an orepass are

- Controlling material size entering the orepass,
- Controlling material flow,



Courtesy of Atlas Copco.
Figure 12.8-12 Raise bore machine drilling a pilot hole and reaming an orepass



Courtesy of Australian Contract Mining.
Figure 12.8-13 Alimak raise system

- Continuously monitoring the orepass for wear and stability, and
- Maintaining good ventilation.

Over time, wear of the sidewalls can make an orepass inoperable. With good condition monitoring, orepass refurbishment or replacement can be planned in a proactive and controlled manner. If wear in an orepass goes undetected and makes it unserviceable, the ore-handling system can be disabled for an unacceptable period of time.

Orepass monitoring is an effective tool that can be used to understand the condition and potential life of an orepass. A number of monitoring techniques are available, including quantitative physical surveying of the orepass using probe hole drilling to determine the orepass profile (costly and time-consuming), a cavity monitoring system (e.g., Optech's), cavity autoscanning laser system (e.g., Measurement Devices Ltd.'s), or other laser scanning devices. Other qualitative methods include orepass camera surveying to obtain a video of the orepass, production data to determine a tonnage reconciliation of material movements, and other data captured by operations (i.e., number of hang-up blasts per time period and type of material blasted down during hang-up blasts). Orepass wear is largely caused by structural failure, stress failure, material impact, and abrasion and blast damage (Hadjigeorgiou et al. 2005). The orientation of an orepass relative to dominant structures can have a significant impact on its long-term stability.

The control of material size in an orepass is facilitated using grizzlies, scalpers, mantles, and orepass rings at the entry point. Material flow in the orepass can be controlled by chutes with chains or gates, or a plate feeder at the orepass exit point. When the orepass bottom has no control, a loader is used to control material flow. When oversize material (producing interlocking arches) and/or fines (producing cohesive arches) are introduced to the orepass, the probability of hang-ups increases and can cause delays. This either reduces throughput or increases the cost to maintain throughput by implementing an alternate ore-handling option.

Oversize can be introduced to the orepass by loaders dumping (tipping) elongated, shaped material through the size controls at the entry point of the orepass or from internal fall-off within the orepass. Reestablishment of material flow in the orepass usually requires some form of blasting to provide vibration or sometimes water flow to initiate material flow.

A comprehensive review of methods for releasing blockages is discussed in Hadjigeorgiou et al. (2004). Depending on the severity of the hang-up, it can be a long process to reestablish material flow, and redundant or alternate orepasses should

be considered at the design stage. Choke feeding of material in the orepass can limit internal fall-off by providing confinement to reduce the effect of high-velocity material impacting the orepass walls. In choke-fed orepasses, the material should be continually drawn to reduce the risk of hang-ups.

Fine material can come from backfill falloff within the stope, raise borer cuttings, or highly fragmented development material. The cohesive nature of fine material can cause blockages, especially when the material is not periodically moved in the orepass. The issues with fine material can be magnified if water is simultaneously introduced into the orepass, which increases the potential for a mud rush at the orepass exit point.

Cost Considerations

Cost estimations need to consider the expected life of an orepass and future cost of replacement. Budget capital cost estimates can be obtained from mining contractors or in-house project teams that specialize in raise boring or Alimak raising. Initial capital cost of orepass controls (chutes, feeder, and chain controls) and material sizing systems (grizzlies, orepass rings, rock breakers, etc.) can also be estimated by specialized mining contractors. Ongoing sustaining capital costs for replacement of chutes, plate feeders, grizzlies, orepass rings, liners, and rock support systems need to be factored into overall cost estimates.

Orepass operating costs include mechanical and electrical maintenance on control systems and rock breakers, and drilling and blasting costs associated with clearing orepass blockages. A good orepass design, potentially at a higher capital cost and followed up with disciplined operating practice, will minimize future operating costs and increase productivity of the orepass system.

Advantages and Disadvantages

Advantages of utilizing orepasses include

- A high-capacity system for vertical transport of material,
- Low operating cost, and
- Buffer capacity.

Disadvantages of utilizing orepasses include

- High capital cost,
- Good technical design required;
- Subject to ongoing operational challenges—for example, hang-ups;
- Performance subject to ongoing ground conditions; and
- Difficulty to maintain if ground conditions deteriorate.

CRUSHERS

Material size reduction in many operations is often required for productive and reliable materials handling. By crushing ore underground, an operation can reduce wear on equipment, reduce clearances required in materials-handling processes, increase hoist skip or other conveyance capacity (reduce material void ratio), reduce oversize handling issues such as hang-ups or blockages, and increase secondary crushing and processing productivity.

Crushers are typically required ahead of conveyors and shafts. Where the material is transported to the surface by trucks, underground crushing is generally less justified because a greater size range is capable of being transported in this manner. Although the material needs to be crushed prior to being processed, in most cases the crushing process is more cost efficient when undertaken on the surface.

In operations where underground crushing occurs, uncrushed material is usually delivered to the primary crusher by dump trucks, wheeled loaders, trains, or orepasses. A feeder device such as a plate feeder or apron feeder can be used to control the rate at which this material enters the crusher. The feeder often contains a preliminary grizzly or screening device, which ensures an appropriate material feed size is received by the crusher.

Safety Considerations

The modern crushing plant includes the following safety features:

- Safety guards around all moving equipment (Boyd 2002).
- Emergency stop facilities on components where personnel access is required (Boyd 2002).
- Dust abatement, suppression, or collection equipment: Dust emissions must comply with the latest regulations for the jurisdiction. Designers must make provisions for the installation of equipment to control these emissions.
- Spillage prevention and collection facilities: Spillages from feeders, chutes, and conveyors should also be minimized.
- Noise abatement devices and controls: Because crushers, screens, and dust-collection fans all contribute to high noise levels, strategies need to minimize such exposure to personnel (e.g., engineering solutions, restricted access, or personal protective equipment).
- Fire protection systems: Fires are major hazards in any underground mine, and the potential for fires on conveyors is high. Protection through the use of automatic sprinklers or deluge systems is important to managing this risk.

Design and Construction Issues

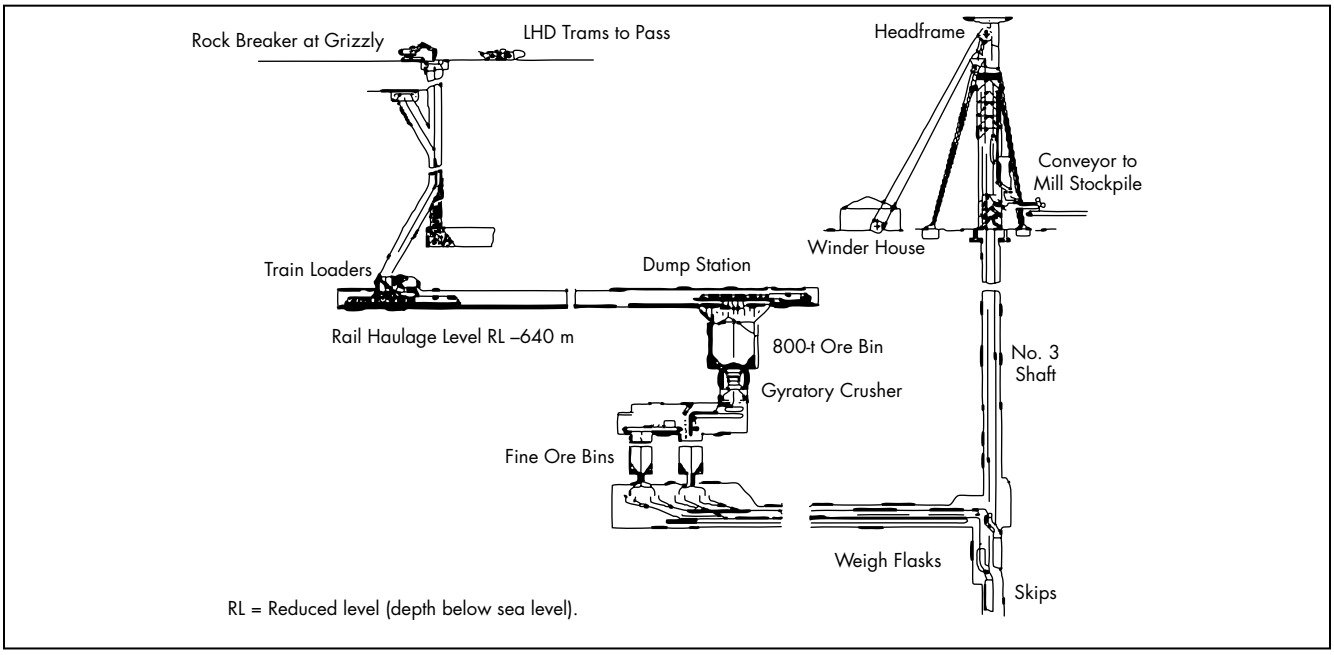
Crushing plant design can be split into three main components: process design, equipment selection, and layout. The first two are often dictated by production requirements and design parameters. The layout can reflect input preferences and operational experience of a number of parties in order to provide a balanced, workable, safe, and economic plant design. Because many underground crushing facilities are located within purposely built rock excavations, it is vital that access and maintenance of all components is sufficiently allocated during the design phase. This should include suitable lifting equipment for major component maintenance or replacement, as large mobile crane access in many underground locations is not possible.

The fundamental goal for a crushing plant design is to provide an installation that

- Meets the required production rates and material sizing,
- Can be constructed within capital constraints,
- Operates at competitive cost, and
- Complies with safety and environmental regulations.

The following critical design parameters should be considered:

- Ore characteristics
- Geographical location



Courtesy of BHP Billiton Olympic Dam.

Figure 12.8-14 Ore-handling system at Olympic Dam mine

- Expected operational life
- Expansion potential
- Safety and environment
- Operability and maintainability

Figure 12.8-14 describes the ore-handling system at the Olympic Dam mine, including the primary cone crusher.

Underground Crusher Location

In underground operations, the excavations required for the installation of crushers and associated infrastructure can be substantial and are often the largest (unfilled) excavations within the mine. When considering an underground crusher layout and location, consideration should be given to

- Local and regional ground conditions,
- Proximity to planned and potential future production areas,
- Current and future stress conditions within the excavation, and
- Proximity to planned and potential future mine infrastructure.

Good geotechnical information is essential in the selection of a crushing plant location and the extent of ground support required.

Life of Mine/Expansion Plans

Planning for expansion should be considered in the design stage for underground crushing facilities. Expansion plans for most crushing plants can be incorporated in the early planning stages at much lower cost than waiting until the mine is up and running, particularly if the cost includes an interruption to normal production activities.

Crusher Selection

Crusher selection is dependent on many operation-specific variables including throughput rates, planned utilization, ore characteristics, material feed size, and required discharge product size. Many of the factors influencing underground crusher selection are common to surface crusher selection and are not covered in detail in this section.

As a guideline, the following can be applied: For a hard-rock mine application below 600 t/h feed rate into the crusher, a jaw crusher is the preferred option as the primary crusher. For feed rates more than 1,000 t/h, a gyratory crusher is the preferred option. Between these capacities, the choice will be dependent on other factors (Ottergren 2003). An example of a primary cone crusher is presented in Figure 12.8-15.

Equipment Overview

The major equipment in a primary crushing circuit usually includes a crusher, feeder, and conveyor. Secondary and tertiary crushing circuits have the same basic equipment items, along with screens and surge storage bins.

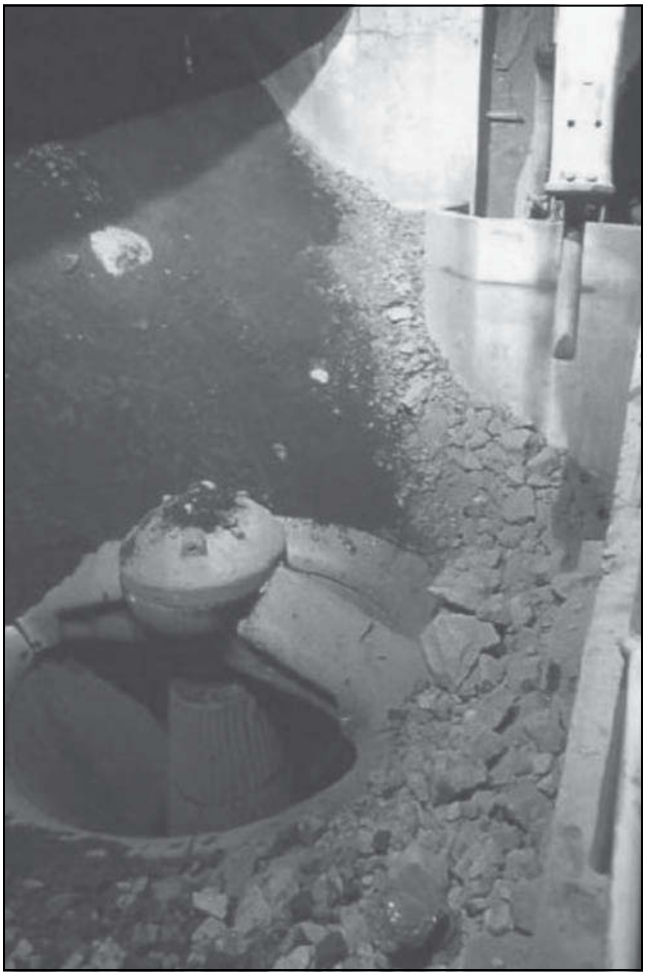
Grizzlies

A grizzly is a sizing device used to ensure a maximum passing size of material and to protect the crusher. Square mesh grizzlies get a more accurate maximum size of material than other shaped grizzlies but can be prone to plugging. Rectangular and nongrided grizzlies are less prone to plugging but can allow material of larger size to pass. Many grizzlies are rigidly mounted so that material must flow through them or be pushed through with equipment. Some grizzlies vibrate to encourage material on the grid to reorient and pass through. Grizzlies can be fed using a loader, truck, or conveyor.

Grizzlies also act as a metal scavenger to trap rock bolts, wire mesh, drill steels, and cables, removing them from the ore stream periodically by a loader from the top of the grizzly.

Rock Breakers

Fixed or mobile rock breakers are often used in conjunction with a grizzly. A modern rock breaker uses a hydraulic arm



Courtesy BHP Billiton Olympic Dam.

Figure 12.8-15 Primary cone crusher and rock breaker at Olympic Dam mine

with a pneumatic impact tip to move material around on the grizzly and impact oversize until it passes. Fixed rock breakers can be operated from a remote operator's cabin or even from the surface to reduce the installation size and ventilation needed underground. Before the advent of the modern rock breaker, oversize material feeding the primary crusher was a greater problem, and primary crushers were designed largely on the basis of gape (minimum dimension of feed opening) rather than capacity.

Additional and Optional Equipment

Other equipment items in crushing circuits can include the following:

- Overhead crane
- Freight elevator
- Service air compressor
- Sump pumps
- Air vacuum cleanup systems
- Rock grapple
- Conveyor belt magnets
- Conveyor belt metal detectors
- Belt monitoring systems
- Belt feeders
- Screw feeders
- Bin ventilators
- Apron feeder to the primary crusher
- Dust collection/suppression system
- Eccentric trolley removal cart
- Personnel elevator
- Air cannons
- Water booster pumps
- Service trolleys
- Conveyor gravity take-up service winch
- Conveyor belt rip detector
- Conveyor belt weigh scales
- Vibratory feeders
- Sampling station

Underground crushing installations are high-capital-cost items, due to fixed equipment costs and large-scale excavations required to house the system. Crusher installation capital-cost estimates should include equipment costs plus the following direct and indirect construction costs:

- Excavation and ground support
- Mechanicals
- Concrete
- Electrical
- Structural steel
- Instrumentation

Indirect costs can fall within a range of 40% to 60% of the direct costs.

Integration Aspects

As mentioned in the introduction, each step in a mining system value chain consists of an upstream buffer, an ore-handling process that upgrades the ore to another value state, and a downstream buffer. The upstream buffer in most operations will be the storage capacity of uncrushed material available for crusher feed. Within typical feed crushing configurations this may be in the form of one or a number of stockpiles, or live capacity within one or a number of ore-passes. Downstream buffer is the capacity of crushed material storage between the crushing discharge and the next step in the materials-handling system (e.g., truck, conveyor or skip load, and transport). Where such upstream and/or downstream buffer is of limited capacity or nonexistent, operational variability around the crusher may adversely affect crusher uptime. Because of the high capital costs of large underground storage facilities for uncrushed or crushed material, it may not be possible to install large buffer capacity at this step of the materials-handling process. Therefore, crushing should not be the operation bottleneck by design, and the crushing and immediate upstream and downstream processes should have capacity in excess of the bottleneck process.

Technology

For the most part, advances in crusher design have moved slowly. Jaw crushers have remained virtually unchanged since about 1950. More reliability and higher production have been added to basic cone crusher designs that have also remained largely unchanged. Increases in rotating speed have provided the largest variation. Production improvements have come from speed increases and better crushing chamber designs.

The largest advance in cone crusher reliability has been in the use of hydraulics to protect crushers from being damaged

when uncrushable objects enter the crushing chamber. Foreign objects, such as steel, can cause extensive damage to a cone crusher and additional costs in lost production. Advancement in hydraulic relief systems has greatly reduced downtime and improved the life of these machines.

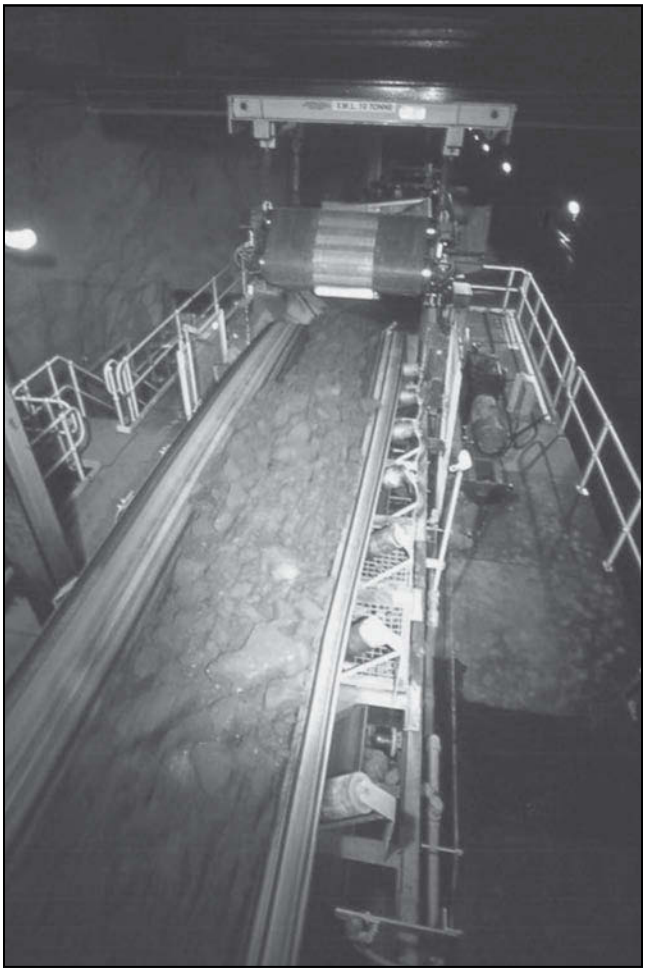
CONVEYORS

Mine conveyor systems have been utilized in underground mass mining for many years. Conveyor systems are commonly found in underground continuous coal mining operations, which use longwall and room-and-pillar methods. More recently, underground conveyor systems have been employed in large-scale, long-life metalliferous mines. Conveyors are more commonly used in operations where mine geometry and production flow allow conveyor haulage to be more efficient or cost-effective than other methods.

Inclined, troughed belt conveyors are more frequently being selected from a range of alternatives, which include shafts and trucks, for ore haulage in underground mass mining projects. Belt conveyor haulage systems are being operated with lifts exceeding those normally associated with truck haulage systems and approaching the limits of shaft haulage systems (Spreadborough and Pratt 2008).

Continued development of conveyor technology has resulted in increased confidence in and reliability of these systems. Reliability of conveyor-based haulage systems as a whole is impacted by the complexity of larger configurations constructed of multiple units or lifts. Modern conveyor installations have proven high availabilities (>85% are achievable in underground operations), which has added confidence to their use in recent times.

Figure 12.8-16 shows a typical conveyor in an underground metalliferous mine operation.



Courtesy of BHP Billiton Olympic Dam.

Figure 12.8-16 Typical conveyor in an underground metalliferous mine operation

Safety Considerations

In the selection of conveying options for an underground operation, several risk areas must be addressed, including

- Belt fires managed by self-extinguishing belt covers and auto detection and suppression systems;
- Belt failures addressed by belt-rip detection systems, monthly belt scans, and concrete bulkheads;
- Guarding requirements to prevent injury to personnel;
- Audible start-up alarms where personnel work near the belt;
- Emergency stop provisions in areas where access by personnel is required; and
- Dust management.

Selection Criteria

The selection of haulage systems for underground mass mining has historically focused on shaft haulage, trucks, and belt conveyors. The application of these alternatives in the Australian mining industry is summarized in Figure 12.8-3. Troughed belt conveyors are shown in this figure to be applied in the range up to 8 Mt/a production rate and 1,200-m lift.

Selection criteria for underground conveyor haulage include

- Inherent safety;
- Capacity (steady state and surge flow);
- Simplicity in design and operation;
- Dimensions (length, height, and width);
- Maneuverability/adaptability to various layouts;
- Cost (capital and operating);
- Reliability/availability;
- Operating life;
- Size of product handled, spillage, and carry-back;
- Dust and noise generation; and
- Automatic, remote operations.

The advantages of conveying material include high automation, minimal operational labor requirements and lower operating costs, and high reliability/availability.

The disadvantages of conveying material include high capital cost (therefore lower probability of duplication), a large footprint, and limited flexibility.

Belt conveyors for underground service usually have a more rugged design and operate at slower speeds than a comparable overland conveyor. Hard-rock mine belt conveyors normally require the ore to be crushed before it is conveyed or at least sized through a grizzly. Reasons for this include increased belt life due to reduced impact from lumps and elimination of tramp material, including rock bolts, rebar, drill steels, and scaling bars, which can seriously affect the integrity of the belt system.

Belt conveyor systems are less flexible than truck haulage and require a high initial investment. This generally means that belt conveyors are the economical choice when the

production rate is relatively high and the transport distance is significant. In certain applications, belt conveyor systems are selected for other reasons. For example, short conveyors are employed underground to optimize feed control to a loading pocket and prevent a run of fines from reaching the shaft.

Elements in a conveyor haulage subsystem can include

- Feed chute of dimensions appropriate for the material received,
- Belt to support the material and to transfer tractive forces to the material,
- Idlers to support the belt,
- Pulleys to resist belt tensions at changes of direction,
- Drives to provide driving and braking force,
- Take-up rollers to provide belt tension for no slip at the drives and for belt sag control, and
- Discharge or transfer chute.

Maintenance equipment at the haulage subsystem will typically include

- Monorails and/or cranes at drives and pulleys for change-out or repair,
- Belt clamps to resist belt tensions when installing or maintaining the belt (spring applied, hydraulic release, tested for holding capacity),
- Belt reel handling facilities, and
- Belt splicing/repair facilities.

Integration Aspects

When considering the conveyor components of the materials-handling system, consideration should be given to the upstream and downstream buffers. The upstream buffer in most operations will be the storage capacity of crushed material available for conveyor feed. In typical crushing/conveying configurations this may be in the form of fine ore stocks within dedicated stockpiles or storage bins. The downstream buffer may be underground storage at transfer points to the next step in the materials-handling system or surface stockpile storage capacity.

Technology

Advances in belt technology and particularly the development of stronger belt carcasses support the potential for further increases in capacity of inclined conveyor haulage systems. Vertical conveyors and hydraulic hoisting represent two potential technologies for haulage from underground mines.

The attraction of vertical conveying is due to the combination of a small footprint, continuous process, and overall energy efficiency. Limiting factors for vertical conveyors include tensile strength of the steel cored belts, safety of belt splices, and required production rate (Pratt 2008). Current precedence for the application of vertical or high-angle conveyors in a mining context is limited to a few hundred meters.

Other special types of belt conveyors include extendable systems, cable belts, and high-angle belts. To date, these special conveyor types have had few applications in hard-rock mines.

Conveyor Costs

Conveyor haulage systems are considered high in capital and low in operating cost compared with truck haulage. Generally, shaft haulage options attract higher capital costs and marginally lower operating costs for a similar-scale operation.

Operating costs will rise for conveyors as the complexity of the system increases with depth as a result of the addition of lifts and transfer points. This does not necessarily favor selection of shaft haulage over a conveyor system because other factors must be considered. Examples of aspects to be considered are horizontal transportation distances, confidence in geotechnical prediction for shaft and conveyor decline options, and other functional requirements of the mine. The actual selection of a haulage strategy for a large, deep mine will require evaluation of the trade-offs in risks and opportunities identified by the various options available.

Limits of Application

The slope of a high lift conveyor group is limited by the slope of the decline/incline development in which the conveyor is installed. This is normally in the range of 1:5.3 to 1:5.4 (10.5° to 10.7°) where the development is carried out using conventional rubber-tired equipment. Length and lift are limited by the belt carcass construction and the strength of the belt splice. The choice of belt construction is also limited by the troughability of the belt. Troughability is the ratio of the cross-belt sag to belt width and generally decreases with increasing carcass strength.

Future Applications

Advances in future designs will be fundamentally linked to the load-carrying capacity of available belt constructions and splice designs. Advances in conveyor technology continue to ensure success of conveyor systems in mass underground mining applications, including improved drive control equipment, belt condition monitoring, improved splice designs for higher strength and longer life, and improved chute designs.

RAIL HAULAGE SYSTEMS

Although belt conveying remains the dominant underground haulage system in coal mines, rail haulage continues to be favored in large tonnage (>5 Mt/a), long-life underground metalliferous operations.

Haulage by rail can be limited to ore and/or waste haulage, or can include integrated rail systems for transportation of personnel, materials, and supplies throughout the mine. Although the following discussion focuses only on the haulage of ore and waste from production areas to the hoisting system, it includes the consideration of interactions with other rail system users where required.

Prior to the emergence of trackless or rubber-tired mobile equipment in underground mining in the 1960s and 1970s, rail or tracked haulage systems were predominant. Many of today's rail haulage mining operations remain a legacy of that period. With improved productivity and performance of diesel mobile equipment and belt conveyors into the 1980s, this equipment has grown in popularity and application as significant methods of underground haulage. However, rail haulage continues to offer competitive operating cost, productivity, and throughput benefits over a range of applications in many underground mining settings.

Rail haulage systems consist of three primary types defined by their power supply: diesel, battery electric, and overhead trolley electric. A diverse range of systems are used around the world, from small-scale selective mines producing less than 100 ktpa to the largest bulk mining operations producing greater than 20 Mt/a.

Health and Safety Considerations

A fundamental consideration in haulage method selection and design is to ensure a safe and healthy workplace. Some of the key potential hazards that require consideration in rail haulage design and potential mitigating actions within a health and safety management system follow:

- High-speed equipment
  - Traffic management plan
  - System access control
  - Adequate lighting
  - Speed limits
  - Comprehensive equipment operating procedures
- Interaction between people and equipment
  - Equipment control systems
  - Physical barriers and guarding
  - Design for visibility
- Unmanned/automated equipment
  - Access control
  - Fail-to-safe control systems
  - Communication systems
- Overhead exposed conductors
  - Permit-to-work process
  - Isolation systems
  - Electrical protection systems
  - Entry height barriers
  - Water and dust management program
- Maintenance hazards
  - Custom-designed and fit-for-purpose maintenance facility
  - Regular inspection and maintenance schedules
  - High standard maintenance equipment
  - Isolation systems
  - Regular track cleaning program
- Mud rushes from chutes and orepasses
  - Water controls at all orepass entry points
  - Orepass monitoring
  - Regular chute inspections and/or continuous remote closed-circuit television monitoring
- Fire
  - Automatic fire suppression systems
  - Emergency response teams and training
  - Good housekeeping

Common and paramount in all the these circumstances are the underlying skilled workplace supervision, customized training programs, skills assurance process, good safety signage, and proactive maintenance programs that meet OEM (original equipment manufacturer) requirements at a minimum.

Selection Criteria

Selection of the preferred haulage system for a mine must consider a wide range of factors, including health and safety, economic, technological, reliability, operability, and flexibility.

Rail haulage can exist either as a stand-alone lateral haulage system or as part of an integrated system. In the case of the former, for example, rail provides for all lateral movement of personnel, materials, ore, and waste in the deep-level South African gold mines and northern Idaho (United States) silver mines. In the case of the latter, rail haulage can operate with mobile equipment, as at the Olympic Dam mine. Ultimate selection needs to match the shape, size, nature, and productive capacity of the ore body within the framework of an economically feasible operation.

The optimal haulage system for a mining operation is dependent on a variety of factors and their interrelationships, the more significant of which are discussed in the following paragraphs.

Health and Safety

Of fundamental importance are the design, installation, and operation of a safe rail system, as discussed previously.

Capital Cost

Rail haulage systems typically have a high capital cost and long construction time compared with alternative haulage systems. This is because economically competitive rail systems are often large scale by nature and require

- Long distances of tunnel development, which takes years to excavate;
- Orepasses and chute systems for train loading;
- Long-term ground support systems;
- High upfront equipment capital purchase costs for locomotives, rail, rolling stock, tipplers, trolley line system, and power reticulation equipment (where installed); and
- Large construction labor and installation costs.

Operating Cost

Converse to capital cost, rail haulage operating costs are among the most efficient per ton of material moved. This is driven by comparatively lower maintenance costs due to a more highly engineered and thus reliable system compared with diesel mobile equipment, requirement for a low number of operators, and better energy efficiency. Spare parts inventories may also be reduced.

Reliability and Operability

Well-engineered rail haulage systems can achieve high system availability (greater than 85% compared to diesel truck fleets, which range from 50% to 75% of total available clock time). The fixed nature of rail systems and repeatability of processes makes it highly attractive for automation, with the potential to deliver increased utilization. State-of-the-art systems, such as LKAB's 26-Mt/a rail system at the Kiruna mine in Sweden, CODELCO's 48-Mt/a Teniente 8 railway at the El Teniente mine in Chile, and BHP Billiton's 8-Mt/a rail system at Olympic Dam in South Australia, are highly automated, requiring high levels of technological support while delivering substantial unit operating cost and productivity benefits (Figure 12.8-17). Due to the fixed nature of rail systems, maintenance can be readily planned and scheduled, spare locomotives can limit production interruptions, while a smaller work force is typically required per ton hauled. Rail haulage systems require specific skill sets to maintain locomotives, high-voltage reticulation systems, communication systems, automation, and control systems.

Production Rate and Scale of Operations

Given the higher capital cost of rail haulage systems, they are typically used in operations with high production rates or long lives, in order to pay back the high upfront costs. Likewise, the high productivities and low unit operating costs offered by rail haulage favor their selection in mines with high throughput rates and/or long lives. However, smaller-scale rail systems



A. LKAB, Kiruna mine, Sweden B. BHP Billiton, Olympic Dam mine, Australia C. CODELCO, El Teniente mine, Chile

(A) Courtesy of LKAB; (B) Courtesy of BHP Billiton Olympic Dam; (C) Courtesy of P. Crorkan-Tait.
Figure 12.8-17 Underground rail haulage systems

are also competitive in mines where the reduced flexibility of a fixed horizon haulage system can be managed. This is particularly relevant in large flat-lying ore bodies where long-haul distances may exist, where larger openings for other forms of haulage cannot be considered, or where diesel mobile haulage systems are avoided because of environmental conditions (e.g., heat, gases, and fire risk).

Ore-Body Geometry

Rail haulage systems are by nature horizontal installations, and their selection must consider the lower flexibility of their fixed nature. In general, larger ore bodies are more suited to rail haulage as production is sourced from one or a few (albeit large) areas as opposed to disseminated, narrow-vein, or dispersed ore bodies where production comes from numerous locations in differing amounts at different elevations. However, rail haulage can be a competitive haulage option in such mines if the fixed nature of rail systems can be managed and matched to other aspects of the operations design.

Production Growth

Although diesel mobile equipment has the flexibility to travel wherever tunnel access is provided (flat or inclined), rail haulage is a constrained system. In order to support growth of operations to new areas or greater depths, rail haulages need to be extended or replicated to support mine growth or extension, or to sustain production from new areas. The 25-Mt/a rail haulage system of LKAB's Kiruna sublevel cave iron ore mine is a good case study, where major rail haulage horizons have been replicated at increasing depth in 300-m vertical increments on 10-to-15-year intervals as the mine has progressively moved its production areas downdip. Consideration at the design stage can enable rail systems to increase their productive capacity, if required, through measures such as adding more rolling stock, upgrading locomotive capacity, or increasing haul speeds.

System Life

The operational life of haulage systems and equipment is a key criterion in selection of haulage systems. Rail systems generally have a longer equipment life than mobile haulage equipment, which requires replacement nominally every 5 to 7 years. High throughput rail systems require longer operational life in order to provide an acceptable return on capital invested.

Mine Plan

Selection of haulage systems must be aligned and complementary to the mine plan. Rail haulage is more typically suited to deeper shaft hoisting operations where ore and waste need to be collected centrally for crushing, transport to, and removal from a single elevation, typically at depth. Development of ore bodies in areas of high topographic relief offers the potential for adit access and efficient rail haulage directly from the mine to the surface processing facility (such as Freeport-McMoRan Copper and Gold's deep ore zone mine at Grasberg in West Papua, Indonesia, and CODELCO's El Teniente mine).

Design Issues

Inclusive in the economic analysis of selecting a cost-effective haulage system are four key aspects for rail haulage design. Inherent to this analysis are capital and operating costs. The following factors need to be evaluated in a fully integrated design approach:

- Capacity
- System layout and geometry
- Materials properties and materials handling
- Equipment

Capacity

The required ore production rate is the primary driver of rail haulage system design. This will determine such key design parameters as tunnel layouts and size, equipment capacity, operating philosophy, the number of units in the fleet, integration with upstream and downstream production process capabilities, and haul speed. Consideration also needs to be given to whether the system is dedicated to ore haulage or is to be shared with movement of personnel and/or materials, and haulage of waste as well as ore.

Due to the fixed nature of rail haulage system infrastructure, consideration should be given at the design phase for the potential and likelihood of future increased production or extension of mining operations to new areas. In such circumstances, engineering designs should consider such key items as the adequacy of tunnel size and extent, maintenance facilities, power supply systems and infrastructure, track quality, and the number and configuration of orepasses and dumping (tipping) points.

System Layout and Geometry

The layout and extent of a rail haulage system requires economic trade-off studies of numerous factors that impact the geometric design, including

- Physical dimensions and depth of the ore body;
- Degree of coverage required by the haulage system beneath the ore-body footprint;
- Primary mining excavation costs (including ground support, final preparation, and drainage);
- Trade-off between optimal rail haulage design and operation, and optimal production mining systems in the mine workings above;
- Single or dual direction of travel;
- Haulage speed (a key driver of productive capacity);
- Radii of curves, single or double rail, tunnel dimensions, gradient, maximization of straights, and super-elevation of curves;
- Number and location of ore and/or waste delivery points into the rail system;
- Number and location of ore and/or waste dumping (tipping) points from the rail system;
- Future operating cost impacts and benefits; and
- Suitability for future expansion.

With an inherent need for wide radius curves to accommodate rail equipment and a preference for straight sections, rail haulage horizons can lend themselves to potential mechanical excavation techniques such as tunnel boring machines. These are commonly applied in civil engineering rail projects and warrant consideration in the preliminary phase of mine rail haulage system design.

The schematic presented in Figure 12.8-18 demonstrates the preference for geometric rail loop layouts, wide radius curves, and long straights.

Material Properties and Materials Handling

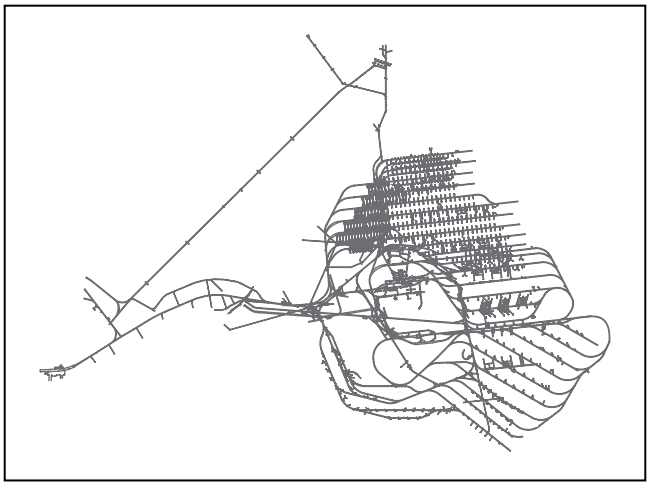
Rail haulage systems must be designed to manage the physical attributes of the material(s) to be hauled. For example, the physical properties of the material will determine the strength of mechanical equipment against wear and tear, and flow properties will impact orepass and chute design and determine the potential need for lining of passes, bins, or chutes for longevity.

The key physical properties to be considered include

- Moisture content;
- Size distribution, proportion of fines (mud-rush hazard), and maximum lump size;
- Specific gravity/material density;
- Abrasiveness;
- Dust generation;
- Presence of deleterious materials, such as tramp steel, cemented backfill, and waste dilution; and
- Changes in material properties over time.

Careful management of these properties has a significant downstream impact on system operability through aspects such as

- Pass and chute maintenance;
- Spillage, which can cause safety hazards, derailments, and significant downtime or equipment damage;
- Train hopper wear and tear;



Courtesy of P. Crorkan-Tait.

Figure 12.8-18 Schematic of Teniente five-rail haulage design layout at El Teniente mine in Chile

- Downtime due to blockages, hang-ups, fines rushes, etc.; and
- Maintainability and maintenance cost.

Equipment

The selection of rail haulage equipment is a key component that must be fully integrated with all the previously discussed aspects of system design. Locomotive prime movers are typically powered by either battery electric, diesel, or overhead electric energy sources. The majority of large-scale, high-production underground rail haulage systems today are overhead DC electric inverted to AC-drive locomotives. Examples of such systems are shown in Figure 12.8-17.

Ore/waste hoppers and tipplers are available in various configurations, with capacities ranging from 1 t to more than 70 t per hopper. Hoppers can be either side or floor dumping (tipping), with a range of chassis, side-door, and floor opening designs available depending on various parameters. Train combinations can be sized to match production and rail system capability with rakes of up to 30 hoppers and total train payloads in excess of 1,300 t.

Rail haulage systems lend themselves to significant levels of automation. Design of remote control systems is critical to operating productivity and reliability, and needs to consider locally available skill sets, technical support, spare parts availability, mine communications systems, and environmental aspects such as water, humidity, dust, radiation, and gases. Train control systems must be fully integrated with the proposed operating philosophy for the haulage system, with the aim of preventing downtime and production delays, and minimizing maintenance.

Other equipment aspects to consider in system design include

- Rail gauge and installation quality;
- Control and communications systems;
- Switch design;
- Train loading points, passes, and chute design;
- Tippler design;
- Drainage and lighting; and
- Signaling.

The maintainability and operability of all equipment requires close attention. This includes access to all parts of the rail system for maintenance personnel and equipment in the event of derailment or breakdown; specialized equipment for rail maintenance; standby equipment, including spare locomotives and hoppers/ore cars; critical/insurance spares; and regular spare parts management.

HYDRAULIC HAULAGE SYSTEMS

Hydraulic ore transport is an often cited but rarely implemented ore transportation system in underground mining applications. While slurry pumping engineering is mature in a multitude of diverse surface applications, its serious consideration in underground environments is hampered by risks associated with

- The need to pretreat and prepare suitable material underground, beyond typical crushing and sizing arrangements;
- Pump and column wear considerations;
- Capital costs and equipment technologies for anything more than very small tonnage rates; and
- Lack of courage to lead change.

One well-established application of underground hydraulic transportation and hoisting is at the McArthur River uranium mine in Saskatchewan, Canada, operated by Cameco (H. Goetz, personal communication). McArthur River produces approximately 47,000 tpa of high-grade (average 17% U_3O_8 [triuranium oxtoxide]) uranium-bearing ore, from a depth of 640 m below the surface. The operation includes full ore preparation underground through a 600-t/d semiautogenous grinding mill, hydrocycloning, and 2 × 13-m-diameter thickeners. The thickener underflow feeds into a 200-m^3 thickener underflow holding tank that supplies the Wirth triplex piston/diaphragm positive displacement pumps (13 million Pa discharge pressure) for pumping up 2 × 100-mm-diameter columns in a single stage to the surface.

The physical characteristics and health and safety aspects associated with handling relatively low quantities of very high-grade uranium ore, with attendant high radiation levels, were a key driver in the selection of hydraulic transportation ahead of more conventional methods.

Hydraulic systems for transporting backfill are described in a separate chapter.

OTHER HAULAGE SYSTEMS

A small number of less common haulage systems are used in underground mines. These are typically selected because of highly specific geological environments, particular material properties, or unique operating systems and processes. These include the following:

- Rope scrapers/slushers—commonly used for ore haulage from operating stopes to orepasses in South African longwall gold and platinum mining operations. Slushers are also used for ore collection, haulage, and delivery into orepasses beneath established block caving operations.
- Monorails—traditionally a materials-handling system. Monorail systems can be either electrohydraulic friction or rack-and-pinion drive, and have been used in moderately dipping applications (up to 14°) in South African gold mines.
- Floor-mounted, rope-driven trap rail haulage systems—applied for haulage of equipment and ore/waste up or down steep inclines of up to 40°.

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CHAPTER 12.9

Hoisting Systems

Peter Tiley

INTRODUCTION

When it has been determined that a mine will include an underground section, it is necessary to review what kind of system will be used to transport workers and material to and from the surface to the underground, as well as to haul ore and waste from the mine to the surface.

This chapter deals with the selection, design, operation, and maintenance of hoisting plants that operate in vertical or near-vertical shafts. The term *hoisting plant* is used to collectively describe the openings and the equipment being considered. Information on the design and technical considerations to be reviewed when selecting a particular component or arriving at an arrangement of the hoisting plant is provided.

SHAFT HOISTING SYSTEMS

A shaft hoisting system consists of the following major sections, starting from the bottom of the shaft:

- A loading station for ore or a service station for workers and material
- Shaft conveyances called skips for ore transport and cages for transporting workers and material
- Ropes that suspend the conveyances
- A shaft that connects the underground to the surface and that is equipped with a system to guide the conveyances as they move in the shaft
- A headframe, located on the surface, that supports either the hoist itself or supports head sheaves that direct the headropes from the top of the conveyance to a ground-mounted hoist and that provides the tipping arrangements for the rock
- A hoist and hoist room

The headframe also typically includes an ore storage area into which ore and waste are temporarily discharged before being transferred to some other surface facility.

HOISTS

Two basic types of hoists are commonly used today: drum hoists, in which the hoist rope is stored on a drum, and friction hoists, in which the rope passes over the wheel during the hoisting cycle. Within each category are several variations.

Drum Hoists

Figure 12.9-1 shows the various arrangements of drum hoists. Drum hoists are usually located some distance from the shaft and require a headframe and sheaves to center the hoisting ropes in the shaft compartment and maintain a rope fleeting angle of less than 2°.

Single-Drum Hoist

The single-drum hoist may be used for balanced or unbalanced operation. When used for unbalanced hoisting, the cost of the electric drive may become quite high for long hoisting distances and high tonnages. This is because the motor must be large enough to produce sufficient torque to handle the combined weight of the rope, conveyance, and payload in the absence of a counterweight.

These costs are, however, offset by the reduced size of the installation because there is one less compartment, conveyance, head sheave, and head rope to accommodate, as well as a smaller hoist foundation and building.

There is virtually no difference in the power costs for balanced versus unbalanced operation, because power costs are for the energy consumed, which is derived from the number of tons hoisted.

As the shaft gets deeper, the difference in the size of the electrical drive for balanced versus unbalanced operation becomes less significant because the mass of the unbalanced rope, which is the same whether there are one or two conveyances, begins to dominate the total unbalance experienced by the hoist.

Single-drum production hoists are unusual because their hoisting efficiency is halved for the same skip size, but they are fairly common when used for service duty in which the hoisting cycle time is not as significant.

In a balanced hoisting system, one rope winds off the drum as the other winds on. When used with a skip or cage in balance with a counterweight, a single-drum hoist can service one or more levels because the location of the counterweight is not important. When used with two skips in balance, the single-drum hoist is only suitable for single-level hoisting. Any rope adjustments to locate the conveyance must be done manually. For shallow shafts with one layer of rope, no dividing of the

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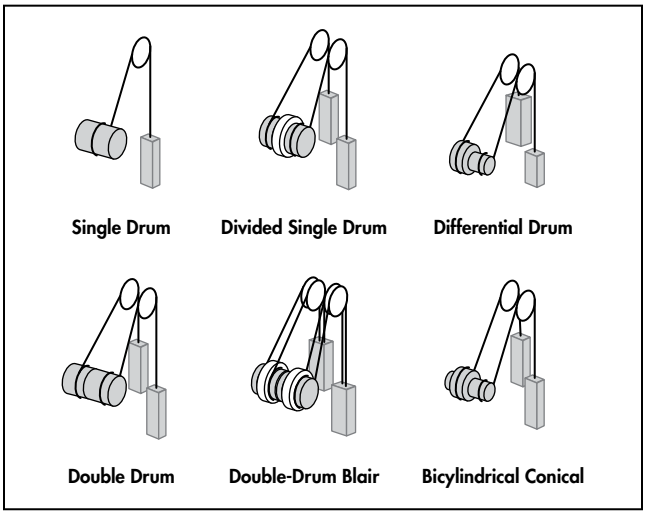


Figure 12.9-1 Drum hoists

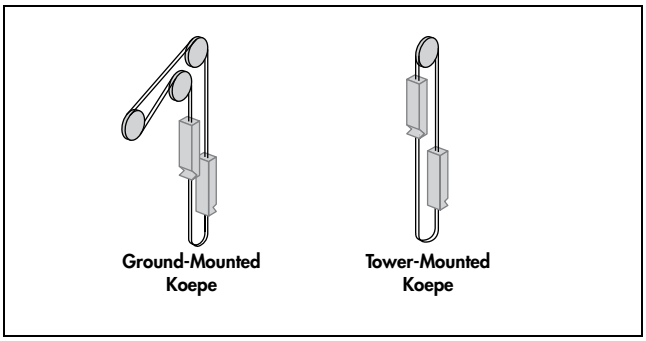


Figure 12.9-2 Friction hoists

drum is required. For deeper shafts, the drum must be divided to wind several layers of rope.

Divided Single-Drum Hoist

This type of hoist is used for deeper shafts with balanced hoisting when several layers of rope must be stored on the drum. Peak horsepower is less than with unbalanced hoisting because the skip weights are balanced. Because the payload and weight of the rope is not balanced, the maximum unbalanced load occurs when the loaded conveyance is at the bottom of the shaft.

Split Differential Diameter Drum

A third, though uncommon, type of single-drum hoist is the split differential diameter drum. In this arrangement, each of the drum compartments is of a different diameter. This type of hoist is used with a conveyance and counterweight in balance or with a skip and cage in balance. If, for example, the counterweight rope is wound on the smaller-diameter drum, it moves less than the main conveyance, and rope adjustment problems are reduced.

Double-Drum Hoist—One Drum Clutched

Although more expensive than a single-drum hoist, the double-drum hoist with one drum clutched has certain advantages. With this type of hoist, it is possible to make quick adjustments to the ropes due to initial stretch. As a service hoist with cage and counterweight, this type of hoist can serve several levels efficiently. As a production hoist with two skips, the ropes can be adjusted to maintain balanced hoisting at any level in a multilevel operation.

Double-Drum Hoist—Both Drums Clutched

The double-drum hoist with both drums clutched has the added feature of allowing hoisting to continue in one compartment should something happen to the other compartment. This is an excellent feature if there is only one hoist available. This type of hoist is also favored during shaft-sinking operations.

Multiple-Rope Hoist—Blair Type

With this type of hoist, each conveyance is suspended from two hoist ropes that are each coiled onto the same drum. The advantage of this is that by doubling the number of ropes per conveyance, the conveyance payload can be increased. This type of hoist was developed in South Africa for deep mines, typically more than 1,800 m (5,900 ft). Another advantage of this arrangement is its ability to dispense with safety dogs on the cage (which are only required for single-rope hoists).

Two such hoists are now operating in Canada, and several more are scheduled to be installed in the near future.

Bicylindrical Conical Drum Hoists

This type of hoist is no longer manufactured, but there are still a few in operation in North America.

The drums on these hoists, both single- and double-drum configurations, are made in two steps with a conical section in between.

The hoists were developed before multilayer winding was practical. Prior to multilayer winding, drum diameters simply increased to match depth. This practice leads to very large diameters (10 m [33 ft] or more), resulting in very high starting torque requirements. By beginning the acceleration on a small diameter, the high starting torque requirement could be avoided.

Friction Hoists

Figure 12.9-2 shows the two possible arrangements of friction hoists. The Koepe or friction hoist was developed by Frederick Koepe in 1877. It consists of a wheel with grooved liner or liners made from a friction material to resist slippage. The hoist rope is not attached or stored on the wheel. In early installations, the hoist was mounted on the ground, and a single rope was wound around the drum and over the head sheaves to the two conveyances in a balanced arrangement. In addition, a tail rope of the same weight per unit length as the headrope was suspended in the shaft below each conveyance. Thus the only out-of-balance load was simply the payload. Friction hoists can be either tower mounted or ground mounted.

Tower Mounted

The main reason for selecting tower mounting is the reduction in cost of the structure, because the rope loads are confined internally. This arrangement also provides a very small footprint that allows the structure to be supported by the headframe's collar concrete in the slip-formed design or by localized piled foundations in the structural-steel design of the headframe.

Because the Koepe wheel diameter is generally larger than the conveyance compartment centers, deflection sheaves

must also be installed in the headframe below the hoist to move the ropes over to match these centers.

Tower mounting is also preferred in cold climates because the conveyance ropes are isolated from the weather.

Ground Mounted

A headframe for a ground-mounted friction hoist must be designed to resist the overturning loads produced by many headropes. These buildings are generally of a structural steel design and require the use of a pair of cluster sheaves to locate the rope centers over the conveyance compartments. Because they must support the entire suspended rope load, rather than simply deflecting the ropes by 10° to 15°, the cluster sheaves are much more robust and are, therefore, significantly more costly than the deflection sheaves used in tower-mounted applications.

Comparison of Friction and Drum Hoists

A friction hoist system differs from a drum hoist system in performance as well as components. Therefore, when deciding which type of hoist to use, it is necessary to compare the two complete systems rather than the two hoists alone. In addition to comparing the total capital costs of the hoist, headframe, ropes, conveyances, and shaft, it is necessary to consider operating costs, maintenance costs, reliability, power supply system, local custom, and individual preference.

The following general statements help distinguish between these two hoisting systems:

- Double-drum hoists are the preferred hoist for shaft sinking.
- Double-drum hoists are the best choice for hoisting in two compartments from several levels.
- Drum-type hoists are best suited for high payloads from shallow depths.
- The limitation on a drum hoist employing a single rope is the ultimate strength of the rope, because large ropes are difficult to manufacture and handle.
- The depth capacity of drum hoists can be extended by using two ropes per conveyance (Blair-type hoist), and with this arrangement, Blair hoists can be used for depths exceeding those of either single-rope drum hoists or friction hoists.
- Friction hoists with multiple ropes can carry a higher payload and have a higher output in tons per hour than drum hoists within a range of depths from 460 to 1,520 m (1,500 to 5,000 ft).
- Friction hoist mechanical operation is very simple, has a low rotational inertia, and is less costly than a drum hoist.
- Friction hoists have a lower peak power demand than drum hoists with the same output.
- The friction hoist can operate on a relatively light power supply.
- Rope maintenance is more intensive and difficult for friction hoists.

Hoist Components

The major components that make up a hoist are the drum or wheel, bearings, gearing, brakes, clutch motor, power conversion, and controls.

Drum

The drum of a drum hoist must be designed to store the amount of rope required for the shaft depth and accommodate the stresses caused by the rope tension loads. The prediction of these rope loads is well documented in the literature.

Although originally provided with no machined grooves (i.e., they were smooth), drums today are generally supplied with parallel grooving, which includes a half-pitch crossover twice per revolution. This grooving pattern was developed by LeBus International, Longview, Texas, to achieve stable spooling in multilayer winding. Drums can also be supplied grooved in a helical pattern, which is a continuous spiral from one end of the drum to the other. This pattern is limited to three layers of rope, because beyond this number of layers, the rope-winding pattern becomes unpredictable. LeBus International's grooved drums, in contrast, have been used to wind as many as 15 layers of rope without a problem.

Rope resonance must be considered in the selection of the grooving pattern. This is discussed in a later section of this chapter.

Wheel

The wheel of a friction hoist is not subjected to crushing forces. As a result, the drums are much lighter and are made wide enough to accommodate the rope-end attachment dimensions, as well as the space for the deflection sheaves.

The linings for the rope treads on Koepe hoists were originally made of wood or leather, which were then replaced by modern plastic.

Bearings

As shown in Figures 12.9-3 and 12.9-4, the hoist drum or wheel is supported on a shaft that rides in two or more bearings. Bearings provide the mechanism that supports the shaft loads, and bearings allow it to rotate at the same time with very little friction.

There are two types of bearings: sleeve bearings, which use a film of oil to keep the shaft out of contact with the fixed part of the bearing, and roller bearings. Sleeve bearings are being supplanted by spherical roller bearings that have the following advantages:

- Low coefficient of friction
- Economy of space
- Simplicity of lubrication
- Practical elimination of wear due to roller point contact
- Maintenance of accurate alignment
- Consistency in design and manufacture

The main disadvantages of the latter are the fact that they are not split, which complicates their replacement, and long order times are involved when bearings larger than about 500 mm (19.7 in.) in diameter are required.

Gearing

Hoists can be driven either directly by large-diameter, relatively low-speed AC or DC motors (90 rpm or less) or through a gear-reduction system by small-diameter, high-speed (400 to 1,800 rpm) motors. With the change in power conversion to variable-frequency drives, the motors have altered to AC machines operating in the 900-to-1,800-rpm range.

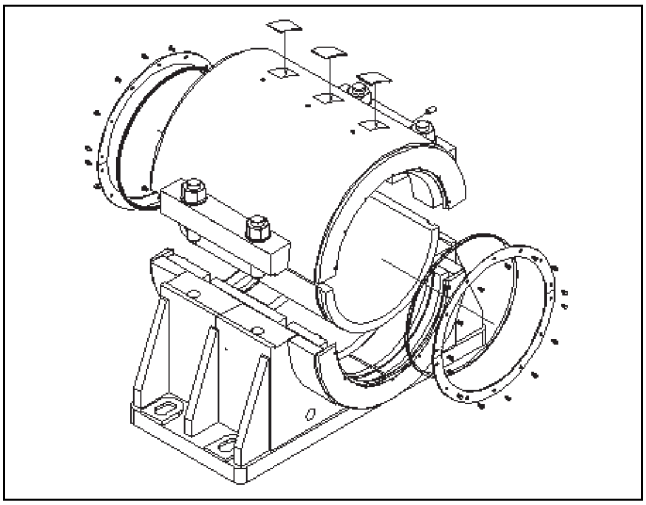


Figure 12.9-3 Sleeve bearing

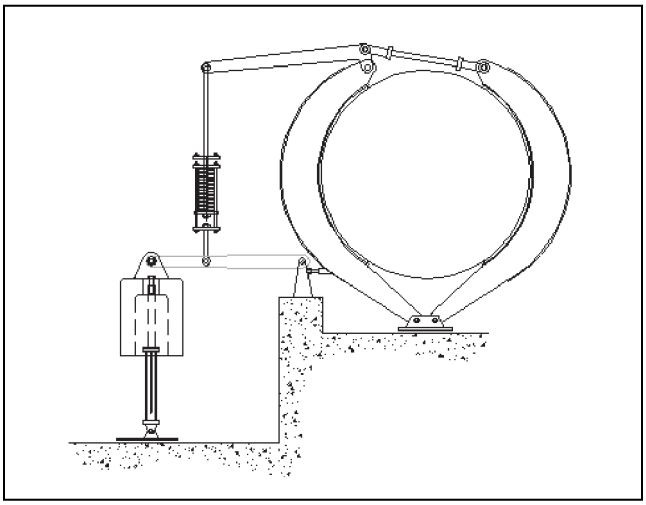


Figure 12.9-5 Jaw braking system

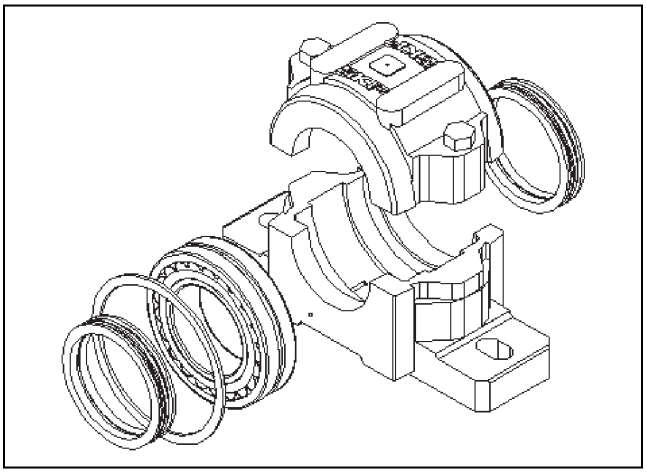


Figure 12.9-4 Spherical roller bearing

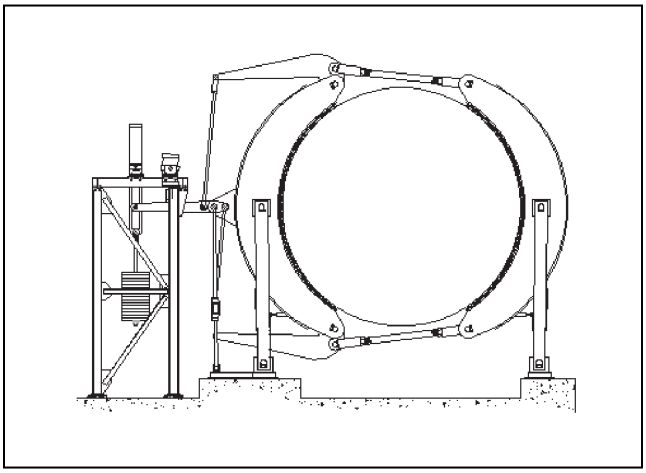


Figure 12.9-6 Parallel motion braking system

Gear reduction was commonly a single-stage system that incorporated a large gear fixed to the drum shaft and driven by one or more pinion gears. This cannot be used practically for the sealed, self-lubricating, multistage reducers now for high-speed AC motors. Reducers are now fully enclosed, multistage, and equipped with oil coolers as well as bearing vibration-monitoring systems. These utilize flexible couplings between the motors and high-speed input shafts as well as between the low-speed output shaft and the hoist drum shaft.

Brakes

A hoist must be equipped with a mechanical braking system to decelerate, stop, and hold the load. Although the electrical motor, drive, and controls can initially decelerate and stop the hoist, this equipment cannot hold the load because of thermal limitations inherent in the drive. Electrical systems also require a source of energy to function. For these reasons, hoists must be provided with mechanical braking systems that ultimately rely on either gravity or spring force to operate.

Drum brakes and disc brakes are the two main types of braking systems. Until the mid-1990s, drum hoists were exclusively equipped with drum brakes in jaw or parallel motion configuration, mainly because they were part of the hoist speed-control system rather than solely for holding or emergency braking functions. Most new hoists, both drum and friction, are equipped with disc brakes.

Figures 12.9-5, 12.9-6, and 12.9-7 illustrate the designs of jaw, parallel motion, and disc braking systems, respectively. Brake control systems are now required to control deceleration rates to very high accuracy to protect passengers from injury, limit the dynamic stresses that arise in the hoisting ropes during emergency stops, and prevent rope slip on Koepe hoists. This level of accuracy is readily achievable with the higher operating pressures and lower inertias associated with disc brakes.

In addition to accurate control of deceleration rates, braking systems must be capable of dissipating the energy absorbed during emergency stops. Disc brakes have lower mass than drum brakes and run at higher surface pressures. Therefore, they tend to heat up more quickly. By manufacturing brake discs in several segments and bolting the segments to the hoist drum or wheel, a disc brake can be designed to expand freely under thermal load and, therefore, avoid distortion or failure.

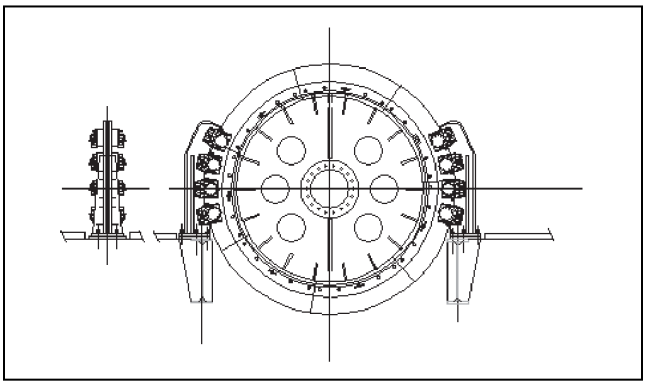


Figure 12.9-7 Disc braking system on a Koepe hoist

Clutch

Double-drum hoists used for unbalanced hoisting, such as when sinking or when operating from several levels, must have at least one of the drums clutch-connected to the drum shaft. This assembly consists of a clutched or floating drum that is supported on the shaft through a sleeve bushing and a clutch arm, which is used to transfer the torque to the drum. The clutch arm consists of a spider or gear that slides axially along the shaft supported by a hub driven by keys or splines or driven by a hexagonal section of the shaft. The drum also incorporates a clutch ring, which has teeth matching the clutch arm on its entire periphery, bolted to the side of the drum. As the spider moves axially, these two sets of teeth engage or disengage. The tooth pitch determines how accurately the rope lengths on each drum can be matched. Driving torque is then transmitted from the clutch spider to the drum through the bolts used to attach the clutch ring to the drum.

As the drum must be prevented from rotating when it is unclutched, it is necessary that the clutch's operating mechanism and the brake holding the drum be interlocked both electrically and mechanically.

Drive Motor and Power Conversion

Hoists have used energy sources such as steam, compressed air, and internal combustion engines to drive them. These have largely been replaced with electrical power.

Electrical drives for hoists originally employed AC wound-rotor induction motors using variable resistance in the secondary for speed control. These systems were then supplanted by DC machines that utilized motor-generator sets to provide DC power. The motor-generator sets were replaced by so-called static drives in the late 1960s. The name *static drive* referred to the fact that the power was converted from AC to DC using thyristors, which are made from silicon wafers and involve no moving parts. The DC technology began to disappear in the late 1980s when it was replaced with AC technology that utilized synchronous motors and variable-frequency sources called cycloconverters. The variable-frequency sources have now been replaced with pulse width-modulated, digitally controlled equipment that utilize transistors (injection-enhanced gate resistors or isolated gate bipolar transistors) for power conversion. The AC motors are now a mixture of synchronous and squirrel-cage designs, with the squirrel-cage design being the preferred option because of its simplicity and ruggedness.

This new generation of variable-frequency static drives also provides AC power with low harmonic content and ensures operation at unity power factor. These drives consist of two inverters back to back. The first converts line frequency AC to DC; the second, DC to variable frequency AC.

Hoist Control Systems

The hoist control system consists of two components: active and passive. The active component starts and stops the hoist, controls its speed throughout the hoisting cycle, and displays the state of the hoist to the operator. The passive component monitors the speed and position of the hoist itself as well as the state of its subsystems, such as the brakes, hydraulic power units, motor ventilation, and similar auxiliary equipment. The passive component also signals the active component when the system is operating outside of its normal range and thereby initiates a stoppage of the system until the fault that raised the alarm is cleared.

All of the required functions are now fulfilled by digital devices call programmable logic controllers. These devices utilize redundancy to a large degree to ensure that they are reliable.

The operator's interface has also evolved to what is called a human–machine interface, which is a graphical display provided on a computer monitor.

HOIST SELECTION

The capacity of a hoist depends on the drum or wheel dimensions, number and size of ropes, speed, and horsepower.

The mechanical components, such as drum, shaft, bearings, gearing, and brakes, are svized to match the rope loading; the electrical components are sized to match the torque required to handle the loads and the speed at which the hoist will run.

The rope is selected to handle the required payload while operating at a reasonable or legislated safety factor.

Selection Procedure for Production Drum Hoists

Sizing the hoist runs through an iterative process, which requires the following initial information:

- Hoisting depth
- Required annual production
- Hours available for hoisting

The process of selection is as follows:

1. Using the depth, calculate the hoisting cycle time. The cycle time will depend on the maximum speed, acceleration and deceleration rates, creep speeds and distances (arriving at and departing the loading pocket and dump), and the loading/dumping time when the hoist is stopped.
2. Varying these items will enable the designer to optimize the selection. Good guidelines are
   - a maximum speed of 5 m/s (1,000 fpm) per 300 m (1,000 ft) of depth up to a maximum of 20 m/s (4,200 fpm);
   - acceleration and deceleration times of 15 seconds;
   - a creep speed of 1.5 m/s and a creep time of 5 seconds; and
   - load and dump times of 10 to 15 seconds, depending on density of the load.
3. Use the cycle time to calculate the cycles per hour.

4. Use the required annual production and available hoisting times per year to calculate the skip payload.
5. Use the skip payload and a reasonable skip tare mass based on 70% of the skip payload to determine the total attached load.
6. Multiply the attached load by 8.5 to determine the required rope strength.
7. Select a rope with this strength.
8. Check that the rope strength also meets the required safety factor (typically 5 but may also have a factor that is depth dependent) when the rope weight is added.
9. Determine the drum diameter based on 80:1 drum-to-rope diameter (this ratio depends on the rope size).
10. Estimate the motor power using kW = (payload × speed)/102, where the payload is measured in kilograms and the speed is measured in meters per second.
11. Estimate the inertia of the rotating parts based on historical data.
12. Carry out a duty cycle calculation to verify the required motor power.

A typical double-drum production hoist cycle, starting with empty skips and loading pocket, no power on the hoist motors, and with the hoist brakes applied, is as follows:

1. Open the loading pocket door and load the empty skip.
2. Close the loading pocket door.
3. Apply power to the hoist.
4. When power is established (torque proving), release the brakes.
5. Accelerate to creep speed, about 1 m/s, and creep away from the end zones, about 2 m (6 ft), to clear dump scrolls.
6. Accelerate to full speed in 15 to 20 seconds.
7. Run at full speed.
8. Decelerate to creep speed in 15 to 20 seconds.
9. Enter dump scrolls at creep speed, about 0.75 m/s (150 fpm), and creep through dump scrolls until the full skip is in fully dumped position or empty skip is in position at the loading pocket.
10. Stop the hoist.
11. Repeat the cycle.

It is good practice to stop the cycle when the empty skip stops at the loading pocket rather than the full skip arriving at the dump point. Headframes are designed to allow for rope stretch and rope length mismatch, which means that the full skip can stop anywhere in about a 1- to 2-m (3- to 6-ft) zone when it is dumping. This is not possible at the loading pocket, because the headrope will stretch when the skip is being loaded and the loading window in the skip is limited in height.

Selection Procedure Drum Hoist for Service

The selection procedure for a service hoist is much simpler than a production hoist because service hoists are sized to handle a maximum payload at a speed that is largely independent of depth. In this case, the following guidelines apply:

- The maximum speed is 8 m/s (1,500 fpm).
- The acceleration and deceleration times are 20 seconds.
- The creep speed is 1.5 m/s (300 fpm), and the creep time is 10 seconds.
- The waiting time is 30 seconds.

The rope size is determined as before, except the cage weight is generally 50% of the payload.

When using a counterweight, it should be sized as the cage mass plus one-half the cage payload, which ensures that the maximum average unbalance for the hoisting cycle will never exceed one-half the payload.

Selection Procedure for Koepe Hoists

The selection procedure for a Koepe hoist is identical to that of a drum hoist, except that the number of ropes must be defined. A good starting point is to use four ropes, as this provides a reasonable limit on complexity while minimizing the size of the ropes.

One item that is different is the mass of the conveyances. As the ratio between the tension in the ropes on the loaded side of the wheel and the tension in the ropes on the empty side of the wheel must be kept below about 1.5, the conveyances must be heavy. A good guideline is to have the conveyance weight equal to the payload and then increase it until the ratio is 1.5 or less.

Koepe hoists often run on rope guides, so provision must be made in the hoisting cycle to slow the conveyance down as it enters the ends of travel where it is guided by fixed guides. These guides are required to keep the conveyances' lateral motion in check when it is traveling in the narrow confines of the headframe and shaft bottom. The guides are tapered to allow for the uncertainty in the conveyance position and are often called "spear" guides as a result.

The conveyance speed when entering these guides is usually kept below 3 m/s (600 fpm), but it can be as high as 6 m/s (1,200 fpm) if the system is properly designed.

The production hoisting cycle for the Koepe hoist is identical to the drum hoist, except for a situation when the skips are running on rope guides. This means that a high creep speed section will occur as the skip slows down to enter the spear guides and then runs in them until it reaches the dump point at which time it will decelerate to a low creep speed.

In addition to determining the hoist's capacity, consideration of the following items is critical for achieving efficient and trouble-free operation of friction hoists: T_1/T_2 ratio, rope tread, deflection sheaves, differential rope loads, and tail ropes.

T_1/T_2 Ratio

The T_1/T_2 ratio is the ratio of the static rope tensions on each side of the drum, where T_1 is the weight of loaded conveyance plus weight of ropes, and T_2 is the weight of unloaded conveyance plus weight of ropes. The maximum recommended value for a particular installation is a function of the coefficient of friction of the tread material, the acceleration and deceleration required, and the angle of lap of the rope on the drum. A typical maximum static value of T_1/T_2 is 1.5. It is also necessary to consider the dynamic loads that occur when the hoist is accelerating and decelerating, both under the normal and braking conditions. Because acceleration will decrease the rope load and deceleration will decrease it, T_1/T_2 can become too high, causing the rope to slip on the drum. When this happens, the coefficient of friction drops and control is lost. There will also be tread damage due to friction temperature rise and abrasion.

The maximum coefficient of friction that should be used when calculating the allowable T_1/T_2 ratio is 0.25. This number is achievable even with wet ropes, as long as they are not overlubricated.

The tension increase during acceleration is given by the factor $(1 + a/g)$; the tension decrease during deceleration is

give by the factor $(1 - a/g)$. Here, a is the acceleration/deceleration and g is the acceleration due to gravity (9.81 m/s^2 [32.17 ft/s^2]). Thus, the dynamic tension ratio is the static tension ratio $T_1/T_2 \times (1 + a/g)/(1 - a/g)$.

The amount of unbalance that the friction can support is given by

$$T_1/T_2 = e^{(\mu \times \theta)}$$

where

e = base of the natural logarithm (2.713)
μ = coefficient of friction
θ = angle of wrap measured in radians (typically π or 3.14 radians, equivalent to 180°)

As an example of the effect of dynamic variation in the rope tension on the margin to rope slip, assume that the actual static tension ratio is 1.5 and that the system has been designed with an angle of wrap of 180°. A coefficient of friction of 0.25 will produce an allowable static tension ratio, T_1/T_2, equal to 2.2, well above the actual static T_1/T_2 of 1.5.

If the hoist is subjected to a deceleration rate of 3 m/s^2, the dynamic T_1/T_2 increases as follows by the factor $(1 + 3/9.81)/(1 - 3/9.81) = 1.88$, giving a dynamic T_1/T_2 of $1.5 \times 1.88 = 2.82$. This is greater than 2.2, indicating that the ropes will slip if the coefficient of friction is only 0.25.

This example shows why braking systems for Koepe hoists must be carefully designed.

Rope Tread

Selection of an appropriate tread material for the drum rope grooves and tread pressure is important. As the rope passes over the drum, the load and thus the strain in the rope changes. This change in strain tends to cause the rope to creep (as opposed to slip) on the drum. A resilient tread material such as polyurethane tends to flow with the change in strain and reduce creep.

Typically, the average tread pressures (i.e., the sum of the rope tensions/projected rope area on the drum) of 2,069 kPa (300 psi) with lock coil ropes, 1,724 kPa (250 psi) with flattened strand ropes, and 1,379 kPa (200 psi) with round strand ropes on plastic treads will give good life.

More recently, nonrotating flattened strand ropes have replaced lock coil ropes in North America. To compensate for the greater tendency of these ropes to wear the tread liners, the liners have been replaced with harder plastics. Although these materials lack the resilience of polyurethane, good rope life can still be obtained provided the hoisting plant is properly designed in the first place. This is explained in greater detail in the sections that follow.

Deflection Sheaves

Deflection sheaves are used when the pitch circle diameter of the friction hoist pulley wheel (sitting above the shaft) is greater than the center-to-center distance of the compartments. In addition to positioning the hoisting ropes, the use of deflection sheaves has the advantage of increasing the angle of contact of the ropes on the wheel, permitting a higher T_1/T_2 ratio before slippage will occur.

However, deflection sheaves have the disadvantage of requiring additional torque during the hoisting cycle, increasing the height of the headframe and putting reverse bending into the ropes, which can reduce their life.

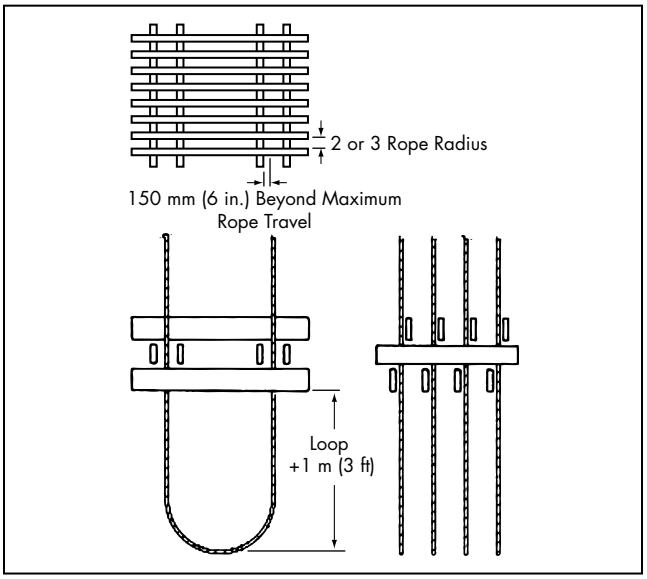


Figure 12.9-8 Tail loop divider

Differential Rope Loads

A great concern with multirope friction hoists is the equalization of loads between the ropes. The two most common causes of differential loading are a difference in rope length or in circumferential lengths of each rope groove on the wheel.

Analysis of this problem and procedures to identify and correct the effects are well covered in the technical literature. In the initial design stage, it is important that steps be taken to reduce its effects. Because there are no hard and fast rules, the following guidelines have been developed in the industry.

The ratio of rope lengths (skip at loading pocket to hoist wheel)/(skip at dump to hoist wheel) should be greater than 50. The total number of revolutions of the drum per trip should be less than 80.

The number of ropes should be kept to a minimum. There will be less difficulty maintaining equal tension on a four-rope hoist than on a six-rope hoist. In addition, a larger wheel will be required for a four-rope drum having fewer revolutions per trip, lower rope bending stresses, and lower sensitivity to tension variations. This requirement may conflict with rope size choice, however.

Rope attachments should have both fine and coarse length adjustment. Rope attachments that incorporate a hydraulic cylinder to balance the loads equally should always be used.

Tail Rope Loop Dividers

To avoid mishap and possible mechanical damage through entanglement, the balance rope loop area should be well controlled in the sump. One method is by constructing a loop divider. The balance ropes should be restricted quite closely in the direction of their lateral movement.

Early contact with the dividers between individual ropes will provide good control in the sump area and also assist in ensuring that adequate stability through the shaft is maintained. Longitudinal movement, on the other hand, should not be overly restricted. The dividers should be designed to avoid trapping any material from spillage. Some operators have found that a wire rope equipped with numerous ultra-high molecular

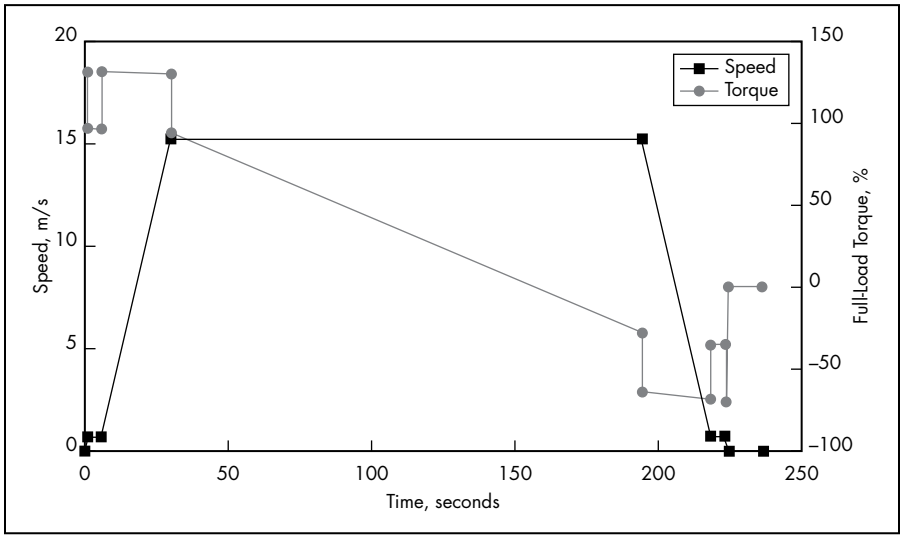


Figure 12.9-9 Double-drum hoist duty cycle

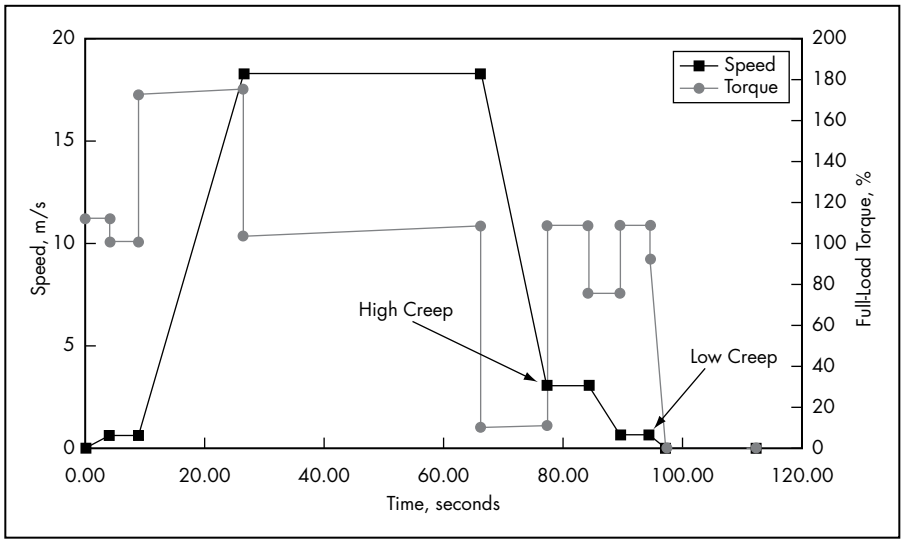


Figure 12.9-10 Koepe hoist duty cycle

weight polyethylene rollers, about 150 mm (6 in.) in diameter and placed in the configuration shown in Figure 12.9-8, has proven to be most successful in this connection. The elevation of the loop divider for good results is one loop diameter plus approximately 1 m (3 ft) above the bottom of the loop. Some operators have placed such dividers at several elevations in the sump area to ensure good contact over the height of the tail loop.

DUTY CYCLES

A duty cycle is a determination of the root mean square (RMS) thermal and torque load on the electrical equipment. The calculations assume that motor current is directly proportional to motor torque.

Duty cycles are usually carried out assuming the motor is force-ventilated. If the machine is self-ventilated, then factors have to be applied to account for the amount of time the motor spends accelerating and decelerating and at rest. These factors differ for AC and DC machines because of the differences in their construction.

The heating effect on each segment of the duty cycle is calculated using

$$\text{heat generated} \sim \text{torque}^2 \times \text{time}$$

The rate of cooling on each segment of the duty cycle is calculated using

$$\text{equivalent cooling time} = \text{cooling factor} \times \text{actual time}$$

The thermal load over the whole duty cycle is the square root of the sum of each segment, that is

$$\text{RMS torque} = \sqrt{\frac{\sum \text{torque}^2 \times \text{time}}{\sum \text{cooling time}}}$$

These results are reasonably accurate for force-ventilated motors but can be misleadingly low for modern self-ventilated machines.

A typical speed-and-torque-versus-time chart corresponding to a duty cycle for a double-drum hoist is shown in Figure 12.9-9 and for a Koepe hoist in Figure 12.9-10.

The drum hoist usually has a significant regenerative portion in its cycle because the large unbalance caused by the rope changes from a positive load to a negative one when the conveyance passes the shaft midpoint. Because Koepe hoists utilize balance ropes, this reversal of load is absent from the cycle.

HOIST UTILIZATION

The previous section outlined the procedure for sizing the hoisting system based on the production or service duty required. Inherent in this procedure is a step for determining how much production time is available each year.

Many mines have multipurpose shafts, that is, the shaft will provide both production and service capacity for the mine. Production and service can proceed at the same time if the shaft is equipped with both a service and a production hoist, as long as the compartments are physically separated by brattice or the personnel are not transported in the shaft while it is in production. It is also possible to use the same hoist for both production and service duty, but this requires a much higher degree of planning and leaves little room for any hiccups in the mining process.

Table 12.9-1 demonstrates the daily examination and testing required on a hoist in North America. In addition to examination and testing, time must also be allotted for maintenance of the hoists, conveyances, shaft, and shaft furnishings. The amount of time required will depend somewhat on the shaft configuration and hoist type, but all of the items listed in Table 12.9-2 are required to some degree.

Experience has shown that the average time available for production hoisting can vary from 12 to 20 hours per day when spread out on an annual basis. For most mines, the production time available will be between 16 and 18 hours per day.

The total hoisting time will also depend on the shift system in practice at the mine. Although most production hoists are automatic, they may have to be shut down during shift change if proper isolation between the cage hoist and production hoist is unavailable or if there is only one combined production/service hoist. Bearing all of this information in mind will permit the mine operator to determine how many hours of actual hoisting is available on an annual basis. Most hoisting plants today will achieve about 95% availability, as long as time has been allotted for their proper maintenance.

Table 12.9-1 Daily hoist examination and testing required in North America

Procedure	Time, minutes
Run through shaft	5
Test overwind and underwind protection	10
Test brakes (dynamic drag)	10
Verify safety devices	20
Examine head- and tail ropes	30
Examine rope attachments	15
Examine hoist and deflection sheaves	10
Total	100

Table 12.9-2 Annual shaft maintenance summary for a shaft with two hoists

Activity	Hours per Activity	Annual Total, hours
Daily hoist examination and testing*	1.5	433.5
Daily rope examination	4	1,196
Weekly hoist, rope, and conveyance examination*	8	208
Weekly shaft equipment and guide examination*	4	208
Monthly rope adjustment and examination	8	96
Monthly shaft equipment repairs†	8	96
Annual rope electromagnetic testing (4 ropes)	32	32
Annual hoist testing	16	16
Annual brake testing	8	8
Biannual rope change	48	24
Total	—	2,317.5
Annual hoisting time with 1-week shutdown	357×24	8,568
Available hoisting time	8,568–1,121.5	6,250.5
Average per day	7446.5/357	17.51
Shaft availability	—	73%

*Will require one crew per hoist.
†May be done on special shutdown.

CONVEYANCES

The transportation of ore, waste, personnel, and material requires the use of a conveyance. Conveyances for handling ore and waste are termed *skips*; those used to carry personnel and material are termed *cages*.

Skips

The skipping cycles discussed above have implicit assumptions as to what type of skip is being used. There are three main types of skips in use today: the overturning (Kimberley), the swing-out body, and the fixed body, as shown in Figures 12.9-11, 12.9-12, and 12.9-13, respectively.

Overturning Skip

The overturning or Kimberley skip consists of a bail frame, a shaft to support the skip, and the skip body itself. The skip body rests on the shaft. The skip is dumped through the action of a bull wheel mounted on each side (at the top) of the skip body. Each bull wheel engages a scroll, causing the skip body to rotate around the shaft as the skip is being hoisted through the dump. The design of the tipping path is complex because the payload is being discharged as it moves along the path. The horizontal and vertical forces that need to be considered in designing the scrolls and their support steel are dependent on the speed at which the skip enters the dump, the inertia of the skip plus payload, the acceleration the skip produced by the curvature of the dumps scrolls, and the gravity forces.

The advantages of Kimberley skips are that there is less shaft spillage when handling fine or wet material and that they can handle larger pieces for a fixed cross-sectional area. The disadvantages are that the capacity is limited because of the height-to-width ratio; the payload ratio is usually lower, particularly with sticky material that tends to build up in the bottom; they generally impose higher stresses on the headframe and bins; and they require more headframe height to dump.

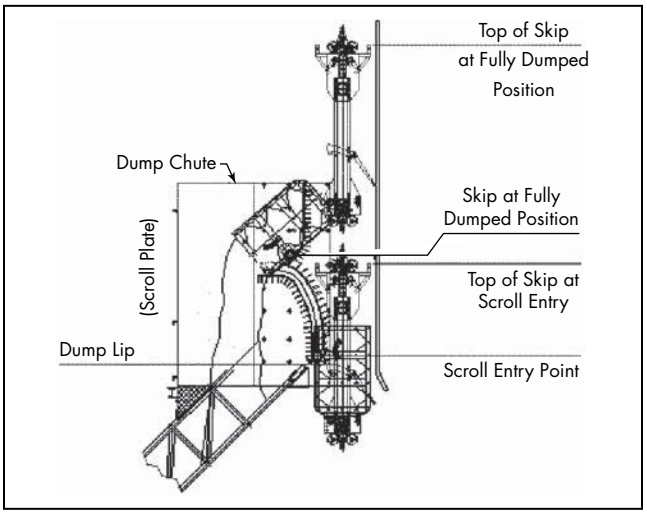


Figure 12.9-11 Overturning (Kimberley) skip

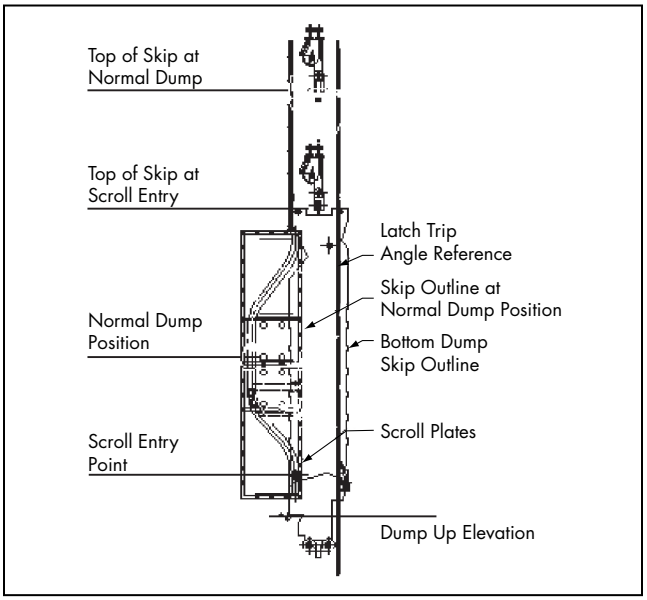


Figure 12.9-12 Bottom-dump skip (the most popular type of swing-out-body skip)

Swing-Out-Body Skip

The swing-out-body skip also consists of a bail frame, a shaft, and the skip body. In this case, the skip body is suspended from the shaft and the skip is dumped by means of rollers mounted on each side (at the bottom) of the body. The rollers engage a scroll and cause the body to pivot away from the bail into the dump. At the same time, a discharge door, which forms the bottom of the skip, opens and the material is discharged. The skip can be dumped by using either a mechanical operator or scroll or by using a hydraulic operator that pulls the skip body out of the bail. These options are illustrated in Figures 12.9-14 and 12.9-15.

For bottom-dumping skips in general, the cross-sectional area is partially determined by the size of the material to be handled. The door opening is sized to be at least three times the dimensions of the largest piece to be handled to prevent possible bridging.

The advantages of swing-out-body skips are that they have a more favorable payload ratio than the overturning type, the tipping path is less complex than for the overturning skips, the dumping distance in the headframe is less than for the overturning type, and they can be built to handle much larger payloads because dumping does not involve lifting the entire mass of the skip body and its payload. The main disadvantage is that muck buildup at the hinge point in the door can prevent the door from closing and thereby place large forces on the headframe scroll or skip body during the closing cycle. In addition, with wet material, there could be spillage or dribble from the door lip; and the bottom door mechanism is more complex than in the fixed-body type.

Fixed-Body Skip

In fixed-body skips, the body of the skip remains fixed and does not move out of the confines of the skip compartment during dumping. Instead, a door at the bottom of the body swings open to form a chute and allows the material to discharge into the dump bin. This type of skip may or may not have a bail.

There are several types of fixed-body skips. The most common are the Rolachute, Arc-Gate, and Sala-type skips. The difference among these types is the mechanism involved with the opening and closing of the door. In the Rolachute skip, the door is held closed by means of a door latch. When the skip enters the headframe, a latch track releases the latch, allowing the door to open and the material to discharge. With the Arc-Gate skip, a roller is mounted on each side of the door. When the skip enters the dump area, the roller engages a scroll, causing the door to swing open. In the Sala-type skip, the door is operated by means of air cylinders mounted on the skip body. Compressed air to operate the cylinders is provided through a yoke that is permanently mounted in the headframe and picked up by the skip as it enters the dump. The Sala design, with its onboard air cylinders, adds additional complications brought about by the requirement to interface with an external source of air.

The cross-sectional area of fixed-body skips is determined by the size of the material to be handled. Because fixed-body skips can be built without a bail, the cross-sectional area of the skip can be increased by 10% to 15% when compared to skips with a bail for the same sized compartment.

The advantages of fixed-body skips are as follows:

- The skip can load and dump through the guides.
- The skip has no bail, thus allowing a larger cross-sectional area to be used for the skip body (this can reduce the overall length of the skip for a given capacity).
- The only moving part is the door that forms its own dump chute.
- The skip requires less travel distance to open the door (none at all with the Sala-type skip).
- This type of skip places less stress on the headframe, skip body, and ropes during the dumping operation.

Both the Arc-Gate and Sala types have the added advantage that no part of the skip, particularly the dump door, protrudes beyond the skip body, thereby avoiding any possibility of hanging up or impacting shaft furnishings should the door fail to close completely.

The disadvantages of this type of skip are as follows:

- Problems can occur with sticky muck, as there is no shaking action when dumping.

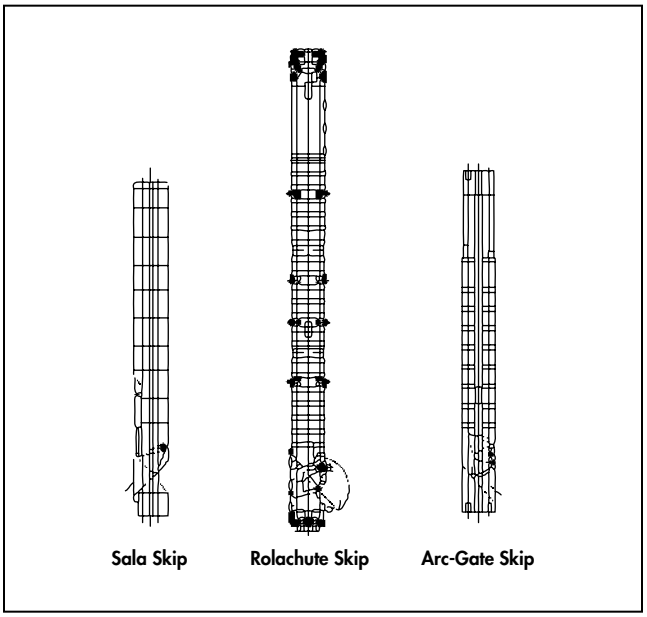


Figure 12.9-13 Fixed-body skips

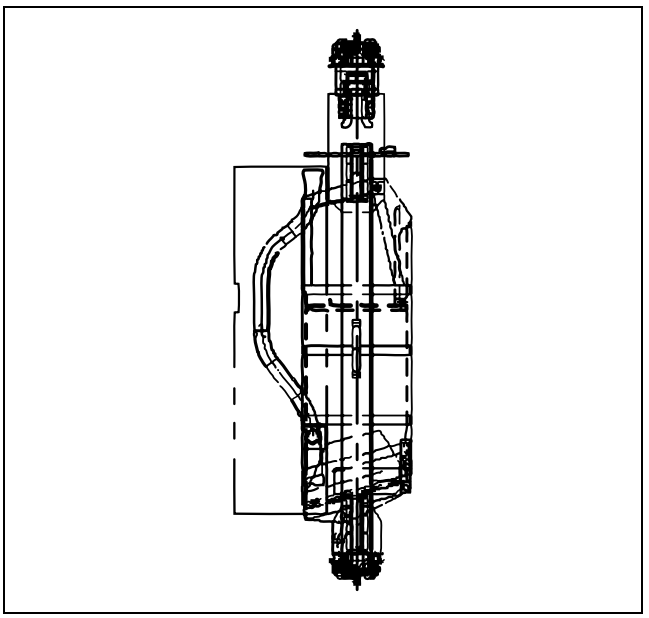


Figure 12.9-14 Dump with mechanical scrolls

- The body necks down at the transition to the dump portion of the assembly, and this can cause hang-ups.
- The entire skip must be removed for replacement of wear plates, not simply the body.
- There is always some dribble into the shaft because the chute does not extend beyond the projection of the skip's body (except with the Rolachute design).

Skip Design Considerations

When selecting a particular skip, the components to be considered during the evaluation process include design methods, body, doors, crosshead, bail, guide rollers, and guide shoes.

Design Methods

Skip components may be designed based on experience and practice or through analytical methods that define the loading being placed on the various components of the skip. The types of loading to be considered are dynamic loading (caused by filling, hoisting, and dumping the skip) and static loading. Factors of safety are usually specified by legislation.

Body

The skip body is designed as a rectangular box with stiffeners. The type of material, its thickness, and the stiffener size and spacing are the variables to be examined. Particular attention must be given to the area on the backside of the skip that receives impact and abrasion from loading.

Skip bodies are usually lined with abrasion-resistant material to increase overall life. Two types of steel and elastomers have been used for skip liners. Abrasion-resistant steels are members of the high-strength, low-alloy family of steels. Manganese steels are used because they have the ability to work-harden during service. Elastomers have the advantage of reduced weight, less ore sticking and buildup, corrosion resistance, and ease of handling during replacement.

The use of aluminum is also increasing, particularly in drum hoist applications. Such skips have a lower mass and allow a higher payload to be hoisted.

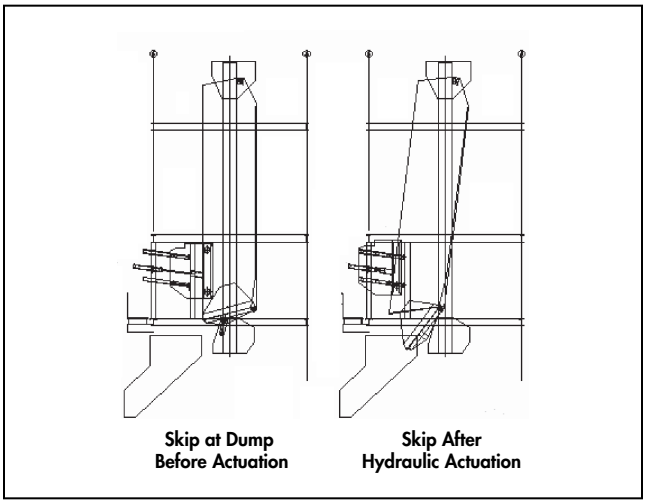


Figure 12.9-15 Hydraulic dump before and after actuation

The impact areas are often lined with heavy rubber materials that were originally designed for use inside of rotating grinding mills. These liners are quite thick, typically 75 mm (3 in.), so they must be accounted for when estimating the live volume of the skip.

Doors

Both swing-out-body and fixed-body skips have doors at the bottom of the skip. Doors must be designed to withstand the impact of large pieces falling on the floor during loading, the static loading of a full skip, and abrasion during unloading.

With swing-out-body skips, the door is hinged at the back, and several problems are associated with leakage paths between the door and the body. These door leakage problems may be solved through the use of rubber sheets, flexible materials, or special modification of plates at the door. Fixed-body skips generally have fewer problems with leakage or door

closure. Problems may occur with both types when they are loaded on the side opposite the dump site. In this case, a vertical stop may be placed for the dump roller to react against. Doors are made and lined with the same material as used in the body.

Crosshead

The crosshead fastens the bail to the rope. It must be designed to take all the skip loads, including acceleration, impact, and service factors. The location of the crosshead above the skip is particularly important during loading. It should not interfere with the flow of muck from the loading pocket. The rope stretch in deep mines can be several feet and must be considered.

Bail

During hoisting, the bail supports the body pivot, prevents the body from moving, and supports the door. At the dump, the bail is subjected to horizontal loading and supports the door. It is thus subjected to both bending and axial loading.

Guide Rollers

The purpose of guide rollers is to keep the guide shoe off the guides. This in turn results in a smoother ride, reducing dynamic loading between the skip, guide, and buntons when traveling.

When selecting guide rollers, there is usually a conflict between a large-diameter tire, adequate shaft clearance, and the cross section of the skip. Left unresolved, the final design usually results in a small-diameter tire. Such tires need frequent replacement. Tire sizes range from 150 to 350 mm (6 to 14 in.) in diameter. Tires are rated on a load-speed basis. High-speed hoists require special guide rollers that are typically made from wire-reinforced rubber. Guide rollers should be part of an assembly comprising springs and shock absorbers in order to work effectively. Systems in which the guide roller is simply fixed to a bracket bolted to the skip are of little use.

Guide Shoes

Guide shoes are used on vertical conveyances to keep the main structure from being worn away by the guides. They are considered as expendable items to be replaced when they are too worn for further use. Clearance between the guide and guide shoe varies from 6 to 12 mm (0.25 to 0.5 in.). Guide shoes can be made from many materials, such as cast iron, mild steel, hardened steel, abrasion-resistant steel, brass, and bronze. The amount of wear to be expected is directly proportional to the load on the guide shoe and inversely proportional to the hardness of the material.

Cages

Cages are used primarily to move personnel and materials entering into and exiting from the mine. In some instances, cages are used to hoist cars loaded with ore and waste. The design and construction of cages are similar to those used for skips.

A cage is merely an enclosed box opened by a door or doors and suspended from a hoisting rope. The dimensions and capacity of the cage are determined by the quantity, volume, weight, and dimensions of the materials and items to be transported.

Cages are much lighter in construction than skips because they are not exposed to the impact loading and abrasion produced when loading and dumping the mine's ore and waste.

Counterweights

In some balanced hoisting systems, a counterweight may be used with either a cage or skip. Because counterweights serve no functional purpose other than to provide for balanced hoisting, their dimensions are quite variable. They are usually designed to fit the space available.

One configuration that has been used for a counterweight consists of a cylinder running in a steel pipe. These systems, although apparently simple, have been found to be hard to maintain because inspection of the pipe is very difficult and correct alignment of the pipes is complicated to determine and monitor.

When used in conjunction with skips on a Koepe hoist, counterweights allow the system to operate from several loading levels, which is not possible with a balanced skipping configuration.

Safety Devices

In North America, cages must be equipped with safety devices that are designed to prevent the cage from falling down the shaft in the event of rope breakage or serious overwind. Different types of safety devices are used for drum hoists and friction hoists.

Drum Hoists

With drum hoists, both excessive overwind and rope breakage can occur. Of these two types of accidents, rope breakage is considered the most serious, and provisions to bring the conveyance to a safe stop in the event of a rope break is required by legislation.

The most common safety device used is the safety dog. A pair of dogs is installed at the top of the conveyance, one pair for each guide. During normal operation, the tension in the hoisting rope keeps the dogs open. Should the rope break, or become slack, a large spring and a series of levers cause the dogs to bite into the guide and stop the conveyance from descending. The dogs dig into the guide to produce a retarding force.

Although dogs are available for use on both wooden and steel guides, only one design, the so-called Wabi single-tooth dog (Wabi Iron and Steel Corp.), which operates on wooden guides, has been demonstrated to produce consistent and acceptable deceleration rates. The dogs used on steel guides are prone to self-energizing, which results in extremely high deceleration rates causing serious injury to personnel riding in the cage.

Friction Hoists

In multiple-rope friction hoist installations, the probability of all hoisting ropes breaking is small. Therefore, overtravel is considered to be more serious, and provisions to bring the conveyance to a safe stop in the event of overtravel are required.

The most common device used is an arrestor gear, consisting of steel frames supported in the hoisting compartment and transported by the conveyance as it travels through the arrestor zone. These devices are located in the headframe and at the shaft bottom. They rely on friction or mechanical deformation of steel to decelerate the hoist.

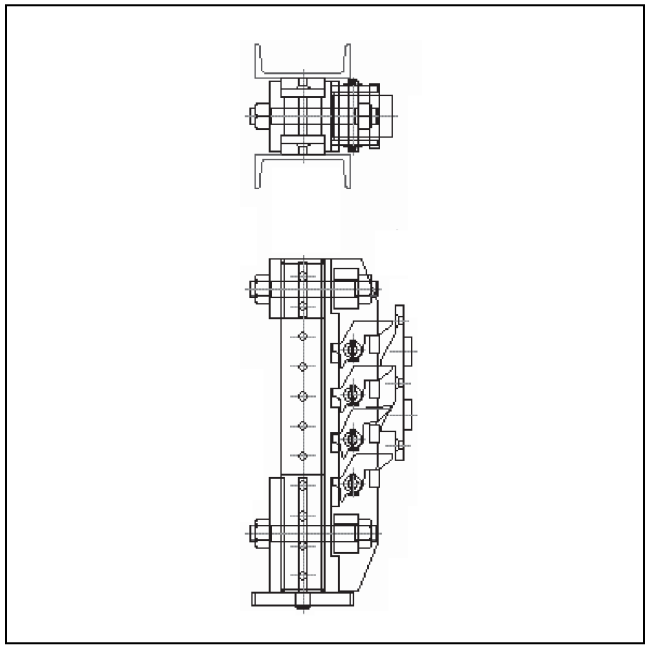


Figure 12.9-16 Catch-gear rack and catch plate

These devices are designed to arrest an upcoming skip at a g-force of 0.9 g's (8.8 m/s^2 or 29 ft/s^2). This level of deceleration ensures that personnel on board will not "float" and thereby injure themselves and ensures that the balance ropes will not overtake the cage, fall back, and likely fail.

In addition to the arrestor gear, the headframe is also equipped with fallback protection. This can be as simple as tapered wooden guides that will jamb the conveyance and hold it until personnel can be evacuated or, more typically, by sets of ratchets that consist of a series of teeth on pivots. This arrangement allows the cage to move up the shaft but prevents it from reversing. Figure 12.9-16 is an illustration of this mechanism.

Loading Pockets

At the loading pocket, ore and waste are transferred from the mine to the skip. The equipment and systems installed can vary from simple manually operated to complex and automated. Two basic types of loading systems are available. These involve measurement by weight or by volume.

The criteria to be considered in the design of all systems include response time, loading time, accuracy of measuring devices, operating and maintenance costs, and amount of spillage.

Although details can vary, each system has the following common elements: a pocket chute with control gates, a measuring pocket, and a skip feed chute with a control gate.

Pocket chutes are generally fabricated from steel with liners. They have an angle of inclination of 45° to 55°. Chute gates can be pneumatically or hydraulically controlled and may be guillotine, ball-and-chain, or undercutting radial arc types.

Skip Loading

Skips are loaded directly from the orepass and are loaded using a measuring flask or a conveyor.

Direct Loading from the Orepass

This method is only used in small operations, typically less than 4 t (4.4 st). The orepass terminates close to the shaft and is equipped with a heavy chain gate, which is raised to allow the ore or waste to fill the skip via a chute. In double-drum installations, the chute will include some kind of diverter mechanism to select between the two skips, as shown in Figure 12.9-17.

Loading via a Measuring Flask

This is the most common arrangement with the flask or flasks installed in a cutout adjacent to the shaft and being fed via a conveyor or sometimes a pan feeder. The flasks, sized to match the skip volume, are filled from the top and are equipped with a door and chute to control and direct the discharge of material into the skip. Loading the flasks is typically controlled by load cells supporting the flask with a volume override. When the material is of a consistent density, such as crushed salt or potash, the systems are often controlled by volume only. The material flows into the flask until the flask is full. At this point, an undercutting arc gate will close the bottom of the storage bin that feeds the flasks. The loading flask is furnished with pneumatically or hydraulically controlled gates. The bottom of the flask is usually inclined at an angle of 60° or greater to ensure rapid loading and complete clean out. Examples of these systems are shown in Figure 12.9-18. With balanced skip hoisting, two loading flasks are provided, one for each skip.

Loading via Conveyor

One of the problems associated with the previously mentioned systems is the difficulty in controlling the loading rate. After the gate is raised to allow the material to flow, there is very little that can be done to regulate the flow, and, as a result, the entire load drops into the skip. This can produce very high dynamic loads on the ropes that support the skip. These loads will reduce the life of the ropes.

To overcome this problem, skips can be fed directly from a conveying system, which can be programmed to fill the skips at a constant rate over a period of 10 to 20 seconds. The conveyors can be equipped with load cells to measure the load accurately. Figure 12.9-19 shows a layout of such a system.

Skip Feed Chute

A feed chute is required to direct the material from the measuring pocket or conveyor to the skip. The angle of inclination of the feed chute is usually 60° or greater.

The clearance between the chute and the skip is approximately 75 mm (3 in.). It has been suggested that the ratio of the chute discharge width to the skip width should not exceed 0.7 for skips up to 1.5 m (5 ft) wide and 0.8 for skips up to 1.8 m (5.9 ft) wide. To reduce spillage, the sides of the chute must be narrower than the opening in the side of the skip body.

Automatic Loading

Automatic loading of skips is used to improve the efficiency of hoisting systems and to reduce labor requirements. Efficiency is improved by ensuring that each skip receives the required load without overloading or underfilling. Labor is reduced when automatic loading is used in conjunction with automatic hoisting. So that automatic loading can be accomplished, a minimum of three control systems must be included. There must be a means of measuring the weight and/or volume of

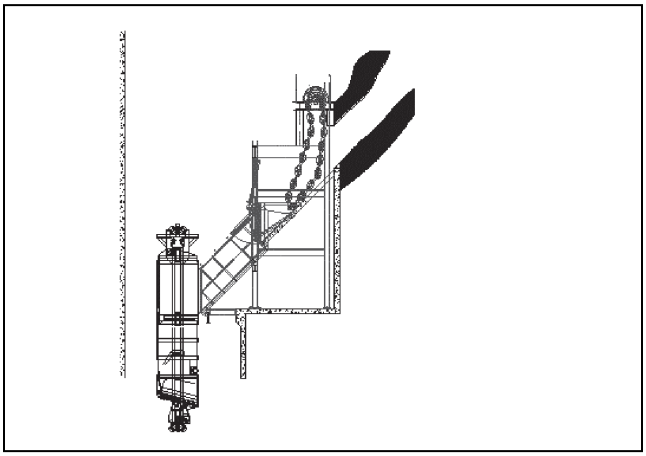


Figure 12.9-17 Chain-gate-controlled skip loading from orepass

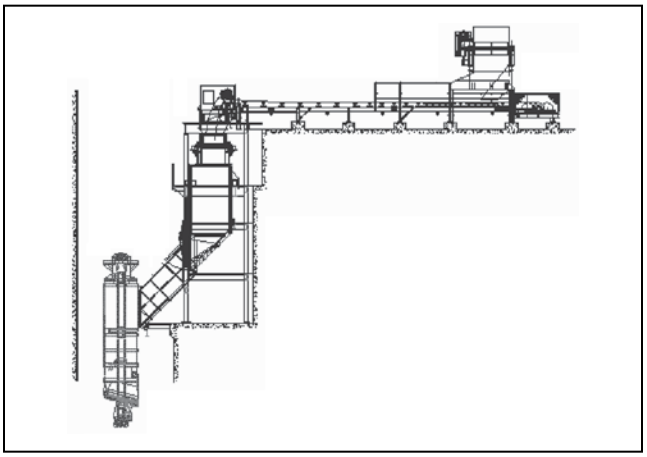


Figure 12.9-18 Skip loading with measured flask

material in the measuring pocket, a means of opening and closing feed chutes to fill and empty the measuring pocket, and a logic system to ensure that the above operations are carried out in the correct sequence to prevent such system failures as double loading and dumping into an empty shaft. With automatic hoisting, the logic system extends to the operation of the hoist.

Spill Pockets

Despite the precautions taken to prevent spillage during the loading and dumping of the skips, a certain amount is inevitable below the loading pocket. This spillage may be confined to the hoisting compartment by lining these compartments down to a spill pocket. According to data available from a number of mines in South Africa, the amount of spillage, expressed as a percentage of tons hoisted, varies from 1.5% to 5%. The larger amounts of spillage are associated with manually operated systems handling uncrushed material.

Two methods of handling spillage in modern hoisting systems are commonly used. One method consists of installing deflectors in the shaft below the underwind position of the skip. These deflectors direct spillage into a small flask similar to the loading pocket. The spillage pockets are then cleaned as required. With drum hoisting, this system can be used at any elevation in the shaft. With a friction hoist, the spillage pocket must be located below the tail rope loops, and a second conveyance, such as the cage, must be used to handle spillage. The second method of handling spillage consists of developing a ramp from the lowest horizon in the mine to the shaft bottom. In this system, any spillage falls directly to the shaft bottom, where it can be mucked out. Regardless of the system used, particular care is required with friction hoist systems to ensure that the tail ropes do not become entangled or affected by the buildup of spill in the shaft and to ensure that the guide rope tensioning weights do not end up resting on the accumulated spill. Most mines today provide ramps to the shaft sump for cleaning the spill.

ROPES

The basic purpose of the hoisting rope is to connect the conveyance to the hoist. It is selected primarily based on safety, compatibility, life, and costs. Safety requirements when hoisting personnel, or where persons may be endangered by hoists and their appurtenances, are usually determined by legislation.

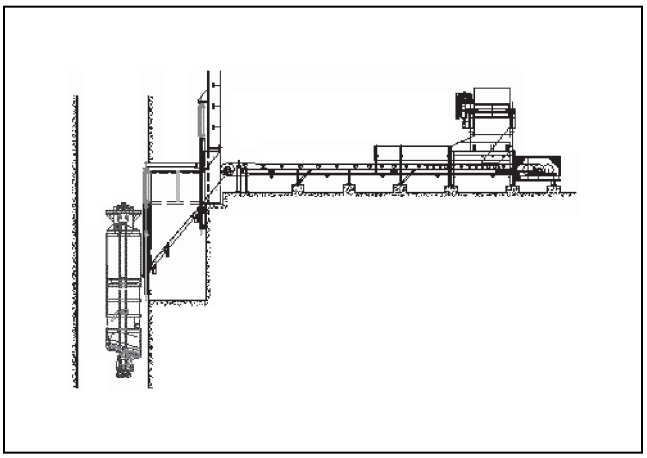


Figure 12.9-19 Skip loading with conveyor and chute

The life of a rope is usually expressed as the number of trips it will make; it is affected by rope construction, hoist and sheave dimensions, type of loading, shaft atmosphere, and maintenance. Costs of initial purchase, maintenance, and rope replacement, including lost production, are normally considered. In addition to hoisting ropes, the hoisting systems designer must be concerned with balance ropes (for friction hoist application), guide ropes, rubbing ropes, and rope attachments.

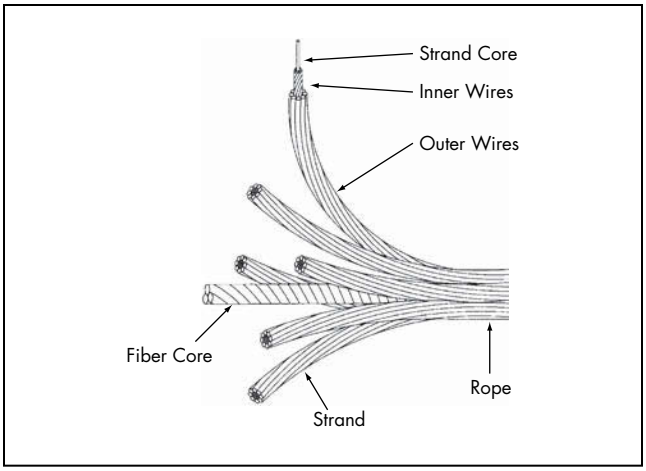
The types of rope construction commonly used for various hoisting applications are as follows.

Drum hoist ropes

- Production duty: Lang lay flattened strand 6 × 30 fiber, or wire metal or plastic core
- Shaft sinking duty: Multistrand, nonspin, 34 × 7, wire metal core

Friction hoist ropes

- Head ropes:
  - Full locked coil
  - Multistrand, nonspin, plastic cushion, 29 × 7, plastic core
  - Round strand, regular lay, plastic core



Source: Anon. 1980.
Figure 12.9-20 Exploded view of a rope

- Balance ropes: Multistrand, nonspin, 34 × 7, plastic core
- Guide and rubbing ropes: Half-lock coil, wire metal core

In this section, a general description of the various types of ropes, plus the factors to be considered to ensure correct application and selection, is presented.

Rope Construction

In the construction of most wire ropes, a number of individual wires are wound around a core to form a strand. The strands are then wound around a core to form the rope. Figure 12.9-20 is an exploded view of a rope showing its components.

Some ropes are constructed without strands by winding consecutive layers of wire around the inner layers. The factors to be considered in rope construction are the wire, strands, cores, and lay. By varying these factors, ropes with different characteristics can be constructed.

Hoist Ropes

The four types of ropes generally considered for hoisting applications are round strand, regular lay; flattened strand, Lang lay; multiple strand, nonspin; and locked coil. Typical properties for these types of rope are given in Table 12.9-3.

Round Strand, Regular Lay Ropes

The wires in these ropes are laid up (i.e., twisted together) to forms strands in the opposite sense to which the strands are laid up to form the rope. The wires on the outside surface lie approximately parallel to the axis of the rope.

Under tension, the wires tend to untwist in one direction and the strands tend to untwist in the other direction. With careful design, the rope will show little tendency to twist when loaded.

Flattened Strand, Lang Lay Ropes

The strands in these ropes are formed around a triangular core and have the wires laid up to form strands in the same sense as the strands are laid up to form the rope. The wires on the outer surfaces of the strands lie at approximately 45° to the axis of the rope. This construction was developed specifically for multilayer winding on hoist drums.

When compared to a round strand, regular lay rope, the flattened strand, Lang lay rope

- Is more flexible because the wires lie parallel to the rope near the rope center, not on the surface;
- Has wires at the outer surface of the rope that are less subject to "plucking" by adjacent turns as the rope coils on the drum; and
- Is designed so that contact loads at the surface of the rope and between strands are distributed over many wires, thereby reducing internal nicking of wires.

One disadvantage of a Lang lay rope is that it tends to untwist when loaded. The two types of lay ropes are illustrated in Figure 12.9-21.

Multiple Strand, Nonspin Ropes

These ropes were developed to overcome the torsion forces developed by Lang lay ropes. They are more susceptible to the crushing forces associated with multilayer winding on a drum and have a lower strength/weight ratio than Lang lay flattened-strand ropes.

This type of rope is used on drum hoists for shaft sinking, where the required service life is relatively short and the conveyances run on rope guides and, thus, low-torque ropes are required. More recently, their design has evolved to include plastic cushioning between layers to reduce internal nicking. The resulting ropes are used as head ropes and balance ropes on Koepe hoists.

Full Locked Coil Ropes

This type of rope is completely different from both round and flattened strand rope. The center or core of the locked coil rope consists of a concentrically laid strand of round wires. Around this core lies one or more layers of X-shaped cross-sectional wires, the outer layer always being Z-shaped and interlocking. The lay of the various layers can be arranged so that the rope does not have a tendency to untwist when loaded. In addition, the structure is compact and stable and better able to withstand crushing stresses than any other rope, and the line contact seals between the wires in the outer layer keeps lubricant in and corrosive environment out.

On the debit side, the specially shaped outer wires cannot be made as strong as round wires, and the rope is less flexible than stranded rope. Larger sheaves and drums are required for ropes of equivalent strength.

The main use of these ropes is on Koepe hoists running on rope guides in deep mines. Figure 12.9-22 shows the cross section of a full locked coil rope.

Rope Selection

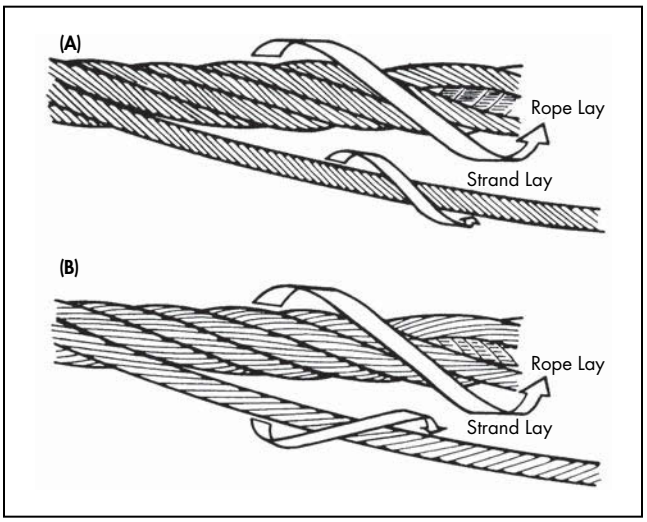
For the initial selection, the following four requirements should be considered: strength, resistance to fatigue failure, abrasion resistance, and resistance to crushing or distortion. The choice should be made after correctly estimating the relative importance of each of the above requirements. Naturally, strength is the overriding concern.

Rope Selection for Drum Hoists

This segment discusses the selection of the size of rope to meet operating conditions for drum hoists used in vertical and slope applications. In the United States, the Code of Federal Regulations (CFR), CFR Title 30, Part 56, specifies minimum requirements for the following items that are relevant to drum hoisting: factor of safety, ratio of drum/sheave diameter to rope diameter, and fleet angles.

Table 12.9-3 Typical properties of rope types

Rope Type	Characteristics and Applications	Comparative Strengths, kN (for 38-mm diameter)	Unit Mass, kg/m
Round strand, regular lay	Used on sinking and Koepe hoists	980	6.1
	Tend to nick in multilayer applications		
	Available from wires up to 1,960 MPa (284,000 psi)		
Flattened strand, Lang lay	Used on drum hoists and shallow Koepe hoists		
	Best for multilayer applications	1.135	5.9
	Very high strength, easy to perform nondestuctive testing		
	High torque under load		
	Available from wires up to 2,000 MPa (290,000 psi)		
Multiple strand, nonspin	Used on sinking and Koepe hoists	1.000	5.7
	Low torque, internal nicking unless plastic fillers incorporated into the design, low elastic modulus, and high initial stretch		
	More difficult to perform nondestuctive testing		
	Available from wires up to 1,960 MPa (284,000 psi)		
Locked coil	Used on drum and Koepe hoists	1.400	8.4
	Near-zero torque, lower unit strength, high elastic modulus		
	Available from wires up to 1,570 MPa (228,000 psi)		



Source: Anon. 1980.
Figure 12.9-21 Rope types: (A) regular and (B) Lang lay

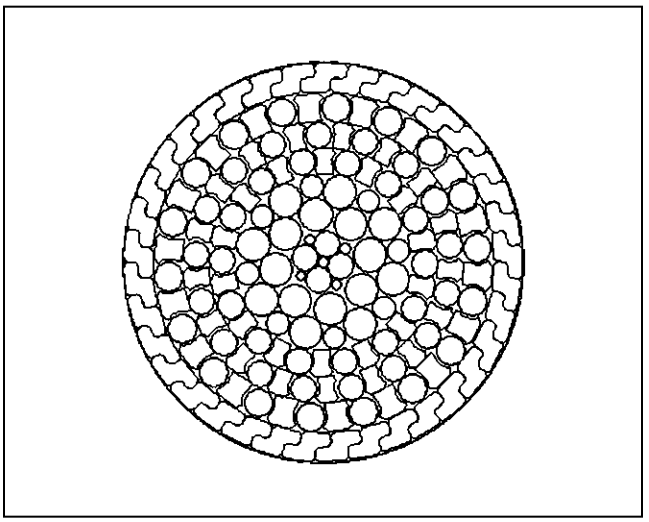


Figure 12.9-22 Cross section of full locked coil rope

Factor of safety. Mandatory requirements by the Mine Safety and Health Administration (MSHA) are covered in CFR Title 30, Parts 55 and 56, and Section 57.19-21 for metal and nonmetal underground mines and underground coal mines, which require the following minimum rope strengths, where L is the maximum suspended rope length in meters:

- For rope lengths less than 914 m (3,000 ft): minimum strength = static load × (7.0 – 0.0033*L)
- For rope lengths in excess of 914 m (3,000 ft): minimum strength = static load × 4.0

Drum/sheave diameter. Mandatory requirements for drum and sheave diameters have now been removed from the regulations, but insofar as the old regulations are still common practice in the United States, as well as Canada and Europe, they are repeated here.

Drum and sheave tread diameters should be

- Not less than 60 times the hoist rope diameter for slope or inclined shaft applications,
- Not less than 80 times the hoist rope diameter if the hoist ropes are 25 mm (1 in.) in diameter or greater or not less than 60 times the hoist rope diameter if the hoist ropes are less than 25 mm (1 in.) in diameter for vertical shaft applications, and
- Not less than 100 times the hoist rope diameter for locked coil ropes.

Fleet angles. The fleet angle refers to the angle made by the headrope between the drum and sheave. If this angle lies outside of fairly small limits, the rope will tend to either pile up at the drum flange or jump several pitches when it starts to return across the drum. Practice has found that the fleet angle

should be between 0.5° and 1.5° for ungrooved drums but can be between 0.25° and 2.0° with drums grooved in the parallel, so-called LeBus pattern.

South African Code of Practice

The most recent codes of practice relating to the selection of headropes as well as assessment of rope condition are those produced by the South African Bureau of Standards (SABS). These are called SANS 10293 (1996) and SANS 10294 (2000) and are worth obtaining to better understand all facets of mine hoisting ropes.

Rope Selection for Koepe Hoists

A friction hoist differs from a drum hoist in that the rope is not wound on and off the drum but passes around the hoist drum, with the friction between the rope and drum providing the force necessary to move the rope. Friction hoists may be mounted in the headframe tower directly over the shaft, or they may be mounted in a hoist house at ground level. The friction hoist drum is often referred to as a wheel.

Regulations and other calculations and considerations involved in rope selection for friction hoists are discussed here.

Factor of safety. Mandatory requirements by MSHA are covered in CFR Title 30, Parts 55 and 56, and Section 57.19-21 for metal and nonmetal underground mines and underground coal mines, which require the following minimum rope strengths, where L is the maximum suspended rope length in meters:

- For headropes with lengths less than 914 m (3,000 ft): minimum strength = static load $\times$ (7.0 – 0.00164*L)
- For headropes with lengths in excess of 914 m (3,000 ft): minimum strength = static load $\times$ 5.0
- Balance or tail ropes = 7.0 on weight of rope (static load)
- Guide or rubbing ropes = 5.0 on weight of rope (static load)

Bending ratio. The required bending ratios are the same as for drum hoists when stranded construction ropes are use. The bending ratio for locked coil ropes must, however, be larger because these ropes are much stiffer in bending than stranded ropes.

Wire Rope Industries of Canada recommends the following minimum bending ratios for locked coil ropes, although all such specifications have been removed from CFR requirements:

- Less than 25 mm (1.0 in.) 100:1
- 25 to 32 mm (1.0 to 1.25 in.) 100:1
- 38 mm (1.5 in.) 110:1
- 51 mm (2.0 in.) 120:1
- Greater than 51 mm (2.0 in.) 130:1

Rope Types

Lang lay ropes work satisfactorily in installations where the hoisting distance is less than 750 m (2,500 ft); at greater depths, the spin upsets the rope structure, resulting in misplaced strands, broken wires, and so forth.

Ropes of nonrotating design, such as multistrand, nonspin, or full lock coil, should be used if the hoisting depth is greater than 750 m (2,500 ft).

Tail Rope Orientation

Any object falling down a mine shaft is affected by the angular velocity of the earth and will move toward the east wall. A tail rope behaves in a similar way. If the shaft compartments are arranged in an east–west direction, the tail rope loops will open and close freely. However, if the compartments are north and south, the ropes could foul the separators at the shaft bottom. Because tail ropes are required for friction hoist installations, the orientation of compartments should be considered. The above described action results from a physical principle known as the Coriolis effect and is a function of the hoist speed and the longitude of the mine.

Tail Rope Loop Diameter

When the tail rope hangs in a shaft beneath a pair of conveyances, there is a curved segment at the shaft bottom which is called the tail loop. The diameter of this loop is determined by the spacing of the conveyances. The rope, however, will twist if the conveyance spacing is less than the natural loop diameter of the rope. This natural loop depends on the rope construction and will vary between 30 and 60 times the rope diameter. The stiffer the rope construction, the bigger the loop.

Guide and Rubbing Ropes

Ropes used as guides and rubbing ropes should be of half-locked coil construction. These ropes are manufactured with a core of concentrically laid round wires covered with one or two layers of shaped wires. The outer layer of locked coil construction is always composed of interlocking wires of Z shape.

Some of the advantages of half-locked coil ropes as guides and rubbing ropes are as follows: their smooth surface reduces friction, they have a greater degree of rigidity, and they use wires with large cross sections on their outer layer. This is critical because one broken wire on the outer layer is cause for the rope to be replaced. A broken wire will catch in the guide rope bushing on the skip, which could lead to a major mishap.

Rope Sag

A rope between a ground-mounted hoist and a head sheave sags and takes the form of a catenary. With a heavy rope and a light load, the sag can be significant and cause problems with the rope impacting the sheave deck or the hoist building front wall, if not considered.

The shape of the catenary is given by

$$y = 2*H/q*\sinh(\bar{r} + a)*\sinh(\bar{r})$$

where

y = vertical coordinate along caternary
H = horizontal tension in the cable
q = unit mass of the catenary
$\sinh$ = hyperbolic sine function
$\bar{r}$ = horizontal coordinate along the catenary
a = horizontal spacing of catenary end points

Inclined (Slope) Haulage

The selection of ropes for an inclined shaft differs little from selection for a vertical shaft. The three major differences are calculation of rope tension, sizing of sheaves, and abrasive wear.

Calculation of Rope Tension

The tension in the rope is the sum of the gravitational, frictional, and acceleration forces. The gravitational force is the sum of the weights of the car or cars, payload, and the rope multiplied by the sine of the angle of the incline. The friction force is commonly taken as 2.5% of the weight of the car and payload plus 10% of the rope load.

Sizing of Sheaves

Because of the space restrictions underground and the need for portability, deflection sheaves are commonly smaller than used elsewhere for hoisting ropes. Minimum drum diameter to rope diameter D/d ratios are presented in Table 12.9-4.

Abrasive Wear

Abrasive wear is more likely to be a problem in a slope than in a vertical shaft, and this potential problem should be addressed in rope selection. One approach is to select ropes with thicker wires, such as 6×7. Some designers increase the factor of safety to allow more wear on the rope before it needs to be replaced.

Rope Operating Practices

The correct wire rope properly installed on well-designed equipment maintained in good working condition provides the foundation for satisfactory rope performance. To attain and sustain the required rope performance, it will be necessary that the rope be properly operated and adequately maintained. Well-developed rope-operating practices should consider the following: rope storage, rope installation, rope changing procedures, rope cuts, lubrication, and inspection procedures.

Rope Storage

Whether the rope must be stored temporarily or indefinitely, it should be indoors, where it can be shielded from weather, corrosive fumes, and excessive heat that might dry the lubricant or fiber core. The rope reels must be turned 180° every 4 months so that lubricants do not settle to the lowest point, thereby allowing unprotected steel to rust.

Rope Installation and Handling

Ropes should be wound onto the drum under load. The direction of coiling should be determined by the accepted rule-of-thumb method, as shown in any rope handbook. Before cutting, the rope supplier or a technical handbook detailing correct cutting procedures should be consulted. This typically requires seizing of the rope on both sides of the cut with hose clamps and cutting the rope with a grinding disc.

Drum Hoist Rope Changing Procedures

Rope changing procedures for drum hoists are relatively easy and are described in rope handbooks. Precautions must be taken, however, to ensure that adequate tensions are applied to provide correct spooling during installation and to pretension the dead turns. Pretensioning the dead turns is particularly important in multilayer winding. Failure to adequately pretension will lead to loose dead turns and catastrophic rope wear.

Adequate pretensioning can be provided by using a special rope-tensioning winch or by doubling down the headrope using a loaded skip.

Table 12.9-4 Minimum D/d ratios for sizing of sheaves

Rope Construction	Minimum D/d
6×26	30
6×31	26
6×36	23
6×41	20
8×36	18

Friction Hoist Rope Changing Procedures

There are several options for changing head and balance ropes on a Koepe hoist. Procedures involve careful planning and special equipment. These procedures are beyond the scope of this document.

Rope Cuts

The end of the rope adjacent to the conveyance suffers premature strength reduction as a result of corrosion and local bending. These ends must be cut off at regular intervals.

In multilayer winders, wear occurs on those areas of the rope that meet at the crossovers, that is, the region of the drum where the rope changes layer. These locations need to be moved regularly (every 3 or 4 months) to avoid premature retirement of the rope.

Lubrication

Atmospheric moisture, shaft water, and chemical fumes are a few of the agents that will cause corrosion. This frequently leads to wire breakage and to the premature removal of mine hoist ropes. In mines with very wet and corrosive shafts, it is distinctly advantageous to apply a lubricant that will penetrate the interior of the rope and that contains a corrosion inhibitor. These must be applied during fabrication of the rope. It is impossible to apply lubricants that can penetrate the rope strands after the ropes are manufactured. Ropes made with fiber cores soaked in lubricants will extend rope life in wet shafts.

Inspection Procedures

To achieve optimum rope performance, an inspection procedure that requires findings to be recorded, fully diagnosed, and translated into appropriate action should be instituted.

The presence of the foregoing not only allows the operator to estimate the remaining safe working life of the rope but can also pinpoint the source causing such damage to the rope.

Ropes should be visually examined daily and tested regularly using electromagnetic devices to detect loss of cross section and broken wires. When the rope's strength is determined to have been reduced below 90% of the original value, the rope must be replaced. Other retirement criteria also exist and are stipulated in CFR Title 30, Section 57.19024.

Rope Attachments

Wire rope end attachments are as important as the rope to which they are fastened. Therefore, the selection of the correct type of attachments should be based on an understanding of how they affect ultimate rope serviceability and efficiency.

Forming a loop in the rope or attaching a fitting to the rope are the only ways of attaching something to a wire rope.

Loops are made either by splicing the rope to itself or by the use of clamps or wedges. Fittings secured directly to wire rope can be applied by cold forming (snagging) of the metal in the fittings, either by pouring a liquid material, such as molten zinc, or by a wedging arrangement. In this section, some of the most commonly used attachments are described and are shown in Figure 12.9-23.

Factor of Safety

The factor of safety for rope attachments is not specifically covered in CFR Title 30, Part 57. However, it is considered good engineering practice to design the rope attachments to be stronger than the rope, that is, the rope will fail before the attachments. British codes require a factor of safety equal to 10 based on static loads.

Cappels

Cappels are rope fittings attached directly to the rope. These types of fittings consist of a pair of wedges grooved to suit the particular rope diameter and interlocked to ensure complementary movement. A number of bands driven over the diverging exterior surfaces of the cappel limbs provide the initial compressive force to ensure that the wedges grip the rope. The limbs' internal surfaces have been machined to fit the external surfaces of the wedges. A safety block is fastened on the end of the rope, protruding beyond the bottom of the wedges. Several types of cappels are used for hoist ropes, including the wedge and loop.

Thimbles

In North America, many mine hoist ropes on drum hoists are terminated with a wire rope thimble and U-bolt cable clamps. With this type of attachment, a loop is formed in the rope and the thimble placed within this loop. The free end of rope is then clamped to the long end to secure the thimble.

Thimbles are made of cast steel, whereas clamps, shackles, and pins are made of 1.5% manganese steel.

Thimble Cappels

These attachments represent a significant improvement over the basic thimble without the complication of installing bands to secure the wedges. These devices are self-locking, using the force on the rope to drive a wedge into a housing that incorporates an extension with a pin to connect the cappel to the conveyance drawbar.

Swivels

When winding with balance ropes, the lay length tends to shorten as each conveyance in turn approaches the bottom landing. It is for this reason that swivels are provided to allow the ropes to spin and regain their normal length. Shaft conditions and loadings influence the type of swivel used.

Attachment Maintenance and Inspection

If thimbles and clips are to be used, then the manufacturers' recommendations for the correct number of clips, amount of turnback, clip positions, and correct torques should be followed closely. If a wedge-type connection is being applied, then care should be taken to properly clean the rope surface. Frequent applications of cement dust and subsequent steel brushing will remove all the surface lubricant.

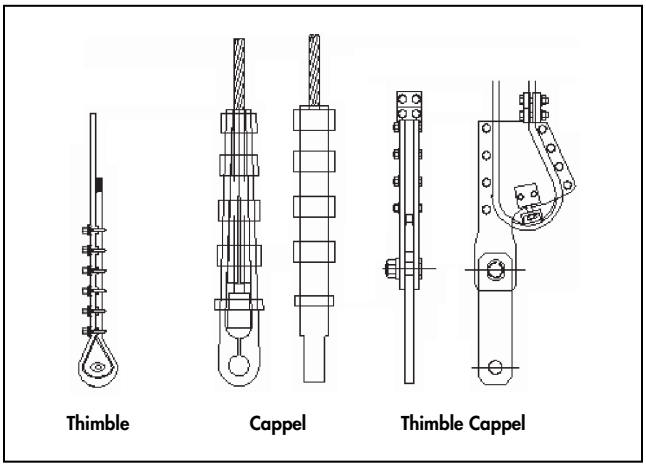


Figure 12.9-23 Rope attachments

In some jurisdictions, statutory mining regulations require that all types of attachments be inspected both visually and nondestructively on a regular basis. In other parts of the world, removal criteria for attachments have been established.

SHAFT AND GUIDE SYSTEMS

A shaft is a vertical or inclined primary opening in rock that provides access to and serves various levels of a mine. Primary openings are those that are considered to be permanent and require a high degree of safety. Chapter 12.4 discusses design and construction details.

Shaft Design Procedure

A suggested procedure for the design of shafts follows. It is important that the steps be followed in the given sequence. The process is iterative and involves working through the process several times before an optimum design is achieved. The shaft design procedure involves the following steps:

1. Define the purpose of the shaft.
2. Identify the location and determine the inclination.
3. Determine the number of hoists required.
4. Determine the size of the conveyances and compartments.
5. Determine the arrangement of the compartments.
6. Determine the exterior shape.
7. Design the interior members (guides, buntons, etc.).
8. Design the shaft lining.
9. Check the ventilation characteristics.
10. Determine the ground stabilization and temporary ground support.
11. Determine the shaft collaring method.
12. Determine the shaft sinking method.
13. Evaluate and modify this procedure by repeating these steps.

Purpose of Shaft

One of the first items to be examined when designing a shaft is to identify its intended purpose. Shafts usually fall into one of the following categories: production (ore and waste handling), service (personnel and materials handling), ventilation (upcast or downcast), exploration (for defining mineral deposits), and combinations of the above.

Location

Generally, the location of a new shaft at the mine site is determined after establishing the following:

- Mine surface layout
- Location, dip, and extent of the ore body
- Number of working levels to be considered
- Location of ore and waste handling facilities
- Water collection sump requirements
- Safety and stability of the shaft pillar
- Future planned expansion of the shaft

The bottom of the shaft, where possible, must also be in a stable formation, and it should be able to facilitate any future deepening. The location of the shaft must also avoid adverse surface features and should satisfy surface layout plus logistics of surface and underground ore and waste handling facilities.

Inclination

The dip of the ore body is the main factor involved in deciding to sink either a vertical or inclined shaft. A secondary factor is the relative ground strength and geologic formations to be encountered by the proposed shaft.

The main advantages associated with vertical shafts are that hoisting speeds are greater, shaft maintenance costs are lower, sinking can be carried out faster, and sinking can be achieved in almost any type of ground.

Inclined shafts (slopes) are generally associated with inclined (dipping) ore bodies, where the length of crosscutting to reach the ore body from a vertical shaft becomes longer with the increase in depth of the shaft. They have the advantage of minimizing development to reach the ore from the shaft and are frequently used in coal mines.

Number of Hoists

The number of hoists (and conveyances) required to meet hoisting demands has a major impact in the design of a shaft. The number of hoists will be determined using the procedures described previously.

Compartment Size

The cross-sectional area of a particular compartment (horizontal area) is dependent on its use. To determine the most appropriate size, it is necessary to list the items to be transported in the compartment, determine their approximate dimensions and weight, indicate their direction of flow, and indicate their approximate quantities. Another important factor to be considered is the size of the shaft-sinking bucket. After having determined the size of compartments for cage, counterweight, and skips, the remaining area is then divided to accommodate ventilation, manway, and pipe facilities.

Size of Skip Compartment

The skip capacity is determined by such variables as the hoisting capacity, material density, lump size, and vertical height available in the headframe. Generally speaking, the skip dimensions and cross-sectional area are determined by the skip supplier. The amount of clearance between the skips and shaft equipment is typically 300 mm (12 in.) for rope-guided skips and 75 mm (3 in.) for skips on fixed guides.

Size of Cage Compartment

Cages are generally used to transport personnel and materials. The following clause of CFR Title 30 (Section 57.19-66 Mandatory) applies in the United States: "In shafts inclined over 45°…each person shall be provided a minimum of 0.14 m^2 (1.5 ft^2) of floor space."

The cage capacity is determined by the cage supplier based on a knowledge of the following factors: the amounts of material to be transported, the dimensions and weights of the major pieces of equipment to be lowered and raised, the number of personnel to be lowered and raised per shift, and the type of conveyance preferred (i.e., single deck, double deck, combination skip/cage, etc.)

After establishing the cross-sectional area for the cage, the cage compartment is sized the same way as the skip compartment to provide similar clearances.

Size of Manway Compartment

From CFR Title 30 (Section 57.11-37 Mandatory): "Ladder ways constructed after November 15, 1979, shall have a minimum unobstructed cross-sectional opening of 610 × 610 mm (24 × 24 in.) measured from the face of the ladder."

From CFR Title 30 (Section 57.11-41 Mandatory): "Fixed ladders with an inclination of more than 70° from the horizontal shall be offset with substantial landings at least every 9.1 m (30 ft) or have landing gates at least every 9.1 m (30 ft)."

In addition to these mandates, it is necessary to provide adequate clearance at the landing platform to allow a person to move from one ladder to the next offset ladder. Generally, an additional floor area of 914 × 610 mm (36 × 24 in.) is provided.

Size of Counterweight Compartment

The counterweight compartment can be of any size or shape as long as it provides sufficient space and clearance for the safe passage of the counterweight with the same clearances as for skips and cages.

Size of Service Compartment

There are no hard and fast design rules for sizing the service compartment except that the cross-sectional area of the compartment should be adequate to provide the following: separate locations for communications (signaling) and power lines; space for locating air, water, pump, and backfill lines; clearance for pipe fittings; and space for pipe column brackets. The compartment must also be accessible from a shaft conveyance, preferably a cage, to allow maintenance of the services in the compartment.

Compartment Arrangement

The arrangement of the compartments in relationship to one another depends on the surface layout, the underground layout, and the type and size of hoist to be used. Because these design details are not known precisely during the initial design stages, one must first make a "best estimate" and be prepared to change the arrangement as the final design evolves.

On surface, the orientation of the compartments in relation to the hoist and headframe is of major importance. Good rope practice for drum hoists requires that the location of the hoist in relation to the shaft and headframe be in line to provide acceptable rope fleet angles.

For friction hoists, the center-to-center distance on the compartments is a major consideration for determining the size

of the drum, the necessity for deflection sheaves, and the type of rope construction for the tail ropes.

Exterior Shape

The exterior shape of a shaft is established by considering ground-stability factors. A circular shape provides better resistance to deformation by lateral pressure. In fair to competent formations, an opening excavated in this form is generally self-supporting. In weak formations, the circular shape is adaptable to a variety of lining materials: concrete, steel segments, or cast-in-place concrete. Competent formations can be excavated to a rectangular shape and be generally self-supporting. An elliptical shape offers better support than a rectangular shape but not as good as a circular one. The geotechnical considerations leading to the selection of an exterior shape are beyond the scope of this discussion, but their importance in the overall design process cannot be overstated (see Chapter 12.4). Also shafts may have different exterior shapes at different depths, reflecting the different strata encountered.

Interior Members

Guides and buntons are the main vertical and horizontal structural elements in a mine shaft conveyance system. The primary function of these members is to facilitate running of the shaft's conveyances. Their characteristics influence not only speed of operation and amount of maintenance required, but also production and ventilation costs. Therefore, the choice of sets and guides is of great importance to achieving operational cost savings. To minimize construction costs and the time to install guides, bunton spacing is maximized. The current trend is to install buntons at 5-m (16.4-ft) spacing. This results in flexible guides and requires more sophisticated designs and simulation of the conveyance motion to avoid so-called conveyance slamming events, which can lead to destruction of the guides and conveyances.

Shaft Guides

Shaft guides are used in vertical shafts to keep the skips, cages, and counterweights in proper shaft position. The two types of guides in use are fixed guides (wood, rail, and structural steel), and rope guides.

Wooden guides. Wooden guides are supported by the horizontal sets installed in the shaft. They are favored where conveyances require safety devices. This is because the decelerating characteristics of safety dogs on wood are more reliable than on steel or rail. When hoisting on wooden guides, a speed of 11 m/s (36 fps) with a medium-sized skip is the practical upper limit. To achieve speed of this order with large conveyances and longer guides, it is necessary to install wooden guides supported by structural steel backers. These backers, typically wide-flange structural steel, are bolted between the steel buntons, and the wooden guides are then bolted to them. This arrangement avoids the problems associated with fastening wood guides directly to wood or steel buntons. Wooden guides are subject to changes in dimensions due to changes in moisture content. They are also subject to wear to a greater extent than steel guides. Wooden guides generally are not used in shafts where fire hazards exist, such as in coal mines.

Until recently, the wood used for these guides was either British Columbian fir or Australian Karri wood. Both of these products are now in scarce supply. Wooden guides are now available in composite wood materials, akin to plywood. The only disadvantage of the composite guide is its tendency to swell when wet; therefore, it is not suitable for wet shafts.

Rail guides. Rail guides are common in inclined shafts. These guides vary in size, with the larger rails giving a wider bearing surface for the wheel. The steel rail provides a harder surface but presents a smaller wear area; this creates more wear and maintenance on conveyance guide shoes and wheels. They have also been used to guide conveyances (usually in shafts equipped with Koepe hoists) in vertical shafts but suffer from their higher lateral flexibility, which leads to excessive motion of the conveyance as it moves in the shaft.

Steel guides. Structural steel tube sections have virtually replaced wood as the material of choice for guides in production shafts, where higher speeds and heavy loads require a fairly rigid and stable section. With steel guides, the conveyances are usually equipped with adequately sized guide roller assemblies to minimize wear between the guides and shoes on the conveyance. When hoisting on steel guides, a speed of 18 m/s (3,500 fpm) is a practical upper limit.

Rope guides. Rope guides are also used for guiding conveyances in vertical shafts. These are generally used with multirope hoisting systems where the effect of hoist-rope torque is minimal or nonexistent. Half-locked coil ropes are used for this service. Rope guides do not require any intermediate support, thereby eliminating the steel sets required for fixed guides. Because rope guides are not able to withstand the horizontal forces of loading and unloading, the conveyance must be supported by other means in these areas. Fixed guides are used at the extremes of travel, and a retractable guiding system can be considered at intermediate levels. There are several schemes by which rope guides are suspended and tensioned, and the selection of any one of these will depend on the particular installation. These methods will be reviewed in a later section.

Because rope guides eliminate the need for horizontal buntons, the ability of the shaft to handle airflow is enhanced. In a large shaft, with multiple conveyances traveling at the same time, the aerodynamic effects can cause unexpected rope deflections. These effects should be studied during the design stage. Because rope guides are the smoothest of all guides, hoisting speeds up to 20 m/s (3,900 fpm) are feasible. Discussion of the detail design of rope-guided systems is provided in a following section.

Shaft Buntons

Shaft buntons are horizontal members used to divide the shaft into compartments. They also support and carry the shaft guides, pipes, and cables. Buntons can be of any cross-sectional shape and can be installed in a shaft at any spacing. However, the correct design and layout of buntons and their installation at a suitable spacing minimizes the air resistance of shafts and, hence, reduces ventilation operating costs. Steel buntons are available with aerodynamic cross sections that minimize the resistance to airflow.

Guide and Bunton Structural Design

The design of guides and buntons has progressed to the point where their design is no longer empirical. The design requires the use of computer-based simulation of the motion of a

conveyance as it moves up the shaft. This allows the designer to optimize the design and reduce the amount of steel that is required for this equipment.

In addition to using simulation software, the designer can also use guidelines detailed in SABS 0208-4 (2001) to provide a check on a proposed design before submitting it to the rigor (and expense) of a computer simulation.

Guide Alignment

One of the required inputs to either the guideline or computer simulation is the degree of alignment expected for the installation. This means how far off of plumb, or straight line, one expects, or allows, the guides to be.

A practical limitation for this has been found to be ±3 mm (±0.125 in.) over 5 m (16 ft). To verify that this has been achieved, it is necessary to either install two plumb lines for each guide the entire length of the shaft or to utilize sophisticated instrumentation developed in South Africa for this purpose. This latter option is a more reliable method, as it does not rely on measurement and recording by individuals with its inherent inaccuracy.

Guide Rope Design

Guide ropes are common in high-speed shafts operating large conveyances from a single horizon. This situation is common in North America in coal, potash, trona, and salt mines.

It is first of all necessary to calculate what resistance to lateral motion the guide ropes provide. The lateral restoring force produced by a guide rope is termed its *stiffness*. This term refers to the force produced by moving the guide rope laterally in the shaft as opposed to bending the rope in an arc.

Calculation of this stiffness in kilograms is depicted in Figure 12.9-24. As this figure shows, the restoring force at any point along the guide rope is simply a function of the tension in the guide rope at that point and the distance the force is from the end of the rope.

The more difficult problem comes in calculating what forces arise as the conveyance moves up or down the shaft. These forces can be put into one of the following categories:

- The Coriolis force, which is the result of the conveyance moving radially through the earth
- Impulsive forces from the buffeting produced as conveyances pass at mid-shaft or cross a ventilation opening
- Steady aerodynamic forces caused by the shape of the conveyance
- A twisting moment arising from the head- and tail ropes

Coriolis force. When a body descends in a shaft, it moves toward the east wall; when it ascends, it moves toward the west wall. This motion is a result of the different circumferential velocities that exist between points along any radius of a rotating sphere. The *apparent* lateral acceleration of the object, as it moves at a constant speed in the radial direction, results in a force on any structure, such as a guide rope, trying to restrain it. This force is called the Coriolis force and is proportional to twice the product of the object's radial and angular velocities. The force is required to increase or decrease the object's lateral velocity to match that of the guide rope, and all other matter at the same radial location. The magnitude of the Coriolis force is given by

$$F = 2\,\Omega\,V$$

where

Ω = rotational speed of the object about the earth's axis
V = radial speed of the object in the shaft

Aerodynamic and buffeting forces. Conveyances will experience short-term impulsive forces called buffeting forces when they pass one another as well as steady aerodynamic forces as they travel the entire length of the shaft. The buffeting forces have been found to be inconsequential and do not need to be considered when designing a shaft. The aerodynamic forces that arise as conveyances move through a shaft are, however, very significant and need to be determined. They arise because the flow of the air around the conveyance is asymmetrical and will cause lift to occur. The lift generally acts toward the closest walls because the air velocity is higher in the reduced space between the conveyance and shaft wall. It is difficult to determine the lift coefficient. Some recent research has provided a means for calculating this coefficient and has thereby enabled the designer to determine reasonable estimates for the level of force that a conveyance might encounter. More accurate determination of the lift coefficient requires the use of computational flow dynamics to model the flow around the conveyance in the shaft or scale-model wind-tunnel testing of the shaft.

The force on the conveyance is calculated from

$$F = \tfrac{1}{2}\, C_L \rho V_r^2 A_c$$

where

C_L = lift coefficient
ρ = air density
V_r = relative velocity of the conveyance and ventilating air
A_c = the area of the skip facing the section of shaft wall

Forces are typically in the range of 100 to 500 N (20 to 100 lb).

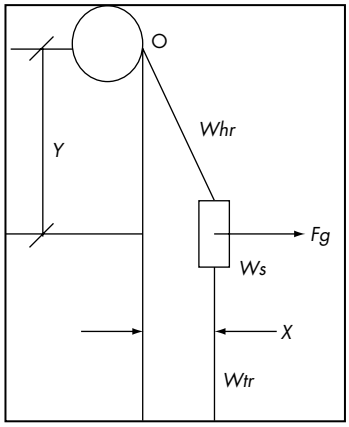
Rope Torque

Both headropes and tail ropes induce torque and, therefore, twisting of the conveyance because of the natural tendency of a rope to untwist when under tension. The data provided above shows that headropes are available with very low torque constants. The ropes used for balance, or tail ropes, are typically designed to minimize the torque that arises as load is applied to them. This minimum is still significant and is the reason why the ropes are attached to the bottom of the conveyance using swivels. This arrangement reduces the torque to a function of the friction in the swivels. The swivels are basically a rolling contact bearing that has a static friction, or stiction, of about 1.5 to 2 times the rolling friction. Based on the data provided by the manufacturer of the swivels, each tail rope will typically produce a torque of the order of 50 N·m (40 lb-ft). This will be developed when about 250 m (800 ft) of tail rope is suspended below the skip.

Guide Rope Location

Figure 12.9-25 illustrates the two most common arrangements of guide ropes. Option A has all four guide ropes along one side of the conveyance, while Option B has one rope at each of the four corners of the conveyance.

Although both options offer the same resistance to lateral motion, Option B provides significantly better resistance to

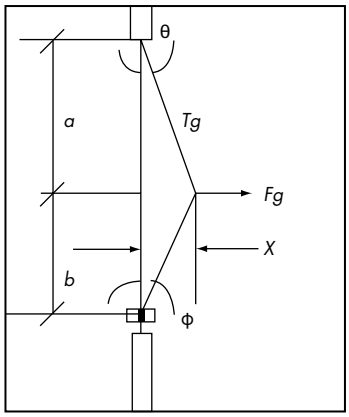


Skip and Ropes

Moments about O:
$Fg \times Y = (Whr/2 + Ws + Wtr) \times X$

Spring rate of Skip and Ropes:
$Ksr = Fg/X = (Whr + Ws + Wtr)/Y$

where
- Fg = Force in Horizontal Direction
- Whr = Weight of Headrope
- Ws = Weight of Skip
- Wtr = Weight of Tail Rope
- X = Displacement of Skip
- Y = Location in Shaft
- Ksr = Spring Rate due to Skip and Rope Masses



Guide Ropes and Tensioning Weights

- Gw = Guide Rope Weight/Unit Length
- Tw = Tensioning Weight Mass
- Tg = Guide Rope Tension at Skip
- Fg = Applied Force on Guide Rope
- θ = Guide Rope Angle at Top
- ϕ = Guide Rope Angle at Bottom
- a = Location of Skip from Top
- b = Location of Skip from Bottom
- Kg = Spring Rate due to Guide Ropes
- $Tg = Tw + b \times Gw$

  and
- $Fg = Tg \times (Sin(\theta) + Sin(\phi))$
  $= Tg \times (X/a + X/b)$
  $= Tg \times (X(a + b)/a \times b$

  hence,
- $Kg = Fg /X = Tg \times (a + b)/a \times b$

Figure 12.9-24 Guide rope stiffness calculation

twist. In fact, Option A will cause twist for any force that acts in a direction parallel to the plane of the guide ropes.

Guide Rope Tensioning

To keep conveyance motion to a minimum, guide ropes should be tensioned to the maximum amount permitted by local regulations. This typically permits operation with a minimum safety factor of 5.

Tensioning of ropes is possible by the following two means: hanging heavy weights (called cheese weights) from the bottom of ropes, and anchoring the bottom of the rope to a beam in the shaft and tensioning the rope in the headframe using either large springs or hydraulic cylinders. The use of hydraulic cylinders simplifies rope changing and reduces the need for shaft excavation. The most reliable method is still the use of tensioning weights, because this approach automatically compensates for rope stretch and temperature fluctuations.

If cheese weights are used, it is necessary to clean out the shaft bottom on a regular basis, or the weights will become buried in spill and drop their tensioning force.

Shaft Lining

Shaft lining may be comprised of shotcrete, cast-in-place concrete, or steel and cast-iron tubbing. When a lining is required, the selection depends on a number of design considerations that relate to the strata being excavated. For further discussion, see Chapter 12.4.

Shotcrete

Sprayed concrete, 150 mm (6 in.) or more in thickness, may be used to control immediate raveling and weathering of the shaft walls. To date, shotcrete has not been used extensively to provide long-term support nor has it been used as the only means of lining in shafts.

Concrete

The use of concrete for permanent support is common in most new shafts. Shafts may be fully lined or partially lined with "concrete rings" with areas of open ground between the rings. Common spacing used for ringed shafts is 1.2-m- (4-ft-) high concrete rings with a 1.2-m (4-ft) open area between, resulting in a 2.4-m (8-ft) set interval. Concrete is generally placed with a minimum thickness of 300 mm (12 in.), but this will increase with depth for linings designed to withstand the hydraulic head encountered in formations that include aquifers.

Steel or Cast-Iron Tubbing

Although the use of concrete as a lining material is becoming more common, there are certain geologic conditions where steel or cast-iron tubbing is a more appropriate

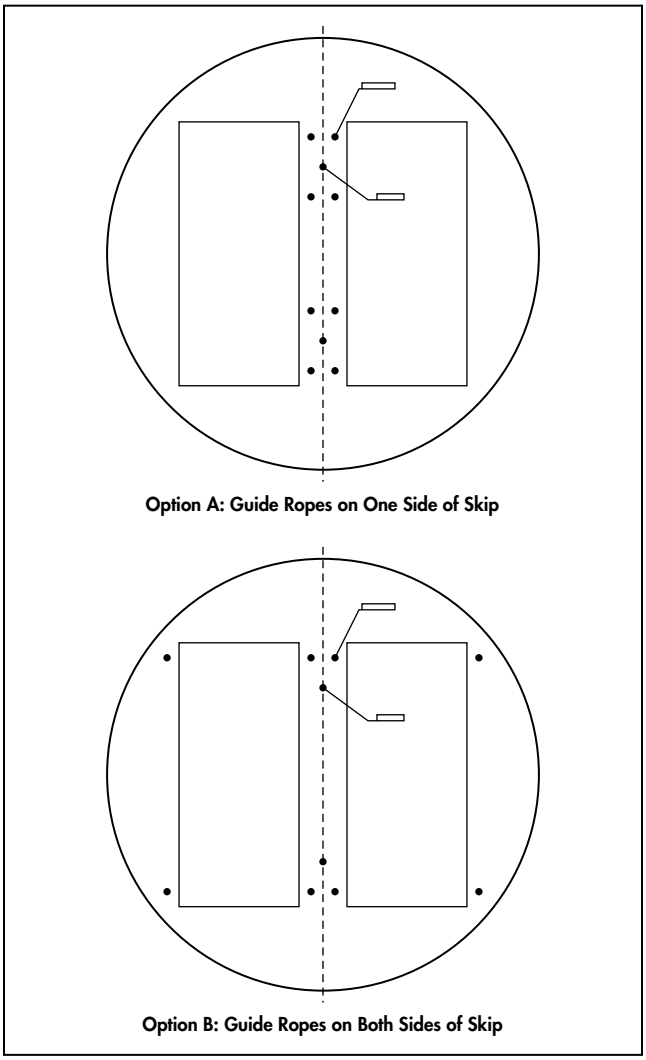


Figure 12.9-25 Guide rope locations for Options A and B

material. Concrete is suitable when water pressures are less than 1,700 kPa (250 psi) and when the stress field is uniform and compressive in nature. Steel is more suitable for a completely watertight lining and is capable of withstanding tensile stresses. These stresses could arise if the shaft is subjected to bending loads caused by subsidence or buckling loads caused by a nonuniform horizontal stress field.

Steel lining can be butt-welded to make it permanently watertight, and this has a higher buckling safety factor than tubbing. Steel is particularly useful as a lining for shafts sunk by drilling methods. Cast-iron tubbing consists of several segments bolted to form a ring and requires a lead gasket to be placed between segments. This type of tubbing is much stronger than the steel liner and is typically used to provide support when crossing aquifers are encountered at considerable depth and, hence, high pressure.

Design Considerations for Shaft Lining

The design requirements for lining a structure are governed by the required duty of the shaft and environmental conditions in which it is constructed. Careful consideration of the following points should be made in the design of structural lining: the precise behavior of the rock mass in situ, the self-supporting action of the rock itself, the effect of separation of the lining due to certain vertical and horizontal movements of the surrounding rock formation, the construction of watertight linings, and the economics of the lining structure.

The design of steel or cast-iron linings is a much more complex process, because the buckling characteristics of an elastically embedded hollow cylinder with external pressure must be evaluated.

Ventilation Characteristics

The shaft is generally the largest single contributor to total mine air resistance when it is equipped with rigid guides for hoisting purposes. The resistance consists of the following two components: frictional resistance caused by viscous drag in the thin air layer at the shaft walls and the periphery of the vertical elements (guides, pipes, cables, ropes, etc.) of the shaft; and shock resistance due to the transverse sets (buntons and dividers) completely immersed in the air stream, thereby obstructing the flow and recurring at intervals throughout the shaft.

These transverse elements create turbulence in the airstream with a corresponding loss of flow energy. The turbulent waves below buntons and conveyances often contribute much more to the total shaft resistance than the skin friction of the shaft walls. The forces caused by the buntons are parallel to the airstream and are proportional to velocity pressure, frontal area, surface texture, and shape. Because the frontal area of all the buntons in a shaft is likely to be more than the frontal area of the cages and skips, the aerodynamic design of the bunton is of importance in the design of a mine shaft.

A reduction in shaft resistance can be achieved through improving the aerodynamic effect of horizontal obstructions such as buntons, minimizing the frontal area of the set members, and increasing the spacing between the horizontal obstructions. Practical experience has indicated that there is little ventilation advantage to be gained from spacing greater than 5 m (16 ft).

A general treatment of mine ventilation is offered in Chapter 15.3.

HEADFRAME

The basic purpose of a headframe installed over a shaft is to support the sheave wheel over which a hoisting rope passes for raising or lowering conveyances or, in the case of a tower-mounted Koepe hoist, to support the hoist and all of its suspended loads. The construction of a headframe is also necessary to allow dumping of hoisted materials aboveground. Structurally, there are two types of headframes: steel headframes, which are typical for drum hoists, and slipformed concrete headframe structures, which are typical for tower-mounted Koepe hoists. These designs are shown in Figures 12.9-26 and 12.9-27, respectively.

Construction Materials

Headframes may be constructed of timber, steel, or concrete. Modern trends in high-capacity hoisting have necessitated the construction of very large headframes, which effectively preclude the use of timber as a construction material.

Steel Versus Concrete

Both concrete and steel can produce economically viable, structurally compatible, and visually appealing headframes

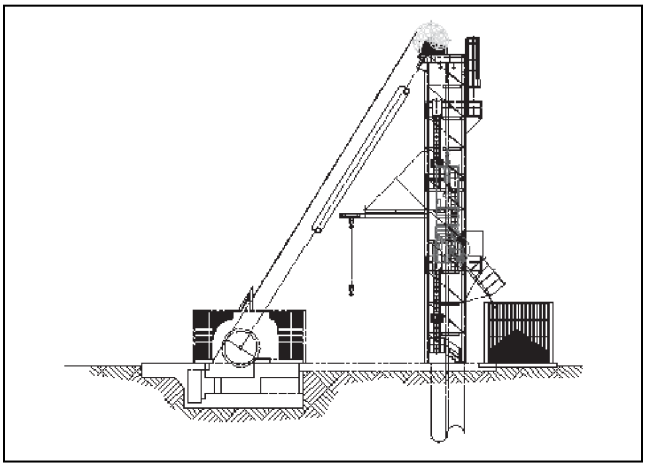


Figure 12.9-26 Steel headframe

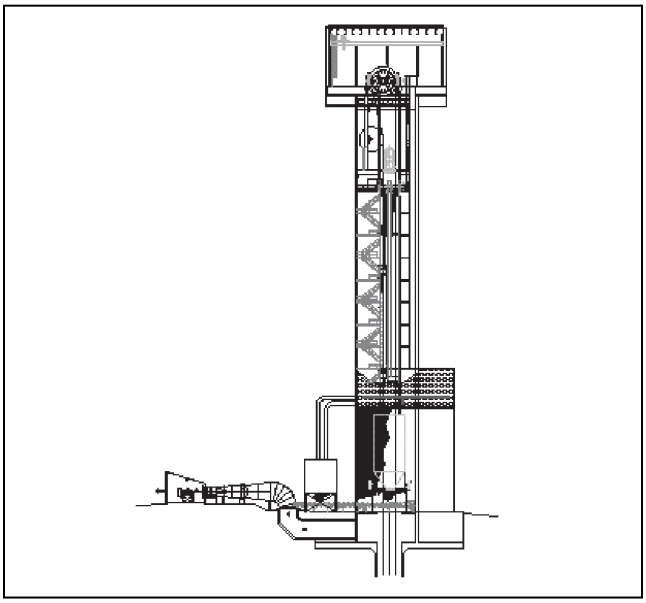


Figure 12.9-27 Concrete headframe

for any new development. The advantages of concrete structures include the following:

- Effective use can be made of space within the structure for tower-mounted friction hoists.
- When used in conjunction with ground-mounted drum hoists, the hoist may be located at any position around the headframe.
- Maintenance costs are low.
- A high damping effect of the massive concrete structure occurs on shock and vibration.
- The collar layout and the interconnection with service buildings is simple.
- Future expansion is possible.
- The structure is durable and resistant to corrosion.

The disadvantages of concrete structures include the following:

- Erecting in remote areas is difficult.
- Supporting the structures in poor ground is difficult.
- Concrete offers no salvage value.

For steel structures, advantages include the following:

- Foundation loads are lighter.
- They are adaptable to minor changes.
- They offer no salvage value.
- Capital-cost saving in remote areas is possible.
- The structure can be prefabricated into transportable lengths for on site assembly.
- Construction is speedy.

The main disadvantage of steel is that it is prone to corrosion and, thus, is not suitable as a production headframe for corrosive ore (e.g., salt and sulfides) unless the steel is treated and regularly maintained.

Design Considerations

A successful headframe design must be based on an overall integrated hoisting system design, which involves a critical examination of the impact of at least the following items: loads, foundation, mine services, sinking provisions, equipment monitoring facilities, conveyance and rope handling, heating and ventilation, and others such as local mining regulations.

Loads

All headframes should be designed to withstand a combination of loads, comprised of the following:

- Dead load consisting of the weight of the headframe, sheave wheels, ore bins, and contents
- Live load from hoisting at maximum capacity
- Breaking loads of the ropes when the conveyance is stopped by the crash beams or if it is jammed in the shaft (breaking stresses introduced by an overwind are usually transferred to the structural system, often through an arrestor gear, and thus the supporting structure must be critically analyzed with a thorough understanding of the arrestor mechanism)
- Wind load, the intensity of which depends on the location, height, and shape of the structure
- Snow load
- Temperature and seismic stresses, taken from published data relating to the geographic area of the shaft location

Foundation

The foundation should be capable of withstanding and absorbing stresses imposed by the structure and associated loads. Foundation design in areas where the ground must be frozen by artificial means before the shaft is sunk require much more consideration than normal because the ground will heave when the freeze is removed, which can lead to tilting of the headframe.

Mine Services

The design of the headframe must suit surface layouts for providing the essential mine services. When possible, it should be located close to the shaft collar house, waiting area, lamproom, first-aid room, hoist house, maintenance shop, changing house, and administrative offices. The type and design of the headframe must also take into account the shaft's internal arrangement, position of the hoist with respect to the shaft, and the position of the sheave steel.

Facilities for the following key concerns must be considered and adequately developed during the conceptual design stage:

- Arrangement of skip dumping area in relation to ore and waste storage and handling facilities
- Clearance required for installation, maintenance, and removal of conveyances
- Lifting equipment required for maintenance of conveyances
- Access required for maintenance of conveyances, dump areas, and so forth

In the case of headframes for drum hoists, provision must be made for

- Lifting equipment and access for maintenance of sheaves;
- Access and anchorage facility to support the ends of hoist ropes when doubling down for pretensioning drum end coils, and
- Subcollar loading access for double-deck cages.

The design of headframes for friction hoists should address the following:

- Supporting facility at predetermined elevations for hoist or balance ropes and conveyances or counterweights during installation and changes
- Clearance for installation of conveyance arrestor gear
- Provisions for handling shaft ventilation air and for sealing machinery rooms
- Arrangement of power conversion equipment adjacent to hoist motors to minimize runs of heavy bus bars
- Lifting equipment access during changing of headropes and tail ropes
- Support, accommodation, inspection, removal, and replacement facilities for rope guides where used
- Ventilation to specific machines and areas
- Controlling moisture, dust, and temperature where required
- Accommodation of sinking arrangements
- Accommodation of all services for air, water, communications, and power lines to the shaft and to the headframe-mounted equipment
- The accommodation of stairwells and elevators
- Lighting and lightning protection

Sinking Provisions

After the decision has been made to sink a shaft, considerations should be given to the type of surface sinking plant (such as headframe and hoist) that is to be used for shaft sinking. In particular, it should be determined whether the sinking plant is to be temporary or is to make use of the permanent headframe and/or hoist.

Temporary headframe. Sometimes a temporary headframe is used during shaft sinking by the contractor. These headframes are generally made in modular form from structural steel. Headframes of this type are portable and can be reused many times.

Design criteria for a temporary sinking headframe are the same as for a permanent headframe. Its height depends on the method of rock disposal and size of bucket. The sinking headframe should embody features for dumping buckets or skips, for protecting personnel on the surface and in the shaft from falling pieces of rock while dumping, and for minimizing the work of top personnel in dumping buckets and removing broken rock. The design of temporary headframe structures should accommodate an adequately sized storage bin and should provide sufficient clearance for a rock disposal vehicle.

Permanent headframe. Current trends favor installation of a permanent headframe for the sinking operation, as it tends to be more economical and time saving. The permanent main skeleton can be designed to facilitate erection and installation of the sinking sheaves at a suitable elevation. The structure can then be finished and equipped during the shaft sinking operation. The use of the permanent structure as a sinking headframe can make the permanent hoisting facilities available at an earlier date following completion of the shaft-sinking program.

Equipment Monitoring Facilities

Two main hoisting incidents, which can be detrimental to personnel and equipment and require monitoring, are overtravel at the shaft extremities (i.e., sump, headframe) and overspeed during the duty cycle. To protect against these incidents, it is mandatory that monitoring and safety devices be installed in the headframe.

Operation of all hoists is monitored by digital equipment designed for this purpose and often termed *safety controllers*. These are driven by the hoist drums and they check hoisting speed, overtravel at each end of the wind, and deceleration in the end zones. In addition to these dynamic checks, track limit switches must be incorporated to protect against a safety monitor losing its synchronism with the hoist. The function of track limit switches is to provide independent overtravel protection in the headframe.

Ultimate protection against overwind is provided for all Koepe hoists by conveyance arrestors. These are installed at the ends of travel in each hoisting compartment. The headframe arrestors are designed to retard an ascending conveyance from the speed of a person riding at a rate of 0.9 *g*'s. Arrestors are also installed at the shaft bottom and are designed to retard a full descending conveyance at a rate of at 1 *g* or higher. It is not practical to install arrestors in the headframe to stop a hoist from full speed. For example, a hoist traveling at 18 m/s (3,600 fpm) would require a stopping distance of 18.5 m (60 ft) when decelerating at 0.9 *g*'s. This amount of overtravel provision would be very expensive to provide and would still not guarantee absolute protection because a runaway hoist can easily reach speeds in excess of 50 m/s (10,000 fpm).

Where applicable, the headframe should accommodate installations of a crash beam and catch gear. With drum winders, the crash beam is located below the sheaves in the headframe. In the event of an excessive overwind, it stops the conveyance from traveling beyond this point and is designed to withstand the breaking strength of the headrope. Should a conveyance on a Koepe hoist be involved in an overwind situation in which the hoist ropes break, the conveyance and tail ropes will fall down the shaft, causing damage. To prevent this, fallback protection should be provided in the headframe to "catch" the conveyance.

Conveyance and Rope Handling

In the design and layout of a headframe, provisions must be made for the handling and changing of conveyances, hoisting ropes, and rope guides (if applicable). Friction hoisting often requires the use of very large skips and cages. Because of their size, the service and handling of these conveyances, and the tying off of the headropes and tail ropes associated with them, considerable problems can arise. In the friction hoist headframe, these can be facilitated through the installation of an adequately sized overhead traveling crane. More

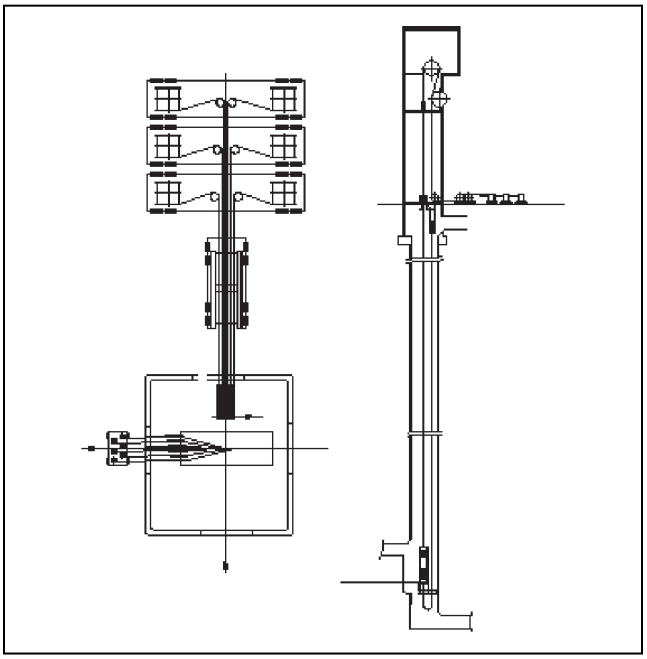


Figure 12.9-28 Multirope changing reeving diagram

recently, special equipment has been designed to permit the removal or installation of all of the headropes at the same time. The process requires a special winch that is designed to pull all of the headropes at once and transfer them on or off of numerous rope-holding winches. A reeving diagram for this arrangement is shown in Figure 12.9-28. In the drum hoist headframe, handling and changing of conveyances and ropes are relatively easy and are usually achieved through the use of mobile cranes, rope blocks, and tugger hoists.

Heating and Ventilation

In warm climates, the headframe may be left open. In colder climates, however, it is necessary to design the headframe as an enclosed structure with adequate provisions for a heating system. In tower-mounted friction hoist systems, with all of the hoists, motors, controls, and so forth enclosed in the headframe, it is necessary to provide a ventilation system to supply filtered air for both cooling and heating.

Some headframes located over downcast shafts are pressurized and must be equipped with air locks to isolate the movement of personnel and equipment into and out of the collar house.

Mining Regulations

Mining operations, including the design and construction of the headframe and shaft facilities, are subject to local, state, and federal regulations that relate to, among other things, safety of permanent structures and environmental protection. These regulations affect the design and construction of mine facilities.

Height

Because of increasing public awareness of the aesthetic impact of industrial development, the height of the headframe should be considered. This is especially true for locations in flat terrain where any large structure can be seen for long distances. The height of a headframe is generally based on the storage bin capacity requirements for ore and waste, the clearance required for the loading facility under the storage bin, the requirement that the resultant force from headframe loadings falls within the backleg post, and the need for providing minimum clearance for conveyance overtravel beyond its normal travel to the head sheaves. Of the above, the determination of the minimum clearance for overrun distance is the principal design consideration. The determination of the minimum clearances required in headframes is described in the following paragraphs.

Operating Allowance

An operating allowance is required to allow for variations in repeatability of the final stop position of the conveyance in the headframe. In multilevel hoisting with drum hoists, this tolerance compensates for permanent rope stretch, differences in drum and rope diameter, and clutching accuracy. In the case of Koepe hoists, the allowance compensates for creep during resynchronization and also allows a reasonable gap for rope adjustment due to permanent rope stretch.

Rope Stretch Allowance

This is included to avoid contact with the track limit switch or encroaching on the safety controller's overwind protection limit when empty conveyances are hoisted.

Stopping Allowance

A stopping allowance is needed to ensure that clearance remains between the top of the conveyance and the first obstruction in the headframe should the conveyance exceed its normal operating limits and require the use of emergency braking to stop.

Rope Tension Equalization

The ropes on a Koepe hoist operate at slightly different diameters because of uneven wear of the rope treads. To minimize the effects of this variation, it is recommended that the ratio of the maximum to minimum suspended rope length should not be more than 50.

Spacing of Koepe Wheel and Deflection Sheave

When a headrope is bent in an arc, the wires that make it up move relative to one another to accommodate the different lengths at the outer and inner radii of the surface of the rope. When the rope straightens again, the wires do not assume their original positions immediately, primarily because there is friction between them. The headrope passing over the hoist wheel is bent one way, straightens when it leaves the wheel, and then bends the other way when it passes over the deflection sheave. If the time taken to pass from the wheel to the sheave is less that 0.5 seconds, the wires will not have time to realign between these reverse bends and will eventually produce distortion and premature failure of the rope. Allowance for this spacing must be made when laying out the headframe.

Rope Harmonics and Vibrations

Vibrations in hoist ropes can be observed as "yo-yo-ing" at the collar, and, in the case of drum hoists, as the rope whips between the hoist and the head sheave. The causes of these vibrations are jerking operations, such as uneven braking, and the horizontal and vertical movement of the rope at the crossover points on the drum. These vibrations cannot be avoided and normally do not cause problems. However, there have

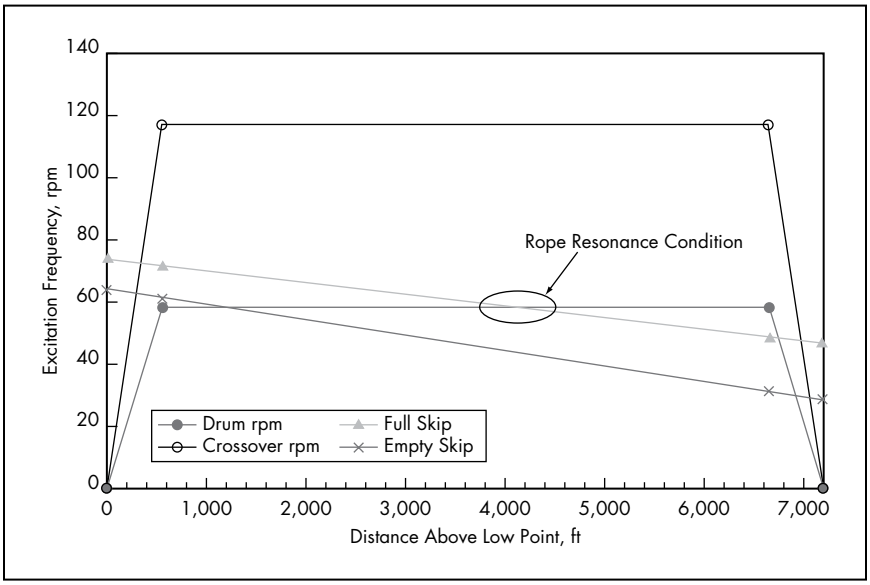


Figure 12.9-29 Rope resonance plot

been cases where the frequency of the crossover motion is a harmonic of the natural frequency of the flight rope (i.e., the length of rope between the drum and head sheave), and severe and dangerous whipping has occurred. Such a situation is most likely with a long, flat span of rope between the hoist and head sheave.

The hoist designer should analyze the harmonic behaviors of the ropes and try to select the type of drum grooving that will avoid harmonic vibrations. This selection requires two calculations. The first calculation determines the natural frequency of the flight rope (which varies with rope tension and therefore decreases linearly as the conveyance ascends the shaft). The second calculation determines the frequency spectrum excited by the crossover spacing. If the excitation spectrum includes harmonics that match the natural frequency of the flight rope, alternative crossover geometries will have to be considered. If suitable alternatives are not found, the hoist speed will have to be altered or idler towers will have to be added between the drum and the head sheave.

A typical plot of excitation frequency and flight rope natural frequency is illustrated in Figure 12.9-29. This plot illustrates that a rope resonance condition would exist if the rope crossover occurred once per revolution but not if it occurred twice per revolution with the crossovers spaced 180° apart.

Calculations for Minimum Clearance

The following details show how the required clearances are to be calculated.

Elastic Rope Stretch for Skip

The headropes stretch when a skip is loaded. This stretch is locked into the rope as it coils on the drum when the loaded skip is raised and is relieved as it uncoils when the empty skip is lowered. The drum will, therefore, make more revolutions when raising a full skip than an empty one. Because the skip distance is measured by revolution of the drum, the hoisting cycle is only accurate for a loaded skip. The headframe will usually be fitted with a "clutching-mark" magnet switch located at a distance above the "skip-at-dump" magnet switch equal to the elastic rope stretch. The stretch (S) can be calculated from

$$S = P*L/(A*E)$$

where

P = payload
L = rope length from the drum to the skip at the loading pocket
A = rope area based on its nominal diameter
E = rope elastic modulus in tension

A good guideline for elastic rope stretch is 300 mm (1 ft) per 300 m (1,000 ft) of shaft depth.

Clutching Accuracy

A clutch includes a ring with a finite number of teeth. The clutching accuracy is, therefore, limited to plus or minus the tooth pitch. The ability for the operator to compensate can be determined from

$$C = \pm\pi*D/(2*N)$$

where

C = clutching accuracy that can be achieved
D = pitch circle diameter of the headrope
N = number of teeth in the clutch ring

Unequal Drum and Rope Diameters

Inequalities in drum or rope diameters between two drums of the same hoist will be amplified by the number of revolutions the hoist makes as it travels up the shaft. This difference is given by

$$\Delta T = \pi*N*(\Delta Dd + 1.7*\Delta Rd)$$

where

ΔT = difference in travel
N = number of revolutions of the hoist
ΔDd = difference in drum diameters
ΔRd = difference in rope diameters

Unequal Rope Lengths

If the ropes on two drums of a hoist are not exactly the same and the drums wind more than one layer of rope, then the drum with the longer rope will operate at a larger average diameter. As a result, this drum will take fewer revolutions to raise a skip than the drum with less rope. This difference is proportional to the number of extra turns times the constant π.

For example, consider the case where the rope and drum diameters are equal but there is one more turn on one drum than the other. Assume also that there are four layers of rope, the PCD of the first layer is d = 4,653 mm, and the PCD of the fourth layer is D = 4,915 mm. One drum has 6 turns on the first layers when the skip is at the loading pocket and 18 turns on the fourth layer when the skip is at the dump, and the other drum has 7 turns on the first layer when the skip is at the dump and 19 turns on the fourth layer when the skip is at the dump.

$$\text{difference in rope lengths} = \pi * d = \pi * 4.653 = 14.6 \text{ m}$$

$$\text{difference in skip travel} = \pi * (D - d) = \pi * (4.915 - 4.653) = 0.8 \text{ m}$$

As these calculations demonstrate, when one rope is damaged and needs to be shortened, the same length must be cut off the other rope.

Stopping Distance

If a conveyance travels beyond the track limit switch (which must be checked at least daily) it will initiate an emergency stop using the brakes to stop and hold the hoist. The allowance to be made for the conveyance overtravel during this event is given by

$$S = 2 * V^2/(2 * D)$$

where

S = stopping distance
V = hoisting speed permitted by the safety controller in the end zone (typically 0.5 m/s or 100 fpm)
D = deceleration rate achieved by the braking system

The factor 2 is included in the equation to account for the dead time of the brakes.

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PART 13

Underground Extraction

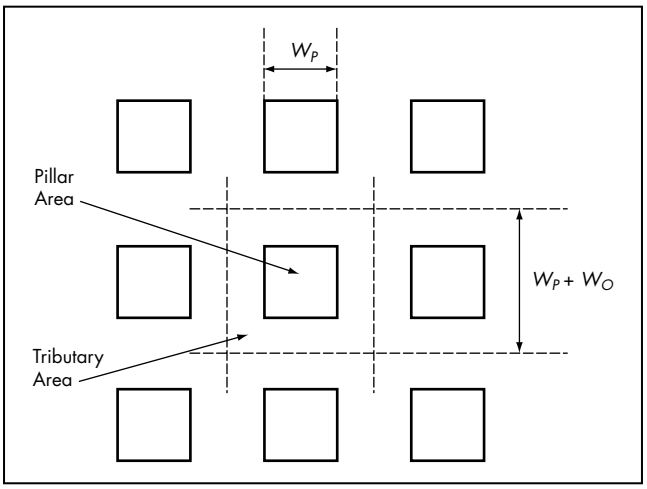
CHAPTER 13.1

Room-and-Pillar Mining in Hard Rock

Richard L. Bullock

Room-and-pillar (R&P) mining is a mining method whereby a series of rooms (horizontal openings) is extracted, leaving pillars of ore, rock, or coal in place between the rooms. Figure 13.1-1 shows a plan view of an ideal R&P mine. The square blocks are pillars; the spaces between the pillars are rooms. In hard rock, pillars are usually much smaller horizontally than rooms. In soft rock and coal mines, they are usually much larger.

The R&P method has been used widely and often—for example, in lead/zinc/copper ore bodies of mines in southeast Missouri (United States) for 150 years; in mines in Charcas (Mexico) for more than 100 years; in copper mines in Poland for nearly 50 years; and in underground rock quarries and salt mines worldwide for more than 100 years. The proliferation of this method for such a long period of time suggests that it is low cost, versatile, and safe.



Source: Zipf 2001.

Figure 13.1-1 Plan view of a room-and-pillar mine

ORE-BODY CHARACTERISTICS OF R&P MINING

The R&P method is normally employed on fairly flat seams or ore bodies, but may in some cases be used in seams that angle up to 45°. Ideally, the ore body or seam to be mined should be very large laterally and fairly uniform in thickness. However, many R&P mines in hard-rock ores have nonuniform ore horizons. For example, in some of the R&P mines in the Old Lead Belt of Missouri, the ore horizon varies in thickness from 3 to 91 m (10 to 300 ft). In the Viburnum Trend mines of Missouri, ore varies in thickness from 3.5 to 30 m (12 to 100 ft).

The strength of both the back/roof and the ore are important. Obviously, roof strength must be sufficient to stand open for the width of the room when the room is opened. Similarly, ore strength determines the size of the pillar required to support the open span of the back. Thus pillar size is not fixed but rather is determined by the sum of (1) stress in the pillar due to the weight of the overburden above the opening, (2) additional stress in the pillar due to removal of the ore that originally supported the current back, and (3) tectonic stress-related forces that may still remain in the rock. The ore in the pillar must be strong enough to compensate for these collective stresses.

Farmer (1992) and Zipf (2001) described the design of stope pillars and barrier pillars normally used for a new R&P mine in a new mining district. The caveat "new mine in a new district" is important, since, in opening a mine in an already-mined district, decisions regarding pillar size and room size for a given stope depth and ore strength have already been made. Yet it is nevertheless important to understand how to design stope (panel) pillars and barrier pillars for a new mine.

TRADITIONAL STRENGTH-BASED PILLAR DESIGN METHODS

This section is adapted from Zipf (2001). Significant research has been done on pillar design for both R&P and longwall mining. Many ground-control and pillar-design experts have contributed (e.g., Holland 1964; Salamon and Munro 1967; Hardy and Agapito 1977; Mark 1999). Their studies include verification of pillar-design formulas and formula constants by statistical verification with a large database of both failed and nonfailed coal mine incidents from Australia and South Africa (Galvin et al. 1999; Gale and Hebblewhite 2005). However, with the exception of the work of Hardy and Agapito, these studies

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concern coal, and there has been no similar verification for hard rock or even other softer minerals such as trona, salt, oil shales, or borates. The purpose of the following discussion is to extend this work to the design of pillars for all minerals, not just coal.

R&P mines may cover only a hundred square meters and contain just a few pillars, as is typical for small zinc deposits. Or they may cover many square kilometers, as is typical for coal, trona, and limestone mines. In an R&P mine, large arrays of pillars are typically grouped into panels, which in turn are surrounded by barrier pillars. The small pillars within a panel are sometimes called panel pillars. In developing layouts, the mining engineer must develop appropriate dimensions for room spans, panel pillar widths, panel sizes, and barrier pillar widths. Developing these dimensions requires evaluation of not just pillar strength but also the consequences of pillar failure, which can happen anywhere in the layout at any time.

Traditional strength-based pillar design requires estimates of pillar stress and pillar strength. The safety factor for the pillar is then calculated by dividing pillar strength by pillar stress. What constitutes an acceptable safety factor depends on the tolerable risk of failure. Safety factors of 2 are typical for pillars in main development headings or panels during advance mining. Safety factors of 1.1 to 1.3 are typical for panel pillars after retreat mining. Safety factors much less than 1.0 are possible for panels where pillar failure is the eventual intent.

Pillar Stress

All equations in this section can be dimensioned as preferred by the user, as long as dimensions are consistent with the equation constants.

According to traditional R&P design methods, in-situ vertical stress is (Farmer 1992)

$$\sigma_s = \lambda z \tag{13.1-1}$$

where
λ = unit weight of rock
z = depth of mining horizon

The tributary area method then provides a first-order estimate of the average pillar stress. For the square R&P system shown in Figure 13.1-1, the average pillar stress is

$$\sigma_{pa} = \sigma_z \left(\frac{W_p + W_o}{W_p} \right)^2 \tag{13.1-2}$$

where
W_p = pillar width
W_o = opening width

For rectangular or irregular-shaped pillars, where R is the extraction ratio, the average pillar stress is

$$\sigma_{pa} = \sigma_z \left(\frac{1}{1-R} \right) \tag{13.1-3}$$

$$R = \frac{A_M}{A_T} = \frac{A_T - A_P}{A_T} \tag{13.1-4}$$

where
A_M = area extracted (i.e., total room area)
A_T = total area of the ore body
A_P = pillar area

The tributary area method assumes that the mined area is extensive and all pillars have the same dimensions. It also ignores the deformation properties of the surrounding rock mass relative to the pillar rock. However, pillar stress is generally higher at the center than at the edge of a panel. Coates (1981) partially solved this problem by developing the following relation for average pillar stress that accounts for the width and number of pillars across a panel and the relative mechanical properties of the pillar and rock mass:

$$\sigma_{pa} = \sigma_z \left\{ \frac{\left[2R - K_0 \frac{H}{L} \frac{(1-2\upsilon_{rm})}{(1-\upsilon_{rm})} - \frac{\nu_p}{(1-\upsilon_p)} K_0 \frac{H}{L} \frac{E_{rm}}{E_p} \right]}{\frac{H}{L} \frac{E_{rm}}{E_p} + 2(1-R)\left(1 + \frac{1}{N}\right) + 2\frac{RB}{L} \frac{(1-2\upsilon_{rm})}{(1-\upsilon_p)}} \right\} \tag{13.1-5}$$

where
K_0 = ratio of horizontal to vertical stress
H = mining height
L = lateral extent of the mined area
E, υ = elastic constants
rm = rock mass
p = pillar mass
B = individual opening width
N = number of pillars across the panel

Although this equation is based on two-dimensional elasticity theory and therefore applies only to long, narrow rib pillars, it illustrates the behavior of average pillar stress. As the E_{rm}/E_p ratio increases, average pillar stress decreases due to the bridging effect of the stiff rock mass. Similarly, as panel width L decreases and the H/L ratio increases, the average pillar stress decreases.

As with the tributary area method, the solution offered by Equation 13.1-5 gives only average pillar stress for all panel pillars and does not give changes in pillar stress across the panel. Boundary-element method programs such as ExamineTAB (Zipf 2001), MUL SIM/NL (Zipf 1992a, 1992b), or LAMODEL (Heasley 1997, 1998) are needed to calculate changes in pillar stress across a panel or within a pillar.

Pillar Strength

Over the past several decades, a large amount of rock mechanics literature has addressed pillar strength in both coal and metal/nonmetal mines (Obert and Duvall 1967; Hoek and Brown 1980; Bieniawski 1992; Brady and Brown 1993). Much of this work is empirical and addresses two issues: (1) the size effect, whereby rock strength decreases as specimen size increases; and (2) the shape effect, whereby rock strength decreases as the width-to-height ratio decreases. Using energy considerations, Farmer (1985) developed theoretical expressions relating strength to size. When failure occurs in a brittle manner, as it does in most rocks, strain energy within the specimen transforms to fracture surface energy, which is constant for a particular rock. Based on energy conservation,

$$SED * V = FE * A \tag{13.1-6}$$

where
SED = strain-energy density
V = volume
FE = fracture-surface energy, which is a material constant
A = fracture surface area

Table 13.1-1 Values for constants in empirical pillar-strength formulas

Source	*a**	*b**	α	β	Comments
Bunting (1911)	0.7	0.3	—	—	Pennsylvania anthracite
Obert and Duvall (1967)	0.78	0.22	—	—	Laboratory rock and coal
Bieniawski (1968)	0.64	0.36	—	—	South Africa coal
Skelly et al. (1977)	0.78	0.22	—	—	West Virginia coal
Greenwald et al. (1939)	—	—	0.5	0.83	Pittsburgh seam mines†
Holland (1964)	—	—	0.5	1	U.S. coal mines†
Salamon and Munro (1967)	—	—	0.46	0.66	South Africa coal mines‡
Hardy and Agapito (1977)	—	—	0.60	0.95	U.S. oil shale mines†

Source: Zipf 2001.
*a and b are dimensionless.
†Use with English units only.
‡Use with metric units only.

Rearranging gives

$$SED * \frac{V}{A} = SED * L = FE = \text{constant} \tag{13.1-7}$$

where L is the characteristic dimension or length of the rock specimen.

Assuming that laboratory-scale failure is mechanistically similar to full-scale pillar failure, then

$$SED_S * L_S = SED_P * L_P \tag{13.1-8}$$

where

S = subscript denoting laboratory-scale specimen
P = subscript denoting full-scale pillar

Since strain-energy density at failure is proportional to the square of stress at failure,

$$\sigma_S^2 * L_S = \sigma_P^2 * L_P \tag{13.1-9}$$

or

$$\frac{\sigma_P}{\sigma_S} = \left(\frac{L_S}{L_P}\right)^{1/2} = \left(\frac{V_S}{V_P}\right)^{1/6} = \left(\frac{V_S}{V_P}\right)^{0.1\,7} \tag{13.1-10}$$

where

σ_P = strength of the full-scale pillar
σ_S = strength of the laboratory specimen
V = volume, which is proportional to L^3

This theoretical relationship accounts for the size effect in observed rock strengths. The following empirical strength formula, proposed by Hardy and Agapito (1977) for oil shale pillars, follows this general theoretical form and provides some experimental confirmation. It also includes an additional term for pillar shape.

$$\frac{\sigma_P}{\sigma_S} = \left(\frac{V_S}{V_P}\right)^{0.118}\left[\frac{W_P}{H_P}\frac{H_S}{W_S}\right]^{0.833} \tag{13.1-11}$$

where

W_P = width of pillar
H_P = height of pillar
H_S = height of specimen
W_S = width of specimen

Classic empirical pillar-strength formulas usually follow one of two general forms:

$$\sigma_P = \sigma_{S'}\left(a + b\frac{W}{H}\right) \tag{13.1-12}$$

$$\sigma_P = K\frac{W^{\alpha}}{H^{\beta}} \tag{13.1-13}$$

where

σ_S = strength of a cubical pillar ($W/H = 1$) at or above the critical size
K = constant characteristic of the pillar rock
a, b, α, β = constants that account for the shape factor

Formulas by Obert and Duvall (1967) and Bieniawski (1968) follow the first form; formulas by Salamon and Munro (1967) and Holland (1964) follow the second. In these forms, the size effect is accounted for directly by the unit pillar strength σ_S or the rock constant K. The constants a, b, α, and β, which account for the shape factor, show reasonable agreement, as shown Table 13.1-1.

Several methods exist for estimating σ_S and K in Equations 13.1-12 and 13.1-13, respectively. One is to consider the strength of a cubical pillar ($W/H = 1$) at or above the critical size, where critical size is the size beyond which rock mass strength remains relatively constant. For coal pillars, the critical size is widely recognized to be about 0.9 m (36 in.). For coal mines in the United States, the recommended strength value for a cube of coal this size is σ_S = 6.2 MPa (900 psi) (Mark and Chase 1997). This unit strength for a full-scale cube of coal can then be adjusted for shape effects using the Mark–Bieniawski relation (Mark 1999):

$$\sigma_P = \sigma_{S'}\left(0.64 + 0.54\frac{W}{H} - 0.18\frac{W^2}{HL}\right) \tag{13.1-14}$$

which reduces to the original Bieniawski relation when pillar width W equals pillar length L.

If laboratory-scale strength data (σ_S) is available, the full-scale strength of a cube of rock mass (σ_S), from Equation 13.1-10, is

$$\sigma_{S'} = \sigma_S\left(\frac{V_S}{V_{S'}}\right)^{0.17} \tag{13.1-15}$$

This unit strength for a full-scale cube of the rock mass can then be adjusted for the shape effect using the Obert–Duvall relation:

$$\sigma_P = \sigma_{S'}\left(0.78 + 0.22\frac{W}{H}\right) \quad (13.1\text{-}16)$$

Finally, the Hoek–Brown failure criterion can also provide an estimate of the strength for a full-scale cube of the rock mass. For most rock masses with good to reasonable quality,

$$\sigma_1' = \sigma_3' + \sigma_c\left(m_b\frac{\sigma_3'}{\sigma_c} + s\right)^a \quad (13.1\text{-}17)$$

where

m_b, s, a = constants that depend on the rock mass quality
σ = uniaxial compressive strength of the intact rock pieces (equivalent to σ_S)
σ_1' = axial principal stress
s_3' = confining principal stress

The constants m_b and s are estimated using a rock-mass classification index called GSI, or Geological Strength Index, which is equivalent to Bieniawski's rock mass rating (RMR) system, assuming a dry rock mass. For a complete discussion of the Hoek–Brown failure criterion for rock masses, see Hoek et al. (1995).

Barrier Pillar Design

The previous formulas for pillar stress and pillar strength apply mainly to pillars within a large array, or so-called panel pillars. The barrier pillars surrounding this array of panel pillars also require sizing. The tributary area method can provide a conservative estimate of barrier pillar stress, if we assume that the panel pillars have all failed or are all mined and therefore carry no overburden stress. Alternatively, numerical methods such as MULSIM/NL or LAMODEL can estimate barrier pillar loads without need for conservative assumptions inherent to the tributary area method.

The empirical pillar strength formulas shown earlier apply to pillars with width-to-height ratio $W/H < 5$. For barrier pillars and other pillars in coal mines with $W/H > 5$, Salamon's "squat" pillar formula applies:

$$\sigma_P = \sigma_{S'}\left(\frac{5^b}{V_P^a}\right)\left(\frac{b}{e}\left(\left(\frac{W/H}{5}\right)^e - 1\right) + 1\right) \quad (13.1\text{-}18)$$

where

$b = 0.5933$
$a = 0.0667$
$e = 2.5$

For hard-rock and other noncoal mines, an equivalent squat pillar formula does not exist. It is necessary to extrapolate the Obert–Duvall relation to high W/H ratios or use numerical methods and the Hoek–Brown failure criterion to estimate the required barrier pillar size.

Koehler and Tadolini (1995) reviewed no fewer than 10 empirical and observational approaches to barrier pillar design in coal mines. Most approaches calculate minimum barrier pillar width as a function of depth, and some include seam thickness. Coal strength is generally neglected. Equivalent empirical approaches to estimate barrier pillar size in noncoal mines do not appear to exist.

Summary of Traditional Strength-Based Pillar Design Methods

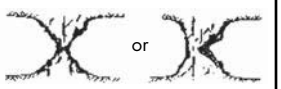
For R&P coal mining, the Analysis of Retreat Mining Pillar Stability (ARMPS) method and program developed by Mark and Chase (1997) applies. This method incorporates the tributary area method and empirical strength formulas for sizing coal pillars during advance and retreat mining, and includes adjustments to pillar stress for factors such as side-loading from previously mined panels. It can also determine barrier pillar size. Again, an equivalent program does not appear to exist for noncoal R&P mining, but the ARMPS method should apply with suitable adjustments to the input parameters.

With traditional strength-based pillar design methods, the mining engineer can determine panel pillar size and barrier pillar size, but not panel dimensions. Operational considerations such as equipment and productivity determine the panel width and usually set it as large as possible. Based on strength considerations alone, a narrow panel requires a narrow barrier pillar and a wide panel requires a wide barrier pillar, and rock mechanics factors do not affect the panel-width determination. Maximum panel width can be determined by the size of air blast that an operation can withstand, but this is not a rock mechanics factor. Other rock mechanics considerations are needed to determine maximum panel width as well as panel pillar and barrier pillar sizes.

Typical R&P Sizes for Metal Mines

Within the bounds of roof and pillar safety, larger rooms enable use of larger equipment and increase the productivity of the stoping operation. Case studies completed many years ago showed that the average productivity of three mines with average room size of 41 m^2 (440 ft^2) was 21.7 t, or metric tons (23.9 st, or short tons) per worker-shift; in contrast, the average productivity of three mines with average room size of 158 m^2 (1,700 ft^2) was 71.5 t (78.8 st) per employee shift (Bullock 1998a). While safety should never be compromised, larger rooms are more productive, providing that they do not get too high to safely maintain the back and the loose rock on high pillars and ribs. Many limestone producers typically have rooms 12.2 m (40 ft) high in a single pass.

Examples of typical R&P mines with very high rock-quality designations (RQD = >85) are the mines of the Viburnum Trend (Carmack et al. 2001). Room widths are held to 11 m (36 ft), but are typically 9.8 m (32 ft); final heights after multiple passes are preferred to be <15 m (50 ft). Roof bolting is done typically in every intersection and elsewhere as needed. Historically, pillar sizes were 8.5 × 8.5 m (28 × 28 ft) on 20-m (60-ft) centers; then panel pillar sizes of 11.6 × 23 m (38 × 76 ft) were adopted for wider areas of mining where pillar extraction is planned upon retreat from the area. Relative pillar strengths in the mines are monitored using a rating system of 1 to 6, where 1 denotes no visual stress and 6 denotes pillar failure (Figure 13.1-2). This classification system is important not only for pillar recovery but also for pillar maintenance. Pillars whose rating reaches 3 must be reinforced with grouted rebar bolts to halt further deterioration if they are to maintain their integrity for a long period. The rock strength of samples in this area depends considerably on mineral content, but varies from ~70 to 200 MPa (10,000 to 30,000 psi). A similar but less defined and seemingly more conservative pillar rating system was developed by Westmin Resources Ltd. (Lunder and Pakalnis 1997).

Pillar Rating	Pillar Condition	Appearance
1	No indication of stress-induced fracturing. Intact pillars.	
2	Spalling on pillar corners; minor spalling of pillar walls. Fractures oriented subparallel in walls and are short relative to pillar height.	
3	Increased corner spalling. Fractures on pillar walls more numerous and continuous. Fractures oriented subparallel to pillar walls and lengths are less than pillar height.	
4	Continuous, subparallel, open fractures along pillar walls. Early development of diagonal fractures (start of hourglassing). Fracture lengths are greater than half of pillar height.	
5	Continuous, subparallel, open fractures along pillar walls. Well-developed diagonal fractures (classic hourglassing). Fracture lengths are greater than half the pillar height.	
6	Failed pillar, may have minimal residual load-carrying capacity and be providing local support to the stope back. Extreme hourglassed shape or major blocks have fallen out.	

Source: Carmack et al. 2001.

Figure 13.1-2 System for rating pillar strength

Some resources exist for which mineral continuity is so consistently inconsistent that no amount of surface prospecting can accurately identify ore continuity, and thus pillar and stope layouts cannot be preplanned. The Mississippi Valley type zinc deposits of central and east Tennessee (United States) are of this type. If this is the case and it is the first mine in a new mining district, an underground test mine is required so that better geotechnical information can be obtained on which to base plans for room widths and pillar widths and height, all of which greatly affect mine operating cost. Structural information can then be seen and accurately mapped, data needed to determine joint and fracture information can be accurately measured, in-situ stress measurements can be taken, and large core samples can be gathered for laboratory analysis.

ORIENTATION OF ROOMS AND PILLARS

Pillar orientation is affected by in-situ stress. The orientations of both rooms and pillars are affected by the extent of dip.

Pillar Orientation Due to In-Situ Stress

As in all mining methods, mining engineers who are planning R&P operations must be aware of probable in-situ stress within the rock before mining. Any significant horizontal stress in a particular direction must be taken into account by orienting the room advance and the direction of rectangular pillars to give the most support in the direction of stress. In the early phase of development, research should be done to determine the magnitude and direction of inherent stress. However, very shallow R&P operations may have very little horizontal stress, in which case the orientation of rectangular or barrier pillars is not a concern, particularly if a mine has been opened from a hillside with adits where nature could have relieved horizontal stresses eons ago.

In sharp contrast to this condition, some deep R&P mines have tremendous problems, not only with horizontal stress but also with rock that absorbs a large amount of energy and then fails violently. In such operations, not only is pillar orientation important, but the sequence of extraction and how it takes place is also important. Such an operation is documented as a case study by Korzeniowski and Stankiewicz (2001).

R&P Orientation Due to Dip

R&P mining is usually done on fairly flat strata but need not be limited to that. Shallow dips, of up to ~8%, have little effect on the layout of the R&P stope. If the mine is developed from a shaft, rooms should be laid out so that haulage of the load goes downgrade. If the mine is developed from an adit or decline, of course, this would not be the case. As to the orientation of rectangular pillars, it probably makes sense to orient the long side of the pillar in the direction of the dip.

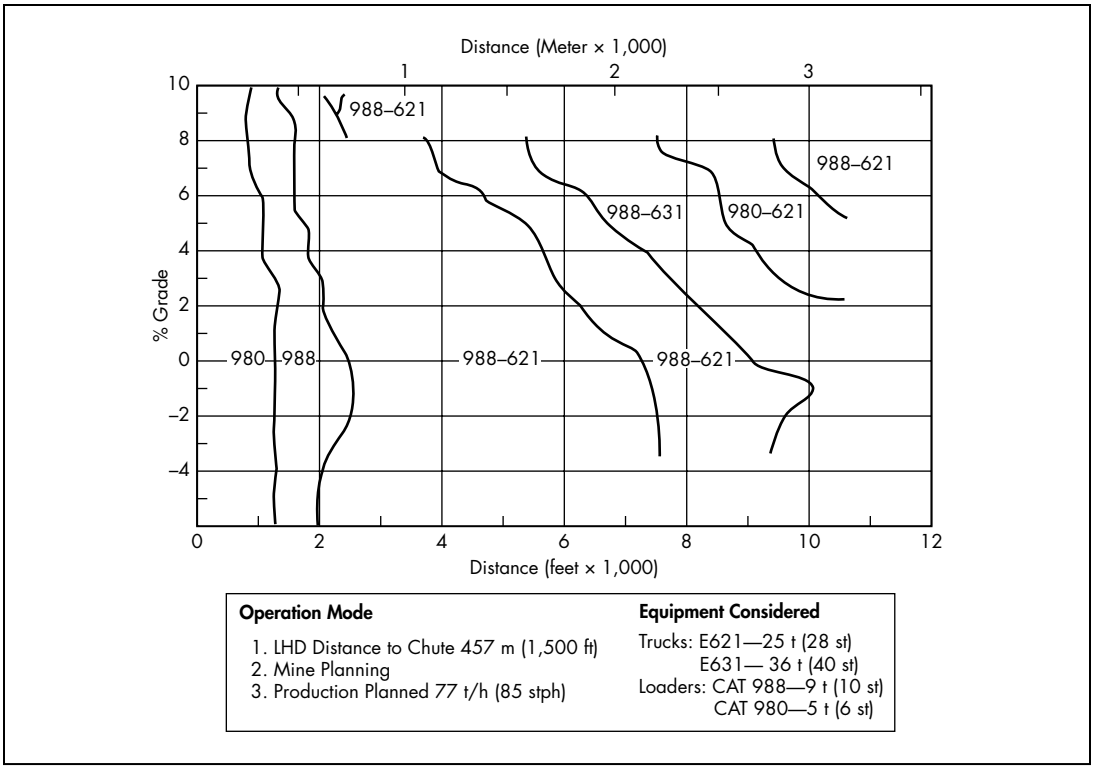
Steep dips of 35% to 40% (~20°) are best mined in a series of parallel slices in steps, each following a level contour of the dipping ore body. As each round is blasted, much of the rock cascades down to the next level, where it can be loaded. This so-called cascade mining method was first reported for the Mufulira copper mine in Zambia (Anon. 1966). However, in this case the pillars were removed almost immediately in a second cycle of mining, leaving the hanging wall to cave as mining retreats along the strike.

Very steep dips of >20° do not allow operation of trackless equipment. Manufacturers have proposed back-mounted, cog-wheel-driven jumbos that can drill stope rounds under these conditions, but the ore so mined does not flow by gravity and must be removed by scraper, conditions being too steep for any other type of loading equipment. Thus many operators over the years have resorted to drilling with handheld jackleg drills and using rope scrapers (slushers) to move ore from steep slopes to the haulage equipment (Christiansen et al. 1959).

R&P MINE HAULAGE DEVELOPMENT

For hard-rock R&P mines, the production shaft is normally developed somewhere near the centroid of the ore body (unless the production opening is by adit or decline, or the ground will be allowed to cave). The objective of production development is to minimize the haul cost for the ore to the shaft. Rail haulage should be kept as straight as possible, with long-haul grades of <2%, although within the rooms themselves grades of up to 5% are permissible. In a trackless haulage operation, however, these numbers may differ and the following apply:

- Keep grades as low as possible and long hauls at <8.0%.
- Keep haul roads as straight as possible. Locate pillars in such a way that roads need not deviate around them. Lay out all main haul roads before mining, and lay pillars out from this plan.
- Keep haul roads in excellent condition. Use adequate crushed stone and keep them well graded and dry. Water



Source: Bullock 1975.

Figure 13.1-3 Equipment and operations planning from a materials-moving simulation program

not only causes potholes, but also lubricates rock to the point that it can cut tires.

In laying out a mine development plan, Spearing's rules of thumb should be kept in mind (Spearing 1995; Bullock and Hustrulid 2001):

- Keep intersections off of the main drift six widths apart.
- Avoid acute-angle turnouts that create sharp bullnose pillars.
- Separate near-parallel ramps and declines from the main drift by at least three times the diagonal width of the main drift.

In planning a trackless/rubber-tire haulage system for an R&P mine, consider the various methods of moving rock from the working face to the crushing/hoisting facility. The versatility and flexibility of modern hydraulic excavators and powerful rubber-tired equipment can make it difficult to choose the optimal method. The following guidelines can help:

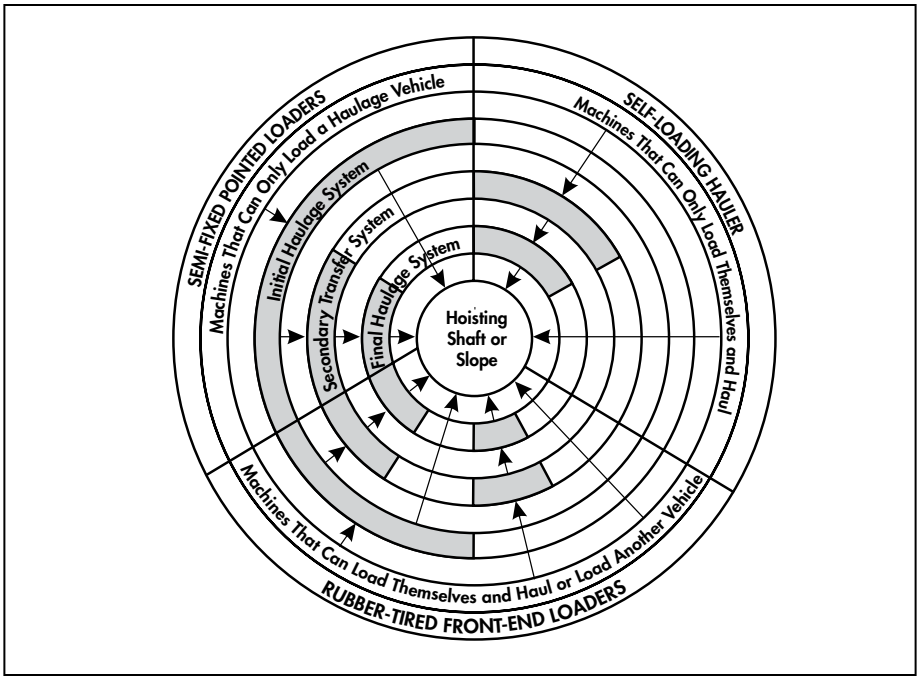
1. When the mine starts operation from the bottom of a decline or shaft, the haul distance is short. Unless hauling is up a decline to the surface, a rubber-tired load-haul-dump (LHD) unit is probably optimal.
2. As the mine works out from the dump point, the LHD unit should eventually start loading a truck. Just one truck suffices as long as the loader can load the truck, then load itself and follow the truck to the dump point by the so-called load-and-follow (LAF) method.
3. As the distance between the mine and the dump point continues to increase, eventually the LHD unit should stay in the heading and load multiple trucks, enough to keep itself busy. At this point, the LHD unit is no longer optimal, and a front-end loader (FEL) may be required. As distance or haulage grade increases, larger trucks and FELs may be also required.

Examples exist (Bullock 1975) where true wheeled loaders have been successfully modified and adapted for both LHD and FEL operation, and perform efficiently at all three stages.

Figure 13.1-3 shows a graph generated by a materials-moving simulation program used in the late 1970s in the Viburnum Trend lead/zinc/copper mines. The program optimizes equipment selection based on least cost. At that time, the equipment manufacturer charged less capital cost per ton of capacity for the 25.4-t (28-st) truck than for the 36.2-t (40-st) truck. When it took this into account, the simulation found it cheaper to run a fleet of smaller trucks at the greater distance and grade. This is not the normal expected result, and thus the simulation proved its effectiveness.

If a mining zone has more than one level and the main haulage is on the bottom level, then various combinations of LHD units to the orepass or FELs with trucks hauling to the orepass are possible, and at the bottom of the orepass, automatic truck-loading feeders can transfer the ore to trucks or rail-mounted trains. If the upper ore body is small and an automatic ore chute cannot be justified, then the ore can be allowed to fall to the ground and an LHD unit or FEL can load it into another truck. Figure 13.1-4 shows all of the possibilities for moving material from an R&P face to the final ore pocket at the shaft. For a complete explanation of how to optimize these variable conditions, see Bullock 1975 or Bullock 1998b.

Clearly, then, in an expanding R&P mine, the optimal method or combination of methods for moving ore constantly changes. But for every condition and distance there



Source: Bullock and Hustrulid 2001.

Figure 13.1-4 Multiple methods of moving ore from a stope face to a hoisting conveyance

is one optimal least-cost method, which can be determined by computer simulation of the underground mine environment (Bullock 1975; Gignac 1978).

Another R&P concept is worth noting. In some older Tennessee mines, the haulage ways along which ore is transported are somewhat narrow. As haul distance increased, it became desirable to use larger-haulage trucks, but such trucks could not be accommodated on the narrow haulage ways. To solve the problem, Savage Zinc, in their Gordonville mine, hooked side-dump, powered truck-trailers (built by Gulf Transport of Australia) together into a mini-truck train, thus increasing the payload per trip and optimizing production for load-haul conditions.

If a conveyor system of haulage is used for the main distance haulage, then the preceding advice for mine layout still applies to the general layout of the mine, except conveyance starts from the decline of the mine entrance where the main conveyor carries mine production to the surface. In these cases, a hard-rock mine usually has a semimoveable crusher or breaker underground before the feeder to the conveyor system. The system of haulage, either LHD unit or FEL/truck, hauls broken rock from the faces to the central receiving point at the crusher. The crusher then feeds the conveyor. The crusher must be moved periodically—about 6 to 12 months—depending on how fast mine production moves the faces away from the central hauling point.

At one time, rail haulage was the principal method of gathering ore from the faces to the production shafts. In the Old Lead Belt of Missouri, more than 556 km (300 mi) of interconnected railroad was used to bring ore into two main shafts from what was originally about 15 mines. Today in the United States, very few R&P metal mines use rail haulage, but many coal mines still do, and in other countries many metal mines still do as well.

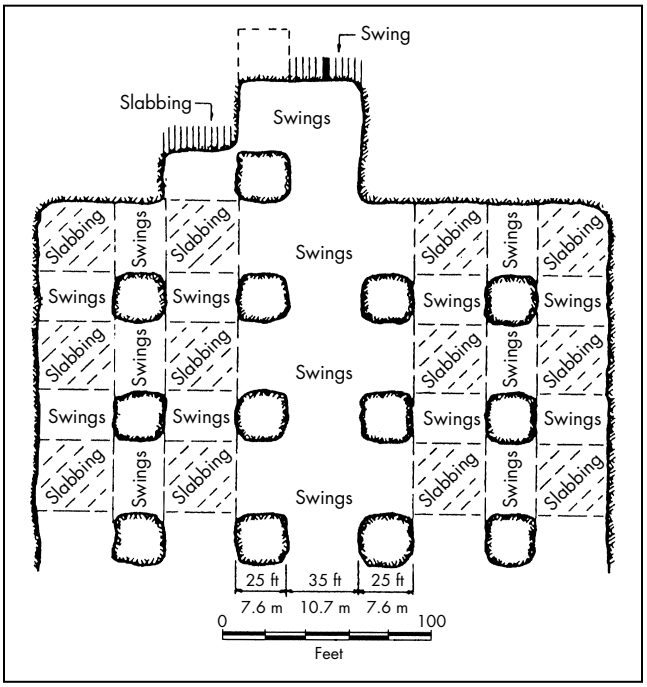
SINGLE-PASS OR MULTIPLE-PASS EXTRACTION

Two approaches exist to mining ore body or valuable rock: (1) taking the entire ore-body thickness in a single pass or slice, or (2) removing the ore body by multiple passes or slices. The choice of approach of course is related to the overall thickness of the ore body to be extracted and to concerns for safety and efficiency.

Normally, in mining bedded deposits for aggregate, the total thickness of the desired horizon is known and the decision concerning slice thickness can be made in advance. However, in metal ore deposits, particularly in Mississippi Valley or collapsed Breccia type deposits, most of the time the thickness of the total mining horizon is not known except where the formation has been penetrated by diamond drill holes and identified, although a few meters away from the hole the formation may very well be different. For both aggregate producer and metal-ore producer, the best approach is first to mine what is thought to be the top slice through the ore body. The thickness of this first mining step depends on the available equipment and on the height of ground that can be mined and maintained safely and efficiently. The advantage of first mining what is thought to be the top slice is that whatever back and rib/pillar scaling and reinforcement is required can be reached easily and safely, probably at distances not more than 6.1 m (20 ft). However, some aggregate producers mine 12.2 m (40 ft) or more in one pass by using high-mass jumbo and extendable-boom roof-scaling equipment.

R&P Rock Breaking for Single-Pass Stoping

In most hard-rock R&P mines, extraction is done by drilling and blasting the face. The initial pass of drilling and blasting is usually done by drilling either burn-cut or V-cut drill patterns. However, only about 40% of the rock should be broken with drilled swing patterns (rounds), breaking to only one free face.



Source: Bullock and Hustrulid 2001.

Figure 13.1-5 Optimizing the drilling and blasting of R&P stoping using 40% pattern swing drilling and 60% slabbing

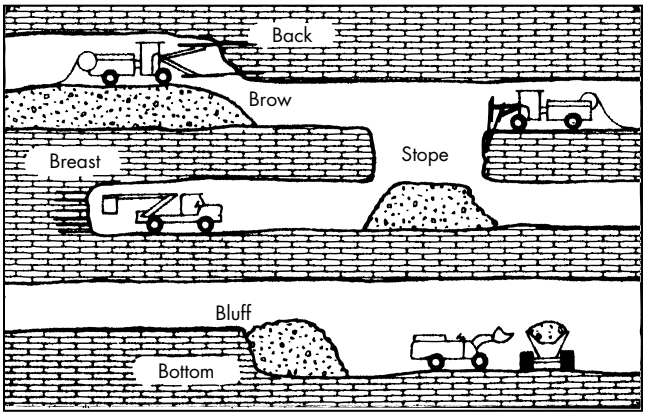
The other 60% of the rock should be broken by slabbing—that is, by drilling holes parallel to the second free face as it is exposed (Figure 13.1-5), which minimizes the cost per ton of rock broken and maximizes productivity.

Particularly in the R&P mining of aggregates and metals, extraction is often done by mechanical means. In many trona and potash mines, and some salt mines, all extraction is done mechanically.

R&P Rock Breaking for Multiple-Pass Stoping

After the first pass is completed in a metal mine for a given stoping area, the back and floor should be sludge-sampled with either pneumatic or all-hydraulic drills. Some operations do this with a stope rock-breaking jumbo and some with an all-hydraulic jumbo designed for this role. The purpose of this sludge-sampling prospecting is to determine what ore remaining in the back and floor needs to be removed by additional slices. If ore is found in the back and floor, then the ore in the back should be removed first. After it is removed and additional back-prospecting reveals no more ore, the back should again be made secure and safe, and ore in the floor can be taken.

Methods of mining ore in the back vary depending on the thickness of the ore yet to be mined and the height of the original stope. If the original stope height was no more than 7.6 m (25 ft) and the back slice thickness was no more than ~2 m (7 ft), most extendable-boom jumbos can reach high enough to drill the brow with breast (horizontal) drilling. Drilling horizontal holes increases the chance of leaving a smooth back, which requires less maintenance than when holes break to the free face of the brow. This is especially true for smooth-wall blasting or flat-seam deposits. A jumbo feed should not be tilted up for drilling near-vertical holes and for drilling uppers



Source: Adapted from Bullock and Hustrulid 2001.

Figure 13.1-6 Various types of stoping action involved in R&P metal mines

to break a brow, particularly in bedded deposits that exist in so many R&P mines.

If ore thickness in the back is greater than the conditions cited previously or if room height is already at the maximum that can be safely maintained, then the back slice should be mined by first cutting a slot in the back at the edge of the entrance to the stope (Figure 13.1-6). This slot should be between the pillars and reach the height of the next slice or the top of the ore body. Care must be taken not to damage the rock that will form the top of the new pillars.

Smooth-wall blasting can be used to advantage in this area. A mine dozer (usually a small dozer of D-4 or D-6 size) begins by pushing rock up and making a roadway for the jumbo to travel on the top of the rock pile. If a dozer is unavailable, an FEL can also do a reasonable job of building the roadway. The jumbo then drills breast (horizontal) holes in the brow and follows the ore zone wherever it goes throughout the stoped area. Even if it goes into the solid beyond the original stoped area, this is not a problem, except that a loader now must load the ore as it is broken. About 75% of the ore must remain in the stope rock pile until back-mining is complete, which can be a disadvantage if the mine must produce immediately. When the top of the ore is finally reached, the back should be made safe since miners are still working close to the back. This type of mining has been practiced in the Tennessee zinc and Viburnum Trend lead/zinc districts. It is not uncommon for several passes of ore to be mined from broken rock piles. In these cases, either ore must be loaded from the bottom edges of the rock piles to make room for new broken rock or a loader must go up on the rock pile and load out the excess rock.

If after the first slice is taken through the stope it is then discovered that a thick, continuous ore zone lies above this first slice—a zone of perhaps 15 to 20 m (50 to 65 ft) or more—then an entirely different approach is required. It may be preferable to create a development ramp to what is now the top of the ore, mine the top slice from this new ramp, and then put through a slot raise from the bottom to the top level that can be slabbed out to make room for long-hole drilling and blasting the ore body down to the level below. Again, the pillars should be presplit or smooth-wall-blasted to protect them. If the eventual height of a pillar can be anticipated, the size of the pillar at its base should be determined taking this

additional height into account. If it is not taken into account and safety factors in the design are insufficient, then the pillars may need reinforcement with grouted rebar bolts to give them the required strength and stiffness.

When all back ore is removed from the stope and the final back is made completely secure, bottom ore can be removed from the floor. This is best done by first cutting a ditch or short decline in the floor, at the entrance to the stope, to the depth of the desired bluff or bench. Bluffs can be carried very thick and are limited only by the height required for the loading equipment to work safely. It is common practice to carry bluffs up to 9 m (30 ft) in the lead mines of the Viburnum Trend area using 7–11-t (8–12-st) loaders. Beyond this height, the safety of the loader operator may become an issue.

Bluffs at least 4 m (13 ft) thick are usually drilled and blasted with downholes by small-surface quarry-type drills. Bluffs of less than this thickness can be drilled out with face jumbos, drilling breasting, or horizontal holes (sometimes called lifters and splitters). These procedures can be repeated over and over until the bottom of the ore is reached.

When it is known beforehand that bluff mining will be done, for the initial pass the pillar width must be left large enough to accommodate considerable height should it be needed. If additional ore is not found, then the pillars can be slabbed down to smaller size. However, again the precaution: in both removal of back ore and taking up of the bottom, the pillars most be protected with presplitting or smooth-wall blasting. Pillar design is discussed in more detail in the following section; the point here is that, unless the original mine plan specifically allows for taking down the back and taking up the floor or specifies a very large safety factor in pillar design, the width-to-height ratio required for safe operation will be exceeded by multiple-pass mining. It cannot be overemphasized that everything must be done to protect pillar integrity during the first of what may become multiple passes. Unless caution during presplitting or smooth-wall blasting is observed, blasting fractures may extend into the pillar as far as 2 m (6.6 ft) or more. As more slices are removed, these fractures begin to open, reducing pillar size and subjecting the pillar to failure.

METHODS OF PILLAR EXTRACTION IN HARD-ROCK MINES

The mining engineer should anticipate some percentage of pillar removal beyond what is cited in the original mine plan and stope layouts. If future pillar extraction is planned—whether by partial slabbing, removal of a few high-grade pillars, or complete removal incorporating some system of backfilling—then what is left after the first pass greatly influences what can be done in the future. Too often, too much is extracted on the first pass, so that the entire area is weakened and must be reinforced just to keep the mine open, much less enabling the selective robbing of pillars.

The first and most important thing to do is to install a complete network of convergence stations throughout the affected area. Experience shows that regular monitoring of such a network over time can reveal what rate of back–floor convergence is acceptable and what rate will lead to massive failure. Convergence in some mines can be as high as 1.25 mm per month without the back breaking up and failing (Parker 1973). Convergence in other mines, such as those of the Old Lead Belt of Missouri and the Viburnum Trend, can be as low as 0.1 mm per month and still cause problems. However, the following are good general guidelines:

- Convergence of 0.0254 mm (0.001 in.) per month is not significant.
- Convergence of 0.0762 mm (0.003 in.) per month indicates a serious problem, but is controllable with immediate action.
- Convergence of 0.1778 mm (0.007 in.) per month indicates that acceleration is getting out of control and the area may be lost.

Several methods exist for pillar removal of R&P stoping:

- Slab some ore off of each pillar containing the high-grade portion of the ore during retreat from the area.
- Completely remove a few of the most valuable pillars, but leave enough pillars untouched to support the back.
- In narrow stopes, completely remove all pillars in a controlled retreat.
- Use massive backfill methods to remove some or all pillars.

Placing cemented rock fill around pillars, all the way to the back, to properly support the stope between the solid and the backfill creates a pressure arch over the intervening pillars, which can then be removed. Economics permitting, the area can be backfilled if necessary, and the pillars that were encapsulated or trapped in the original fill can be mined from a sublevel beneath the pillar by long-hole blasting of the pillar into the sublevel area. (Lane et al. 1999) The total backfill prevents any future subsidence.

All of the previous methods (except sublevel extraction) were used in the final mining of the Old Lead Belt of Missouri in its last 25 years, and much more intensely in its last 10 years. This R&P mining district was mined for 110 years before finally shutting down.

Sometimes when pillars are partially removed without proper planning, or when pillars that are too small are left after first-pass mining, the pillars begin to break up and serious convergence accelerates uncontrollably. One of two things must be done to save the area: (1) massive pillar reinforcement, if there is time before collapse; or (2) massive backfill in the entire area.

Among examples of massive pillar reinforcement, fully grouted rebars were placed in more than 300 pillars in the one of the R&P mines of the Viburnum Trend. Weakly (1982) has documented the method employed, reinforcing pattern, cement grout mixture, convergence instrumentation, and results. In the Old Lead Belt areas, pillars to be reinforced were wrapped with used hoist cable, with a load of 5.5 t (6 st) placed on each wrap (Wycoff 1950), but this reinforcement method is not as fast, economical, or effective as use of fully grouted rebars in the pillars.

Among examples of massive backfill are the Leadwood mines of the Old Lead Belt, where this method was used only about 132 m (425 ft) below surface. However, the back was thin-bedded dolomite, interbedded with shale and glauconite, and badly fractured and leached. The roof bolt was originally developed in these St. Joe mines in the late 1920s as a means of tying the layers together to act as a beam (Weigel 1943; Casteel 1964); the roof bolts were used with channel irons to form a crude truss. Even though the rock in the pillars provided poor support because it contained bands of

high-grade galena, for economic reasons pillar removal and slabbing took place over a period of 25 years. Occasionally local cave-ins would occur after an area had been pillared, but since the cave-ins were beneath uninhabited St. Joe–owned land, they were of no real concern. However, when slabbed pillars between two smaller cave-ins began to fail and a third and fourth cave-in occurred in more critical areas, there was a considerable amount of concern. Initial extraction of some of the area involved multiple-pass mining and room heights were mostly 6.1 to 12.2 m (20 to 40 ft). Final mining of the area had resulted in ore extraction of approximately 95%. To stop caving in the third and fourth areas, in around 1962 more than a million tons of uncemented, cycloned sand tailings were put into the mines, filling the rooms nearly to the back. The backfill was very successful in controlling the converging, failing ground.

The end result of use of these methods is comparable to the overhand-cut-and-fill practice of deliberately mining pillars very small and immediately filling in around them. This method, known as postpillar mining, was used in the Falconbridge nickel mines (Canada) (Cleland and Singh 1973), the Elliot Lake uranium mines (Canada) (Hedley and Grant 1972), and in some mines in San Martin and Niaca (Mexico) and San Vicente (Peru). The end results for the two mining methods, with very small pillars encapsulated in sand tailings, are similar.

VENTILATION

Most of the literature on ventilation design for R&P mining is written for coal mines. However, a few considerations are unique to metal and aggregate R&P mines. R&P metal or aggregate mines differ from R&P coal mines in the following respects:

- Everything is larger. It is not uncommon to have entry drifts of 9 × 9 m (30 × 30 ft) and rooms stoped out to 12 × 15 m (40 × 50 ft). A lot more air or a stream of high-pressure directed air is required to meet the required minimum velocity across the working face.
- Stopings are difficult to build and the total force against them can be enormous. Many R&P mines rely on auxiliary fans to pick up air from the main ventilation drifts and carry enough air through vent tubing to serve the needs of the active face.
- Ventilation doors are similar in design to airplane hangar doors. Again, the total force against these doors, which must be controlled automatically, is enormous.
- Air stratification in large stopes can be a problem.
- Diesel equipment is used extensively, and the exhaust must be diluted to meet particulate standards.
- Ventilation fans can be placed underground, if it is beneficial to do so.

THE EFFECT OF CHANGING MARKET CONDITIONS ON R&P MINE PLANNING

Market conditions are important in planning for R&P mining, even during the stages of feasibility planning. An advantage of R&P mining for metal or any commodity that is gradational in value is that mining methods and approaches are flexible and can be changed quickly in response to changing market conditions—more quickly, in fact, than is possible for other mining systems.

Another advantage of R&P mining is that, if the ore body is continuous, new faces are continuously being opened. A stope only four pillars wide can have as many as 15 faces open simultaneously for drilling and blasting. Imagine how many faces would be exposed to drilling and blasting if a stope were 10 pillars wide. For metal mines, this offers a great deal of flexibility; one can mine the grade of ore that is most desirable at the current market price. For at least short periods of time in each stope unit, it is usually possible to work only the higher-grade or lower-grade faces of a particular mineral, although doing so usually has a drastic effect on the grade within a few days. During initial mining of the high-grade Fletcher mine in southeastern Missouri (United States), after operation for only 3 to 4 years, approximately 50 to 70 faces were open for mining on any given day although only 10 to 12 were actually worked. One can then select the faces to be mined so as to maintain the grade of lead, zinc, or copper ore that can best be handled by the concentrator and still optimize the financial objective of the mine.

Spare equipment can be put into reserve stopes to increase production if the remaining materials flow can handle the added capacity. However, if this is done too often or for too long, mine development must also be accelerated. Old stopes, if maintained, can be reactivated quickly for mining of lower-grade minerals that have become financially worth mining.

Lower-grade ore can be left in the floor or the back of stopes until price rises, at which point the reserves are readily available for quick mining. This technique is often overlooked by those unaccustomed to planning R&P metal mining, where mineral values are gradational. The optimal technique is first to mine the better areas of the mineral reserve and maintain a grade of ore that satisfies the economic objectives at that time. Then when economics change, the lower-grade areas left as remnant ore reserves can be mined.

In spite of these considerations, however, even in R&P mining, drastic changes in the rate of mining (momentum) cannot be assumed to be free. It often takes several months with an increased labor force to regain a production level that seemed easy to maintain before a cutback in mine production. If spare equipment is used to increase production, maintenance will probably convert from a previous preventive-maintenance on-shift schedule to a breakdown overtime schedule, at least until permanent additional equipment can be obtained. Nevertheless, the necessary changes can be made.

As discussed in the section on pillar robbing, another method for achieving economic flexibility with R&P mining is to slab or remove high-grade pillars. Then, even in the later years of the mining operation, some "sweetener" is left to blend with the lower grade. Although not unique to R&P mining, this technique is easier to accomplish in an R&P operation than in other more complex mining systems.

CAPITAL AND OPERATING COSTS FOR HARD-ROCK R&P MINING

Capital and operating costs for hard-rock R&P mining are usually considerably less than for other hard-rock mining systems at the same daily production. However, panel caving is not comparable because R&P mines are not normally built to produce the equivalent lower tonnage per day.

Table 13.1-2 shows estimated preliminary capital costs for R&P mines. The original estimates (Stebbins and Schumacher 2001), based on 2000 data, included cost contingencies of 7.2%

Table 13.1-2 Estimated preliminary capital costs for R&P mines developed from shafts (mid-2009 dollars)

Daily tonnage, t (st)	1,200 (1,323)	8,000 (8,819)	14,000 (15,432)
Total capital cost	$39.8 M	$125.0 M	$152.8 M
Total capital cost/daily tonnage	$33,200	$15,600	$10,900

Source: Adapted from Stebbins and Schumacher 2001.

Table 13.1-3 Estimated direct operating costs for hard-rock R&P mining (2008 dollars)

	Cost, US$	
Activity	**Per metric ton**	**Per short ton**
Drill & blast labor	0.47	0.43
Loading & hauling	0.73	0.67
Maintenance labor, repair, parts, & tools	1.84	1.67
Indirect labor	1.14	1.04
Overtime	0.15	0.14
Other materials & tires	0.64	0.58
Sundry supplies	0.49	0.44
Diesel fuel	1.60	1.45
Drill steel & bits	0.26	0.24
Grease & oil	0.76	0.69
Roof supplies	0.13	0.12
Tire recapping	0.18	0.16
Other outside services	1.10	1.00
Blasting supplies	1.11	1.00
Underground prospecting	0.38	0.35
Fringes on direct labor	0.55	0.50
Total direct operating cost	11.53	10.46

to 7.6%. The contingencies were removed, the percentage of the remaining total was calculated for each element of capital cost, elemental capital costs were escalated using the proper indexes for the time period and totaled, and a 25% contingency was added to the total. These costs include all equipment, preproduction development, mine-surface facilities, engineering and project management, working capital, and cost contingencies.

Table 13.1-3 shows estimated direct operating costs for R&P mines. The estimates are based on 1998 data from six highly productive R&P mines that vary in daily production from about 2,300 to 6,300 t (2,500 to 7,000 st) per day. The estimates were escalated to 2008 using the proper mining-unit inflation index for each cost element, and a 20% contingency was added. It is noteworthy that the six mines have very large rooms, are highly mechanized, and are considerably more productive in tons per worker-shift than is average for the industry (the latter is about 40–50 t [44–55 st] per worker-shift).

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CHAPTER 13.2

Room-and-Pillar Mining in Coal

Jerry C. Tien

INTRODUCTION

After the coal seam is reached by means of a slope or shaft, the room-and-pillar (R&P) mining method can be used to extract coal. Room-and-pillar (also called bord-and-pillar) mining is an "unsupported" mining method used to extract mineral deposits that are roughly tabular and generally flat-lying such as coal. Using this system, the coal seam is extracted across a horizontal or nearly horizontal plane while leaving "pillars" of unmined coal to support the overburden and leaving open areas or "rooms" underground.

This method is termed *unsupported* because no artificial pillars are used to support openings. The roof is supported by natural pillars left standing—usually in a systematic pattern—and generous amounts of roof bolting and localized support are also usually used. The key to successful R&P mining is selecting the optimum pillar size, which varies depending on many factors, varying between 40% and 60% of the seam volume and proper pillar layout. With proper planning, this method provides a very effective means for extracting coal.

The productivity of R&P mining operations experienced a distinct step-change in the latter part of the 20th century as a result of increasingly powerful mining equipment, technical progress, and rationalization of efforts, and is now an indispensable method in underground coal mining. Worldwide today, it is estimated that at least 60% of coal mined underground uses the R&P mining method, although the exact amount of coal mined using the R&P mining method is difficult to obtain—the 60% figure is estimated by examining major coal-mining countries. For example, the world's number one coal-producing country, China, has at least 85% of its coal mined using R&P mining; more than 95% of China's coal is mined underground. More than 60% of U.S. coal mined underground comes from the R&P method (including 10% coal from longwall operations). It is estimated that less than half of Australian underground coal is mined using the R&P method. South Africa produced 43% of that country's total coal production using R&P mining in 2008. It is also estimated that at least 60% of India's coal comes from R&P mining.

In the mid-1980s, R&P mining using either drilling and blasting or a continuous miner was the dominant method in underground coal mining, producing about 87% of all the coal mined underground in the United States. Following the growth of longwall methods, this dropped to approximately 60% in 2007 (including the coal from longwall gate road development). Even longwall mining operations require highly efficient R&P units for timely entry development, producing an estimated 20% of total coal production. It probably will be difficult for R&P to compete costwise with longwall operations in all circumstances, but with its modest capital requirement and versatile operation, the R&P method has been able to compete head to head in terms of cost per ton of coal extracted with longwall mining in many cases.

ROOM-AND-PILLAR MINING

Early mines using the R&P method were developed more or less at random, with pillar sizes determined empirically and headings driven in whatever direction was convenient. For example, in Australia in the early days, the rooms were laid out systematically to suit hand shoveling and endless rope haulage. In modern coal mines, however, rooms and pillars are typically laid out orthogonally in a regulated pattern (Figure 13.2-1).

In addition to support, coal pillars are also expected to restrict surface movement, protect critical airways from the effect of high load distribution and abutment stresses, form ventilation partitions for airflow distribution, act as yield (sacrificial) pillars to provide temporary support in high lateral-stress situations, and as shaft pillars to protect air shafts and other critical mine structures. Barrier pillars can be used to form a load-bearing framework to minimize the impact of ground movement from adjacent workings, to isolate major faults and local joints, to protect critical roadways, to separate adjacent mine leases, and most importantly, to provide a barrier to large masses of water-filled old workings.

R&P mining has two basic operations: entry development and coal production. The former is to develop mine entries to access coal seams and construct the necessary support infrastructure to facilitate coal production. Although in inclined coal seams it might be necessary to excavate either above or below strata to access the coal seam, most entries are in the

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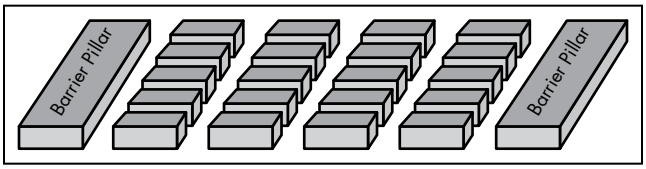


Figure 13.2-1 Room-and-pillar layout with five entries and long barrier pillars

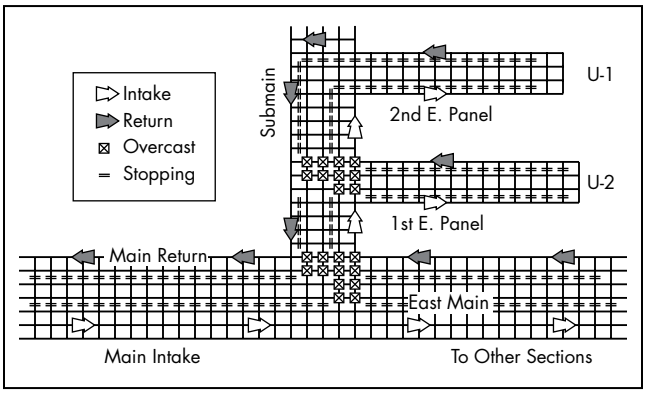


Figure 13.2-2 Mine section with two operating units, seven-entry main, five-entry submains, and four-entry panels

coal seam itself, which also produces coal in the process. Both horizontal and inclined R&P operations require other support operations such as roof bolting, ventilation, face cleaning, and transportation.

The mine layout has to be carefully planned to accommodate all functions for maximum efficiency. Development openings (entries) and production entries (rooms) closely resemble each other—both are driven parallel and in multiples, and connected by crosscuts to form pillars. Driving multiple entries simultaneously is necessary for high production and operational efficiency; it also facilitates ventilation, roof bolting, and coal transportation routes in the face area.

Usually, a hierarchy of entries exists in underground coal mines. Main entries are driven mainly to divide the property into major areas and usually serve the life of the mine for ventilation as well as for worker and coal transport. Submain entries can be regarded as feeders from the mains that subdivide each major area. From the submains, panel entries take off to subdivide a block of coal into panels and rooms for orderly coal extraction. It can be seen in Figure 13.2-2 that the East Main has a 3-2-2 or intake-neutral-return arrangement, and the submain off the East Main has the same arrangement (intake-neutral-return) but in 2-2-1 layout. The number of intake entries exceeds the number of return entries indicating it is a blowing system (with a fan on top of an intake air shaft) where neutral air travels away from working faces.

The number of development entries in any area can vary between three and seven, or as many as eleven in some cases; the exact number is often a balance of the following factors:

- Expected load on the pillar due to its depth or other geotechnical concerns
- Coal production
- The system that supports the expected production (number of pieces of mining equipment that are operating, labor and material transport, ventilation, etc.)

It is critical that a proper balance be achieved between the optimum coal extraction ratio and the size of pillar necessary to support the overburden.

Geotechnical Design for R&P Mining

Because of the complicated and varying nature of rock strata, it is customary to start the basic pillar design by assuming (1) rock strata to be homogeneous, uniform, and horizontally bedded deposits; (2) a uniform pillar layout; and (3) the rock materials are competent. Other factors such as pillar shape (uniformity of distribution and cross section and height–width ratio), true pillar geometry (outside dimensions minus an outer shell of distressed materials that in effect support very little load), and true mechanical behavior of pillar materials also have to be considered.

Bieniawski (1987) suggested the following eight-point general pillar design approach:

1. Data collection to (a) obtain a uniaxial compressive strength of roof rock and coal (σ_c); (b) spacing, condition, and orientation of geological discontinuities; and (c) groundwater condition.
2. Determine roof span and support methods.
3. Estimate pillar strength (σ_p) by considering size effect using

   $$\sigma_p = \sigma_1 [0.64 + 0.36 (w/h)]$$

   where σ_1 is the strength of a cubical specimen of critical size or greater (e.g., about 3.3 ft for coal), w is pillar width, and h is pillar height.
4. To obtain pillar load using

   $$S_p = 1.1H\left(\frac{w + l}{l}\right)^2$$

   where S_p is pillar load (psi), H is the depth below surface (ft), w is pillar width (ft), and l is the roof span (ft).
5. Select a proper safety factor, F (ranges between 1.5 and 2), $\sigma_p = FSp$, and solve for pillar width.
6. Determine extraction ratio (ER) using

   $$\text{ER} = \frac{(w + l)^2 - w^2}{(w + l)^2} = 1 - \left[\frac{w}{w + l}\right]^2$$

   If the ER is not acceptable and needs to be increased by decreasing the pillar size, select a new width and repeat Steps 3 through 6. Figure 13.2-3 shows that a 50 × 50 ft pillar with 20-ft width will result in nearly 50% ER.

   The optimum ER must be a compromise between the safety of the structure and the cost of mining and conservation of resources. Considering all factors, the extraction ratio for R&P mining ranges from about 50% to 60% for underground coal mining, as shown in Figure 13.2-3.
7. Cross-check results by using the Holland formula ($\sigma_p = \sigma_1[0.778 + 0.222w/h]$) and the Obert–Duvall formula ($\sigma_p = \sigma_1\sqrt{w/h}$). The Holland formula is valid for pillar width-to-height ratio and the recommended safety factor is between 1.8 and 2 (Figure 13.2-4).
8. Exercise engineering judgment by considering a range of mining and geological parameters to assess the various options for mine planning in accordance with the local mining requirements.

For example, a coal mine in the Pittsburgh seam (United States) is currently mining at 500-ft deep with stable pillars and roofs of w = 40 ft, h = 10 ft, l = 18 ft, and k = 5,580 psi

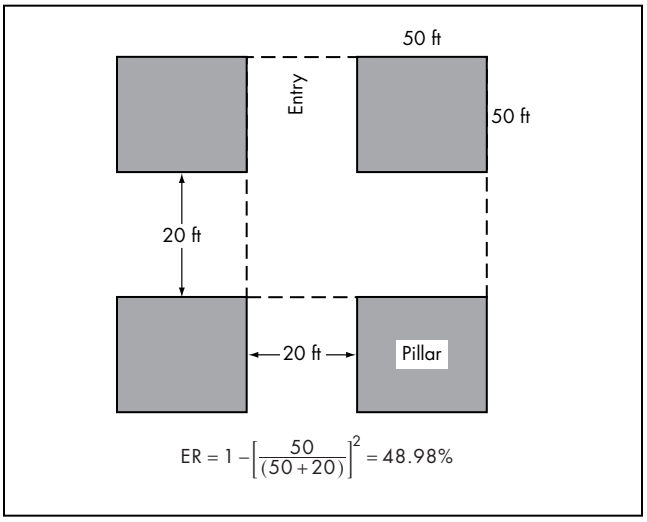
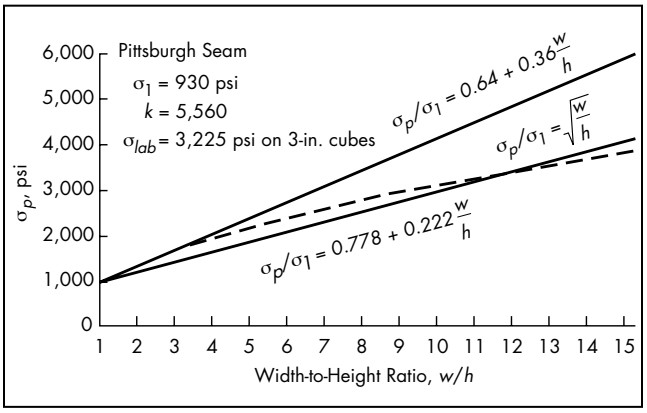
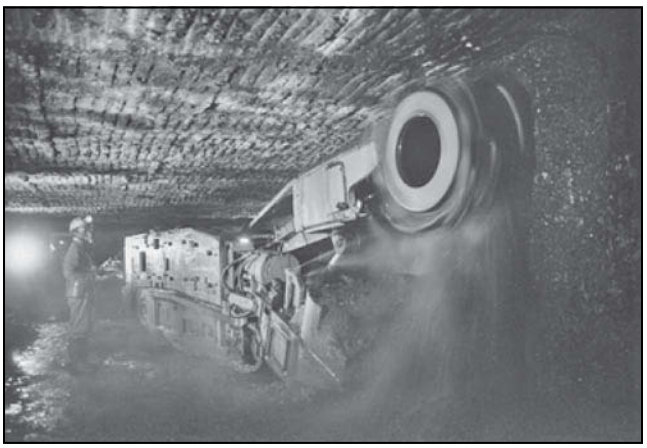


Figure 13.2-3 Extraction ratio calculation for a 50 × 50 ft pillar with a 20-ft-wide entry



Source: Bieniawski 1987.

Figure 13.2-4 Pillar strength in the Pittsburgh seam versus *w/h* ratio as predicted by three formulas



Courtesy of Joy Mining Machinery.

Figure 13.2-5 Continuous miner at work

for the Pittsburgh seam. K is a constant for actual pillar material, defined by $k = \sigma_c\sqrt{D}$, where σ_c is the uniaxial comprehensive strength of rock specimens tested in the laboratory having a diameter or cube size dimension of D (in.). Mining is planned at 1,000 ft. Calculation shows that pillar stress (S_p) = 1,150 psi. For a 1,000-ft depth, using proper pillar strength (σ_p = 1,934 psi), safety factor (F = 1.67), and extraction ratio (0.52), the resulting pillar width will be 65 ft.

After coal is mined, it is hauled away using shuttle or ram cars to a properly located breaker station (usually 200–250 ft away from the working face) where it is crushed and transported to the surface by conveyor.

Practical limitations exist on R&P mining in coal operations. That is, at depths greater than 1,300–1,600 ft, this method may become very difficult or uneconomic to practice, owing to excessive roof pressure and the larger pillar sizes that are required. However, depending on other parameters, there are always exceptions. For example, KGHM is mining copper in Poland by the R&P method below 3,300 ft.

Continuous Mining Method

With a continuous mining method, separate unit operations of drilling, cutting, blasting, and loading are replaced by a single high-performance continuous miner (CM). As the steel drum (cutterhead) spins in a circle, the tungsten carbide steel teeth or cutting bits break the coal from the face. The coal falls onto a steel apron where moving arms load the coal onto a short conveyor on the back of the CM and transport it to a shuttle car waiting at the rear (Figure 13.2-5).

The shuttle car is driven to a feeder breaker three to four breaks (180–280 ft depending on pillar sizes) away and loads the panel conveyor, which carries the coal to the surface. After one working face is completed, the CM then moves to the neighboring face and the process is repeated.

A typical CM measures approximately 33 × 11 × 4.5 ft (length × width × height, respectively). The depth of each cut in a working face has increased today to 30–40 ft ("deep cut") because of more powerful and reliable machines, but this requires multiple passes and resetting of the miner to complete cutting in one face. Efficiency of the miner operator and coordination of other auxiliary functions such as face cleaning, establishing ventilation, and roof bolting are all key factors in determining the face advance rate and productivity. Remote-controlled CMs are now the norm in the industry, and they have significantly enhanced face productivity and safety.

A typical (standard) continuous mining section uses one set of equipment with one crew operating in a panel of three to seven entries. The section is equipped with one CM, two to three shuttle cars or ram cars, one roof bolter, and a section scoop for auxiliary jobs. In addition to a section foreman, a CM section would have nine to ten workers: two CM operators (one operating and one assisting/relieving), two roof bolters, three shuttle car operators, one mechanic, and one general laborer).

In addition to the standard unit, U.S. coal mines have also been using either a double or a super unit arrangement: The former uses two sets of equipment and two or nearly two crews operating side by side in a panel with 8–13 entries; the latter uses two sets of equipment and one crew plus two extra workers operating side by side in a panel with 4–13 entries. Depending on the mine's specific condition, either method has been used successfully in developing long-distance mine mains with multiple entries.

Since the late 1970s, mining machines progressed steadily from 440 V to 950 V with an increase in productivity. By 1990 virtually all CMs were powered by 950 V. The demand for larger, more-powerful mining equipment prompted the need for increased voltage even further for a CM. Since input voltage varies directly with mining rate and motor torque varies with the square of the voltage, higher voltages for a given size cable also lead to a lower power voltage drop. With further improvement in shielding technology, high-voltage miners have become available. For example, Joy Mining Machinery's 14 CM series high-voltage (2,300 V) continuous miner can mine up to 38 t/min, depending on the seam thickness. Special design, use, and maintenance precautions are needed to ensure an equivalent level of safety when high-voltage systems are used in permissible areas. In the United States, mining equipment operating above 1,000 VAC requires a petition for modification including an approval examination by the Mining Safety and Health Administration for each high-voltage CM deployed inby (the distance between where one stands and the face). CMs capable of cutting 19.7 ft of coal seam have also been produced.

Ongoing improvement in CM performance in both productivity and reliability has brought about a dramatic increase in coal production. Under certain conditions, CM units have been able to deliver more than 3,000 t/shift on a regular basis.

Conventional Mining (Drilling and Blasting) Method

Conventional mining is cyclical, employing mobile mechanized mining equipment to carry out production unit operations in five distinct steps:

production cycle = cut + drill + blast + load + haul

A cutting machine cuts a 5–7-in. slot or kerf at about 10 ft in length to improve breaking during blasting. A drill is then used to drill holes into the coal face in a predetermined pattern, which are then loaded with permissible explosives. After the coal is shot and fragmented by a qualified shot firer (following strict safe-blasting procedures), it is scooped up and loaded into a waiting shuttle car that transports the coal to a conveyor-belt loading head (feeder breaker) for transport to the surface. In some smaller mines, the coal is hauled in rubber-tired trailers drawn by battery-powered tractors. After the first working area is completed (depending on specific practice, this could be the first heading on the right in a panel if the mine works from right to left direction), the crew moves to the adjacent workplace and the steps repeat, as shown in Figure 13.2-6. Coal mined at the face is transported using a shuttle car or ram car to the breaker where it is transferred to the conveyor belt.

Several other auxiliary functions must also accompany production functions: face cleaning, roof control, proper face ventilation, cables and equipment advances, and movement of materials and supplies. As a result, multiple entries are necessary such that all functions can be performed at separate faces without delays.

Depending on the face advance rate and traveling distance, as well as the change-out points due to multiple hauling units, the location of the dumping point (breaker) needs to be located in a position that minimizes haulage travel time and congestion.

In the United States, haulage entries where coal is transported are separated from both intake and return airways, thus

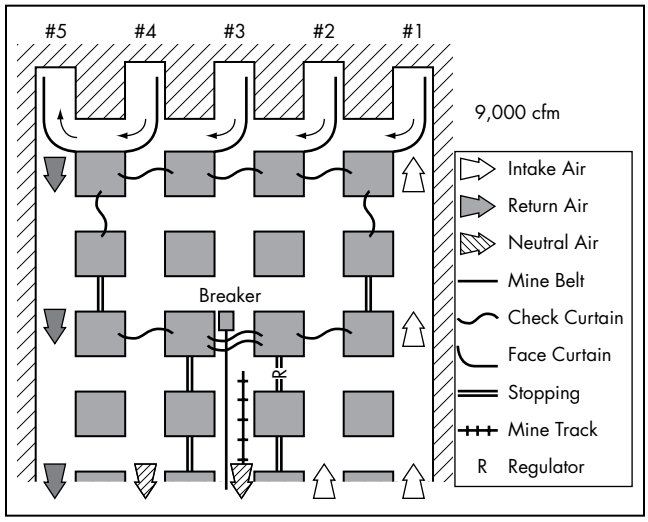


Figure 13.2-6 A five-entry unit with mining progressing from right to left

the introduction of "neutral" entry airways, which should not be used in the face area but be channeled directly into the return airway.

Increased safety concerns, efficiency demand, and the need to operate several pieces of mining equipment simultaneously have caused this conventional mining method to fall out of favor, and it is slowly being replaced by the continuous mining method. In 2008, only 0.6% of all coal mined underground in the United States was mined using this method.

KEY AUXILIARY FUNCTIONS

To provide a highly productive mining environment while maintaining safety, a well-coordinated mining plan that incorporates several key support functions is critical. The top two such functions are effective ground control and proper face ventilation.

Ground Control in Coal Mining

Ground control is concerned with both the short- and long-term stability of mine accesses and entries as well as subsidence control. Although no method of roof control yet devised has proven to be universally acceptable for the wide range of strata conditions, perhaps the most significant development in coal mine ground control during the 20th century was the introduction of roof bolting in the late 1940s and 1950s (Figure 13.2-7). Being able to provide effective and efficient ground control is essential to ensure safety and productivity in underground coal mines.

The theory of the interaction between roof bolts and the rock strata is complex and several models are available: beam theory (roof bolts bind together several weak strata into one stronger "beam"), suspension theory (weak members of the strata are suspended from a strong anchor horizon), and the keying-effect theory (roof bolts act much like the keystone in an arch). All seem to be able to explain at least part of the support phenomenon under certain conditions.

From an engineering standpoint, roof bolts are inherently more effective than the wooden timbers they replaced, they greatly accelerated the transition to trackless, rubber-tired face haulage, and they significantly facilitated coal production in underground coal mining.

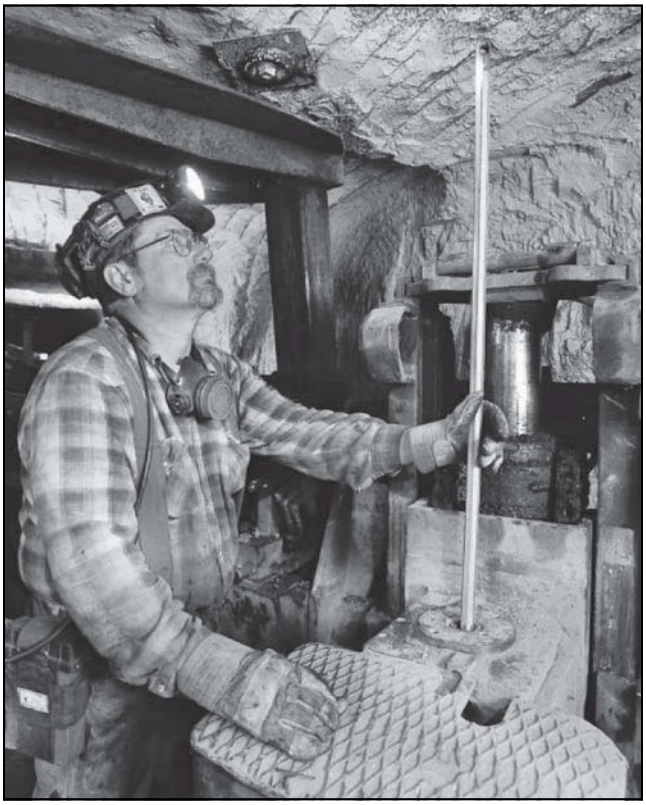


Figure 13.2-7 Roof bolting has become an integral part of the mining cycle

Steel bolts, usually 4–6 ft long and 0.6–1 in. in diameter, are inserted in holes drilled into the roof strata by a roof bolter and are secured as primary roof support by friction or resin. Various types of bolt systems are commercially available (frictional, tensioned, nontensioned, fully-grouted tensioned) and accepted by the United States and world mining industry.

The tensioned bolt system (also referred to as an active system) includes the conventional point-anchored mechanical bolt, a tensioned combination bolt, partially grouted tensioned rebar bolt, mechanically anchored resin-assisted bolt, partially grouted cable bolt, and so on. The nontensioned "passive" bolts include the fully grouted resin bolt and resin-anchored cable bolt. The fully grouted tensioned bolt combines the advantages of both the tensioned and nontensioned bolt. Roof channel, roof mat, roof pan, wire mesh, and shotcrete have also been used in combination with roof bolts for surface restraint. They are used to support small pieces of loose rock or as reinforcement for shotcrete, and the latter to provide support to the rock surface between bolts. Shotcrete provides the added advantages of reducing the rock exposure, thereby minimize weathering effects.

Today, all major mining countries have mandatory safety and ground control standards for underground coal mines. In the United States, specific regulations on roof support standards are set forth in Subpart C, Title 30, Part 75 of the Code of Federal Regulations (CFR 2010). Regulations stipulate that "no person shall work or travel under unsupported roof" underground, and "the roof, face and ribs of areas where persons work or travel shall be supported or otherwise controlled to protect persons from hazards related to falls of the roof, face or ribs and coal or rock bursts; no person shall work or travel under unsupported roof." Additionally, there are detailed regulations dealing with roof bolting, including installations, conventional roof support, pillar recovery, and automated temporary roof-support systems.



Source: Barczak 2005.

Figure 13.2-8 Cribs and steel plates used to reinforce roof support

Additional supporting systems, either temporary or permanent in nature to meet specific geological conditions, are commonly used. Wooden cribs or posts, trusses, cable bolt, steel set, and wire mesh or shotcrete have all been successfully applied in underground coal mines. Wooden cribs (Figure 13.2-8) are prop timbers or ties laid in alternative cross-layers (i.e., log-cabin style) to effectively provide significant reinforcement capacity over a large deflection range. However, crib support systems restrict airflow, generate floor heave, and permit large roof deformations that can lead to failure. In other words, they have to be used carefully.

Roof truss as a means of secondary or supplemental mine roof support has been receiving increasing acceptance in recent years, particularly in bituminous coal mines. In those entries where long life is required or doubtful performance from other forms of support seems likely, trusses in many cases have provided the necessary protection for continuing operations. Steel trusses employ a combination of steel rods anchored into the roof to create zones of compression and provide better support for weak roofs and roofs over wide areas.

The aforementioned roof bolts used to reinforce the roofs of mines, although effective, are relatively expensive and by definition inflexible. Cable bolts use strand cable coupled with properly designed couplings. These are not only more economical, they provide relative flexibility as compared to the steel bar. This support system has been recognized as an effective ground control system in underground coal mines.

Steel sets are also commonly used as long-term support of underground openings such as inclined mine entries, main travelways, belt entries, and so on. The operator generally connects the steel set with tie rods, inserts wood blocks between the arch sets as lagging, and backfills the voids between the

steel set and roof. Under some extreme geological conditions, yieldable arches are used to provide effective ground support in areas of excessive ground movement or faulted or fractured rock. They are designed such that when the ground load exceeds the design load of the arch as installed, yielding takes place in the joints of the arch permitting the load to settle into a natural arch of its own and thus bring all forces into equilibrium. The arch is often stronger after yielding than before due to increased joint overlap.

Ventilation

Proper ventilation is critical in any underground coal operation because of the presence of methane and coal dust as well as diesel particulate matters from diesel-powered equipment. Control of these hazardous pollutants has become increasingly challenging because of increasing mine sizes, coal production, and the use of diesel-powered mining equipment in recent years.

Engineering control techniques commonly used for ventilation purposes range from simple dilution with sufficient air quantity, water sprays and water additives, and local dust collectors to complex drainage systems to remove the gases prior to mining in many gassy mines. Nevertheless, dilution remains the most common method used. In general, the reduction in pollutants is roughly proportional to the increase in airflow; therefore, providing sufficient airflow underground remains the first priority in controlling the underground atmosphere.

However, providing the necessary additional air quantity can be difficult and expensive because, for a given fan pressure (H, in. water), the quantity of air (Q, ft^3/min) delivered is inversely proportional to mine resistance (R):

$$Q = \left(\frac{H}{R}\right)^{1/2}$$

where R is determined from the physical characteristics of mine airways: airway frictional factor, airway perimeter, air-traveling distance, and average airway cross-sectional area shown as

$$R = \frac{KOL}{5.2A^3n^2}$$

where K is airway frictional factor (lb·min^2/ft^4); O is airway perimeter; L is traveling distance, which increases with increasing mine size (ft); A is average cross-sectional area (ft^2), and n is the number of entries.

In general, R&P mining will have low mine resistance because of multiple (parallel) entries. However, this can be both advantageous as well as challenging; the air needs expensive channeling systems (continuous stopping lines and overcasts) to effectively "push" fresh air to the working faces.

In the United States, mines using the R&P method have always been laid out in a structured and hierarchical fashion—intake, return, and neutral (belt and track) airways are always clearly defined and grouped by stopping lines in all mains, submains, and panel layouts, as shown in Figure 13.2-2 (where all three intake, two neutral, and three return airways are separated by stopping lines.) A great number of overcasts (or undercasts) are used to separate the intake, neutral, and return airflows. The number of entries at an intersection will determine the number of overcasts (or undercasts) for effective air distribution. For example, as many as 16 overcasts at one single intersection may be necessary to fully separate a 7 × 5–entries intersection.

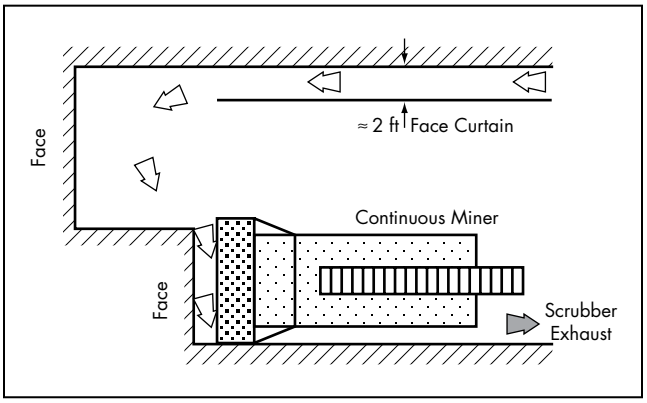


Figure 13.2-9 Scrubber at work

The power (AHP, or air horsepower) required to circulate a certain amount of air quantity (Q) under H (in. water) can be calculated using the following equation:

$$\text{AHP} = \frac{HQ}{6{,}345} = \frac{(RQ^2)Q}{6{,}345} = \frac{RQ^3}{6{,}345}$$

Alternatively, it is proportional to the cube of air quantity delivered. In other words, double the air quantity quadruples the mine pressure, which requires eight times the amount of energy compared to that needed before the change.

In the face area, U.S. federal regulation (CFR 2010) requires that 9,000 cfm of fresh air is to be delivered to the last open crosscut, properly installed face curtain (or brattice) in each individual heading used to push air to the deepest penetration, and check curtains must be designed to isolate the dusty air generated in the feeder breaker area from being blown to the face area, as shown in Figure 13.2-6.

Perhaps the most prolific success story in face ventilation since the 1980s has been the use of a machine-mounted flooded-bed scrubber (Figure 13.2-9). Flooded-bed scrubbers are devices installed on CMs to collect dust-laden air through several inlets near the front of the continuous mining machine. Inside the scrubber, the dust-laden air passes through a knit wire-mesh filter panel and is wetted with water sprays, where the dust particles are captured by the water. This filtered air then enters a demister where water is removed before the treated air is discharged at the back end of the miner.

Today, scrubbers are widely used on CMs in the United States and are highly successful in reducing coal dusts in the face area. Efficiencies for removal of respirable dust typically run from 90% to 95%, provided that screens are washed relatively frequently.

Many factors influence scrubber efficiency: scrubber capacity, face ventilation method (blowing or exhausting curtain or tubing), amount of fresh air reaching the beginning of the face curtain or tubing (a minimum of 3,000 cfm by regulation), ventilation setback distance (distance from the end of the face curtain or tubing to the deepest penetration), and CM location. Water-spray locations will also affect scrubber capture, with studies showing that by placing water sprays outby (away from the face), the scrubber inlets between the spray inlets and the rear end of the machine make these water sprays act as small jet fans. This arrangement will potentially induce additional airflow alongside of the machine and provide best results.

Another way to reduce dust includes displacement, which uses the airflow in a way that confines dust and methane sources and keeps them away from workers by forcing the dust and/or methane downwind of the worker. In addition, modifying the CM cutting cycles by leaving roof coal in place until it can be cut to a free face generates less dust and sometimes provides better extraction control.

OTHER ESSENTIAL AUXILIARY SERVICES

Other auxiliary operations are essential to the performance of an operating unit: coal haulage, power supplies, pumping, materials and supplies, stopping lines, and overcast construction.

A CM is not truly continuous but often only operates for less than half of the time, with the other half either employed for tramming (from location to location) or waiting for haulage vehicles to unload coal. Different methods have been used to minimize this nonproductive time in order to realize CMs' true production potential in the face area. The proper number of haulage vehicles for the face and an optimum distance between the face and the feeder breaker are the most commonly used measures to minimize a CM's waiting time. Face haulage uses either an electric-powered vehicle with an umbilical cable, rubber-tired shuttle cars, or battery-powered ram cars. Today it is common to use three haulage vehicles with well-planned travel routes to maximize efficiency, a compromise between optimum coal haulage and minimum congestion.

Flexible conveyor trains (FCTs) have also been used in some mines to minimize the CM's waiting time. They are electric-powered, self-propelled conveyor systems that provide continuous haulage of material from a CM to the main mine belt. The FCT uses a rubber belt similar to a standard fixed conveyor. The FCT's conveyor belt operates independently from the track chain propulsion system, allowing the FCT to move and convey material simultaneously. Available in lengths of up to 570 ft, the FCT is able to negotiate multiple 90° turns in an underground mine infrastructure, thereby greatly facilitating face haulage.

In addition to face haulage, proper planning is also critical to ensure an efficient haulage system to transport coal from the face area to the panel belt and ultimately to the surface through panel and main conveyor belts.

To prevent explosions from settled coal dust on the floor, rib and roof areas of mine entries are covered with pulverized limestone (inert) dust or rock dust. Studies show that during an explosion the rock dust disperses, mixes with the coal dust, and prevents flame propagation by acting as a thermal or heat sink (i.e., the rock dust reduces the flame temperature to the point where combustion of the coal particles can no longer occur). Under U.S. federal law (CFR 2010), all underground areas of a coal mine, except those areas in which the dust is too wet or too high in incombustible content to propagate an explosion, must be properly rock-dusted to within 40 ft of all working faces. Rock dusting is usually carried on a third or idle shift to not interfere with routine production.

In coal, methane drainage is still an art, but some of the drilling technology (e.g., directional drilling from both surface and underground) borrowed from the petroleum industry has been practiced at some gassy mines since the mid-1990s and has obtained quite satisfactory results; its performance is still continually being assessed.

PILLAR EXTRACTION

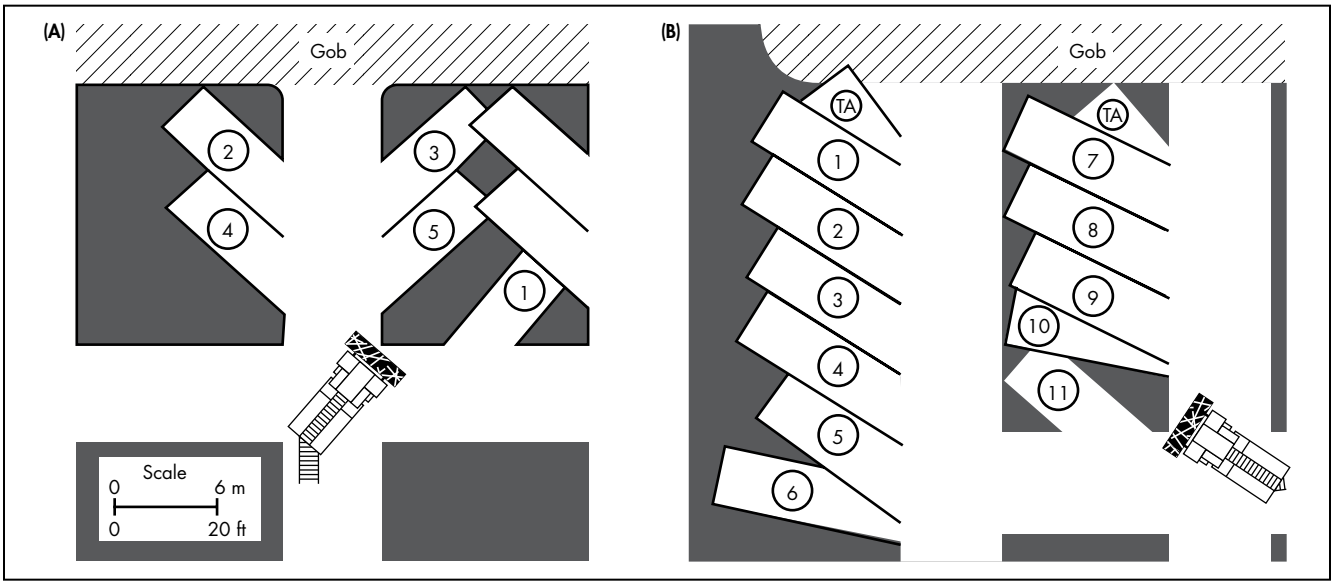
Only after careful review of pillar performance to date and pillar-extraction plans, and upon approval from regulatory agencies can coal pillars can be systematically removed after primary mining to maximize coal production. Some of the commonly used extraction methods are the "Christmas tree" or "split-and-fender" extraction methods. The former is favored because it does not require place changes and bolting. A third system is the "pocket-and-wing" method, which can fully extract large pillars. The general procedure is to extract one row of pillars at a time, leaving the mined-out portion, or gob, free to subside. Whereas extraction of all the coal in a pillar is a desirable objective, partial pillar-extraction schemes are more common. These are usually accomplished by CMs (or hydraulic mining with augers) driving into pillars according to predetermined extraction plans (Figure 13.2-10).

Pillar-extraction plans must be properly designed according to site-specific geotechnical and operational parameters (equipment and timber availability and cost, pillar dimensions, major geological features/structures, roof competency, seam thickness, extraction span or cavability, gob edge, and depth of cover) to ensure pillar stability during extraction. Case histories attributed to roof falls have found that both weak rock and competent immediate roof strata cause pillar failures. Hazards associated with pillar extraction also tend to intensify with depth; failures include both squeeze (nonviolent gradual pillar failures) and bumps, both of which are generally more severe at depth. Design of pillar-extraction schemes requires geotechnical input from specialists.

SYSTEM RATIONALIZATION AND OPTIMIZATION

Traditionally, coal productivity has always been measured by tons per workshift or similar measures based on total tonnage. Other factors such as size distribution, dust generation, and waste management also need to be considered at the planning stage to meet specific contractual demands and processing capacities at the surface preparation plant.

In the face area, CM cutting, loading and tramming capabilities, miner coal haulage, face layout, and so on must all be carefully planned and matched in order to achieve optimal overall efficiency. Materials handing still represents a significant component of the operational cost. A study on coal mining production costs in the United States (Chugh et al. 2002) shows that, of the total production cost, labor (with benefits) constitutes 47%, mining-related costs (supplies, maintenance, power, and other direct costs) uses another 29.5%, with the remaining designated to depreciation, taxes, and other expenses. Of this total production cost, half goes to face production, outby construction uses another 25%, coal processing and waste management uses 15%, and the remaining 10% is allocated to transportation. The same study indicates that face haulage is still the major bottleneck to productivity despite the use of more powerful high-voltage CMs. The study compares a Joy 14CM15 miner (995 V) with a high-voltage 14CM27 miner (2,300 V) in the Illinois coalfields. Results show that the high-voltage miner outperforms its counterpart in cutting and loading rate (by 29.89%), tons per car (by 8.64%), and tramming rate (7.84%), but they perform equally in tons per shift. (True productivity gains of the high-voltage miner were not fully realized because of the batch haulage system delays.) In most countries, the major impediment to productivity is still the time required to install roof supports.



Source: Mark and Zelanko 2001.
Figure 13.2-10 Commonly used pillar extraction methods

SUMMARY

With its low capital investment and great flexibility, R&P mining continues to be an indispensible method in underground coal mining. In some cases it is the only method to access remnant parcels of coal that would otherwise be abandoned using longwall mining. It is also essential in gate road development in longwall mining, and under special circumstances, with proper planning, it may (in cost terms) even compete with longwall mining.

Although the basic methods of operation in R&P mining remain similar, there are a variety of systems with minor variations from location to location depending on local specifics. It is all a matter of system optimization—what would be an optimal number of entries or haulage method in one location may not be so in a different location depending on advance rates, ground control concerns, and dust and methane emissions that dictate ventilation practices. The optimum plan for a site is a trade-off of such factors. For example, the more entries, the higher the coal production, although at the expense of advance rate. On the other hand, fewer entries will cause congestion but will provide faster advances. Modern high-speed computers and simulation packages are ideal for modeling these scenarios for optimum performances.

A CM must be matched with a compatible haulage system in order to realize its full potential. Options could be a high-voltage miner matched with a continuous haulage system, a batch haulage system, or a surge car. Another benefit is that the high-voltage miner generates 43% less dust; using surfactants reduces that by an additional 6%.

Underground coal mining has always been a cost-competitive industry and will continue to be so in the foreseeable future. Results from continuing efforts to reduce production costs and/or increase productivity to stay competitive have been quite impressive. According to the U.S. Energy Information Administration (2010), in the period from 1949 to 1999 the underground coal productivity per worker-hour rose from 0.68 t/h (short tons per hour) to 3.99 t/h, a 587% increase, and to 4.04 t/h in 2004, a 594% increase from 1949 and a historical high. This rate decreased to 3.17 t/h in 2008.

The continuing improvement in equipment technology, system monitoring, automation, and system optimization in R&P mining make this method more productive and indispensible in underground coal mining.

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CHAPTER 13.3

Shrinkage Stoping

Jack Haptonstall

INTRODUCTION

Shrinkage stoping is a vertical overhand stoping method in which most of the broken ore remains in the stope to provide a working floor for the miners. The broken ore also provides wall support until a stope is completed and ready for drawdown of the remaining ore.

In general, the method is used in steeply dipping (>50° dip) narrow ore bodies with regular boundaries. Ore and waste (both the hanging wall and the footwall) should be strong, and the ore should not be affected by storage in the stope. The method is labor intensive and cannot readily be mechanized. It is usually applied to ore bodies on narrow veins or to ore bodies for which other methods cannot be used or might be impractical or uneconomical. The method can be easily applied to ore zones as narrow as 1 m (3.1 ft), but it can also be successfully used for extraction of much wider stopes. The method is most efficient when drilling of the ceiling or back is done with uppers instead of horizontally.

Shrinkage stopes are mined upward in horizontal or inclined slices. Usually about 35% of the ore derived from the stope cuts (the swell) can be drawn off (shrunk) as mining progresses. As a consequence, no revenues can be obtained from the ore remaining in a stope until it is finally extracted and processed for its mineral values.

Logically, ore broken in a shrinkage stope should be free flowing, and broken ore should not pack or re-cement itself together. Neither the ore nor the adjacent country rock should contain undue amounts of clay or other sticky material that would cause the broken ore to hang together in the stope and make it difficult or impossible to draw. The ore should not readily oxidize, which may cause the broken ore pile to re-cement itself and consequently "hang up." Also, oxidation of the ore minerals may have an adverse effect on recoveries of values in subsequent beneficiation of the ore. To avoid mining excess amounts of waste as dilution from the stope back, ore developed for shrinkage stoping should be fairly continuous along the strike of the vein or ore body. In many cases, however, it may be possible to mine around waste areas and leave them in place as random pillars.

In designing for shrinkage stoping, consideration must also be given to the plunge or rake of the ore body, especially where an entire ore body or ore shoot may be mined as a single stope rather than as preestablished stope panels with defined vertical endlines. A stope with a shallow plunge or rake (<50°) may be difficult to mine by shrinkage methods because the ore moves away too quickly from the predeveloped extraction system, occasioning the need for additional development of the system. Further development, especially raising, is also often required when the ore abruptly extends beyond stope endlines.

Following is a summary of the parameters for the application of shrinkage stoping to a particular ore body (Boshkov and Wright 1973; Lucas and Haycocks 1973; Morrison and Russell 1973; Lyman 1982):

- **Ore characteristics:** The ore should be strong and nonoxidizing, should not pack or stick together, and should not spontaneously combust.
- **Host rock characteristics:** The host rock should be characterized by moderately strong walls that are free of clay or geologic structures such as faults, joints, and so on.
- **Deposit shape:** The deposit can be almost any shape, but it should have uniform, definitive boundaries.
- **Deposit dip:** The dip should be greater than the angle of repose (>39°) and preferably steeper than 50°.
- **Deposit size:** Width of the ore deposit should be narrow to moderate; the deposit can be of any length and height.
- **Ore grade:** Average ore grade should be moderate to high.
- **Production rates:** Production rates should usually be small to moderate.

Vertical longitudinal sections of typical shrinkage stopes are shown in Figures 13.3-1 through 13.3-3.

DEVELOPMENT AND PREPARATION

Areas selected for shrinkage stoping are generally developed by drifting in the vein or ore body on two levels that are spaced vertically at predetermined intervals that customarily

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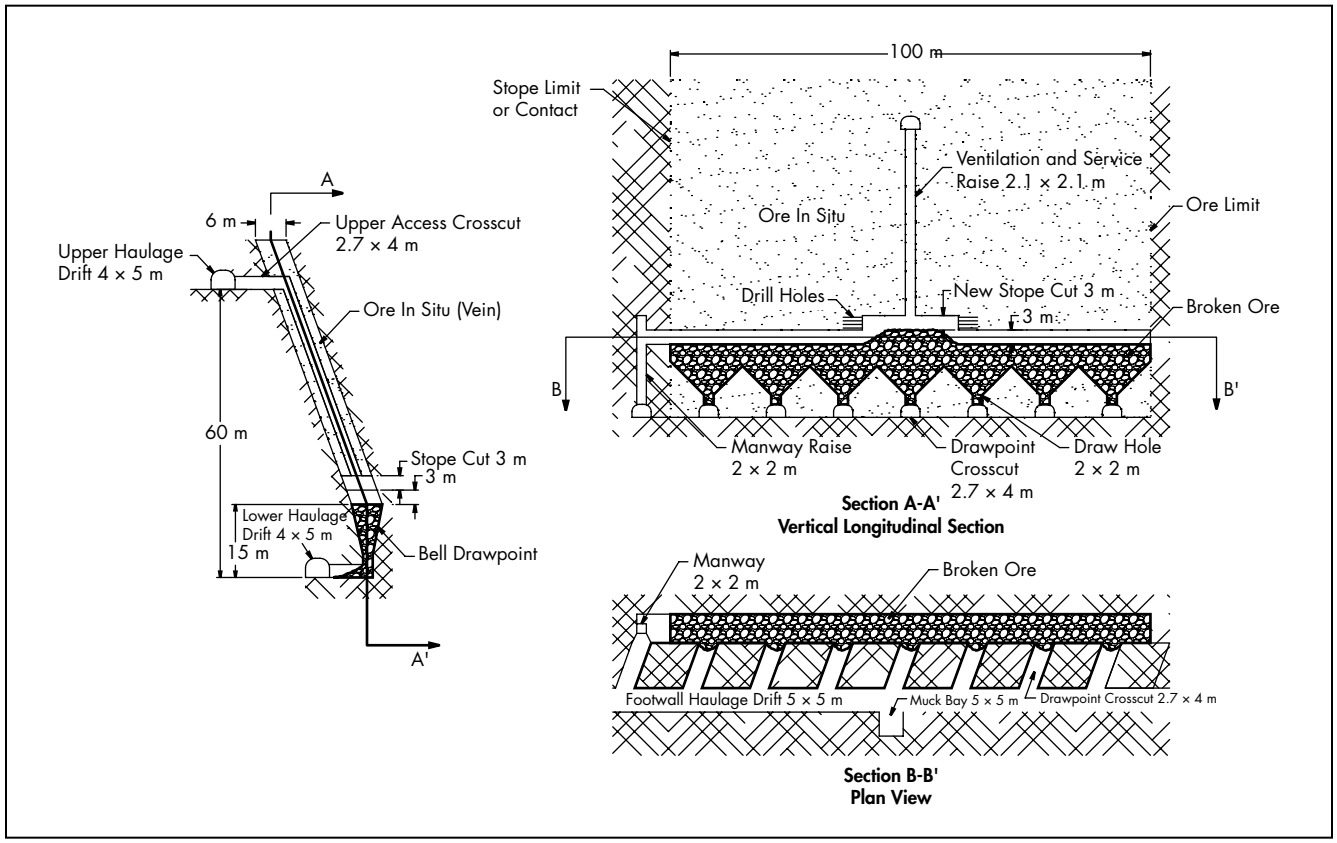


Figure 13.3-1 Idealized shrinkage stope with drawpoint extraction

vary between 30 and 180 m (100 and 600 ft). After a viable ore body has been established, the next phase is to develop one or more raises to establish vertical ore continuity and provide a ventilation opening and access to the stope. In some cases the raises are not developed, and access is only from the sill level below the stope.

Raising may be done conventionally with platform-type raise climbers or mechanically with raise-boring equipment. Drifting is normally done using conventional drill-and-blast track or trackless methods.

Stopes may be prepared with extraction raises on 7.6- to 9-m (25- to 30-ft) centers the full length of the stope block. Each raise is fitted with a chute (except for manways and/or service ways) that is normally of timber construction. Extraction raises are belled out and hogged over (mined over or excavated) to form the undercut for the start of the first stope cut (see Figure 13.3-2).

Another method used for preparing a stope is to blast down at least two cuts from the ore zone, clean up the broken ore, and install stull timbers or timber sets in the sill drift below the stope. Timber chutes or chinaman chutes are installed at approximately 7.6- to 9-m (25- to 30-ft) intervals as part of the timbering. A chinaman chute is a timber platform constructed below the caps of the timber sets; ore is shoveled from the platform into cars spotted below it.

Both of the foregoing types of stope preparation are still used, but on a very limited basis. A more common method for preparing shrinkage stopes in modern operations is to drive a sill drift along the vein or ore body as well as an extraction drift in the footwall of the ore zone and parallel to it. The footwall drift is normally situated about 7.6 to 15 m (25 to 50 ft) from the ore body. Subsequently, draw-hole extraction crosscuts are driven from the footwall drift into the sill drift on 7.6- to 15-m (25- to 50-ft) centers. The back of the ore body sill drift is then drilled and blasted down, and the swell is extracted via the draw holes, either with rail-mounted mucking machines or with load-haul-dump (LHD) units. See Figures 13.3-1 and 13.3-4).

Obviously, other variations for the preparation of shrinkage stopes have been used successfully. These may include preparation with short crosscuts into the waste footwall from the ore sill drift, driving finger raises back up to the ore and preparing the first stope cut from the finger raises. Another variation might be the elimination of the construction of chutes or drawpoint crosscuts and drawpoints, electing instead to extract ore from the ore sill drift with remotely controlled LHD units.

STOPING OPERATIONS

When an ore block has been developed for shrinkage stoping, a manway is usually constructed in the raise to the next level. One or more manways and service ways are usually constructed with timber on one or both ends of the stope. Often a timber slide is installed in one of the manways for hoisting and lowering supplies and equipment into and out of the stope; hoisting and lowering are often done with an air or electric single-drum (tugger) hoist, which is installed in the level below the manway. After the manways, ventilation raise(s), and service ways have been established for a stope, mining can commence.

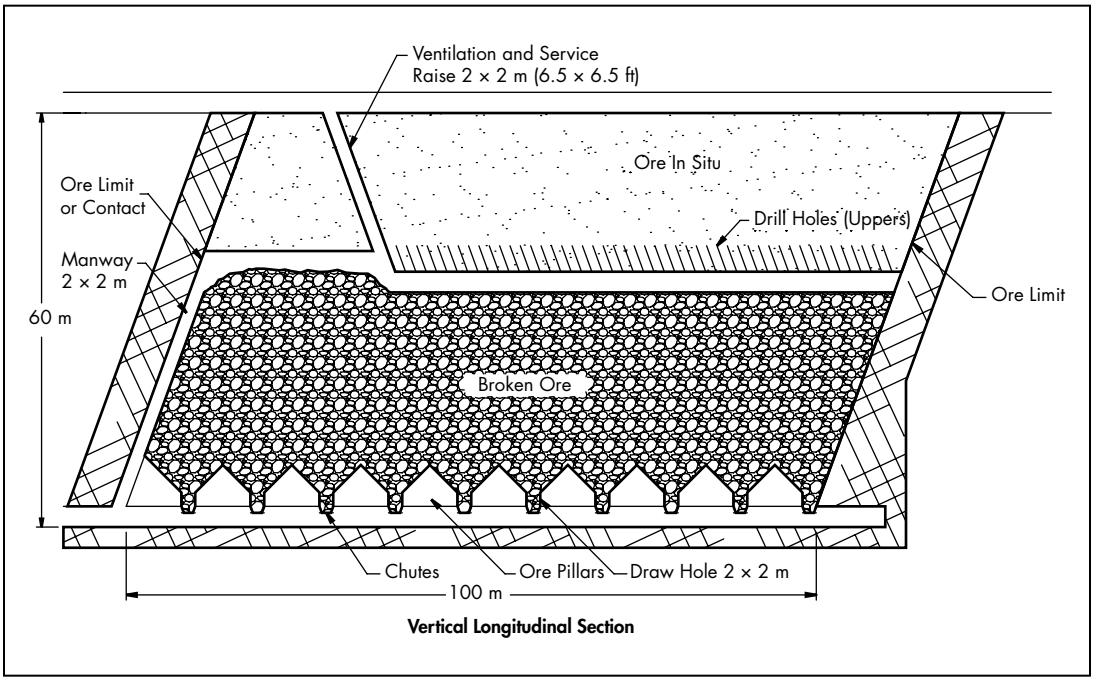


Figure 13.3-2 Idealized shrinkage stope with chutes

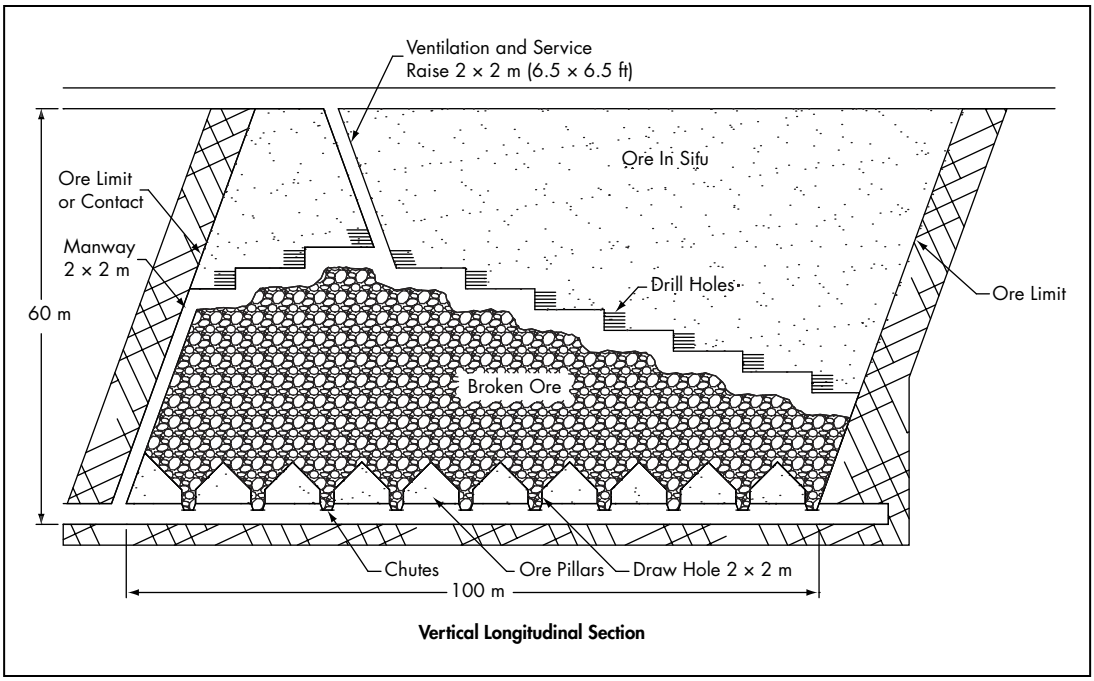


Figure 13.3-3 Idealized rill shrinkage stope with chutes

Drilling of a shrinkage stope back is usually done with handheld stoper or jackleg drills, although mechanized drill wagons or jumbos have been used in wide (>5.0 m [>16.4 ft]) stopes. Back stoping (vertical or near-vertical drilling) is the preferred mode of operation, but breasting down (horizontal drilling) is also common. Upholes are generally 1.5 to 2.4 m (5 to 8 ft) in length, and usually all drill holes are loaded and a complete cut is blasted simultaneously. Breasts are drilled with 1.8- to 3.0-m (6- to 10-ft) holes or normally are blasted once per shift. Drill holes are loaded with ANFO (ammonium nitrate and fuel oil) products, water gels, and slurry blasting agents. Primers are usually water gel sticks or dynamite, and holes are initiated using nonelectric, electric, or fuse-blasting systems.

After a cut or round has been blasted in a stope, drawdown (emptying out) of the 35% swell is necessary. Leveling of the muck pile is required as a next step to provide a work platform and facilitate drilling the next vertical or horizontal cut. Leveling of the ore in the stope can be done with hand

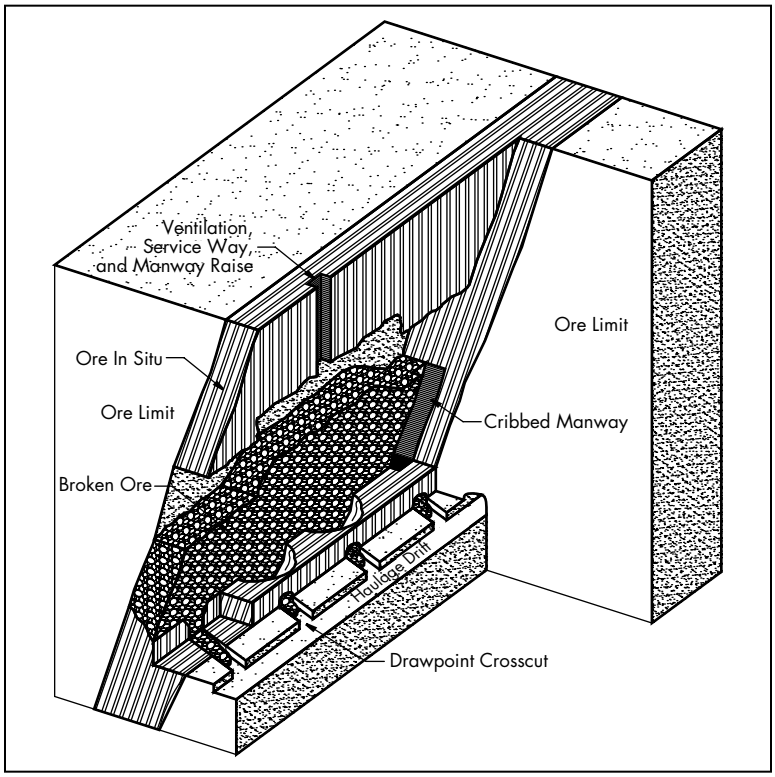


Figure 13.3-4 Isometric view of shrinkage stope with LHD drawpoints

shovels for small stopes, with two- or three-drum air or electric slushers for stopes that are longer and/or wider, and with LHD units for large stopes. After leveling, construction of manways and service ways and drilling of the next stope cut are done to continue the mining cycle.

Variations for establishing openings for manways, ventilation raises, or service ways may include strategically cribbed timber raises, steel culverts or rings, or even timber sets within the confines of the stope. These installations may be desirable during the mining phase but may create safety problems or nuisances during ore drawdown due to the collapse of the structures for these facilities. Pinning, stulling, or wedging these installations to the stope walls may prevent their destruction during drawdown. However, if the construction materials are dragged down with the broken ore, they can cause hang-ups in the extraction drawpoints or chutes.

A stope should have strong self-supporting walls to permit the application of shrinkage stoping. Excessive dilution through scaling of walls can preclude use of the method. Good mining practice and government regulations dictate a minimum ground support program for stope extraction. Shrinkage stoping has inherent dangers, merely because workers must be in the excavated area to drill, load rounds, and install ground support and stope services—all while standing on a broken ore pile. Many serious accidents have occurred when voids in the broken ore pile suddenly collapsed, dragging miners and equipment down into the void. In other cases, a chute or draw hole over which miners were working had mistakenly been pulled, with the same result. The requirement that miners use safety belts in a stope may reduce the danger, but it also reduces the workers' flexibility.

Ground support is usually minimal in shrinkage stopes, especially when the method is applied in areas of a vein or ore body that are steeply dipping, have strong ore and wall rocks, and have few structural features (e.g., faults, fractures, or joints) in the walls or within the vein or ore body. However, modern mining practice is to err on the side of safety, and so in most cases ground control is a primary function. Rock bolting of stope backs and ribs with mechanical or grouted rock bolts is the preferred method of ground support in shrinkage stopes. However, in very narrow stopes (e.g., <2.0 m [<6.6 ft] wide), correct installation of bolts in the walls is quite difficult because the bolts must be placed perpendicular to the planes of the hanging wall and footwall. More often than not, bolts in narrow stopes are installed at acute angles to the walls, which diminishes the effectiveness of the installations. Stulling of the ribs and back is still practiced; recent South African innovations in stull installation have made it a much more attractive option.

Stulling is simply the installation of timber props from wall to wall or floor to back. Typically, a board headpiece is inserted between the head end of the stull and the rock hanging wall, and the stull is secured by pounding wedges between the rock footwall and the butt end of the stull. In the past, the butt ends of stulls were almost always installed in a hitch cut from the rock with a hammer and chisel. The invention of the Jackpot headpiece has resulted in efficiency improvements in the installation of stulls. The Jackpot is a hollow steel can that fits over the butt-end of a timber stull and is pumped up hydraulically to pressurize the stull against the ribs or back of the stope. The unit eliminates the need to cut a hitch in rock to accommodate the stull, cut the stull to a precise length, and then try to tighten the installation with wooden headboards

and wedges. Leaving pillars in areas of low-grade ore or waste within the stope boundaries is a very acceptable approach to supplementary wall support. The pillar should be shaped in such a way that ore does not remain on top of it on drawdown of the stope.

Sampling of narrow shrinkage stopes is usually done by taking either channel or chip samples at systematic intervals (e.g., 1.5 m [5 ft]) from the stope back, ribs, or face. Samples are usually taken by hand; sometimes mechanical means can be applied in the process. When high sulfides are present, X-ray analyzers can be used for sampling. In wider stopes, drill sampling of the back and ribs can be done. The drill sample holes may crisscross the stope back according to a predetermined pattern and may also be drilled into the stope walls. Drill cutting samples are collected in a canvas sample bag or other container through a hose and funnel or other device.

STOPE DRAWDOWN

Shrinkage stope drawdown usually results in a consistent and steady supply of ore to the process plant and/or mine stockpiles. If a mine has several shrinkage stopes in various phases of the mining cycle, there is no reason the ore flow to the plant or stockpiles should ever diminish after completion of the initial stopes in the mine plan. Maintaining a steady ore flow from shrinkage stoping is largely dependent on engineering the optimum production rate for a given operation.

Completed stopes can be drawn down from strategically placed chutes and from drawpoint crosscuts, or they can be remotely mucked with LHD units. In some mines, slusher trenches developed below the stopes have been used effectively to remove broken ore from shrinkage stopes. Haulage from the stope extraction points can be done using trackbound or trackless equipment. Chutes should be robustly designed and well constructed to ensure that they will not be destroyed during the extraction phase.

Shrinkage stopes usually should be drawn down evenly so that if the stope walls do peel or slough, the waste material remains atop the pile and does not add dilution to the broken ore. After a stope drawdown has started, ground control of the walls, pillar recovery, ore hang-ups, and so on, is minimal. Reentry of miners into a stope during the drawdown phase is not recommended.

One of the most dangerous jobs in a mine is the drawdown of shrinkage stopes, especially where the ore contains blocky or sticky material that may cause the broken ore to hang up in the stope. Stope hang-ups are often removed by washing them down with water, by destroying them with explosives, or by blowing them down using high-pressure air (blowpipes). In the past, miners entered completed stopes to pick them down by hand, but as has been stated previously, this is unsafe and not recommended.

A summary of the advantages and disadvantages of the shrinkage stoping method is shown in Table 13.3-1.

VARIATIONS AND APPLICATIONS

Variations of shrinkage stoping include inclined or rill shrinkage and long-hole shrinkage. Large pillars or small isolated ore blocks may sometimes be recovered using shrinkage methods. Certain underground openings, such as short shafts, may also be excavated using shrinkage methods.

A mine that employed shrinkage stoping as a primary mining method was the now closed Homestake gold mine in Lead, South Dakota (United States). The Homestake ore bodies were mainly fracture fillings situated in the limbs of folds. “Bullpen” stopes (i.e., fixed-minimum-width shrinkage stopes laid out in a transverse direction to the long axis of a large ore body) 15 m (50 ft) wide were developed across the great Main Ledge ore body. Stopes were mined over timbered sills, where strategically located chinaman chutes were constructed for ore extraction. Stopes were mined overhand for about 21.5 m (70 ft) to within 9.1 m (30 ft) of the next level. Ore pillars 7.6 m (25 ft) thick were left between the primary stopes, and these, along with the crown pillars left over in the primary stopes, were subsequently extracted with square-set stopes. Homestake abandoned this type of shrinkage stoping just before World War II.

A second example of large-scale shrinkage stoping is the shut-down Idarado mine located between Ouray and Telluride in Colorado (United States). The Idarado ore bodies were mainly steeply dipping veins with excellent ore continuity. Shrinkage stopes were mined along the veins’ full width, spans that varied between 1.5 and 7.6 m (5 and 25 ft). The consistent ore continuity permitted the mine planners to divide the veins into stope panels generally 122 m (400 ft) long and about 61 m (200 ft) high. Panels were prepared over a slusher trench situated about 6.1 m (20 ft) above the primary on-vein drift. Raises on 7.6-m (25-ft) centers, developed from the slusher trench, were belled out and interconnected above the slusher trench to form the opening for the first stope cut. Ore extracted from the stope was slushed from the extraction raises to a chute in the exact center of the stope. The Idarado operations were closed in 1979.

Variations on the Idarado system were practiced at the Morococha and Casapalca mines of the Cerro de Pasco Corporation in the Andes Mountains of central Peru in South America and at the Tayoltita mine in the state of Durango, Mexico. Stopes in these mines were prepared conventionally over the main on-vein development level by driving 7.6-m- (25-ft-) long raises on 7.6-m (25-ft) centers. The raises were belled out to form the opening for the first stope cut about 5 m (16 ft) above the main level. On completion of this step, each raise was fitted with a timber chute for ore extraction. (See Figures 13.3-1, 13.3-2, and 13.3-4.)

In all of these examples, a raise was first developed through each ore block or stope panel for ventilation, manway, and service way. Manways from the sill level were carried as timber-cribbed raises within the stope, as stulled and lagged raises on the ends of a stope, or—as in the case of the Idarado mine—as boreholes located 3 m (10 ft) inside the footwall of the stope. In the vein mines, drilling was done with pneumatic handheld stopers or jackleg drills, whereas in the Homestake mine, drilling was done with bar-and-column Leyner-type drills.

The terms *inclined shrinkage* and *rill shrinkage* refer to an application in which multiple faces or benches for drilling are carried along the back of the stope as it is mined upward (Figure 13.3-3). This system was common practice at the Pachuca mine in the state of Hidalgo, Mexico. Stopes were developed conventionally over pillars as described previously, and chutes on 7.6-m (25-ft) centers were installed on the extraction raises. Pachuca miners preferred to use jackleg drills rather than stopers, and with the rill system multiple faces were available for breast stoping in a given shift (see Figure 13.3-3).

Table 13.3-1 Advantages and disadvantages of shrinkage stoping

Advantages	Disadvantages
Shrinkage method is easily adapted to narrow veins.	Method requires steep dip (>60°).
Stopes do not require backfill.	Method requires strong ore and walls.
Stope extraction system is usually simple.	Stoping is labor intensive and efficiencies are low at about 3 to 9 t (3.3 to 9.9 st) per worker-shift.
Stope drawdown provides a steady, consistent supply of ore.	Only 35% to 40% of ore is available for extraction during mining cycle.
Application of method requires a minimum investment in equipment and machinery.	Very little mechanization of stoping is possible.
Stope development is uncomplicated and minimal.	Excessive clay or oxidation of ore may result in stope hang-ups or problems in ore dressing.
Reasonable selectivity is possible.	Ground control is sometimes difficult to install.
Ground support of ore and walls is minimal.	Risk exists of losing stope if drawdown is not properly controlled.
Gravity aids drawdown of stopes.	Mining costs are usually moderately high to very high.
Mining recovery is good (75% to 100%).	
Dilution is low (0% to 20%).	

Source: Morrison and Russell 1973; Lucas and Haycocks 1973; Hamrin 1982; Lyman 1982.

Long-hole shrinkage (Figure 13.3-5) is developed in the manner previously described. The method was applied with limited success at the Stillwater mine near Nye, Montana (United States). In this method, drilling of the stope block is done from vertical raises driven through the ore block or panel on 15- to 30-m (50- to 100-ft) centers. Raises in this application are preferably developed with raise climbers or with cage raising techniques. The raise climber platform or the cage serves both as the entry/exit vehicle for the raise and the platform from which to drill and blast the stope block. Long holes are drilled from the raise, parallel with the strike of the ore zone, and these are also loaded and primed from the climber or cage platforms. Initiation of the stope blast is done from a safe area on the service level below the stope.

Shafts, winzes, or other large underground openings may be developed through shrinkage methods. Often these excavations are done as described previously for the long-hole shrinkage application. In certain situations, conventional shrinkage of a shaft, winze, or other excavation may be necessary.

SAFETY, EFFICIENCIES, AND COSTS

Shrinkage stoping has largely been discontinued in most countries because it has numerous safety hazards, is very labor intensive and largely inefficient, and is generally expensive. Coupled with these reasons is that more than 60% of the mined ore remains in the stope until stoping is completed, which may take many months and sometimes even years. During this period, amortization of the investment required to develop and mine an ore block by shrinkage stoping may outweigh any advantage in applying the method to a particular operation (see Figure 13.3-5).

Typically, the efficiencies in a modern shrinkage stoping operation are in the range of about 0.3 to 1.2 t (0.3 to 1.3 st) per worker-hour. In western operations, very few miners have experience in the development and operation of shrinkage stopes, but the method is still widely practiced in Latin America and in developing countries elsewhere in the world.

Costs in a typical North American shrinkage operation are currently on the order of US$25 to US$50/t (US$55/st), with more than 50% of the costs assigned to labor. Operations in Latin America or Asia may enjoy lower costs because labor is less expensive, but efficiencies are generally not equal to those of North American operations.

CASE STUDY: EAGLE RIVER MINE

The Eagle River gold mine is owned and operated by Wesdome Gold Mines of Toronto, Ontario, Canada. The mine is located near the northeast corner of Lake Superior, 50 km (31 mi) west of the town of Wawa, Ontario. It is accessible by gravel roads connecting Trans-Canada Highway 17 near Wawa to the site.

The Eagle River mine has been in production for about 12 years. Historic production from the mine is about 22,900 kg (711,000 oz) of gold from mining and processing about 2.4 Mt (2.64 million st) of ore. The mine is currently being operated at a production rate of 100,000 t (110,000 st) per year.

Geology

The Eagle River ore deposit occurs in the Mishibishu greenstone belt, which is a broad arcuate belt about 60 km (37.2 mi) long and 18 km (11.1 mi) wide. The area in which the Eagle River deposit occurs contains magnesium-rich tholeite, calc-alkaline andesite, dacite, and minor rhyolite. The ore occurs within shear-hosted near-vertical quartz veins that have mineable widths of 1.5 to 7.5 m (4.9 to 24.6 ft). The ore zones of the Eagle River mine are of a series of quartz lenses that to date have been traced over a strike length of 2.5 km (1.5 mi) and a depth of 650 m (2,132.5 ft). At least four distinct ore zones or shoots have been developed, and these vary in length from 50 to 250 m (164 to 820 ft).

Ore Reserves and Mining Method

Proven and probable ore reserves at the end of 2007 were 253,000 t (278,300 st) at an average grade of 12.9 g gold/t (0.37 oz/ton). The mining method, formerly all shrinkage stoping, was changed to long-hole stoping in 2000.

Details regarding shrinkage stoping are as follows:

- Stope heights are 100 m (328 ft) with drawpoint scoop-tram/truck mucking and hauling.
- Two raises with doghole accesses are driven on the boundaries of ore blocks, which can be up to 100 m (328 ft) long.
- Two people work a stope, breasting toward the middle. Breasts are 2.5 m (8.2 ft) high and 3 m (9.8 ft) long and fill a void measuring 3.5 m (11.4 ft).
- Uppers at one end of a stope provide the start for the next lift.

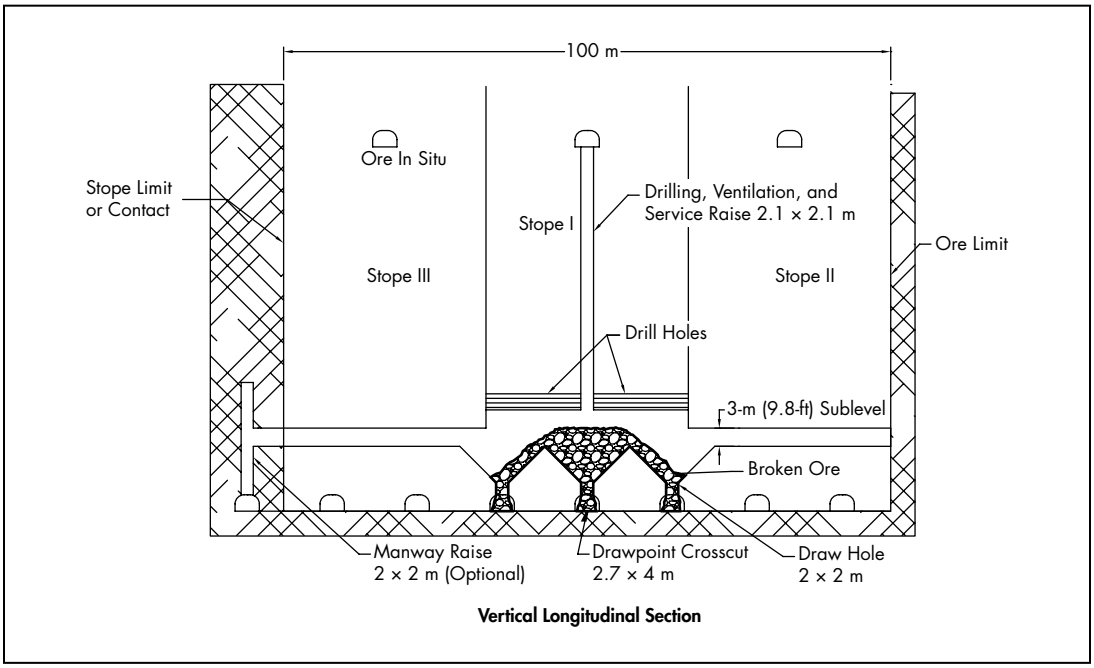


Figure 13.3-5 Idealized long-hole shrinkage stope

- Dogholes are staggered at 10-m (32.8-ft) intervals in both access raises. No stull timbering is required. Ladders provide access from dogholes, at least one of which is always available because of stagger.
- Productivity is 15 t (16.5 st) per worker-shift.

Details regarding long-hole stoping are as follows:

- Sublevels are spaced at 15 m (49.2 ft); jackleg and slusher subdrifts are driven from a central mill hole.
- Slot raises are developed as drop raises.
- Drill holes are 50-mm- (2-in.-) diameter downholes on a 3×3 m (9.8×9.8 ft) pattern.
- Productivity is 40 t (44 st) per worker-shift.
- Retreat mining is practiced with haulage and drawpoints on main levels spaced at 60-m (196.8-ft) intervals.

Equipment, Productivity, and Costs

Equipment includes 3.8-m^3 (5-yd^3) diesel LHD units, 25-t (27.5-ton) trucks, two-boom drift jumbos, scissors lifts, raise climbers, jacklegs, and stoper drills.

Productivity is about 15 t (16.5 st) per worker-shift in stopes. The shrinkage mining cost is unavailable. The long-hole mining cost is about US$40/t (US$44/ton).

SUMMARY

The labor intensity of shrinkage stoping as well as the limited mechanization possible for the method precludes its wide-scale application in most modern mining operations. However, in some applications, the method may be appropriate—for example, in a mine in which the ore occurs in very narrow veins and cannot be economically extracted using other methods.

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CHAPTER 13.4

Sublevel Stoping

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INTRODUCTION

This chapter refers to the generic mining method of sublevel stoping. The most commonly used sublevel stoping mining methods are sublevel open stoping, long-hole open stoping or blasthole stoping, and vertical crater retreat (VCR). Variations of this method include vein mining, transverse stoping, Avoca, longitudinal, and other less-used methods such as slusher mining of uppers. The shrinkage stoping method is also a variation of sublevel stoping and is discussed in Chapter 13.3.

Definitions

Following are definitions of general terms for this mining method, also shown schematically in Figure 13.4-1.

- **Span:** The length of the stope along the strike.
- **Width:** The perpendicular distance between the footwall and the hanging wall.
- **Height:** The distance along the exposed hanging wall and not the vertical height between levels.
- **Longitudinal pillar:** A pillar aligned along the strike of the stope.
- **Rib pillar:** A pillar aligned transverse of the stope, perpendicular to the strike.
- **Sill pillar:** Horizontal pillars that separate levels or stopes.
- **Dilution:** The reduction of ore grade due to mixing of ore with barren rock.
- **Internal dilution:** Rock that must be mined because of the geometry of the ore body and the requirement to mine rectangular areas. The term is synonymous with *planned dilution.*
- **External dilution:** Dilution caused by sloughing or failure of stope walls and back and is outside the blasted stope boundary. External dilution is defined as the external waste tonnage divided by the ore tonnage. The term is synonymous with *unplanned dilution*.

Sublevel stoping, in the absence of consolidated fill, employs pillars to separate the individual stopes to reduce the potential for wall slough. Sublevel stoping requires a straight/linear layout of stope and ore boundaries. Inside of the stope, everything is ore with no chance of recovering small mineralization in the wall rock. This method requires knowledge of the ore boundaries as shown in Figure 13.4-2.

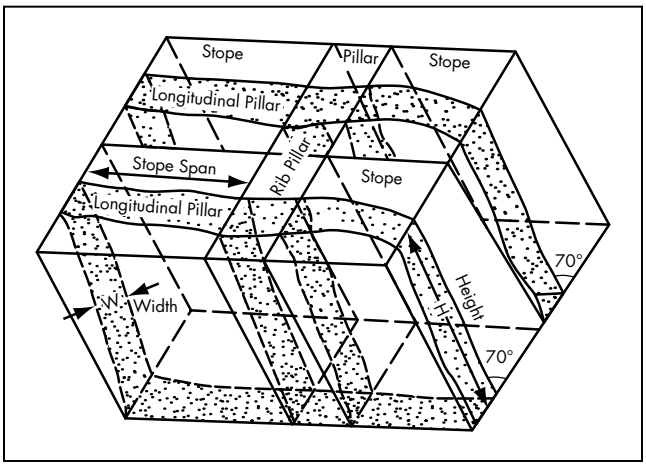
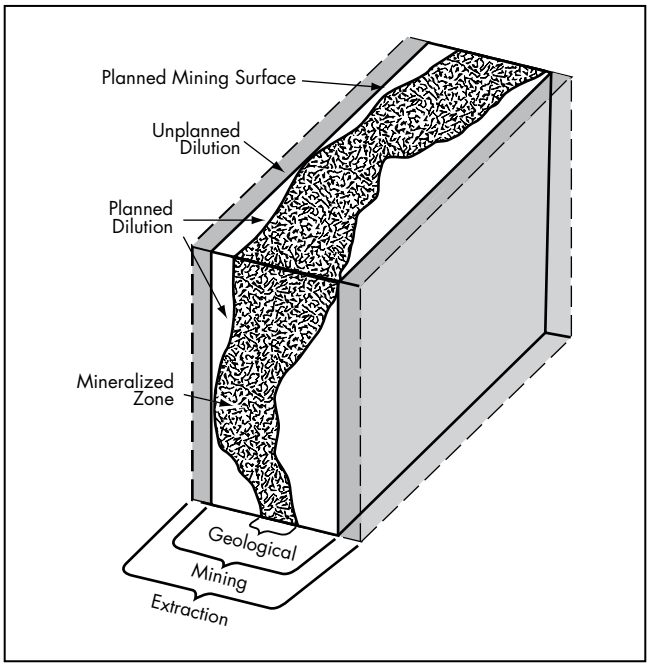


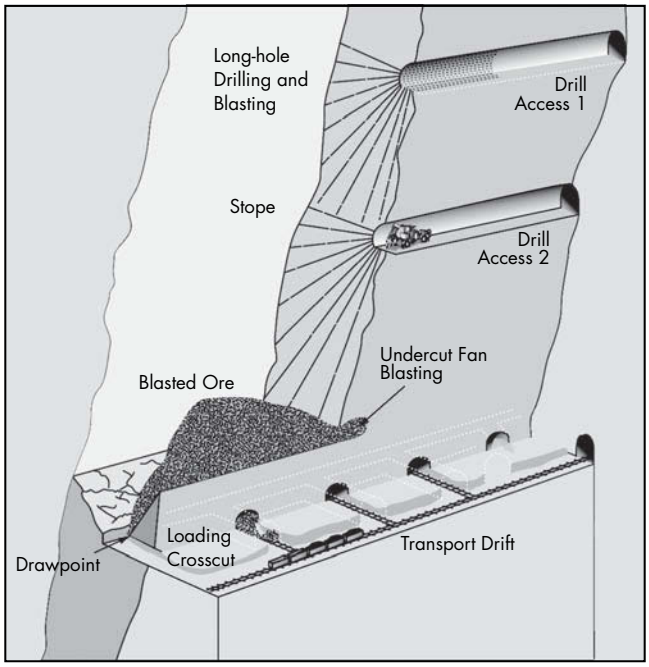
Figure 13.4-1 Terms used in sublevel stoping

Sublevel stoping with no fill is a mining method in which ore is mined and the stope is left empty. The result is a large void that requires individual pillars be placed to separate the stopes. Sublevel stoping is largely restricted to steeply dipping ore bodies (50°–90°) with a competent hanging wall (HW) and footwall (FW). Figure 13.4-3 shows the general approach to sublevel stoping whereby ring drilling is used from levels generally spaced ~20 m apart in a vertical dimension. Level spacing is largely limited by the length of the production holes, which range in diameter from 50–75 mm and maximum lengths of 25 m. This can be modified if in-the-hole (ITH) drills or top hammer tube drills are used. Characteristic with the sublevel stoping method are the intermediate levels, which largely differ from long-hole (blasthole) stoping as depicted in Figure 13.4-4 where the intermediate level has been removed. In sublevel stoping the mining is accomplished from individual levels at predetermined vertical intervals. These intervals are largely governed by

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Source: Scoble and Moss 1994.
Figure 13.4-2 Defining dilution



Source: Hamrin 2001.
Figure 13.4-3 Sublevel open stoping

- Ore geometry, in order to minimize internal dilution by enabling the extraction of irregular ore bodies,
- Rock mechanics constraints in terms of minimizing the external dilution through wall slough, and/or
- Operational restrictions such as drilling equipment constraints.

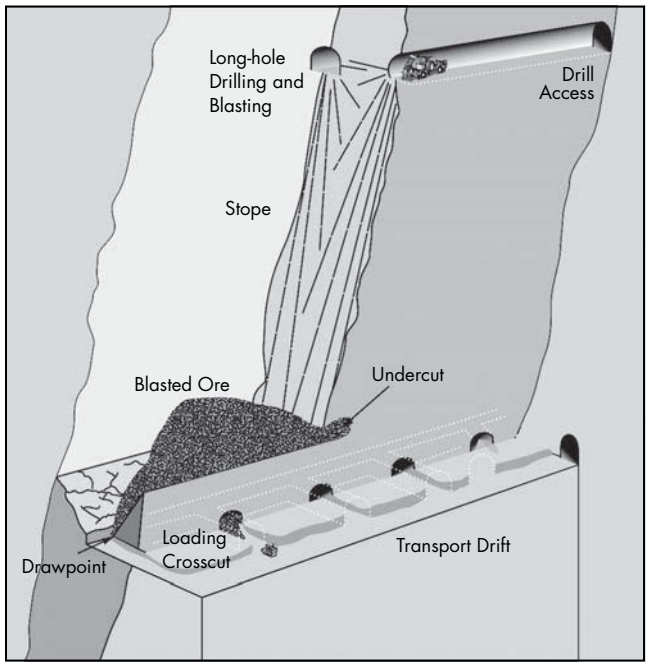
Sublevel and long-hole methods require blasting into a "vertical slot/free face," whereas a VCR, shown in Figure 13.4-5, differs in terms of blasting to a horizontal free face, which is largely confined due to the muck remaining within the stope as only the swell is drawn.

Variations of the sublevel method include narrow vein mining/Alimak, Avoca, longitudinal, sublevel retreat, and transverse stoping, as well as historical methods such as slusher and track mining (Haycocks and Aelick 1998).

SUBLEVEL STOPING REQUIREMENTS AND CONSTRAINTS

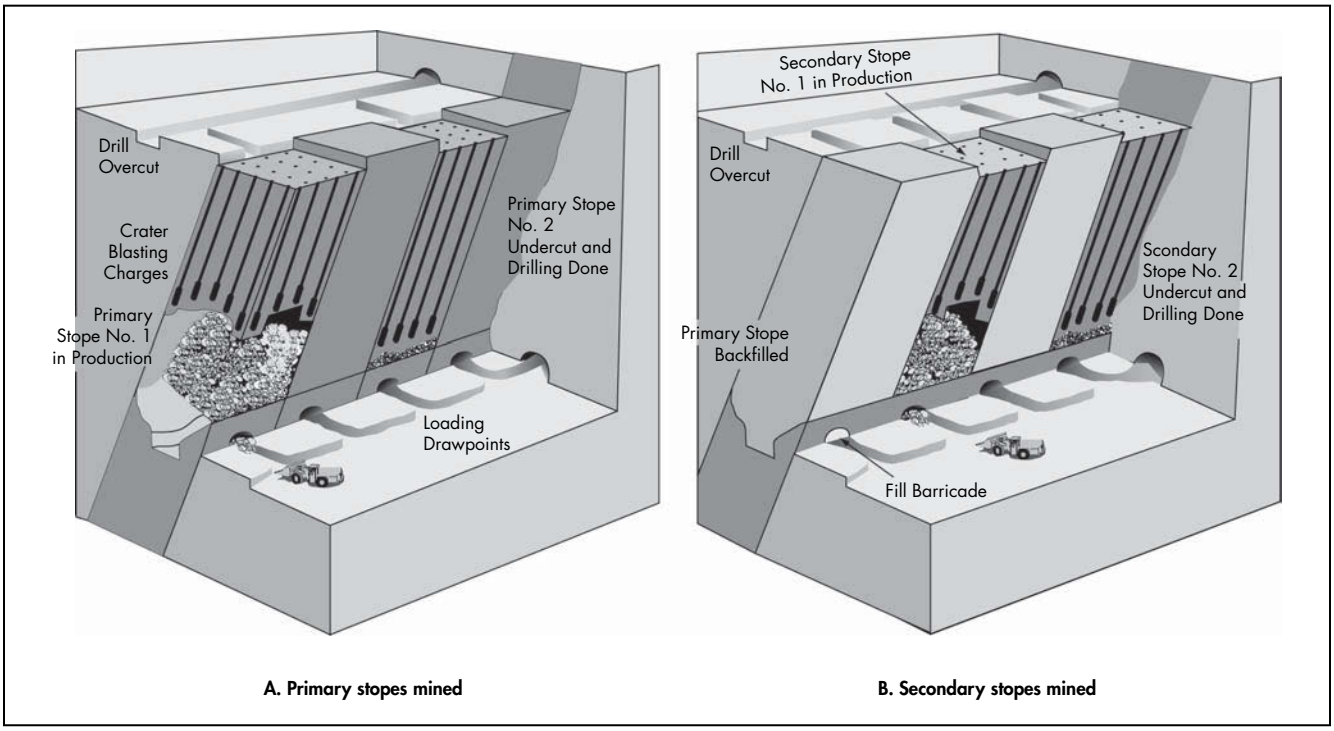
The following variables must be addressed in sublevel stope designs:

- **Size:** The minimum width generally ranges from 3 m to 6 m; however, in isolated cases it reaches 1.5 m (Clark and Pakalnis 1997) and lower (0.8 m). The width is governed by the production blast pattern, which, with the use of 50-mm blastholes, is typically 1.2 × 1.2 m, and the stope layout is based on this spacing (Figure 13.4-6).
- **Shape:** The shape is preferably tabular and regular in shape from level to level.
- **Dip:** The dip is preferably greater than the angle of draw, which typically is in excess of 50° in practice. The concern also is that a shallow hanging wall dip will result in a less-stable HW configuration because of gravity influences and increased wall exposures between vertical stope horizons, all resulting in increased potential for external dilution.



Source: Hamrin 2001.
Figure 13.4-4 Long-hole or blasthole stoping

- **Geotechnical:** This requires a moderate to strong ore strength and generally a competent HW-FW as these will be exposed and affect the level of external dilution. The ore will determine the potential pillar sizes, hole squeeze, and block size that affect production stope productivity (Pakalnis 2002).
- **Stope spans:** Since this is a nonentry method, stope spans can be larger. The span should be designed to control



Source: Hamrin 2001.

Figure 13.4-5 Transverse stoping

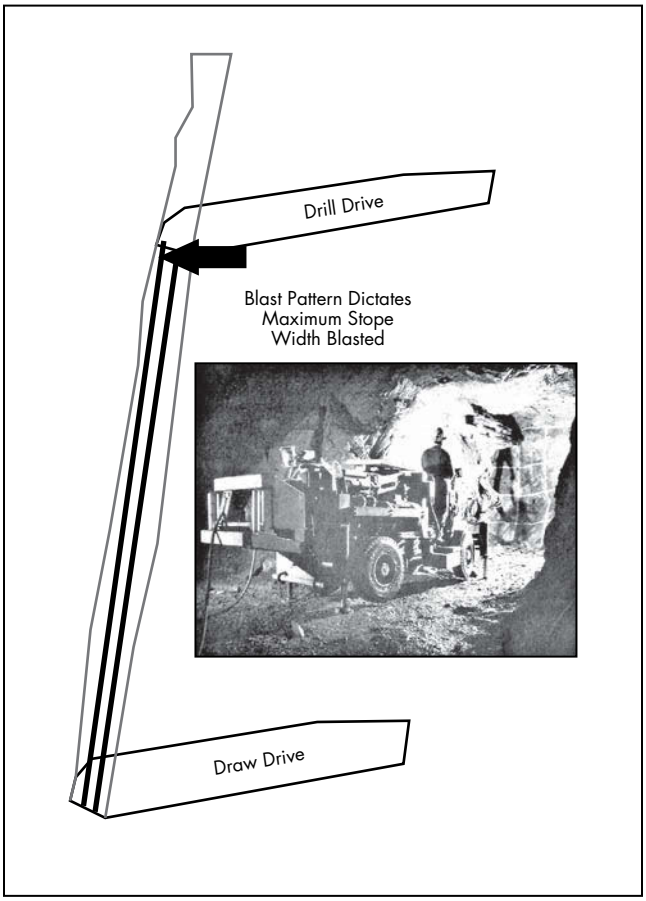


Figure 13.4-6 Sublevel stope layout

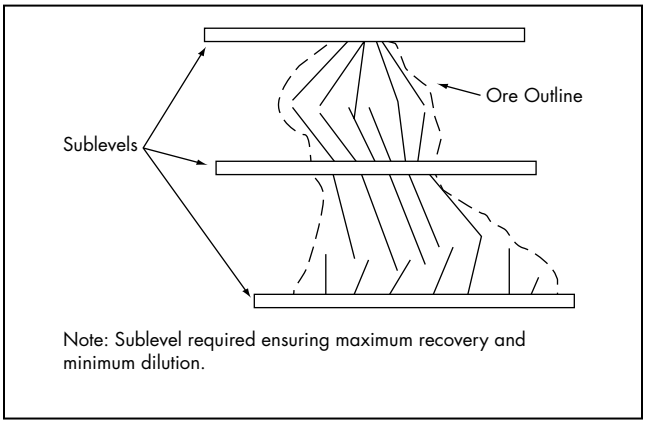


Figure 13.4-7 Geometric constraints

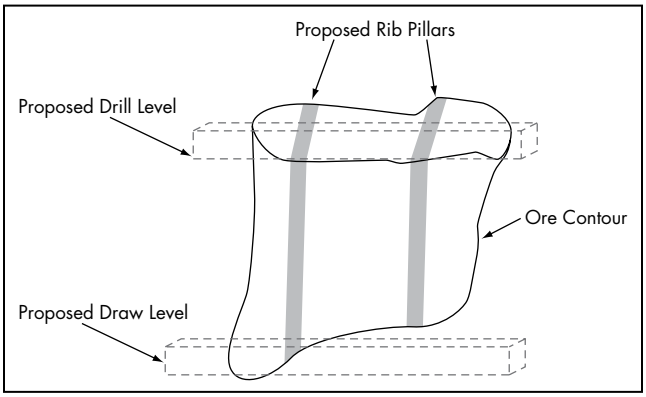


Figure 13.4-8 Initial stope planning

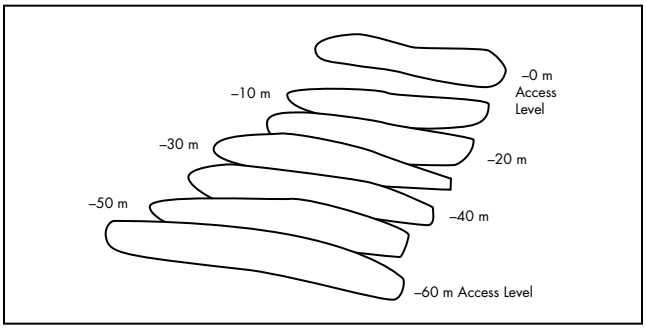


Figure 13.4-9 Composite plan section of ore body used to design blast layout

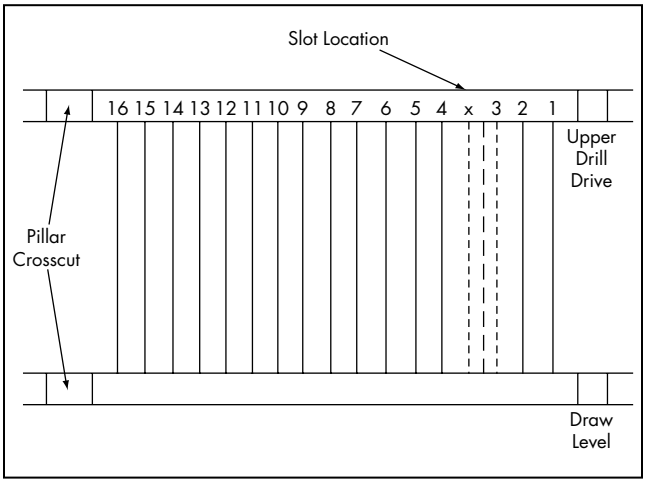


Figure 13.4-10 Longitudinal section of blasthole layout

external dilution and avoid stope collapse and air blast. Span length is governed by HW rock mass quality and generally is in the range of 30+ m with the stope height (inclined) in excess of 30 to 60+ m.

- **Pillar size:** The purpose of the pillars is to support the crosscuts and divide up the stopes. The size of the pillars is dependent on induced stresses, structure, rock mass, and operational constraints.
- **Selectivity:** Selectivity is limited because waste zones can be incorporated as pillars. Changes in ore-body geometry outlines are difficult to address unless the ore body narrows to the next pillar or sublevel where the drill pattern can be modified (Figure 13.4-7).

DESIGN CONSIDERATIONS FOR SUBLEVEL STOPING

General Design Guidelines

The design of a sublevel stope starts with an engineered layout that incorporates the geometry of the stope, stope span, stope height, pillar dimensions, drill levels, and draw levels (Figure 13.4-8). This layout is then superimposed upon the ore contours (plan) as defined from the upper drill drive to the lower draw level horizon. The example shown in Figure 13.4-9 is a schematic of a 60-m long-hole stope (151-mm blastholes) with geologic contour intervals shown every 10 m in the plan.

The resultant longitudinal composite is shown in Figure 13.4-10 employing a ring burden (distance between drill rings) of 3 m from pillar/stope boundary to pillar/stope boundary.

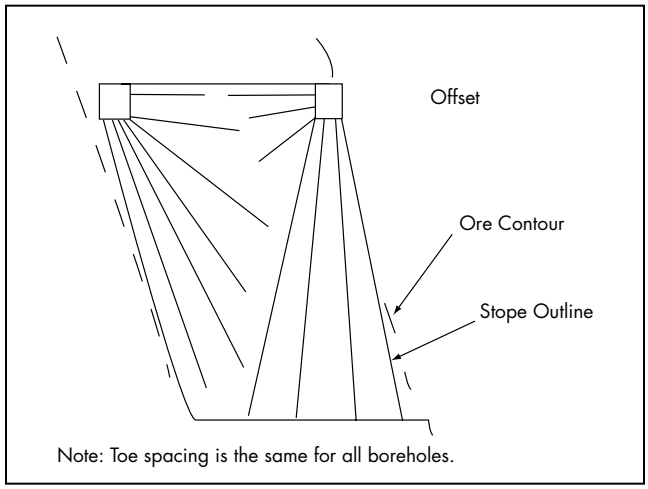


Figure 13.4-11 Ring section

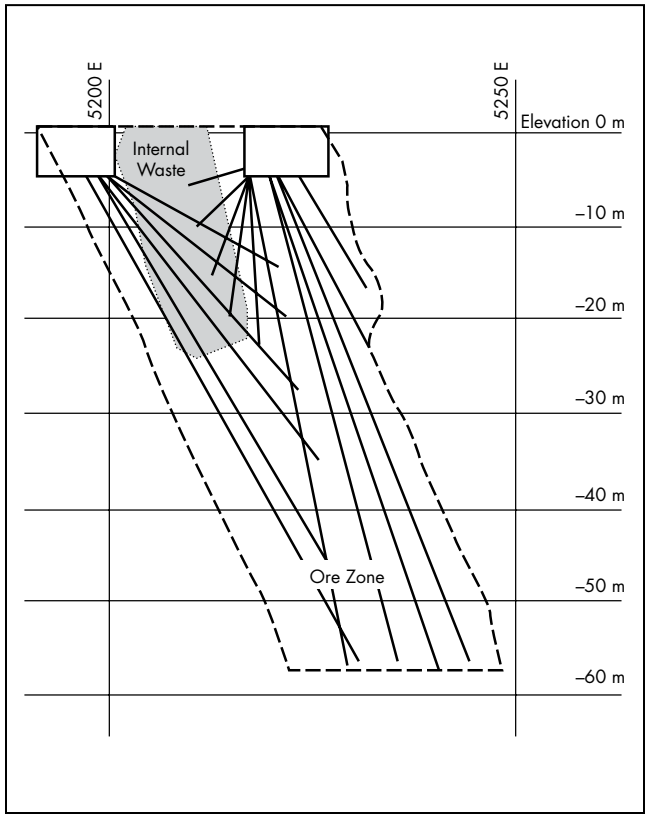


Figure 13.4-12 Engineered ring section, looking north (example)

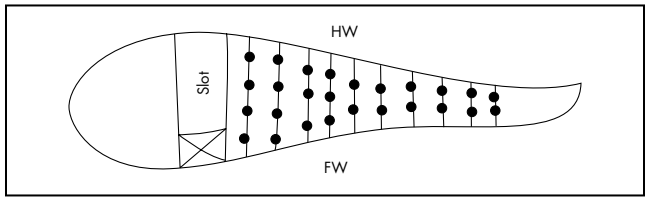


Figure 13.4-13 Location of slot raise (plan view)

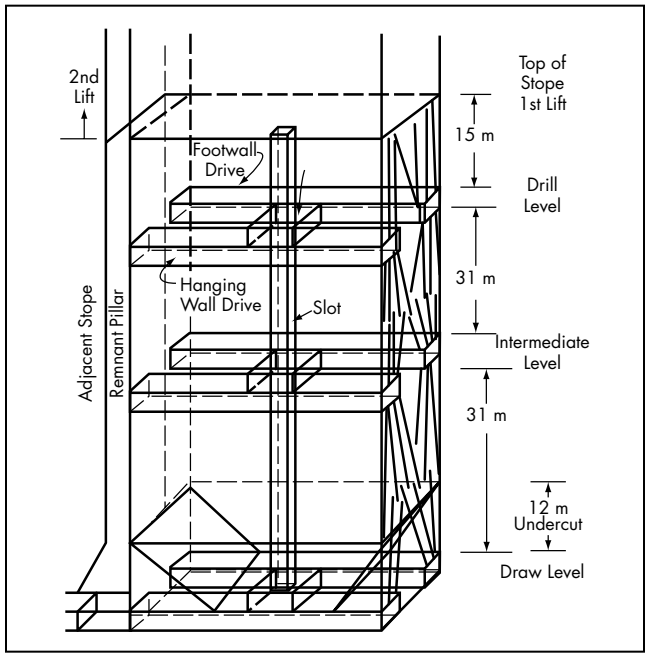


Figure 13.4-14 Sublevel open stope

The ring section is shown in Figures 13.4-11 and 13.4-12 with toe spacing (distance between toes of blastholes) of 4.2 m. The ring section incorporates the geological ore outline as defined by the geologic level contours with the stope outline coinciding with the drilled-and-blasted layout.

Development Considerations

Sublevel stoping uses long-hole drilling employing extension drill steels to achieve the appropriate blasthole depth. When ring drilling is used, the entire cross section of the stope is drilled with holes that radiate from the drill drive. The drilling pattern is matched to the shape of the ore body and location of the drill drift. Parallel holes are drilled when the drill drive can be silled out from the HW-FW, but this largely is constrained by the stability of the exposed working back. Two principal drill systems exist: top hammer and in-the-hole hammer. Both require long-hole rock drills equipped with extension steel in 1.2–1.8-m-long sections. Top hammer drills are more suited for narrower ore bodies (sublevel stoping), while ITH hammers are more suited for wider ore bodies (long-hole stoping). These will be discussed in a later section.

The blast layout for the individual rings will incorporate the ring number, hole number on that particular ring, the amount of explosive required (kilograms), delay interval, angle of hole to be drilled, length of hole to be drilled, and the depth of collar (stemming) to be used.

A slot raise must be developed in order to accommodate the swell of the blasted muck. It generally is developed at the extremity of the stope as shown in Figure 13.4-13, and subsequently the slot raise is enlarged FW to HW to open up the area for blasting. Generally, one cannot blast rings into a narrower void, so the slot should be located in the largest area of the stope.

Ore Handling Considerations

Ore handling in sublevel stoping involves removal of the ore at the bottom of the stope, and historically it involved track and/or slushers to remove the muck. This process is now conducted largely by trackless mining equipment such as scoop trams used for drawpoint loading into mine trucks and/or orepasses as shown in Figures 13.4-3 and 13.4-4.

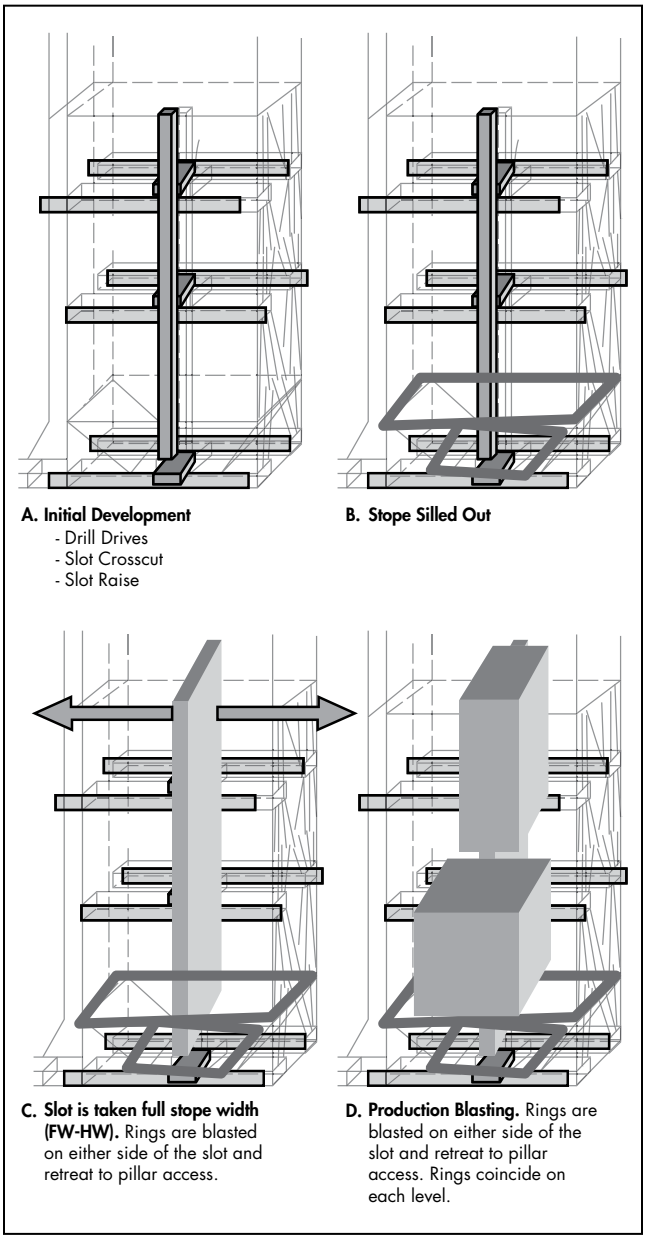


Figure 13.4-15 Sublevel open stope development

Sublevel Stoping

Sublevel stoping design is schematically shown in Figure 13.4-14, and the sequence of development and extraction sequencing is shown in Figure 13.4-15. The dimensions noted in the figures are typical of sublevel stoping dimensions and are employed solely to assist in the description of the method and not intended for design, as the dimensions of a stope are based on the geometry of the ore body and operational constraints.

This mining method employs sublevels located approximately 20–30 m apart. The distance between sublevels is largely governed by the length of hole that can be drilled with

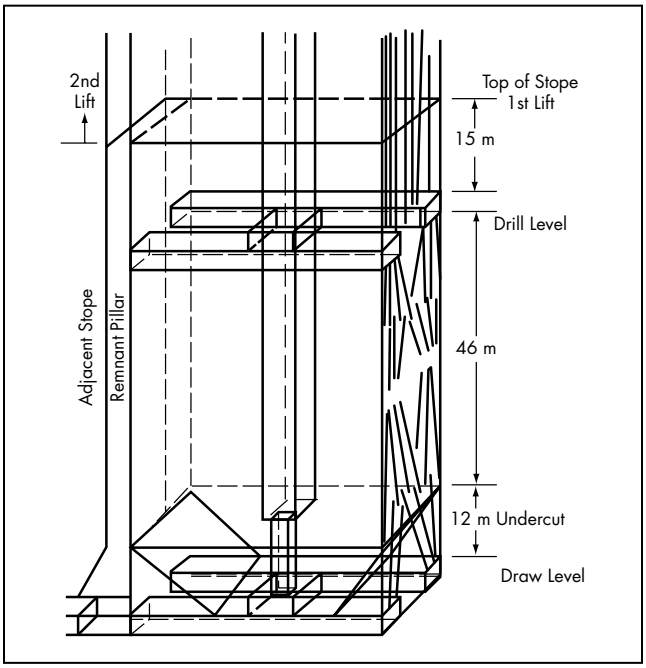


Figure 13.4-16 Long-hole open stoping

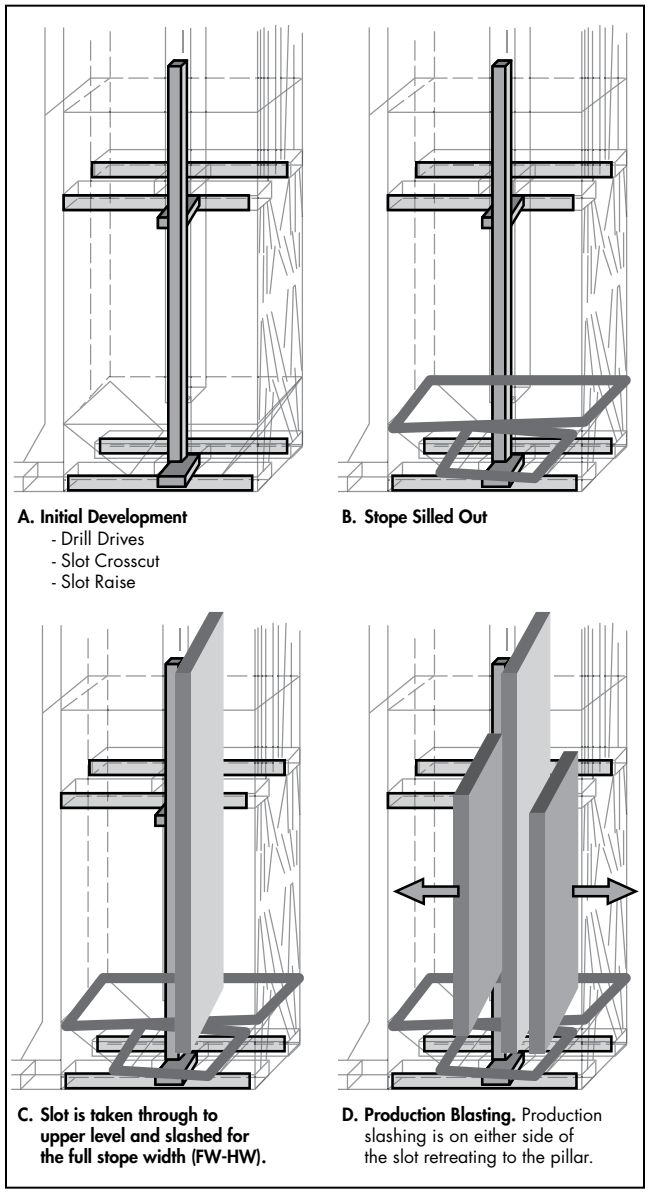


Figure 13.4-17 Long-hole open stoping development/mine sequence

minimal drill-hole deviation (under 2%). The drill-hole diameter ranges from 50–75 mm using top hammer drills, which restricts the length of the hole to generally under 30 m with blasthole burden and toe spacing between approximately 1 × 1 m and 2 × 2 m (typically 1.2 × 1.2 m). Modern tube drills (top hammer) at 100 mm in diameter are able to drill 35–40-m-long holes.

Generally, if the stope width (HW-FW) is greater than 15 m, an FW and HW development drive as shown in Figure 13.4-14 is used; otherwise, only a center drive in the middle of the stope is developed.

The initial development is shown in Figure 13.4-15A, whereby the drill drives, slot x-cut, and raises are driven. The drill drives are comprised of the draw level, intermediate level, and the upper drill level. The undercut (Figure 13.4-15B) is silled out for a vertical height of approximately 12 m above the draw level. The height of undercut or void can be minimized through the use of programmable detonators, ensuring that sufficient void is created for the subsequent blast. The undercut serves the purpose as well to ensure breakthrough of the holes from the upper drill drive. A 2 × 2-m slot is bored/blasted to the full length of the level above the upper drill level (Figure 13.4-15B), which is subsequently slashed to 3.7 × 3.7 m for the full stope height and width from FW to HW (Figure 13.4-15C) to provide sufficient void space for the subsequent rings to be mined. Production blasting (Figure 13.4-15C) is comprised of individual rings blasting into the void for the full stope width on either side of the slot. This assumes pillar access exists on either side of the slot. Normally the production rings blasted from the intermediary level correspond to a similar set of rings on the upper drill level (Figure 13.4-15D) to ensure that a void from draw level to upper drill level exists. The geometry shown in Figure 13.4-14 employs a ring burden of 1.5 m and toe spacing of 2.1 m. The stope is normally drilled off prior to commencement of blasting, and only the holes that are scheduled for the blast are loaded. The upholes from the intervening levels must ensure interleaf coverage of approximately 1 to 2 m. The example shown in Figure 13.4-15 uses 15-m-long drill holes with uppers and downholes, and 1–2-m interleaf coverage.

Long-Hole Stoping

Long-hole (blasthole) stoping development is shown in Figure 13.4-16 with the sequence of development and extraction shown in Figure 13.4-17. The subsequent examples given are typical of long-hole stoping dimensions and are employed solely to assist in the description of the method and not intended for design, as these dimensions change based on the geometry of the ore body and operational constraints.

Long-hole stoping largely eliminates the intermediary level with the draw and drill horizon interval governed by the length of hole that can be drilled with minimal drill-hole

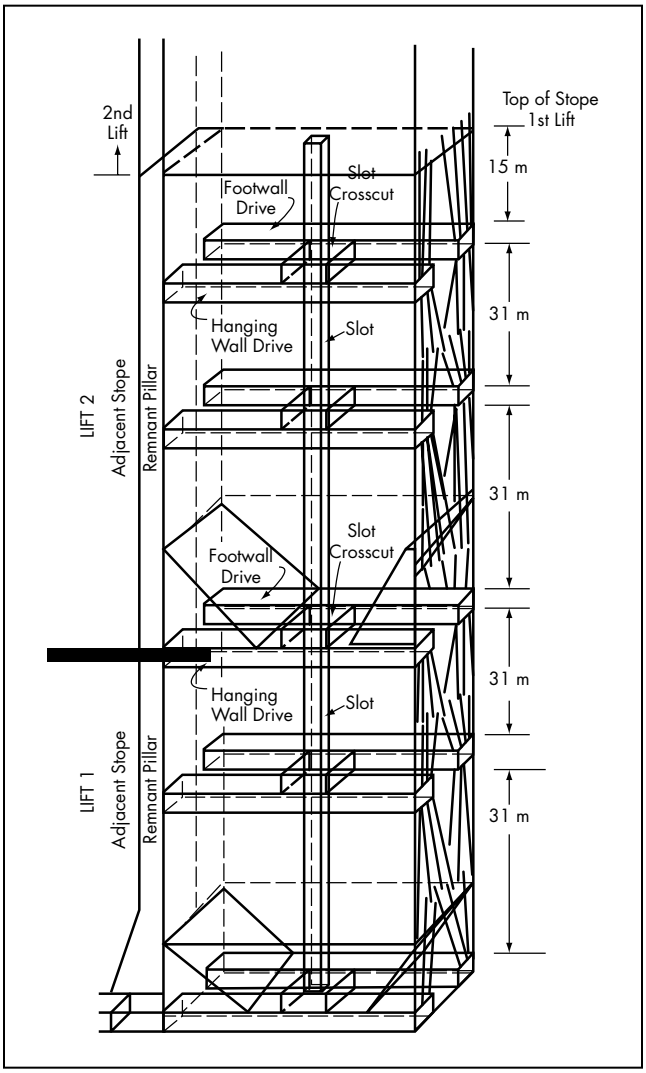


Figure 13.4-18 Piggyback stope

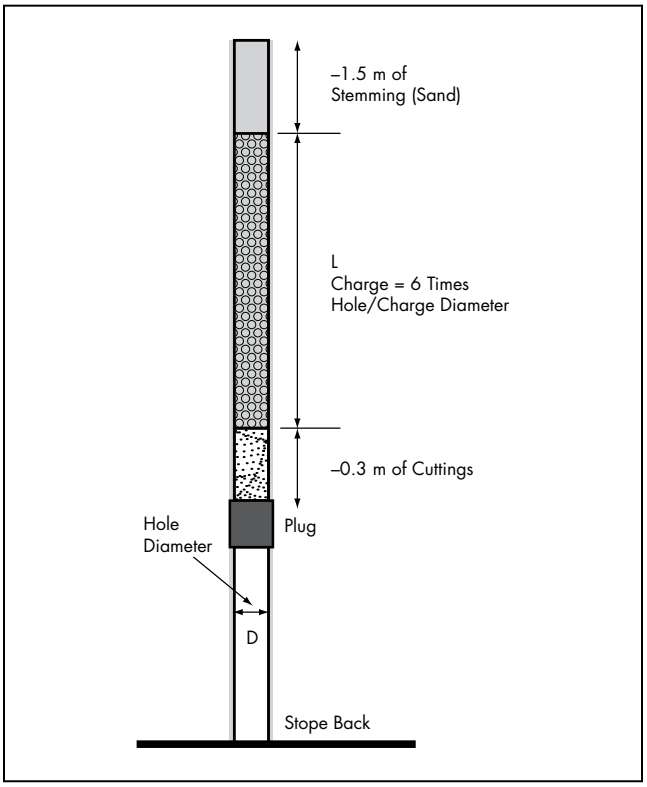


Figure 13.4-19 Typical cross section of a VCR-charged hole

deviation (under 2%). The drill-hole diameter ranges from 75 to 150 mm using ITH hammer bits, thereby enabling the lengths to approach 30–60 m in length with blasthole burden and toe spacing approximately 3–4 m^2 (3 × 3 m). The development is as shown in Figure 13.4-17. Generally, if the stope width (HW-FW) is greater than 15 m, an FW and HW development drive is used as shown in Figure 13.4-16; otherwise, a center drive is driven in the middle of the stope.

The initial development is shown in Figure 13.4-17A, whereby the drill drives, slot crosscut, and raises are driven. The drill drives are comprised of the draw level and the upper drill level as the intermediary level has been removed. The undercut (Figure 13.4-17B) is silled out for a vertical height of ~12 m above the draw level. A 3.7 × 3.7-m slot is bored/blasted to ~12 m above the upper drill level (Figure 13.4-17B), which is subsequently slashed to 6.1 × 6.1 m for the full stope height and width from FW to HW (Figure 13.4-17C). Production blasting (Figure 13.4-17D) is comprised of individual rings blasting into the void for the full stope width on either side of the slot. This assumes pillar access exists on either side of the slot. The geometry shown in Figure 13.4-16 employs a ring burden of 3 m and toe spacing of 4.2 m with 150-mm-diameter blastholes. The stope is normally drilled off prior to commencement of blasting. The example shown in Figure 13.4-17 uses 46-m-long downholes and 15-m upholes.

A variation in the above sublevel and long-hole mining methods is to use nonconsolidated backfill above the upper drill drive of Lift 1 and subsequently drawing out from the level that serves as the draw horizon for the level above (Lift 2) as shown in Figure 13.4-18. This negates the need for cones in ore and consequently maximizes ore recovery as shown in Figures 13.4-15 and 13.4-17. The cones can be eliminated with the use of remote mucking equipment; however, the equipment will be traversing under potentially extended, unsupported spans (see Figure 13.4-5).

Vertical Crater Retreat

As shown in Figure 13.4-18, a VCR is a variation of long-hole open stoping where the "free face" is not a vertical slot but a "flat back" at the base of the block to be mined. Spherical charges are used to break the ore into slabs as shown in Figure 13.4-5 and have a length/diameter (L/D) ratio of 6:1. Field testing has shown that a ratio of explosive column length (L) to hole diameter (D) of 6 or less will behave similarly to a spherical charge. Blasting is carried out in horizontal slabs with only the swell being mucked at the drawpoint. This is a form of shrinkage stoping where the broken stope muck provides passive support to the stope walls. The ore is recovered at the base of the stope through drawpoints. Similar requirements and constraints to that of sublevel stoping exist except for the need for a competent HW-FW due to the option of maintaining the stope full of muck.

Development is similar to that of long-hole stoping, requiring an upper drill horizon and draw level, and it is

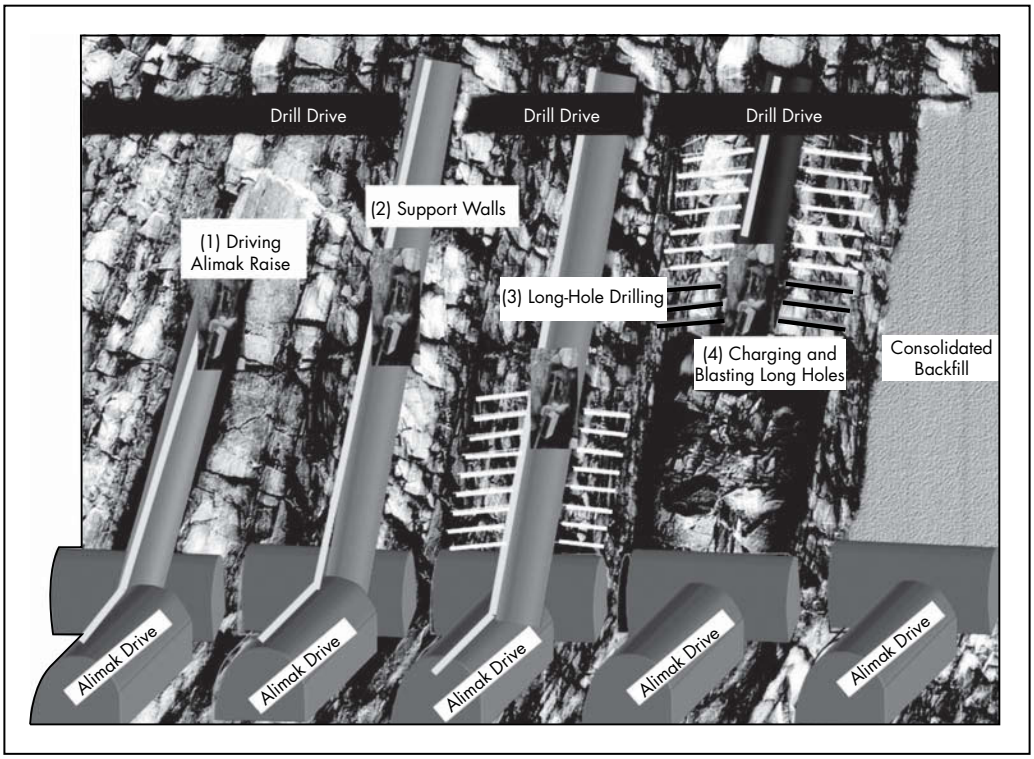


Figure 13.4-20 Alimak raise mining

generally recommended to sill out at the drill horizon to provide drill coverage for the entire block.

The vertical separation between drill and draw level is largely a function of the ore regularity and drill accuracy as detailed in general for the long-hole mining method. The dimensions are similar to that of long-hole mining where ITH drills are employed with heights ranging from 30–60 m and 75–150-mm drill diameters are used. A typical loaded blasthole for VCR is shown in Figure 13.4-19 employing a single deck charge.

Advantages of the VCR are the high productivity associated with this bulk mining method and the ability to mechanize. The ability to only muck the swell enables support to the stope walls. An advantage of this method over shrinkage is the nonentry and high mechanization associated with VCR.

Disadvantages of this method are the extensive pre-stope planning and development that is required prior to commencement of production mining, as the stope must be largely drilled off prior to bench blasting. Similar disadvantages to that of shrinkage mining exist in having the broken ore within the stope until the end of mining of the block.

VARIATIONS ON SUBLEVEL STOPING

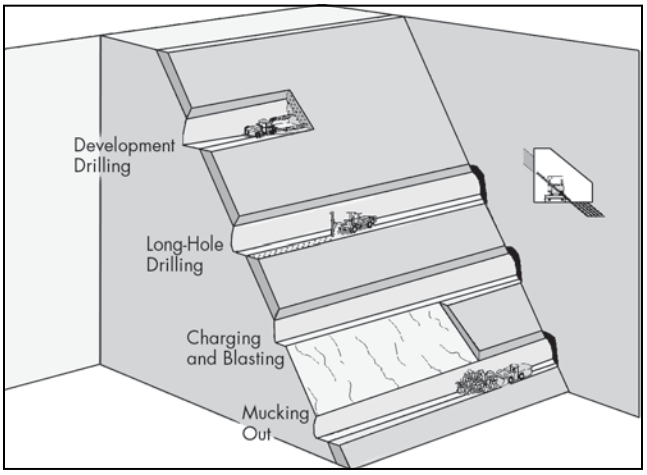
The sublevel mining method has variations that have been implemented and will be discussed in the context of its similarity with sublevel stoping.

Vein Mining

Vein mining—also termed *Alimak mining*—has been employed within narrow vein ore bodies as detailed in the Namew Lake mine (Canada) case study (Madsen et al. 1991). Access to the ore is gained by a bored raise/Alimak such as that shown in Figure 13.4-20. The diameter of the raise is approximately 2–3 m and extends from draw level (Alimak drive) to upper drill drive as shown in Figure 13.4-20 (item 1 in the figure) and spans the length of the ultimate stope span with similar constraints as those detailed for sublevel stoping. Support may be in the form of cable bolts (item 2) in the HW providing the final wall support on stope extraction. The ore is drilled laterally by conventional drills, long-hole jumbos, or other methods and ranges from 5 m to 15 m in length to the adjacent stope (item 3) with blasting from the draw level vertically to the upper drill level (item 4). An intervening pillar may be left between stopes or the stope mined from one Alimak raise to the next depending on the geotechnical constraints. The major advantage is the ability to mine narrow ore bodies with minimal horizontal development. The vertical height of the Alimak is largely limited by operational and geotechnical constraints and reaches heights of 30 to 100 m. Blasthole sizes are generally 50–75 mm with burdens and spacing similar to that of sublevel stoping (1–2 m^2).

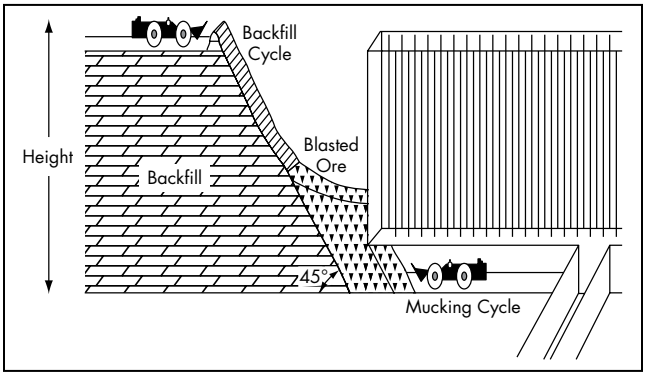
Transverse Open Stoping

Variations of sublevel stoping with delayed fill are shown in Figure 13.4-5. This mining method is largely used for stope widths in excess of 20 to 30 m or as dictated by geotechnical stable back spans; otherwise, conventional longitudinal or strike mining is used. Figure 13.4-5 shows the objective is to recover the secondary pillars between the primary stoping blocks, which can be excavated by sublevel stoping (general) and subsequently filled with consolidated fill that can be comprised of hydraulic fill, paste, or cemented rock fill. Mining of the secondaries occurs after curing of the primaries to a strength that is able to withstand minimal dilution. Generally, the binder content ratio is 30:1 to 20:1 (fill to cement by volume). Alternatively, a permanent pillar is left behind to confine



Source: Hamrin 2001.

Figure 13.4-21 Longitudinal mining without fill



Source: Caceres 2005.

Figure 13.4-22 Longitudinal mining with fill (Avoca)

the unconsolidated fill with only primaries excavated along the strike. With this variation the secondaries are narrow pillars left behind (approximately 3–5 m). A disadvantage of this method is its inability to follow the variations of an irregular hanging wall dip.

Longitudinal Mining

Figure 13.4-21 shows sublevel extraction employing mucking along the strike (retreat). This is a variation of conventional FW drawpoints as shown in Figure 13.4-21. The stopes with no fill are as shown in Figure 13.4-21 and with delayed fill are as shown in Figure 13.4-22. The delayed fill method of longitudinal mining is also referred to as Avoca mining. Having longitudinal mucking access requires that remote load-haul-dump (LHD) equipment be used. This method is also referred to as sublevel benching.

CONCLUSION

Sublevel stoping accounts for more than 60% of all underground production in North America. This is largely due to the developments of extension steels, hollow tube and special long-hole rock drills, and ITH drilling techniques requiring less development and greater production capacities. Several variations exist; however, characteristic to this method is the development from a top drill drive and removal of muck from a draw level below for a steeply dipping stope. The variations of the method are selected to suit the ground conditions and operational requirements of the mine.

An essential part of sublevel stoping is the stope extraction sequence. The extraction sequence is governed by the development, rock mechanics, tonnage requirements, and, if applicable, fill cycle.

Sublevel stoping is a safer mining method because the operator is never within the stope under the unsupported back. Further, the mining method works on a retreat pattern where the equipment and operator work under a supported back. This mining method is suitable to modern hauling equipment including the use of remote LHD units where the operator is removed from any potential hazard associated with the stope.

An important safety consideration with open stoping is to ensure that drawpoints remain full above the brow of the stope. Adhering to this safety standard largely eliminates the risk of potential air blast due to hanging wall collapse.

The main advantage of sublevel stoping is the efficiency associated with drilling, blasting, and loading operations as they can be performed independently from each other. A high potential exists for mechanization with moderate to high productivities of more than 25 t per worker-shift.

The main disadvantage is the complicated and comprehensive development that is needed and the requirement for regular tabular ore geometries.

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CHAPTER 13.5

Cut-and-Fill Mining

George Stephan

DEFINING CHARACTERISTICS

Cut-and-fill is the broad descriptive term applied to mining methods requiring that excavated voids be filled with barren material to facilitate the continuation of ore production. This fill is required to provide support for subsequent openings or to provide a working platform for further mining. Cut-and-fill methods may be applied in conjunction with other conventional methods, such as blasthole stoping, or they may be methods that stand alone, such as drift-and-fill.

Classic cut-and-fill mining involves the successive mining of horizontal or inclined slices upward through a relatively narrow and subvertical tabular ore body. This is followed by the placement of uncemented waste rock or hydraulically placed sand fill to create a new, higher working level to gain access to ore above and to support the ground below. Raises maintained up through the fill (and occasionally down from upper levels) provide for access, ventilation, ore removal, and the drainage of water from the fill material.

Modern variants may use paste fill and typically use ramp access to allow the use of mechanized mining equipment. These techniques preclude the need for raises.

None of the cut-and-fill methods described herein can match the productivity or the low mining cost of mass mining methods. Cut-and-fill methods are chosen because they are more economically attractive than other extraction methods available for the same situation.

FAVORABLE CONDITIONS

Cut-and-fill methods are applicable to a wide variety of situations, including the following:

- Ore zones are irregular in shape and orientation.
- Ore grade is high and dilution control is critical.
- The precise contacts between ore and waste are structurally critical but not readily visible.
- The waste rock is weak.
- Ore zones are large, but their rock quality is weak.
- Localized underground stability is required.
- Surface disturbance must be minimized.
- The value of the ore makes recovery of support pillars economically viable.
- The reduction of surface waste storage is important.
- The need exists to advance a working platform for the upward mining of the ore body.

Cut-and-fill methods are favored when the ore value is relatively high and ore recovery rate with minimal dilution cannot be satisfactorily accomplished by open stope mining or caving methods. If the openings are small enough and the ground conditions are competent, irregular ore boundaries can be mined with open stope methods without backfill. However, when the openings are larger and the ground conditions are less favorable, backfilling becomes necessary for safe economic production.

Fill methods are useful when regional stability is needed. This is the case where large empty stopes could adversely affect the stability of permanent mine access drifts, shafts, and raises. These stresses could also affect new mine production stopes and surface infrastructure.

There is also the advantage that fill can provide a relatively inexpensive and convenient way to advance a working platform to maintain access to the ore for drilling, blasting, and mucking. Another advantage is that ventilation air can be managed more efficiently because vacant and unused mine openings are closed by backfill.

LIMITATIONS

These factors limit the applicability of cut-and-fill:

- The availability of a sufficient quantity of a suitable fill material type
- The cost of binding agents (if required)
- Production, preparation, transportation, and placement cost of the fill material
- Storage and reclamation facilities to match the mining cycle demand
- Congestion and interruption of production mining activities

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The principal drawback for the cut-and-fill methods is the additional cost of producing, preparation, transporting, and placing the backfill.

Not all readily available fill materials are appropriate; the backfill material must match its intended use. The material that is most readily available and least expensive may not have characteristics that would allow its use as fill material. The material most readily available also may not be usable in the stope designs dictated by the size and shape of the ore body. Further, the least expensive delivery system may not be practical.

Assumptions about fill availability should be verified early in mine planning. Suitable aggregate for cemented rock fill may not be available near the mine site. Both sand fill and paste fill generally make use of tailings from an on-site mill. For sand fill, however, sufficient coarse fraction may not be available within the tailings, either because of the fineness of the grind or because of the need to supply coarse fraction for tailings dam construction. Conversely, the grind may be too coarse to provide sufficient fines for paste fill.

In addition to the added cost, backfilling operations can result in congestion, maintenance, and cleanup problems underground. Cut-and-fill imposes limitations on the sequence and timing of production that, if not carefully controlled, may take precedence over the maximization of production grade. Usually, conventional ground support in the form of rock bolts, wire mesh, and shotcrete still will be required to temporarily support the stope openings during production mining.

The cost of producing and placing fill requires consideration of the mining method trade-offs. Unless the deposit simply cannot be mined by any nonfill method, the additional cost and the advantages of using fill must result in additional ore production (increased recovery), reduced dilution, and/or cash cost reductions sufficient to justify the additional cost of the fill and its preparation and placement.

TYPES OF CUT-AND-FILL METHODS

There are numerous cut-and-fill methods and many hybrid combinations. The following is a basic set of descriptors:

- Vertical progression of the mining
  - Underhand (mining beneath the backfill)
  - Overhand (mining on top of the backfill)
- Mining method
  - Drift-and-fill
  - Postpillar
  - Bench-and-fill
  - Blasthole stoping
  - Uphole, flat-back slicing, or back stoping
- Type of fill
  - Cemented rock fill
  - Uncemented rock fill
  - Sand fill (hydraulic)
  - Paste fill
- Timing
  - Concurrent (placed during the cycle of stope advance)
  - Delayed (placed after the excavation of the stope is complete)

Cut-and-fill mining is extremely flexible, and various combinations of mining methods and fill materials can be used to cope with specific mining situations. However, some combinations are not feasible, such as underhand drift-and-fill with uncemented rock fill. Further discussion regarding possible combinations can be found in the "Cemented Fill Stopes" and "Uncemented Fill Stopes" sections.

METHOD SELECTION CRITERIA

Ore-Body Morphology

The size, shape, orientation, and degree of boundary uniformity of the ore body are the first considerations in selecting a cut-and-fill method. The use of fill for support and for provision of the necessary platform to access the ore provides the operator with great flexibility for adjusting the excavation to irregularities along the boundaries of the ore body. Alternative methods (that do not use fill) may result in unsafe working conditions, excessive dilution, or insufficient recovery. Cut-and-fill methods can be adapted to flat, vertical, undulating, or even en echelon zones and to situations in which the ore-body shape and attitude change quickly along the strike or dip.

Access, Infrastructure, and Development Cost

Typically, overall project cost for access and infrastructure to support cut-and-fill mining is less than the cost for bulk mining methods like block caving. Overall project cost is greater, however, than the cost for room-and-pillar or open stoping.

The use of fill for support, rather than leaving unmined pillars of ore, results in greater recovery of ore because fewer unrecoverable ore pillars are left behind at the conclusion of mine operations. In methods such as shrinkage stoping, much of the broken ore is left (albeit temporarily) in the stope to provide wall-rock support and a working platform from which to mine the whole block of ore. Although the shrinkage ore is withdrawn at the completion of the stope, the use of fill as a working platform and wall-rock support results in a faster return on the cost of mining.

Amenability to Mechanization

The mechanization of cut-and-fill mining has two phases: mechanization of the ore extraction (i.e., the cut) and mechanization of the fill process. The mechanization potential for ore extraction with cut-and-fill methods varies widely. The ground conditions that dictate the use of cut-and-fill mining may also limit drift dimensions or, alternatively, may require flexibility to adapt to varying stope conditions. This reduces the size of trucks and loaders that can be used. Options associated with production and delivery of fill are covered in Chapter 13.6. By itself, the use of fill seldom enables an increase in the level of mechanization.

Dilution

Cut-and-fill methods increase the extractable yield of ore bodies and reduce dilution from boundary overbreak. However, the effect of dilution from fill must be considered in any economic analysis. Fill dilution generally carries no recoverable value, whereas overbreak into country rock may carry at least some value because of its proximity to the ore. Dilution from fill can come from stope ribs where the adjacent stopes have been previously backfilled. If the stope in production is above a previously filled stope, the mucking operations will typically remove a layer of fill material from the floor to clean up as much ore as possible. Less fill will be taken if the floor surface is cemented fill. Conversely, ore can be lost in the floor of a stope if ore is used to build a roadway on top of the fill. Fine

ore, which usually contains a significant portion of value, can be lost into the fill from the cut below if the fill is not well consolidated at its placement.

The dilution potential of various fill materials can differ significantly. Uncemented rock and sand fill typically dilute at the highest rates. Dilution is usually less with cemented rock, sand, and paste fills. However, dilution from cementing agents may have an adverse effect in the metallurgical plant. This difference in behavior must be included when evaluating the relative costs of using these fill materials in the mining operation.

Geotechnical Constraints

The in-situ geotechnical environment—including the rock quality of the ore and country rock, geologic structures, ore-body depth, water, and regional stresses—determines the size and shape of the stopes that can be excavated before placing fill. The ground support required to provide a safe working environment is also dictated by the geotechnical situation. Cut-and-fill methods frequently rely on the initial mining of stopes in virgin ore zones. These stopes are called *primary stopes*. They are subsequently backfilled to provide support for the mining of adjacent stopes, which are called *secondary stopes*. The size, shape, and mining sequence of the primary and secondary stopes are dictated by the geotechnical conditions of the ore and country rock and the mechanical properties of the planned fill material. Sometimes it is advantageous to place higher-strength, higher-cost fill in a primary stope and lower-strength, lower-cost fill in a secondary stope. The flexibility to control and differentiate fill quality and properties according to the application is crucial to mine operation, safety, and cost control.

Sill pillars and crown pillars are used to separate stoping zones vertically. The dimensions of these pillars are determined by the geotechnical environment and the stresses induced by mining. If these pillars contain valuable ore, it may be desirable to place highly engineered fill to facilitate the safe recovery of this ore. The objectives are to use cut-and-fill mine planning to maximize the extraction of the ore body and to avoid sterilizing any portion such that it cannot be safely recovered.

Availability of Fill Material

The availability of fill material (rock, sand, or paste) affects the selection of cut-and-fill methods. The quantity, engineering properties, and cost must be matched to the planned use. For example, if the on-site mill tailings are unsuitable, a nearby source of alternative material must be found. Similarly, if the price for cement is unreasonably high, the cost and availability of the various fill alternatives must be weighed against the engineering requirements of the selected cut-and-fill method. The safest and least expensive combination of fill material and cut-and-fill method must be found and continually reviewed.

The use of tailings for fill is often dependent on mill throughput. An unfortunate side effect of using mill tails, particularly in mines with numerous smaller stopes, is that when stopes stop producing to allow for fill placement, they must rely on continued production from other stopes to provide fill material. The design of the mine must include satisfactory storage capacity for fill because it is usually accumulated at a slow, relatively steady rate—typically as a by-product of the milling operation. When fill is required for placement, it must be delivered at a much greater rate, stope by stope. Careful planning is therefore critical to ensure that a balanced mining/filling cycle is maintained throughout the operation. Failure to maintain a balanced cycle can result in fill shortfalls, leading eventually to loss of production until a proper cycle can be reestablished.

CEMENTED FILL STOPES

The inclusion of a binding agent such as cement in the fill is required when the fill must have sufficient strength to provide support for the excavation of stopes located above, below, or adjacent to the stope being mined. The term *delayed* is used to indicate that extraction of all ore in the stope has been completed before the stope is backfilled.

Occasionally, when the value of the ore in the pillars of a conventional room-and-pillar mine is considered high enough, cemented fill is placed around the pillars to permit recovery of the ore. Placement of cemented fill is also sometimes used to facilitate the later extraction of ore from sill pillars where the fill material is normally uncemented.

The use of hydraulic and cemented fill can cause significant problems with level cleanup, sumps, settlers, and pumps. This is usually worse with cut-and-fill stoping than with sublevel open stoping because there are many more points of drainage to manage.

Blasthole Stoping with Delayed Backfill

Blasthole stoping with delayed backfill is a method that can be applied to ore bodies that are generally tabular and vertical or near vertical. The rock quality of both the ore and the waste needs to be fair to good. Cemented fill for the primary stopes is required when a primary/secondary plan is pursued. Cemented fill allows secondary stopes to be mined subsequently. The secondary stopes may be backfilled partially or completely with uncemented fill.

Longitudinal Blasthole Stoping with Delayed Backfill

Where the ore zone is narrow and the long axes of the stopes are parallel to the strike, the stope orientation is called *longitudinal*. Occasionally the ore is contained in en echelon lenses: narrow parallel zones of ore separated by narrow parallel zones of waste. These conditions require sequences of longitudinal stopes followed by backfilling to prevent stability problems and dilution from the collapse of the waste material between ore lenses (see Figure 13.5-1). The width of the stopes is generally the distance between hanging wall and footwall. The maximum length is generally limited by geotechnical considerations. To minimize dilution from overbreak or slough when the hanging wall is weak, the installation of mechanical support (bolts, wire, shotcrete, or cable bolts) may be required.

Transverse Blasthole Stoping with Delayed Backfill

Where the ore zone has sufficient width, the long-axis orientation of the stopes may be perpendicular to the strike. These are *transverse stopes*. Geotechnical considerations dictate the transverse orientation as well as the stope dimensions. The sequence of lateral and vertical, primary and secondary stope mining, backfilling, and curing requires careful planning (see Figure 13.5-2). After mining begins, the plan seldom permits significant changes; it becomes difficult to adjust the production rate and grade.

Drift-and-Fill

Drift-and-fill is the term applied to projects in which the ore is extracted by excavating parallel drifts through the ore zone and backfilling them before mining the adjacent drift. The drifts are

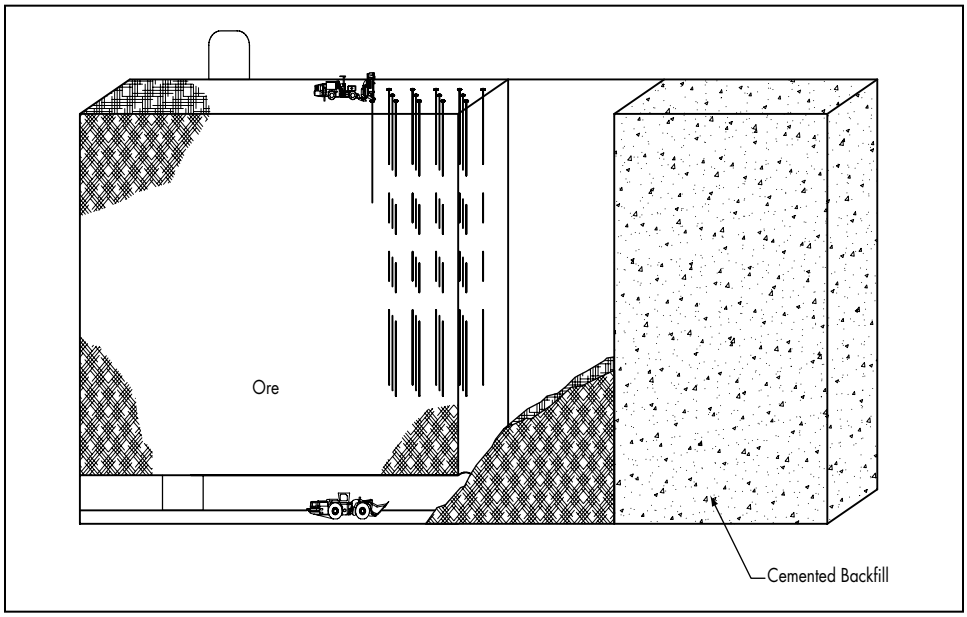


Figure 13.5-1 Typical longitudinal blasthole stoping with delayed backfill

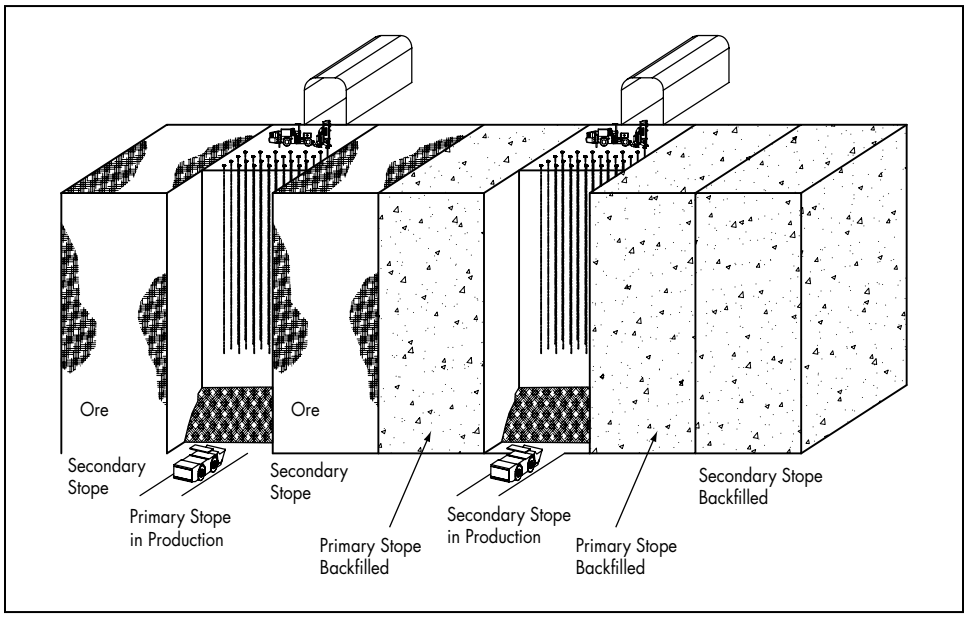


Figure 13.5-2 Typical transverse blasthole stoping with delayed backfill

usually alternately filled with cemented and uncemented rock fill. The orientation of the drifts mined on a level can be longitudinal, transverse, or diagonal to the strike of the ore body. For additional strength, the orientation is sometimes alternated between levels. Excavation of the ore by drifting is less productive and more costly than excavation by blasthole stoping. Therefore, drift-and-fill is usually employed where ground conditions are too weak and the ore boundaries are too irregular for blasthole stoping. Because of its higher associated operating costs, drift-and-fill is best used where the ore grade is high.

Overhand Drift-and-Fill

Overhand drift-and-fill is a bottom-up mining sequence (see Figure 13.5-3). This method is useful where the ore occurs in relatively thin, shallow dipping lenses with weak back or wall rock. The bottom-up sequence may also be desirable because of production grade considerations. Ideally, this method would be used where the rock quality of the ore in the back is not sufficiently high to permit postpillar stoping. This method permits backfilling of secondary stope drifts with uncemented fill, thus reducing costs. The first or lowest cut requires cemented fill in both primary and secondary drifts if mining blocks from below are scheduled to mine up to a previously mined block.

Underhand Drift-and-Fill

Underhand drift-and-fill is a top-down sequence (see Figure 13.5-4). The use of underhand drift-and-fill stoping

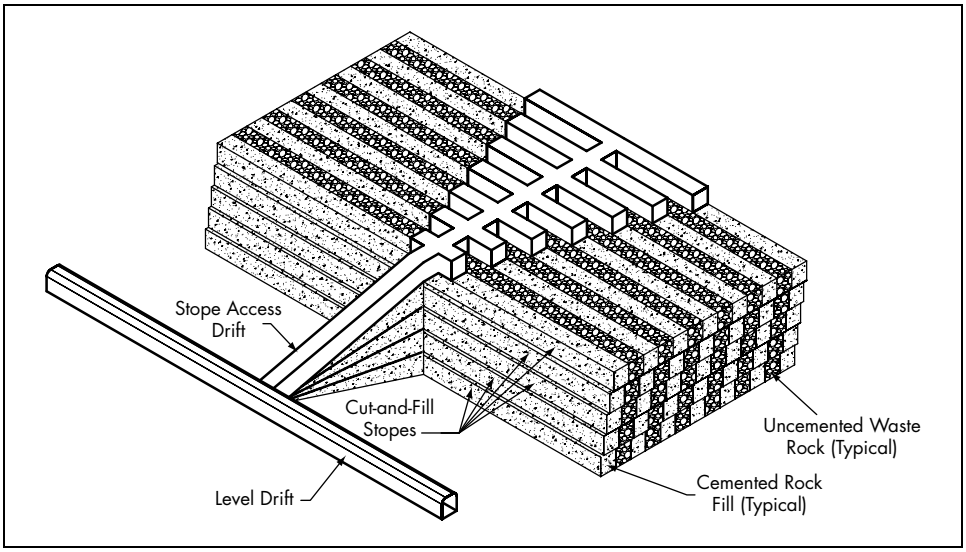


Figure 13.5-3 Typical overhand drift-and-fill

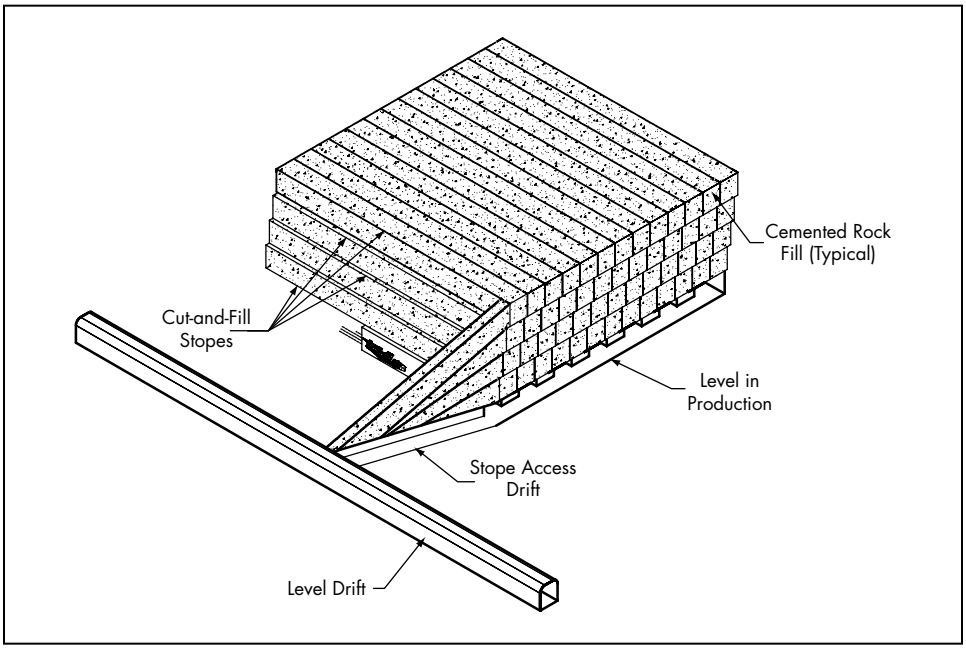


Figure 13.5-4 Typical underhand drift-and-fill

fills the critical need for a method that facilitates the mining of high-grade deposits with weak ore and weak country rock. This method results in the creation of a strong crown pillar of cemented fill beneath which subsequent mining can safely proceed. The initial level may be opened using small drifts that are easily supported. After the upper level is completed and backfilled, succeeding levels below can be opened to more efficient mining widths. Orienting the drifts on each level such that they are not parallel to the overhead drifts reduces the risk of ground falls. The fill material must be carefully engineered and the samples tested regularly. The method may involve the use of mats or tension cables to support and distribute the load above each open drift. The ore zone should be vertical or near vertical to maximize the benefit of employing this method.

UNCEMENTED FILL STOPES

The use of uncemented fill has the advantage of eliminating the cost for cement. Uncemented fill has limitations when used as the primary backfilling material because it lacks the strength of cemented fill. Therefore, it is seldom used where the mining of adjacent stopes or stopes below is planned. The use of uncemented fill dictates a bottom-up mining sequence or the use of substantial sill pillars between levels.

Usually, stope mining with uncemented rock fill has the advantage of being able to use any sort of readily available and inexpensive fill material. Frequently, waste muck from development mining can be used. This relieves the expense of transporting thve waste to the surface for disposal. Alternatively, the rock can be quarried on the surface and delivered to the

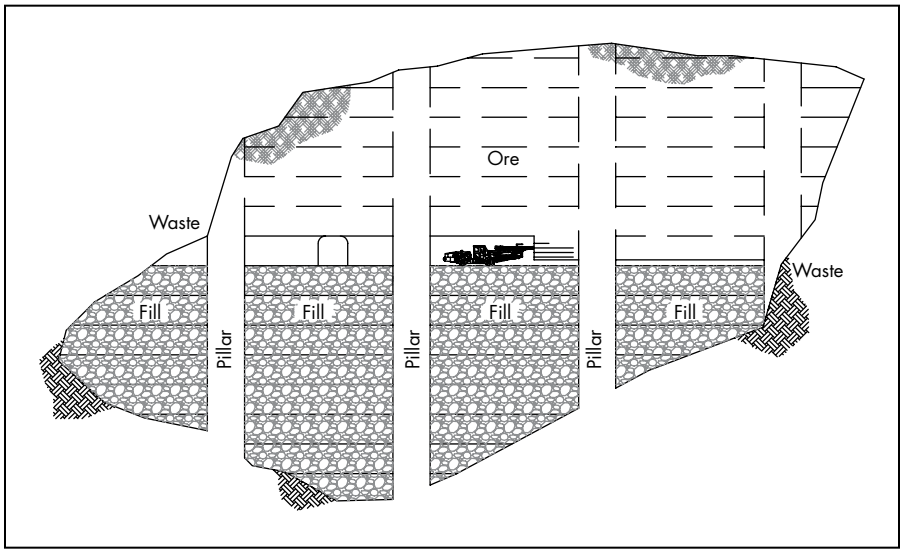


Figure 13.5-5 Postpillar mining

underground stopes via borehole or waste pass. Because of its coarse texture, rock fill is often capped with sand fill or cemented sand fill to provide a mucking surface that is less susceptible to ore loss and dilution.

Postpillar Mining

In horizontal or near-horizontal ore bodies of substantial thickness and width, the placement of uncemented fill serves the primary purpose of providing access to excavate successive slices of ore in a bottom-up sequence. The postpillar method can be described as a room-and-pillar excavation where backfill (usually uncemented rock fill) is placed on the floor so that successive layers can be mined upward through the ore zone (see Figure 13.5-5). The pillars, or at least the upper portions of the pillars, can be preserved and extended upward to continue supporting the back. The pillars can be much taller and more slender when they are surrounded by fill, and so recovery is higher than for nonfill room-and-pillar methods.

Vertical or Near-Vertical Cut-and-Fill Stoping (Conventional or Mechanized)

The use of uncemented fill in narrow-vein stopes that are inclined or vertical provides both stability and a working platform for the upward advance of the excavation of the ore. Hydraulic (sand) fill or rock fill with sand is superior to rock fill alone for this application because greater density is achieved. Frequently, a top layer of cemented fill is used to provide a stronger surface for machine operation and to minimize ore loss and dilution.

Using sand (usually coarse tailings) to backfill stopes was one of the earliest fill applications in mining. It was often used in conjunction with timber support, but the use of timber is now largely supplanted by bolts and shotcrete. The transport, distribution, and placement of fill are as hydraulic slurry via a pipe system. Less frequently, compressed air is used for the transport and pneumatic stowing of the fill. The pipe system prevents congestion of the haulageways and eliminates the need for trucks and loaders. With hydraulic fill, drainage of the excess water must be accommodated with designed filters, drainage raises, sumps, and pumping systems. These features impose substantial operational costs and scheduling limitations. Pneumatic fill systems have increased ventilation requirements and pipe wear as well as the physical limits associated with distribution and the potential health risks of increased airborne dust.

In conventional cut-and-fill stopes, access to the level of the stope cut was gained by timbered or steel-tubbed raises that were maintained and extended up through the backfill from the haulage level (see Figure 13.5-6). In the stope, drilling was accomplished with hand-operated stoper drills for vertical holes or jacklegs for horizontal holes where the cut would be taken by breasting down. Slushers were the principal means for mucking the blasted ore across the stope to the muck raise. From the bottom of the muck raise, chutes were used to transfer ore to muck cars for transport to the shaft by rail. In addition to ore transport, the raises provided manway access, hoisting of supplies, pipe and power reticulation, and drainage of the backfill water.

In the conventional system, it was difficult to bring equipment into or out of the stope; after a piece of equipment had been installed in a stope it remained isolated there, rising with the level of the cut. Low unit cost, light weight, simplicity, and durability were important equipment characteristics; slushers fit this description well. However, they were not as productive for mucking as low-profile LHDs (load-haul-dump units) with buckets.

Overshot muckers powered by compressed air were used at some mines to replace slushers. Overshot muckers are relatively simple to maintain, and ventilation requirements are no greater than those required for slushers.

Eventually, diesel-powered LHDs were placed into service (Figure 13.5-7). With their introduction, ventilation requirements increased and maintenance requirements demanded better access than raises could provide. The cost of LHDs forced the need to achieve higher utilization rates, which in turn made it necessary to allow stope-to-stope movement of LHDs. This change led to the shift from access by raises to access by ramp systems.

Resuing

Resuing is the term applied to the mining method whereby high-grade ore from very narrow veins is extracted selectively before stopes are widened in waste rock to obtain the additional

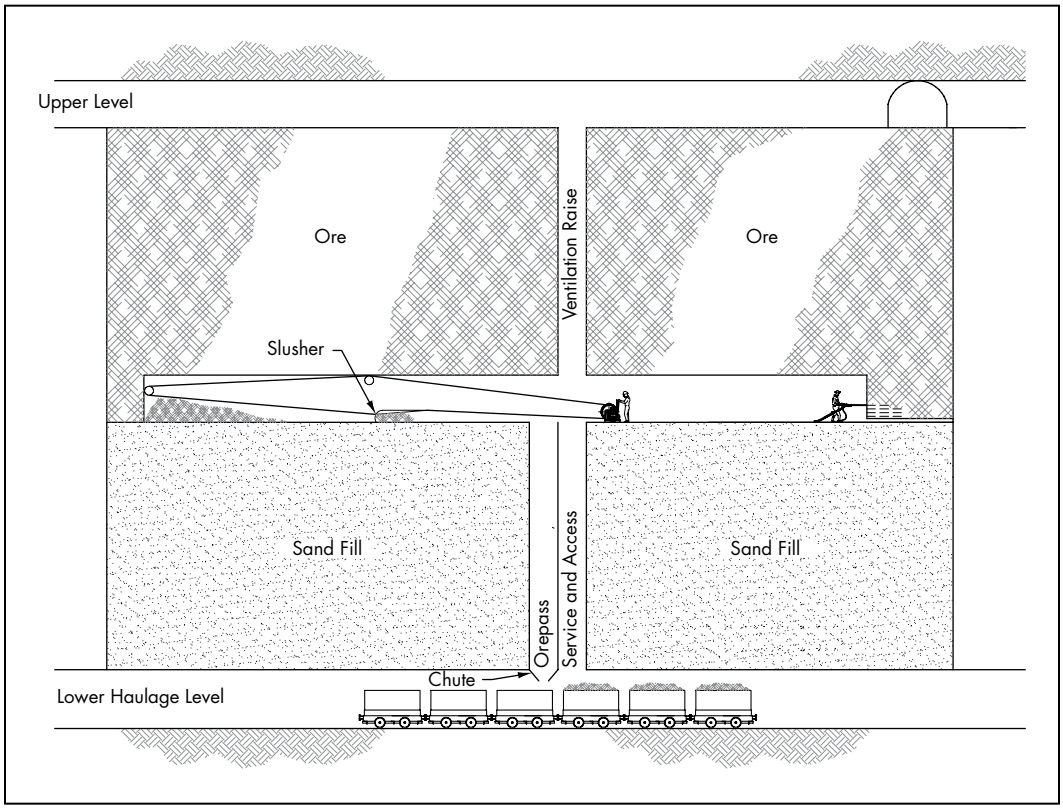


Figure 13.5-6 Conventional cut-and-fill stoping with hydraulic (sand) fill

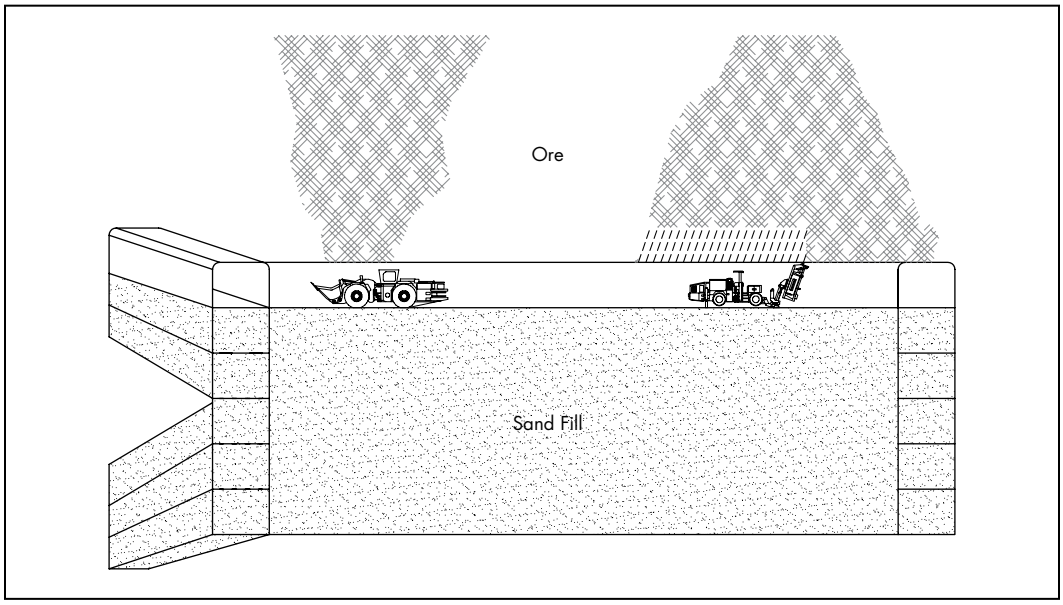


Figure 13.5-7 Mechanized cut-and-fill stoping with uncemented hydraulic (sand) fill

width required for equipment to operate (see Figure 13.5-8). Resuing requires a working platform of fill.

Bench-and-Fill

Bench-and-fill stoping is applied to vertical and near-vertical stopes that have sufficient width, length, and competency of wall rock to permit a system of laterally retreating blastholes followed by advancing waste rock fill (see Figure 13.5-9). This method allows variations in placement of the fill. The amount of waste rock exposed between the blasting and mucking of ore, and the advancing face of waste fill, can be adjusted to suit local conditions. In some cases, the waste fill is advanced such that all open space is completely eliminated. This lack of open space enhances support for weak hanging-wall and

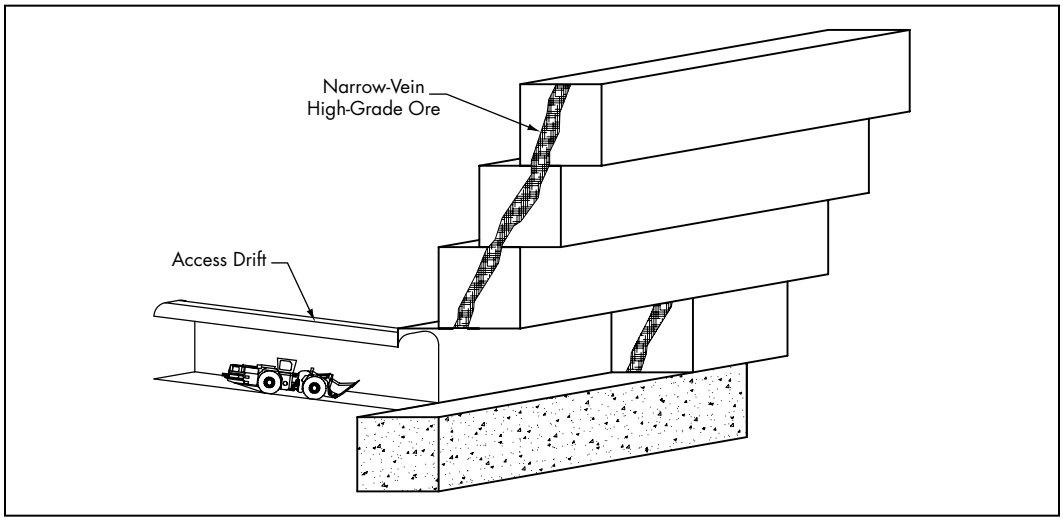


Figure 13.5-8 Resuing

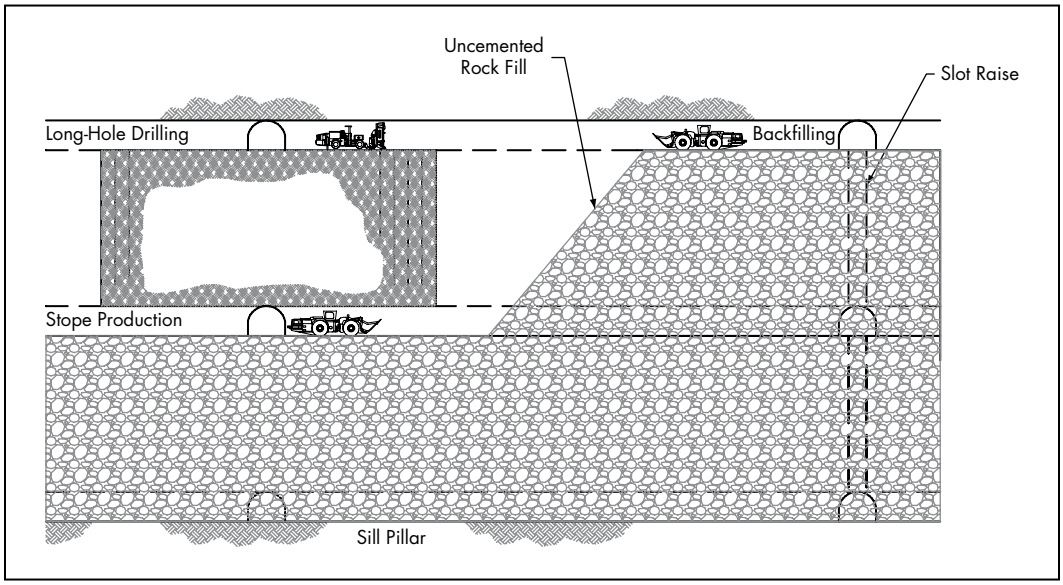


Figure 13.5-9 Bench-and-fill

footwall conditions, thus reducing dilution and ore loss. The use of some cemented fill to augment the uncemented fill can also increase recovery.

EXAMPLE MINES

The mines referenced in this section are using, plan to use, or have used the cut-and-fill methods described herein. Many of these operations use more than one method, and in many cases their specific working methods have evolved over time. For reference and research purposes, the lists include both currently operating and nonoperating mines.

Mines Using Cemented Fill

This group includes mines using longitudinal blasthole stoping with delayed cemented fill, overhand drift-and-fill with delayed cemented fill, overhand drift-and-fill with cemented fill, and underhand drift-and-fill with cemented fill:

- Longitudinal blasthole stoping with delayed cemented fill
  - Garpenberg—Bergslagen, Sweden
  - Zinkgruvan—Bergslagen, Sweden
  - Julietta—Far East Russia
  - K2—Republic, Washington, United States (closed)
  - Kensington—Juneau, Alaska, United States (planned)
- Transverse blasthole stoping with delayed cemented fill
  - Big Gossan—Papua, Indonesia (planned)
  - Cannon—Wenatchee, Washington, United States (closed)
  - Diavik—Northwest Territories, Canada (planned)
  - Greens Creek—Admiralty Island, Alaska, United States

  - Kidd Creek—Timmins, Ontario, Canada
  - Meikle—Carlin, Nevada, United States
  - Mt. Isa—Queensland, Australia
- Overhand drift-and-fill with cemented fill
  - American Girl—Yuma, Arizona, United States (closed)
  - Golden Promise—Republic, Washington, United States (closed)
  - Rosebud—Lovelock, Nevada, United States (closed)
  - Strathcona deep copper mine—Levack, Sudbury, Ontario, Canada (closed)
- Underhand drift-and-fill with cemented fill
  - Apex—St. George, Utah, United States (closed)
  - Bullfrog—Beatty, Nevada, United States (closed)
  - Carlin East—Carlin, Nevada, United States
  - Deep Post—Carlin, Nevada, United States
  - Getchell/Turquoise Ridge—Golconda, Nevada, United States
  - Jerritt Canyon—Mountain City, Nevada, United States
  - Lucky Friday—Mullan, Idaho, United States
  - Rain—Carlin, Nevada, United States
  - Stratoni—Stratoni, Greece
  - Val Chisone—Italy

Mines Using Uncemented Fill

This group includes mines using vertical or near-vertical stoping with uncemented sand fill (hydraulic fill), resuing, postpillar mining with uncemented fill, and bench-and-fill stoping with uncemented fill:

- Vertical or near-vertical stoping with uncemented sand fill (hydraulic fill)
  - Bunker Hill—Kellogg, Idaho, United States (closed)
  - Galena—Wallace, Idaho, United States
  - Groundhog—Vanadium, New Mexico, United States
  - Homestake—Lead, South Dakota, United States (closed)
  - Star—Burke, Idaho, United States (closed)
  - Stillwater—Nye and Big Timber, Montana, United States
  - Sunshine—Kellogg, Idaho, United States (closed)
- Resuing
  - Golden Patricia—Kenora District, Ontario, Canada (closed)
  - Julietta—Far East Russia
- Postpillar mining with uncemented fill
  - Black Cloud—Leadville, Colorado, United States (closed)
  - Christmas mine—Winkelman, Arizona, United States (closed)
  - El Mochito—Las Vegas, Honduras
  - Mantos Blancos—Antofagasta, Chile
  - Milpillas—Sonora, Mexico
  - Rey de Plata—Guerrero, Mexico (closed)
  - Tizapa—Guerrero, Mexico
  - Madero—Zacatecas, Mexico
  - King Island Scheelite—Tasmania, Australia (closed)
- Bench-and-fill stoping with uncemented fill
  - El Peñón—Antofagasta, Chile
  - Ken Snyder—Midas, Nevada, United States (modified)
  - Mt. Isa lead mine—Queensland, Australia
  - Stillwater—Nye and Big Timber, Montana, United States

CURRENT TRENDS

Stope Mechanization

The widespread introduction of mobile equipment (diesel-powered loaders, trucks, drills, and service vehicles) since the 1960s and 1970s has largely replaced the use of jacklegs, stopers, and slushers, the primary underground mining machinery of the prior era. The maintenance and maximum usage of this mobile equipment precludes allowing it to become captive in a single stope. Therefore, access to stope levels is now almost exclusively by ramps rather than by raises.

The demand for ventilation has also been increased because of the increased usage of diesel equipment.

In recent years, production with remote controlled loader operation has been replacing the more costly development of gathering bells and chutes that have been commonly used in blasthole stope mines.

Paste Fill

Mining fill systems based on the use of paste for backfill are supplanting other fill systems for bulk backfill applications. The opportunity to transport and place the material by pipe reticulation substantially decreases congestion in drifts and ramps. The placement rate of paste fill is much faster than placement using trucks carrying fill from a surface or underground batch plant and then having to rely on LHDs and jammers to place the fill into the stope.

The resulting benefits (lower operating costs, lower ventilation demands, and a faster turnaround time for the filled stope) are similar to those achieved by hydraulic systems, but without the water drainage issues. However, paste plants are costly to construct and must be completed during preproduction development.

Cemented Rock Fill

The use of cemented rock fill continues to be a common choice where bulk material handling is not advantageous and where flexibility is important to maintaining operations.

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CHAPTER 13.6

Backfill Mining

Douglas F. Hambley

INTRODUCTION

This chapter discusses the design of systems for backfilling underground mines, opening with a short history of backfilling followed by a listing of the types of backfilling systems to be covered in the chapter, and closing with a discussion of sources of further information. Stoping methods that use backfill are discussed in other chapters, especially Chapter 13.5.

Backfilling of mined openings may be performed for a number of reasons. The first is to provide support for walls of an opening in relatively weak rock so that ore immediately above, below, or adjacent to the opening can be mined. Another possible reason for backfilling is to contribute to the support of the roof and walls of a wide opening. In this case, the backfill will not prevent failure in the roof, but it will minimize the displacement of the broken rock and thereby minimize the potential for progressive upward failure, possibly reaching to the surface. This leads to another reason for backfilling, namely minimization of surface subsidence. A final important reason for backfilling is isolation and disposal of mining/milling wastes considered to be unattractive or an environmental hazard on the surface.

Unless otherwise noted, equations in this chapter are given in consistent units. The reader is expected to include units when performing calculations and provide any necessary conversion factors.

BRIEF HISTORY OF BACKFILLING PRACTICES

The backfilling of mined openings has a long history. It is only since about the 1980s that archeologists have recognized that backfilled bell pits in chalk in Europe that date from 2700 BC were actually mines for flint. The workings extended outward from the base of a shaft in all directions without support until it became unsafe, at which time the workings and shaft were backfilled to surface and another pit begun alongside. This method of mining apparently remained in use in near-surface ironstone and coal mines in the United Kingdom until the early 19th century (Shepherd 1980).

Stowing of wastes in workings to allow the mining of adjacent areas was a common practice in the metal mines in Cornwall (United Kingdom) and the coal mines in Belgium in the mid-19th century. Wastes generally consisted of barren or low-grade rock mined as part of the stoping process or in other areas of the mine away from the ore body or seam. However, if wastes obtained in this manner were insufficient, other materials such as sand or smelter slag could be brought into the mine to act as backfill (Callon 1876).

In the Kolar Gold Fields in India, quarried granite blocks were used as backfill/ground support in overhand stopes until the 1940s. The granite blocks were hand fitted to completely span the width of the stope. The advantage of this was that because the deformation modulus is similar to that of the wall rock, significant amounts of stress could be transferred to the blocks from the wall rock, thereby reducing the potential for rock bursts, which were a severe problem in those deep mines. The disadvantage of this method is that it is only feasible where labor is extremely cheap.

Hydraulic backfilling (fill placed hydraulically) has been used in the coal mines in Pennsylvania (United States) since 1884 (Peele 1941). With regard to metal mines, hydraulic placement of sand-sized tailings appears to have been first practiced in the gold mines on the Witwatersrand in South Africa in 1910, its initial adoption in North America occurring in Butte, Montana (United States) in 1917. As in Pennsylvania, the initial use at Butte was to seal off a mine fire in an area of broken ground where other sealing methods were impractical (Rahilly 1923). General rules were developed early from experience concerning the pipe velocities necessary to convey tailings hydraulically without having the pipes "sand out" (fill up from particle sedimentation due to insufficient flow velocity); however, criteria for flow and identification of the different flow regimes that operate with slurries have only been developed since the early 1950s beginning with Durand and his co-workers at SOGREAH (SOciete GRenobloise d'Etudes et d'Application Hydrauliques) in France and others working independently in the United Kingdom (Abulnaga 2002).

In the late 1950s, the International Nickel Company (INCO) of Canada (now Vale Inco) and Falconbridge Nickel Mines Ltd. (now Xstrata) began experimenting with the addition of portland cement to hydraulic backfill. The purpose was to produce a stronger fill that could be used for support in

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pillar recovery operations and could provide competent mucking floors, thereby eliminating the need for timber floors. In 1961, with financial support from the two aforementioned mining companies, Canada Cement Lafarge Ltd. began a comprehensive study to develop cemented fill technology that was completed in 1970 with the publication of their findings (Weaver and Luka 1970).

Pneumatic stowing was introduced in Polish coal mines in the 1920s and soon afterward adopted in other coal mining areas of Europe. In North America, the use of pneumatic stowing was promoted by the U.S. Bureau of Mines in the 1960s and 1970s but has not generally been adopted. The method was tested at the Sullivan mine in British Columbia (Canada) in the late-1960s and later used in uranium mines in Grants, New Mexico (United States), and in gold mines in South Africa.

The concept of thickened tailings, developed by Professor Robinsky of the University of Toronto and first applied at the Kidd Creek mine in Timmins, Ontario (Canada), in the mid-1970s, was extended by the late Dave Landriault and others at INCO Metals in the 1980s and later at Golder Paste Technology Ltd. to become paste fill when used for backfill underground. In 1975, pump-pack was developed by the British using broken coal and bentonite, which was a thixotropic material and pumped like a toothpaste but set like a low-strength concrete (Schmidt et al. 1976). Paste fill was developed independently in Germany at the Bad Grund mine in the early 1980s (Jewell 2006).

"Flowable fill" has been used in recent years by, among others, the Office of Surface Mining Reclamation and Enforcement (OSMRE) of the U.S. Department of the Interior to remotely backfill near-surface abandoned mines—especially coal mines—that represent a subsidence hazard. Flowable fill refers to a cementitious slurry consisting of a mixture of fine aggregate or filler, water, and cementitious material(s) that is used primarily as a backfill in lieu of compacted earth. This mixture is capable of filling all voids in irregular excavations and hard-to-reach places (such as under and around pipes), is self-leveling, and hardens in a matter of a few hours without the need for compaction in layers.

TYPES OF BACKFILL

As indicated in the history section, the different materials that have been used to backfill mine openings include the following:

- Dry sand and rock fill
- Uncemented hydraulic backfill
- Cemented hydraulic backfill
- Cemented rock fill
- Paste fill
- Pneumatic fill
- Flowable fill

In subsequent sections of this chapter, each of these alternatives will be discussed. For each, the method of placement, drainage requirements, if any, equipment requirements, pipe wear, and other relevant design and operating topics are considered.

Dry Sand and Rock Fill

Dry placement of stope fill materials is often the simplest method of fill placement and has a long history. In metal mines in North America and Australia that used cut-and-fill or square-set stoping, this method dominated until the 1950s and was still taught as a preferred method of backfilling in some mining schools until the late 1960s.

Advantages and Disadvantages

Although it has generally been superseded by other methods, there are still situations where dry placement of fill has advantages over other systems. For example, if a mine does not have a processing plant on site to provide mill tailings, other fill options must be considered. Such fills could include (1) natural surface sands (glacial till in North America, natural dune sands in Australia); (2) open-cut overburden, specifically quarried surface rock, or even rock generated in local civil works; and (3) underground development waste. Dry fill is also the preferred option in evaporite and gypsum mines where the ore or host rock is readily dissolved in water, or in mines in arid locations where water is scarce.

Dry placement of fill has two main disadvantages. First, it has a relatively low density as placed, which means it can undergo significant compression before it reaches its optimum density to provide stope wall support to prevent spalling of stope walls and minimize stope wall convergence. Second, depending on stope configuration, it may be difficult to place dry fill close to the roof of the opening across the full span of the stope roof.

Transport and Placement of Dry Fill

The first consideration, as with all types of backfill, is the means of transport of the dry fill underground. The most efficient means for transport of sand- or gravel-sized particles is a borehole from the surface. Selection of the size of the borehole depends on two considerations: the required throughput in metric tons per hour, and the size distribution of the particles. For coarse material such as gravel, the particle size will govern, because the diameter of the borehole should, as a general but not necessarily universally applicable rule, be a minimum of three to six times the maximum particle size of the gravel in order to avoid hang-ups. For finer material, the governing concern is the required throughput. In general, boreholes with diameters of 150 mm or 200 mm (6 or 8 in.) will be sufficient for most purposes. However, using this size of pipe, one must never let the pipe fill and possibly bridge, as it may plug.

The most common method of transporting dry fill to the workings is by haulage vehicles. In conventional, older mines, this usually meant railcars, which would dump the fill into a fill raise. More recently, haulage trucks have been used and belt conveyors are also being used for sand, gravel, or rock. All of these placement methods require good access to the workings that need to be filled.

Another important consideration with dry placement of granular fill is the method used to place and compact the fill within the stope. Historically, placement was done by hand, which is slow and costly. If the area to be filled is accessible to mobile equipment, spreading and compaction could be accomplished by means of a hydraulic ram and blade mounted on a haulage truck or load-haul-dump (LHD) vehicle. (The blade is pushed laterally against the fill pile by the hydraulic ram to compact it.) Another placement and compaction method that has been tried is a high-speed throwing belt, the main disadvantage of which is the limited distance that the fill travels beyond the discharge point.

In the case of rock fill, placement is not generally a problem if dumping is from the top of the working. Because of

the mass of the rock pieces, gravity flow is usually satisfactory. However, large blocks contain a significant proportion of voids, and such fill may need considerable consolidation before it can provide significant stope wall restraint. This disadvantage is often resolved by filling the voids with cemented hydraulic fill, as discussed later in this chapter.

Uncemented Hydraulic Backfill System Design

Hydraulic transport and placement of backfill is the method most commonly used in metal mines but less commonly in coal mines. Deslimed and partially dewatered mill tailings are the materials most commonly transported underground hydraulically, but surface sand (e.g., glacial till in North America, natural dune sands in Australia) is also transported in this manner. The fine (slimes) portion of tailings is undesirable because of its slow drainage, but also because of the potentially disastrous consequences of liquefaction resulting from vibrations from, for example, production blasts.

Transport of Slurry Solids Content

Mill tailings generally consist of a heterogeneous slurry with approximately 30%–40% solids by mass and the rest water. The solids and the water represent two separate phases that will readily separate if the flow velocity is insufficient. When a heterogeneous slurry is discharged from a horizontal pipe, the solids will settle out with the coarsest particles settling first. As the solids are discharged, they will gradually form a cone around the end of the pipe, with the finer materials toward the toe of the slope. The slope of the cone varies with the solids content of the slurry; with very dilute slurries, the slope may not exceed 2°–3° and the sediments will be loosely packed, but at solids contents of 60%–70% by mass or more, the slope will approach the angle of repose of the particles and the solids will consolidate readily as they drain.

Hydraulic backfill systems typically operate at solids contents of 65%–70% solids by mass or more. Because, as stated previously, the tailings in the mill typically have a solids content of around 30% by mass, it is generally necessary to partially dewater the tailings prior to transport underground. Dewatering is usually accomplished using hydrocyclones or occasionally thickeners or rake classifiers. Hydrocyclones are vastly preferable because they most effectively deslime as well as dewater. One word of caution: The cut point commonly used by geotechnical engineers for drainage computations is the D_{10} size, whereas the D_{20} or D_{50} size is used by designers of hydrocyclones. Failure to appreciate this distinction could result in having a larger fines content than anticipated in the backfill. It is critical to maintain the design flow rate in a hydrocyclone if the cut point is to be achieved in practice.

Drainage requirements. After hydraulic backfill has been placed in a mined opening, it is necessary to drain away the water for a number of reasons. First, water-bearing sand/slime-sized material, especially if it contains excessive slimes and/or excessive water, can liquefy if subject to vibration caused by a seismic disturbance such as an earthquake, by a rock burst, or by production blasting. Second, drainage of the water is essential to consolidation of the fill. Because the fill is used as the working floor for the next lift in overhand cut-and-fill stoping, it is generally desirable to have it available for production traffic within 24 hours.

Prevention of the phenomenon of liquefaction requires essential consideration in the preparation and placement of hydraulic fill—particularly uncemented hydraulic fill—underground.

A general, but not absolute, rule is that the gravitational percolation rate of water through the fill should exceed 10 cm/h (4 in./h). Because drainage of water from the fill is by gravity with a hydraulic gradient of unity, this percolation rate corresponds to a hydraulic conductivity of approximately 3×10^{-3} cm/s (5.9×10^{-3} fpm), which is a typical value for a well-graded, fine-grained sand. To ensure this drainage rate is achieved, it is customary to remove the fines so that no more than 10% of the solids are finer than 74 μm (200 mesh). However, satisfactory backfilling has been achieved with as much as 10% of the solids finer than 43 μm (325 mesh). Also, hydraulic conductivity is temperature dependent, so as mine water temperature increases, fines content can increase and drainage rates remain satisfactory. Importantly, the converse applies. A comprehensive discussion of hydraulic fill drainage mechanisms is given by Thomas et al. (1979).

To facilitate fill drainage, it is customary to erect fences within the stopes being filled to contain the fill and to drain the water through ladderways or drain towers constructed up through the fill. To allow the fill water to drain without loss of fill fines it is customary to place filter fabrics on the inside of fill fences and outside drainage ladderways and towers inside the site to be filled. Historically, burlap (in North America) or hessian (in Australia) was used as the filter fabric. More recently, these materials have been largely replaced by monofilament geotextiles, though the traditional materials, where available, are still acceptable. To select the correct geotextile, the following filter selection criteria developed by the U.S. Army Corps of Engineers may be used:

- The ratio of the D_{85} (85% finer) size of the backfill to the opening size of the geotextile should exceed 1.0.
- The open area of the geotextile should be less than 40% of its total area.

For most tailings, the D_{85} size will range from 149 to 297 μm (0.0069 to 0.012 in.). Selection of the geotextile should also consider the required water flow rate per unit area of geotextile, which depends on the dimensions of the fill fences and drainage towers.

Backfill transport pipe size requirements. With two-phase (solids/fluid) heterogeneous slurries, the transport velocity in a pipeline must be high enough to prevent deposition of coarse particles. On the other hand, an excessive transport velocity in the pipe will result in excessive friction losses and pipe wear. A flow velocity of 1.5 m/s (5 fps) is often but not always considered the minimum necessary to provide satisfactory results. Conversely, a flow velocity of greater than 2.4 m/s (8 fps) may result in excess friction losses and pipe wear. Table 13.6-1 presents hourly placement tonnages that can be achieved at a pipe velocity of approximately 1.8 m/s (6 fps) and 65%–70% solids density by mass for a material with a specific gravity of 2.65.

Pipe friction losses and pump requirements. In mines accessed by a shaft, transport of backfill underground will be by means of either a pipe in the shaft or a borehole near the shaft (preferable to a pipe in the shaft, depending on shaft function) that connects the backfill plant to the workings. In such cases, the elevation difference is usually but not always sufficient to transport the fill to the working place without pumping. However, where access to the mine is by means of a horizontal adit, gravity flow will, with the occasional

Table 13.6-1 Hydraulic fill placement rate versus pipe diameter

Nominal pipe diameter, mm	50	80	100	150	200
Nominal pipe diameter, in.	2	3	4	6	8
Fill placement rate, t solids/h	15–20	35–40	65–75	145–165	255–290

Table 13.6-2 Fill tonnages handled before replacement by various types of pipe and fittings

Type	Standard (Schedule 40)	Heavy-Duty (Schedule 80)	Rubber-Lined
Pipe, t	250,000	800,000	>800,000
Bends and elbows, t	—	>250,000	>1,300,000
Tees, t	150,000	600,000	>600,000

Source: Adapted from Waterfield 1973.

exception, not be sufficient to transport the backfill slurry and pumping will be necessary. Where mine access is by decline, pumping may or may not be necessary, depending on the general mine layout. In all cases it is prudent to calculate the friction losses in the system. Friction losses are greater than those for water alone, but can be estimated from the conventional Darcy–Weisbach equation using the following adjusted friction factor (Abulnaga 2002):

$$f_m = f_L\{1 + 81C_v[gD_i(\rho_s - \rho_L)/(V^2\rho_L\sqrt{C_d})]^{1.5}\} \quad (13.6\text{-}1)$$

where

f_m = friction factor for the slurry mixture
f_L = friction factor for clear water under the same temperature and pressure conditions
C_v = solids concentration by volume
D_i = inside diameter of the pipe
ρ_s = solids density
ρ_L = water density
V = slurry velocity in the pipe
C_d = drag coefficient, which can be taken as 0.43 in most situations

If a pump is found to be necessary, a centrifugal slurry pump is the most likely selection, pump specification being based on head and flow rate requirements. These are relatively low-head pumps, and it may be necessary to install pumps in series to ensure that varying head requirements are met. However, material will flow through the pump impellers even if the head contribution is not needed and the motor not operating. To minimize wear in centrifugal pumps, the following guidelines should be followed (Paterson 2006):

- Discharge velocity less than 8 m/s (26 fps)
- Maximum impeller tip speed for all-metal pumps of 28 m/s (92 fps)
- Maximum impeller tip speed for rubber-lined pumps of 20 m/s (76 fps)
- Range of discharge flow/flow at best efficiency point of 60%–100%

Wear in pipes and line fittings and pipe material selection. Erosion of the walls of boreholes used to transport backfill underground is generally not considered a serious problem, though boreholes must be lined from the surface down to appropriately below the base of surface rock alteration, sometimes referred to as the base of complete oxidation. This is based on the usual situation where the oxidized/altered rock from the surface will be scoured by the fill flow, whereas the significantly more solid rock below will not. On the other hand, wear in elbows, tees, and other fittings and in pipes will result in the need for periodic replacement or selection of more appropriate lining materials. Consequently, the design engineer must consider whether to use

- Rubber-lined, heavy-duty (thick-walled), or standard pipes;
- Victualic, threaded, or flanged pipe couplings;
- Rubber-lined, built-up (heavy-duty), or standard bends, elbows, and tees;
- Heavy-duty rubber hose or rubber-lined pipe connections at horizontal takeoff points; or
- For long-term, high-tonnage applications, pipes and fittings with basalt or similar linings.

In general, wear is most severe at bends, elbows, and tees; however, wear will also occur in straight pipe, especially if the slurry is abrasive and contains angular particles. Wear rates depend on the density of the slurry, the diameter of the pipe, the flow velocity, and the maximum size and abrasiveness of the particles. Unfortunately, wear rates are rarely expressed in terms of these parameters but are more commonly given in terms of the tonnage handled by pipe prior to replacement. If line velocities are not excessive, the tonnages given in Table 13.6-2 can be expected before replacement becomes necessary.

The lack of data in Table 13.6-2 regarding the life of elbows and bends is because tees are much more commonly used for takeoffs because it is simpler to change the side of the tee that is blanked off than it is to uncouple a line to insert a bend or elbow. In stopes, distribution pipes can be either standard steel pipe or plastic pipe and should include tees and valves at 3-m (10-ft) intervals to facilitate even distribution of the fill. Wear is typically on the invert of the fill pipe, and pipe life can be extended significantly by rotating the pipe one-sixth of a revolution at regular intervals.

In multilevel mining, heavy-duty rubber hose connections are preferable to pipe connections at the levels. Only a single hose is required at a level even though there may be multiple branch lines. However, the connections on the ends of the hoses will be bolted flanges rather than victualic or threaded connections.

Backfill Plant Requirements

The first requirement in a typical backfill plant is a bank of hydrocyclones to remove excess water and fine particles from the tailings. The overflow from the cyclones (the fines and excess

water) is typically transported to a surface tailings impoundment for disposal. Many mines once operated on a 5-day week, whereas concentrators typically operate on a 7-day week, and in such cases it became necessary to provide a storage silo capable of holding backfill pouring requirements for 1 or 2 days. This silo is known as a repulper, because the backfill is allowed to drain until needed, at which time water is added at high pressure to the base of the tank to refluidize the fill.

The fill plant will also need transfer pumps and valving, and care should be taken to ensure that the capacities of the transfer pumps are sufficient. Transfer pumps should be able to handle a flow rate approximately 20% greater than the design flow rate for the system (solids plus water), because if the flow rate decreases below the critical value for sedimentation, the inevitable result will be the complete filling of the pipe with solids. Cleaning out a pipe that has sanded up is time-consuming and costly, particularly if the fill contains portland cement.

Current practice is more likely to have both mine and mill operating around the clock, thus eliminating the need for fill storage tanks and the complexities they involve.

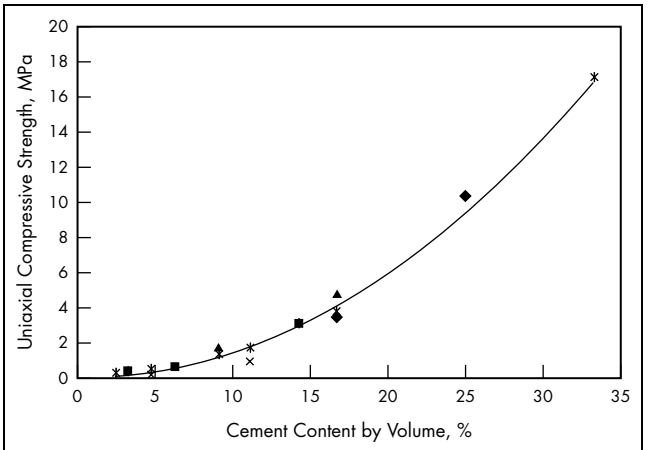
Cemented Hydraulic Backfill System Design

Correctly prepared uncemented hydraulic fill is predominantly sand-sized and must behave as a sand, albeit on occasions a slow-draining sand. Consequently, it has no true cohesion or, until drained, no unconfined compressive strength. For some applications, such as fill placed against unmined rock that will be recovered later, it is often desirable that the fill be able to stand unsupported upon exposure. For such applications, it is customary (but on occasions not at all necessary and possibly detrimental) to add cement to the backfill.

Because cemented fill is the same fill material as the uncemented fill except for the cement, several of the design parameters for a cemented hydraulic backfill system are the same as for an uncemented backfill system. However, slurry solids content must be more carefully controlled, though pipeline velocities remain essentially the same. As is the case for uncemented fill, the tailings are deslimed with a typical D_{10} size of 74 μm (200 mesh). However, the cement provides fines to the mix and the drainage rate is typically somewhat slower than that for uncemented fill. This potential problem is in part negated for the following reasons: (1) portland cement consumes fill water during hydration, reducing the volume of water to be removed; (2) portland cement produces heat during hydration, reducing the viscosity of fill water, tending to increase dewatering rate; and (3) portland cement lubricates the flow so higher placement densities can be used, reducing the amount of water to be removed. In pipeline flow, portland cemented fill flows more uniformly than uncemented hydraulic fill, and this tends to reduce pipeline wear for similar placement rates—coarse fill particles have reduced access to pipeline walls. Optimum solids content for pipeline flow generally ranges from 68% to 72% solids by mass, and closer attention is given to control of this solids content. Costs considered, after portland cement is added to hydraulic fill, the fill operation moves from underground disposal of mining waste to preparation of a quality-controlled engineering material.

Cement Content Versus Strength and Deformation Properties

The strength of a cemented fill depends on the percentage of cement in the fill. Figure 13.6-1 shows the relationship between cement content and strength for several cemented mine backfills in Canada and Australia. Although Figure 13.6-1 shows the cement content as a volume percentage, it is best to express the cement content as a mass percentage of the total dry solids. The equation of the best-fit curve in Figure 13.6-1 is given by



Note: Symbols represent different mines. The mines are not identified because some of these data are from unpublished sources.

Figure 13.6-1 Backfill compressive strength as a function of cement content

$$C_o = 12.1\ P^{2.0448} \tag{13.6-2}$$

where

C_o = compressive strength in kilopascals (kPa) (multiply result by 0.145 to obtain strength in psi)
P = cement content in volume percent

Some mines have experimented with replacing part of the cement with fly ash or ground slag and/or similar pozzolanic materials. In the case of fly ash, there does not appear to be a significant difference in compressive strength for fly ash contents up to 20% of the cement content in fill with cement/tailings ratios between 1:10 and 1:30 by volume. In the case of ground slag, the uniaxial compressive strength increases compared to portland cement only, with the increase approximately proportional to the slag content.

Cement Content Versus Use

The amount of cement required in a cemented fill depends on the strength requirement and the use for the fill. Some examples are given in Table 13.6-3. These should be used for guidance and not adopted as standard. For example, use of cement in mucking floors may have advantages for movement of mucking equipment, but it prevents almost totally the downward flow of fill drainage water.

Freestanding Height

The freestanding height of cemented backfill depends on the compressive strength and density of the fill as follows:

$$H = 2C_o/\gamma g \tag{13.6-3}$$

where

H = freestanding height of the fill
C_o = uniaxial compressive strength of the backfill
γ = density of the fill

Table 13.6-3 Cemented fill cement/tailings ratio versus application

Application	Cement/Tailings Ratio
Bulk filling	1:30
Barrier against adjacent pillar, adjacent stope, or related structures	1:10
Mucking floors	1:6 to 1:12
Water-retaining bulkheads/barricades where no restriction of stope drainage must result—drain pipes through the strengthening plug	1:2

In practice, the known parameter in Equation 13.6-3 is the required freestanding height; the compressive strength necessary for the fill to stand unaided at that height is the unknown. For example, if the required freestanding height is 8 m and the specific gravity of the fill is 1.7, Equation 13.6-3 indicates that the required strength is 133 kPa (19.3 psi). From Figure 13.6-1, the strength corresponds to a backfill containing approximately 3% cement. Alternatively, for a compressive strength of 133 kPa, Equation 13.6-2 gives a cement content of 3.22%, which is equivalent to a cement/tailings ratio of 1:30.

Pump Requirements

Where there is extensive horizontal development on upper levels of a mine, the elevation head may be insufficient to support flow by gravity alone. In such cases, pumps will be necessary to transport the fill over all or part of the underground fill distribution system. Both positive displacement pumps and centrifugal pumps can be used for this application; however, rubber-lined centrifugal slurry pumps are considered a better choice. Pumps with dry glands, which minimize water handling requirements, are available. With centrifugal pumps, the slurry will flow through a pump even if it is not needed to supply head.

Backfill Plant Requirements

Backfill plant requirements are similar to those for uncemented fill, except for the need for a cement silo plus mixing tanks and feeders.

Cemented Rock Fill System Design

Cemented rock fill, as typical of modern practice, was developed at the Geco Division of Noranda Mines Ltd. in the early 1960s to provide support for stope walls that were prone to slough. By restraining stope wall rock, the fill reduced dilution and improved overall regional stability. A cemented rock fill system was developed at Mount Isa in Australia in the late 1960s, but the main reason for the rock fill component was to reduce the quantities of cemented hydraulic fill as portland cement was expensive in that part of Australia, being remote from cement suppliers and prone to disruption of supply during the wet season. In a cemented rock fill system, the main design function of the cemented hydraulic fill is to fill the voids in the broken rock.

Rock Fill Size Considerations

Rock fills should consist of coarse angular fragments of medium to high density, medium to high chemical stability, and high strength and abrasion resistance. Abrasion resistance can be tested using the Los Angeles abrasion test.

The rock fill can be modified to suit a particular application by optimizing the particle size distribution. A competent fill that will stand when exposed on a vertical face should have a low porosity. Fill should be appropriately graded with respect to particle size. The particle size distribution of a rock mass is often represented by the Gates–Gaudin–Schuhmann distribution:

$$P = (d/m)^n \tag{13.6-4}$$

where

P = cumulative percent finer than size d
d = particle size
m = maximum particle size

In the concrete industry, a densely packed fill material is obtained when the exponent, n, in Equation 13.6-4 is 0.5.

A major concern with rock fill is the extent of rock degradation from impacts during freefall passing underground from the surface or from degradation from particles grinding against each other during choke-fed passing. An impact test that was used successfully at Mount Isa in Australia (Kuganathan 2005) consisted of dropping samples of up to 300-mm (11.8-in.) boulders from a height of 16.5 m (54 ft) onto a piece of 16-mm (⅝-in.) thick manganese steel liner plate. A test was considered successful if 75% of the material recovered was coarser than 25 mm (1 in.). The first full-scale drop tests at Mount Isa involved dropping several tens of metric tons of surface-quarried rock about 530 m (1,600 ft) down a vertical shaft, with particle size analyses before and after.

Binder Placement

The stiffener in a cemented rock fill can consist of deslimed mill tailings fill, cemented deslimed mill tailings fill, paste fill, portland cement, fly ash or other pozzolan, or a combination of two or more of these. All of these binders are transported most easily by pipeline to be added to the rock fill as it enters the top or some more-appropriate location in the stope. Standardization of such practice is difficult and large stopes must be considered on a case-by-case basis.

Paste Fill System Design

With paste fill, the tailings become a high-density slurry with a solids content of 75%–80% by mass. The slurry acts as a viscous, Bingham solid, for which a critical shear stress must be exceeded before the material will flow. The critical shear stress for paste fills typically ranges between 250 and 800 Pa (0.041 to 0.116 psi) (Boger et al. 2006).

Because of the viscous nature of the paste and the large amounts of energy required to transport paste in the turbulent regime, it is customary to transport paste in the laminar flow regime (Paterson 2006). The drawback to laminar flow, however, is the potential for sedimentation and blockage in the pipeline, though paste fill, because of its tightly controlled production and relatively high pulp density, is less prone to settling than conventional deslimed mill tailings fill. Two other important attributes of paste fill are (1) the water is bound into the slurry, so that little drainage is required; and (2) the fact that fines removal from tailings may not be required.

Paste Particle Size Requirements

At least 15% of the particles in the paste should be finer than 20 μm (0.00079 in.) (Hassani and Archibald 1998).

In addition, it is customary to add from 3% to 6% portland cement or a mixture of portland cement and fly ash as a binder. If the tailings contain insufficient fines, it may be necessary to add fly ash or additional cement.

Pump Requirements

The critical issue for pumping the paste from the backfill plant to the mine stopes is the initial shear stress that must be overcome before the paste will flow (Abulnaga 2002):

$$P_{st} = 4\,\tau_0\,L/D_i \tag{13.6-5}$$

where
P_{st} = start-up pressure
τ_0 = initial shear strength of the paste
L = pipe length
D_i = inside pipe diameter

The initial shear stress to be overcome, or yield stress, can be measured by the vane shear test. Alternatively, it is becoming more common to estimate the yield stress from a slump test using the following relationship (Boger et al. 2006):

$$\tau_0/\rho g h = 0.5 + 0.5(s/h)^{0.5} \tag{13.6-6}$$

where
s = measured slump
h = height of the slump cylinder

Because of the viscosity and the density of the paste, positive displacement pumps are required. These pumps have the attribute that they can pump against any head up to the design head of the machine while the flow rate remains constant. These pumps should be fitted with pressure relief valves because they will continue to pump against blocked or closed pipes, which could result in a ruptured pipe or a damaged pump.

Pipe Wear

Few data are available regarding the wear of pipes transporting paste fill. However, because the paste moves as a viscous unit, it is expected that the wear rates are less than for conventional hydraulic fills. The wear can be checked in vertical sections of the pipelines using a three-legged caliper probe. It is recommended that readings be taken after 15,000 t (16,535 st) have been transported and thereafter at 50,000-t (55,116-st) intervals.

Backfill Plant Requirements

Components in a paste fill plant typically include tailings storage tanks with a 1-day live storage capacity; paste thickeners for reducing water content from 65% by mass to approximately 20% by mass; silos for cement (and possibly fly ash) storage; weightometer hoppers for tailings, cement, and fly ash (if used); mixers for blending tailings, fly ash, and cement (if used); and positive displacement pumps.

Pneumatic Stowing

Pneumatic stowing is often used in situations where hydraulic filling has significant drawbacks. Examples are situations where it is undesirable to handle the large quantities of water that drain off from the fill or where seams are thin and flat-lying so that hydraulic placement of fill tight to the roof is difficult to achieve. Pneumatic stowing has also been considered as a means of placing broken rock whose ore grade is too low for recovery by conventional means in order to extract a mineral by pressure or flood leaching.

Transport Considerations

For successful pneumatic stowing, the air velocity must be sufficient to move the solids. The air velocity at which the solids begin to move is the critical transport velocity. For vertical conveying, the critical transport velocity equals the terminal velocity, which is given by either Stokes' law or the Newton–Rittinger equation depending on whether the flow is laminar or turbulent. For pneumatic transport, the flow of both air and the solids is turbulent, and the terminal velocity is given by the Newton–Rittinger equation:

$$v_t = [4gD_i(\rho_s - \rho_a)/(3\rho_a C_d)]^{0.5} \tag{13.6-7}$$

where
v_t = terminal velocity
ρ_a = air density

For the fully turbulent regime, C_d has an essentially constant value of 0.44. The air density term in the numerator of Equation 13.6-7 can be ignored. For horizontal conveying, the critical velocity is approximately $0.7v_t$. For satisfactory flow, the air velocity should be four to five times the critical velocity.

Particle Size Distribution

The particle size distribution will affect the velocity necessary to transport the solids and the pipe size. To minimize the possibility of bridging of large chunks, the pipe diameter should be at least three times that of the largest chunk carried. As a practical matter, the maximum lump size should be 100 mm (4 in.) or less, with 75 mm (3 in.) preferred. Considered another way, typical stowing pipe diameters range from 150 mm (6 in.) up to 225 mm (9 in.). Consequently, the maximum lump size should preferably be less than 75 mm (3 in.), and ideally 50 mm (2 in.) or less. A smaller maximum particle size will also result in reduced velocity, power requirements, and pipe wear.

Feeder and Blower Considerations

A pneumatic stowing system has three basic components:

1. A hopper and feeder for the solids
2. An air supply consisting of compressed air from either a pipeline or compressor, or a rotary blower
3. An injection pipe and nozzle

The hopper should be large enough to contain a bucket load of broken material from a typical haulage vehicle. The discharge opening in the hopper should have a minimum dimension of at least three to six times the dimension of the largest piece being transported.

The feeder should be sized to handle the expected hourly throughput of the system. Three important considerations for the feeder, which is generally a rotary airlock type, are the number of vanes on the rotor, diameter of the airlock, and the rotor rpm. For a given flow rate in cubic meters per hour, better flow characteristics are obtained with a larger diameter airlock operated at a lower rpm. Six or eight vane models are available. In some applications, difficulties have been experienced with the eight-vane units.

Table 13.6-4 Estimated fill throughput through different types of pipe before replacement

Type of Pipe	Pipe Life, 1,000 m^3 of material passed
Standard Schedule 40 steel pipe	8 to 30
Twin-wall pipe	20 to 150
Steel pipe with abrasion-resistant basalt lining	200 to 1,500

Source: Adapted from Jerabek and Hartmann 1966.

Except for very large tonnage operations, low-pressure systems using positive displacement blowers provide satisfactory air quantities for stowing. These blowers are noisy and should be fitted with silencers. Injection pipes are typically 150–225 mm (6–9 in.) in diameter. Air velocities in the stowing pipe should exceed 20 m/s (65 fps), and the exit velocity in the nozzle should be about 40 m/s (130 fps) for optimum results. Stowing machines are available to place up to 280 m^3/h (367 yd^3/h).

Solids Loading and System Pressure Drop

An important parameter is the solids-to-air ratio by weight. For most stowing applications this ratio will range from 10 to 40. However, the actual value of the ratio for a given stowing operation will depend on the quantity and size of the solids to be stowed, given that with coarse material the velocity required to keep the solids in the air stream will be higher than for fine material.

The system pressure drop consists of the pressure drop due to the air in the system and the pressure drop due to the solids. The air pressure drop is given by the Darcy–Weisbach equation in the following form:

$$\Delta p_a = p_a^2 - p_2^2 = f\,(L/D_i)(V_a^2/2g)\gamma_{a1}p_a \qquad (13.6\text{-}8)$$

where
p_a = initial air pressure
p_2 = mine atmospheric pressure
f = friction factor = 0.016
L = pipe length
D_i = pipe internal diameter
V_a = pipe air velocity
γ_{a1} = specific gravity of mine air

Based on tests performed by Peter (1952), the total system pressure drop for a given solid/air ratio can be estimated from the following equation:

$$S = (0.2086 - 0.0001\,EQ)R_s + 1 \qquad (13.6\text{-}9)$$

where
S = ratio of system pressure drop to air pressure drop
E = electrostatic charge
Q = airflow rate, m^3/h
R_s = solids/air ratio by weight

Usually the equation shows the product of EQ in the direction of flow for determining the pressure drop. Thus, the presence of statically charged particles increases the pressure drop (Smeltzer et al. 1982). After the system pressure drop is known, the system power requirements can be determined.

Power Requirements

The power required to transport the solids is given by

$$P_b = RTQ\,\gamma_{a1}\ln((R_s\,\Delta p_a + p_2)/p_2) \qquad (13.6\text{-}10)$$

where
P_b = blower power
R = universal gas constant
T = absolute temperature (degrees K in metric units or degrees Rankine in English units)
Q = airflow rate
γ_{a1} = mine air density

Additional power will be required for the feeder as follows:

$$P_f = Q\,\gamma_{a1}\,D_f \qquad (13.6\text{-}11)$$

where
P_f = feeder power
D_f = feeder airlock diameter

It will be necessary in Equations 13.6-10 and 13.6-11 to include appropriate conversion factors to obtain the power in metric or English units.

Pipe Wear

Broken rock stowed pneumatically wears pipe rapidly, especially at bends. For this reason, standard Schedule 40 steel pipe should not be used. Heavy-duty (Schedule 80) pipe could be used; however, there are special multiwall pipes fabricated especially for stowing. Another option is pipes containing abrasion-resistant basalt pipe linings. Estimates of stowing pipe life for different types of pipe are presented in Table 13.6-4. Pipe life can be maximized by rotating the pipe one-sixth of a revolution at frequent intervals.

Flowable Fill System Design

Flowable fill is a self-consolidating, low-strength material used as an alternative to consolidated granular fill. It is an excellent solution for filling inaccessible areas such as abandoned mines and is used for this purpose by OSMRE and the Pennsylvania Department of Environmental Protection (PaDEP), among others. The mix proportions to provide 1 m^3 (1.31 yd^3) of the flowable fills used by these agencies are presented in Table 13.6-5.

Fly ash must conform in general to mineral admixture Class F but must have a moisture content not greater than 20% and a maximum loss on ignition of 6%. Black or brown fly ash will not be allowed unless certified lab results show the loss on ignition to be 6% or less. The sand must be 45%–80% finer than 1.19 mm (16 mesh) and approximately 10%–30% finer than 297 μm (50 mesh). The gravel must consist of hard, dense, durable uncoated gravel or crushed stone with a grain size distribution as follows:

Percent Finer, by Mass	Particle Size
100	19 mm (3/4 in.)
40–70	9.5 mm (3/8 in.)
0–15	4.76 mm (4 mesh)

Table 13.6-5 Flowable fill mixes for backfilling abandoned mines

Mix No.	Portland Cement, kg	Sand, kg	Gravel, kg	Fly Ash, kg	Water, L*
OSMRE No. 1	233	0	0	1,130	470
OSMRE No. 2	219	854	384	531	272
OSMRE No. 3	201	785	0	816	272
OSMRE No. 4	296	0	0	1,186	401
PaDEP	167	0	0	1,186	247

*Water contents may be modified from those shown to obtain sufficient slump for the mixture to flow.

Because flowable fill has a high slump by definition, gravity flow from the mixing tank will suffice for most applications. Other agencies that deal with abandoned mines may use other mixes.

Flowable Fill Equipment Requirements

Flowable fill is generally used to backfill abandoned mines. Consequently, it will generally not be possible to direct the flow of material from underground. Therefore, multiple access points from the surface are usually necessary, and storage silos and mixing equipment must be portable. Portable concrete batch mixers are readily available that can handle up to 153 m^3/h (200 yd^3/h). Portable silos for sand and fly ash storage can also be hired. Another option is to contract with a local concrete Ready Mix supplier, indicating the mix to be supplied.

OTHER CONSIDERATIONS

Remote Placement of Backfills

Backfilling of former mines as a subsidence-prevention measure is becoming increasingly common. In general, it will be necessary to provide multiple pour points to ensure the void will be as completely filled as possible. Similar to discharge from a stacker or conveyor, the solid discharge from a fill pipe will form a cone that is characterized by finer and finer particles as it moves away from the discharge point. The angle of repose of the cone depends on the solids percentage in the slurry—the higher the percent solids, the steeper the cone. Because a supply bore will become useless after the cone fills to the top of the opening, the spacing between supply borings depends on the height of the opening, as well as the percent solids in the fill.

Automated Systems

Because of the generally batch nature of filling and the need to change discharge locations during the course of backfill placement, not all facets of backfilling lend themselves to automatic operation. However, some portions of the cycle, such as starting and stopping of pumps and opening and closing of hopper gates, can be automated by means of interlocks, level controls, and other automatic controls. For example, the Manitoba Division of Vale Inco installed an automated fill pumping system at the Thompson mine in the 1960s. In theory, the valving on the discharge locations could be automated and interlocked with level switches placed below the discharges; however, the result would be a pouring system in the stope that would be significantly more difficult to install and dismantle.

Effect of Cemented Fill in Mill Feed on Processing Plant Recovery

When mucking on a fill floor, it is inevitable that some fill will be mixed with the broken ore. If the fill is uncemented, the only negative effect is some dilution. However, if the fill is cemented, there are likely to be unwanted effects on the concentrator circuit, especially in the case of a silver/lead/zinc ore. Specifically, the cement will significantly increase the pH of the process water because the pH of hydrous cement is approximately 10.5–11. In addition, the presence of calcium carbonate may chemically affect milling processes. Occasionally, the presence of cement will improve mill recovery percentages.

Heat Generation of Sulfide Backfills

Tailings containing pyrite and especially some forms of pyrrhotite readily oxidize, giving off heat and absorbing oxygen. As a consequence, it is not uncommon for the temperature in a stope to rise by 3°–5°C (5°–10°F) after the first lift of fill has been placed. However, the oxidation process may also result in additional cementing of the fill. Noranda took advantage of this at the Horne mine in Quebec when designing the hydraulic backfill system there in the 1930s (Patton 1957). One unfortunate consequence of this was loss of life by removal of oxygen from mine air. Also, it is reported that in the Kimberley mine in British Columbia, iron sulfides in placed rock fill increased fill temperatures to such an extent that fill behind bulkheads became molten.

SOURCES OF FURTHER INFORMATION

As indicated in the introduction, this chapter makes no claim for completeness. The following publications are aimed specifically at design of backfill systems:

- *Handbook on Mine Fill* (Potvin et al. 2005)
- *Paste and Thickened Tailings: A Guide*, 2nd ed. (Jewell and Fourie 2006)
- *Mine Backfill 1998* (Hassani and Archibald 1998)

These publications do not cover pneumatic stowing or flowable fill.

Numerous case histories of backfill systems can be found in the proceedings volumes of the International Symposia on Mining with Backfill that have been held approximately every 4 years since the first one was organized by Ed Thomas, the late Ken Mathews, Jack Barrett, and others at Mount Isa in Australia in 1973 under the auspices of the Australasian Institute of Mining and Metallurgy. Another source of case histories is the proceedings of the International Symposiums on Paste and Thickened Tailings that have been held since the 1990s, which have generally been published by the Australian Centre for Geomechanics.

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CHAPTER 13.7

Cave Mining

Dennis H. Laubscher

INTRODUCTION

The expression *cave mining* will be used in this chapter to refer to all mining operations in which the ore body caves naturally after undercutting and the caved material is recovered through drawpoints. The term encompasses block caving, panel caving, inclined-drawpoint caving, and front caving. Caving is the lowest-cost method of underground mining, provided that drawpoint size and handling facilities are tailored to suit the caved material and that the extraction horizon can be maintained for the life of the draw.

The daily production from cave mining operations throughout the world (in 2008) was approximately 503 kt/d. Table 13.7-1 shows production broken down by layout.

Today, several open-pit mines currently producing in excess of 50 kt/d are examining the feasibility of implementing low-cost, large-scale underground mining methods. Several underground cave mines that produce high tonnages are planning to implement caving heights (or "dropdowns") of 500 m and at depths of 1,500 m. This will result in a considerable change in their mining environments. These changes will necessitate more detailed mine planning, rather than the simple projection of current mining methods to greater depths.

As more attention is directed to mining large, competent ore bodies with low-cost underground methods, it is necessary to define the role of cave mining. In the past, caving has been considered for rock masses that cave and fragment readily. The ability to better assess the cavability and fragmentation of ore bodies, the availability of robust load-haul-dump (LHD) machines, an understanding of the draw control process, suitable equipment for secondary drilling and blasting, and reliable cost data have shown that competent ore bodies with coarse fragmentation can be mined at a much lower cost using caving rather than with drill-and-blast methods. However, after a cave layout has been developed, there is little scope for change.

Aspects that have to be addressed are cavability, fragmentation, draw patterns for different types of ore, drawpoint or drawzone spacing, layout design, undercutting sequence, and support design. Table 13.7-2 shows some significant anomalies in the quoted performance of different cave operations.

It is common to find that old established mines that have developed standards during the course of successful mining in the upper levels of an ore body are resistant to change and do not adjust to the ground control problems that occur as mining proceeds to greater depths or as rock types change. Mines that have experienced continuous problems are more amenable to adopting new techniques to cope with a changing mining situation. Detailed knowledge of local and regional structural geology, the use of an accepted rock mass classification to characterize the rock mass, and knowledge of regional and induced stress environments are prerequisites for good mine planning. It is encouraging that these aspects are receiving more and more attention.

Table 13.7-1 Production per day obtained using different mine layouts in 2008

Mine Layout	Daily Production, kt
Grizzly	0
Slusher	3
Load-haul-dump machine	±500
Total	503

Table 13.7-2 Anomalies in quoted performance of caving operations

Quote	Explanation
96% of ore recovered for 100% mineral extraction	Underevaluation of ore body and dilution zone
Correct drawpoint spacing, but occurrences of 200% overdraw with 30% waste dilution entry	Highly irregular draw and underevaluation of dilution
15% dilution entry in spite of correct drawpoint spacing and uniform fragmentation	Drawpoints being drawn in isolation
Ore from lower 100 m of draw column still reporting in drawpoint, even though 260 m of ore has been drawn	Large range in fragmentation and irregular, high values in dilution zone

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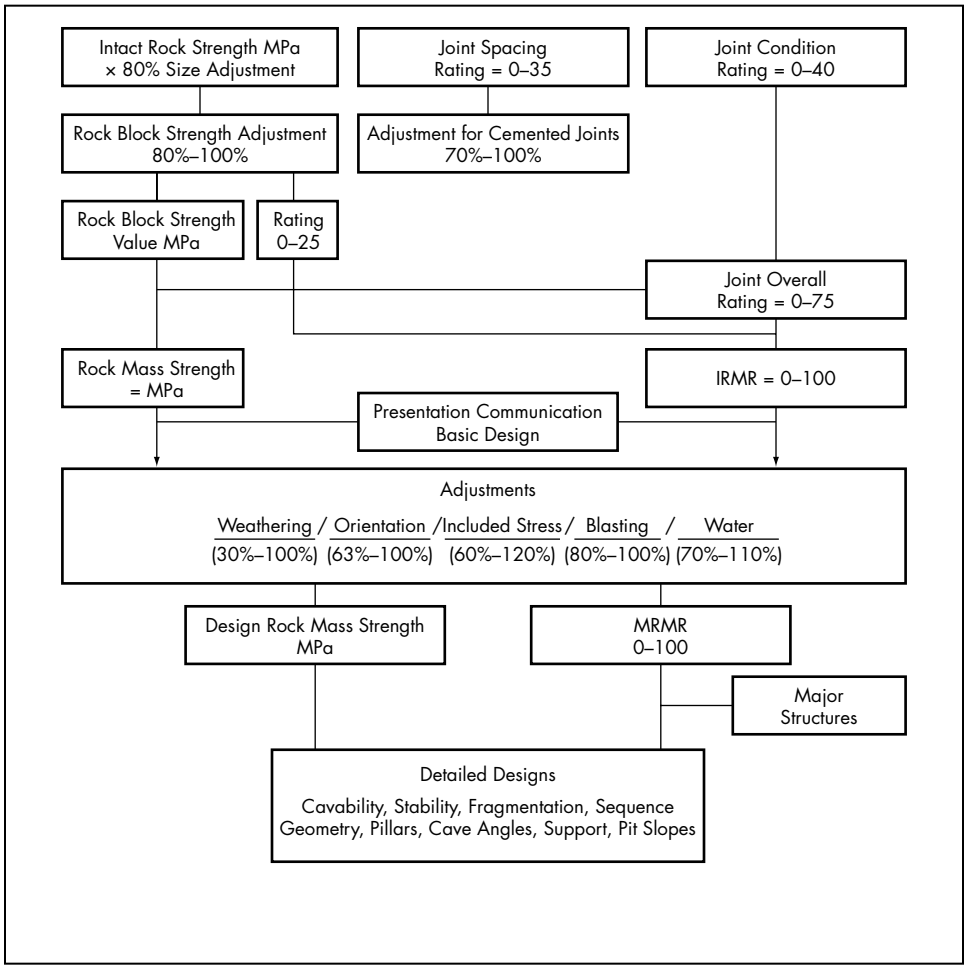


Figure 13.7-1 Flow sheet of MRMR procedure with recent modifications

The Laubscher rock mass classification system provides both in-situ rock mass ratings (IRMRs) and rock mass strength. Such a classification is necessary for design purposes. The IRMR defines the geological environment, and the adjusted or modified rock mass ratings (MRMRs) consider the effects of the mining operation on the rock mass. Figure 13.7-1 is a flow sheet of the MRMR procedure with recent modifications. The reader is encouraged to read the chapter "The IRMR/MRMR Rock Mass Classification for Jointed Rock Masses" by Laubscher and Jakubec (2001). The ratings describe in detail cavability, subsidence angles, failure zones, fragmentation, undercut-face shape, cave-front orientation, undercutting sequence, overall mining sequence, and support design.

FACTORS AFFECTING CAVING OPERATIONS

The 25 parameters that should be considered before implementing any cave mining operation are set out in Table 13.7-3. The parameters in bold are a function of the parameters that follow. Many parameters are uniquely defined by the ore body and the mining system and are not discussed further. The parameters considered later are common to all cave mining systems and need to be addressed if any form of cave mining is contemplated.

Cavability

Monitoring many caving operations has shown that two types of caving can occur—stress and subsidence caving. However, it is better to use the terms *vertical extension* to mean upward propagation of the cave and *lateral extension* to mean the propagation of the cave as a caved block is expanded.

Vertical extension caving occurs in virgin cave blocks when the stresses in the cave back exceed the strength of the rock mass. Caving can stop when a stable arch develops in the cave back. The undercut must be increased in size or the boundaries weakened to induce further caving. High horizontal stresses acting on steep dipping joints increase the MRMR. This was the situation in block 16 at the Shabanie mine in Zimbabwe. A stable back was formed when the undercut had a hydraulic radius of 28 and the MRMR was 64. When block 7 (adjacent to block 16) caved, the horizontal confining stress was removed, which resulted in a reduction in the MRMR to 56 in block 16. At this point, caving occurred.

Lateral extension caving occurs when lateral restraints on the block being caved are removed by mining adjacent to the block. This often results in a large stress difference, leading to rapid propagation of the cave and limited hulking.

Table 13.7-3 Parameters to be considered before implementation of cave mining

Cavability	**Primary Fragmentation**	**Drawpoint/Drawzone Spacing**
Rock mass strength (rock mass rating [RMR]/MRMR)	Rock mass strength (RMR/MRMR)	Particle size of ore and overlying rock
Rock mass structure-condition geometry	Geological structures	Overburden load and direction
In-situ stress	Joint and fracture spacing, and geometry	Friction angles of caved particles
Induced stress	Joint condition ratings	Practical excavation size
Hydraulic radius of ore body	Stress or subsidence caving	Stability of host rock mass (MRMR)
Water	Induced stress	Induced stress
Draw Heights	**Layout**	**Rock Burst Potential**
Capital	Particle size	Regional and induced stresses
Ore-body geometry	Drawpoint spacing and size	Variations in rock mass strength, modulus
Excavation stability	Method of draw—gravity or LHD	Structures
Effect on ore minerals	Orientation of structures and joints	Mining sequence
Method of draw	Ventilation, ore handling, drainage	
Sequence	**Undercutting Sequence (pre/advance/post)**	**Induced Cave Stresses**
Cavability: poor to good or vice versa	Regional stresses	Regional stresses
Ore body geometry	Rock mass strength	Area of undercut
Induced stresses	Rock burst potential	Shape of undercut
Geological environment	Rate of advance	Rate of undercutting
Rock burst potential	Ore requirements	Rate of draw
Production requirements	Completeness of undercut	
Influence on adjacent operations	Shape (lead, lag)	
Water inflow	Height of undercut	
Drilling and Blasting	**Development**	**Excavation Stability**
Rock mass strength	Layout	Rock mass strength (RMR/MRMR)
Rock mass stability (drill-hole closure)	Sequence	Orientation of structures and joints
Required particle size	Production	Regional and induced stresses
Hole diameter, lengths, rigs	Drilling and blasting	Rock burst potential
Patterns and directions		Excavation size (orientation and shape)
Powder factor		Drawpoint
Swell relief		Mining sequence
Support	**Practical Excavation Size**	**Method of Draw**
Excavation stability	Excavation stability	Fragmentation
Rock burst potential	Induced stress	Practical drawpoint spacing
Brow stability	Caving stresses	Practical size of excavation
Timing of support: initial, secondary, and production	Secondary blasting	Gravity or mechanical loading
	Equipment size	
Rate of Draw	**Drawpoint Interaction**	**Draw-Column Stresses**
Fragmentation	Drawzone spacing	Draw-column height
Method of draw	Critical distance across major apex	Particle size
Percentage hangups	Particle size	Homogeneity of ore particle size
Secondary breaking/blasting	Time frame of working drawpoints	Draw control
Seismic events		Draw-height interaction
Air blasts–drawpoint cover		Height-to-short-axis base ratio
		Direction of draw
Secondary Fragmentation	**Secondary Blasting/Breaking**	**Dilution**
Rock, block shape	Secondary fragmentation	Ore-body geometry
Draw height	Draw method	Mining geometry
Draw rate, time-dependent failure	Drawpoint size	Particle size distribution
Rock block workability and strength	Gravity grizzly aperture	Range of particles, unpay ore and waste
Range in particle size, fines cushioning	Size of equipment and grizzly spacing	Grade distribution of pay and unpay ore
Draw control program	Ore handling system, size restrictions	Mineral distribution in ore
		Drawpoint interaction
		Secondary breaking
		Draw control (techniques, predictions)
		Draw markers
Tonnage Drawn	**Support Repair**	**Ore/Grade Extraction**
Level interval	Tonnage drawn	Mineral distribution
Shut-off grade	Point and column loading	Method of draw
Drawpoint spacing	Brow wear	Rate of draw
Dilution percentage	Floor repair	Dilution percentage
Controls	Secondary blasting	Cutoff grade to lant
Redistribution		Ore losses
Subsidence		
RMR/MRMR		
Height of caved column		
Minimum and maximum spans		
Major geological structures		
Depth of mining		
Topography		

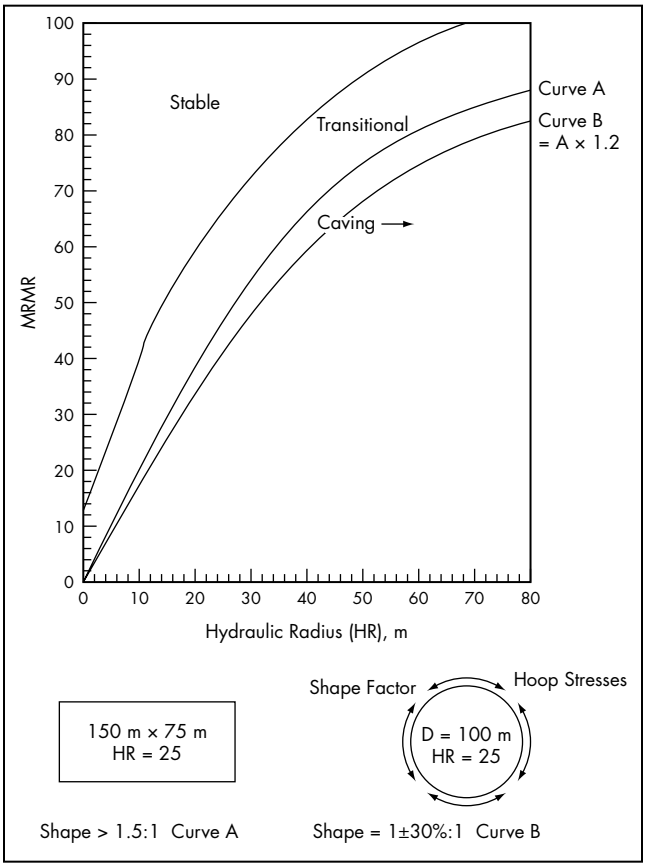


Figure 13.7-2 Stability diagram based on worldwide experience

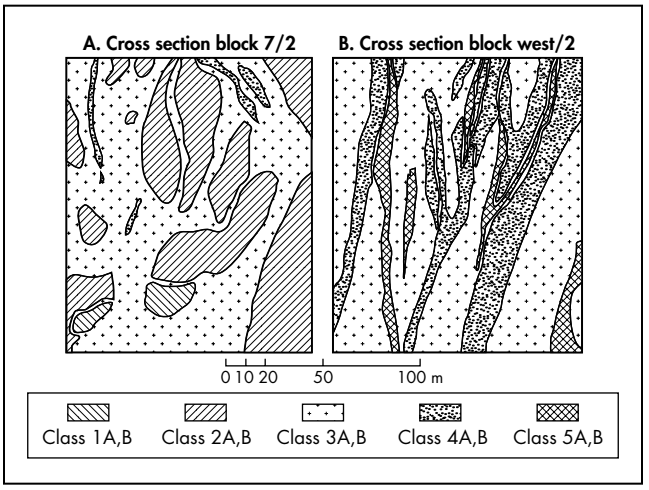


Figure 13.7-3 Geomechanics classification data

Figure 13.7-2, based on a worldwide experience base, illustrates caving and stable situations in terms of the hydraulic radius (area divided by perimeter) for a range of MRMR values. An additional curve has been added to account for the stability that occurs with equidimensional shapes.

All rock masses will cave. The manner of their caving and the fragmentation need to be predicted if cave mining is to be implemented successfully. The rate of caving can be slowed by control of the draw since the cave can propagate only if there is space into which the rock can move. The rate of caving can be increased by a more rapid advance of the undercut, but problems can arise if an air gap forms over a large area. In this situation, the intersection of major structures, heavy blasting, and the influx of water can result in damaging air blasts. Rapid, uncontrolled caving can result in an early influx of waste.

In conventional layouts, the rate of undercutting (RU) should be controlled so that the rate of caving (RC) is faster than the rate of damage (RD) due to abutment stresses. Thus, RC > RU > RD.

However, in areas of high stress, the rate of caving must be controlled to maintain an acceptable level/amount of seismic activity in the cave back. Otherwise, rock bursts can occur in suitably stressed areas (pillars and rock contacts). As advanced undercutting will be used in these situations, damage to the undercut and production levels will not be a problem.

The stresses in the cave back can be modified to some extent by the shape of the cave front. Numerical modeling can be a useful tool for helping an engineer determine the stress patterns associated with possible mining sequences. An undercut face that is concave toward the caved area provides better control of major structures. In ore bodies having a range of MRMRs, the onset of continuous caving will be in the lower-rated zones if they are continuous in plan and section. This effect is illustrated in Figure 13.7-3B, where the class 5 and 4B zones are shown to be continuous. In Figure 13.7-3A, the pods of class 2 rock are sufficiently large to influence caving, and cavability should be based on the rating of these pods. Good geotechnical information, as well as information from monitoring the rate of caving and rock mass damage, is needed to fine-tune this relationship.

Particle Size Distribution

In caving operations, the degree of fragmentation has a bearing on

- Drawpoint spacing,
- Dilution entry into the draw column,
- Draw control,
- Draw productivity,
- Secondary blasting and breaking costs, and
- Secondary blasting damage.

The input data needed to calculate the primary fragmentation and the factors that determine the secondary fragmentation as a function of the caving operation are shown in Figure 13.7-4.

Primary fragmentation can be defined as the size distribution of the particles that separate from the cave back and enter the draw column. The primary fragmentation generated by subsidence caving is generally more coarse than that resulting from stress caving. The reasons for this are (1) propagation of the cave occurs more rapidly, (2) the rock mass disintegrates primarily along favorably oriented joint sets, and (3) there is little shearing of intact rock. The orientation of the cave front or back with respect to the joint sets and the direction of principal stress can have a significant effect on the primary fragmentation.

Secondary fragmentation is the reduction in size of the original particles as they move through the draw column. The processes to which particles are subjected determine the size distribution that reports to the drawpoint. A strong, well-jointed

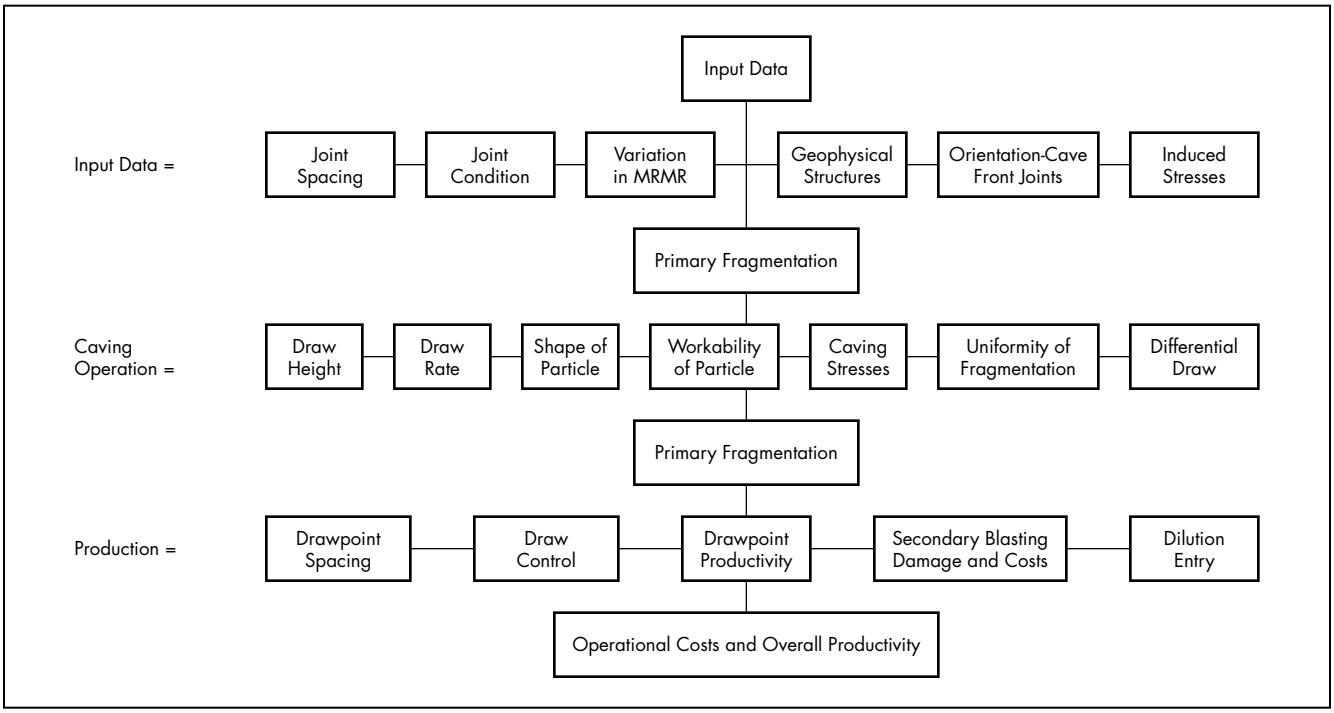


Figure 13.7-4 Input data for calculation of fragmentation

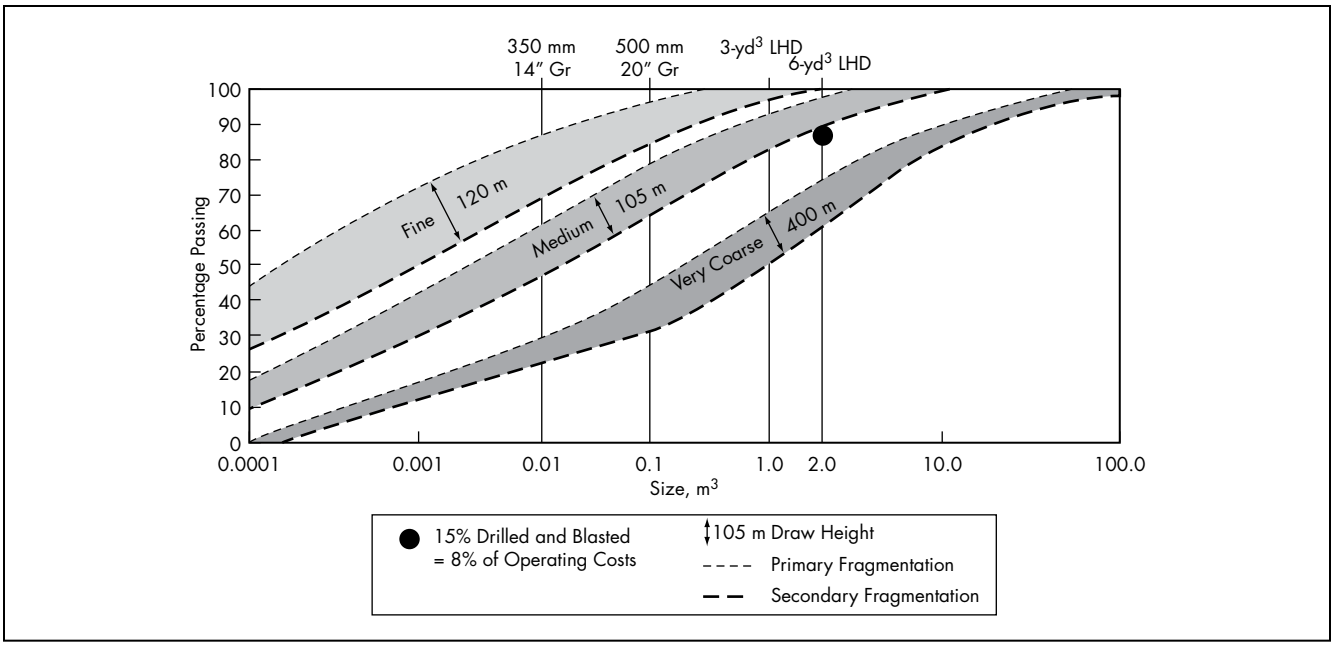


Figure 13.7-5 Size distribution of cave fragmentation

material can result in a stable particle shape at a low draw height. Figure 13.7-5 shows the decrease in particle size for different draw heights and coarse (less jointed) to fine (well jointed) rock masses. A range of RMRs will result in a wider range of particle sizes than that produced by rock with a single rating. This is because the fine material tends to cushion larger blocks and prevents further attrition of these blocks. This difference is illustrated in Figure 13.7-3B, in which class 5 and class 4 material is shown to cushion the larger primary fragments from class 3 material. A slow rate of draw results in a higher probability of time-dependent failure as the caving stresses have more time to work on particles in the draw column.

Fragmentation is the major factor determining drawpoint productivity. Experience has shown that 2 m^3 is the largest block that can be moved by a 6-yd^3 LHD machine and still allow an acceptable rate of production to be maintained. In Figure 13.7-6, the productivity of a layout using 3.5-, 6-, and

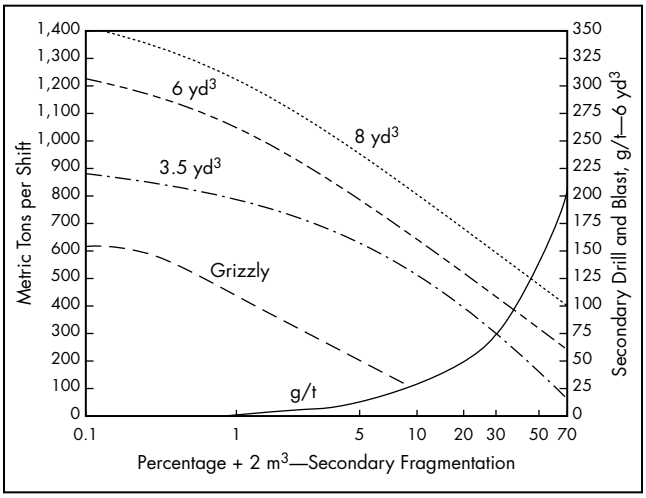


Figure 13.7-6 LHD productivity on a round trip of 100 m with 60% utilization

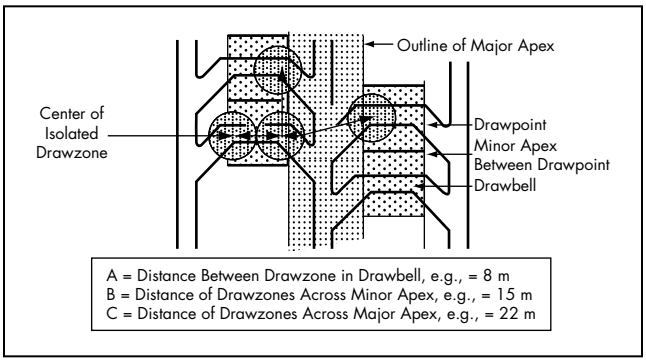


Figure 13.7-7 Maximum and minimum drawzone spacing (isolated drawzone = 10 m, area of influence = 225 m^2)

8-yd^3 LHD machines loading to a grizzly is related to the percentage of fragments larger than 2 m^3. The use of secondary explosives is based on the amount of oversized rock that cannot be handled by a 6-yd^3 LHD.

A computer simulation program has been developed for predicting primary and secondary fragmentation. The results obtained from this program are confirmed by underground observations.

Drawzone Spacing

Drawpoint spacings for grizzly and slusher layouts reflect the spacing of the drawzones. However, in the case of LHD layouts with a nominal drawpoint spacing of 15 m, drawzone spacing can vary from 18 to 24 m across the major apex (pillar), depending on the length of the drawbell. This situation occurs when the length of the drawpoint crosscut is increased to ensure that an LHD machine is straight before it loads. In this case, the major consideration (optimum ore recovery) is compromised by incorrect use of equipment or a desire to achieve ideal loading conditions. The drawzone spacings for 30-m center-to-center production drift spacings are shown in Figure 13.7-7, with the critical distance being across the major apex. The length of the drawbell can be increased from 8 to

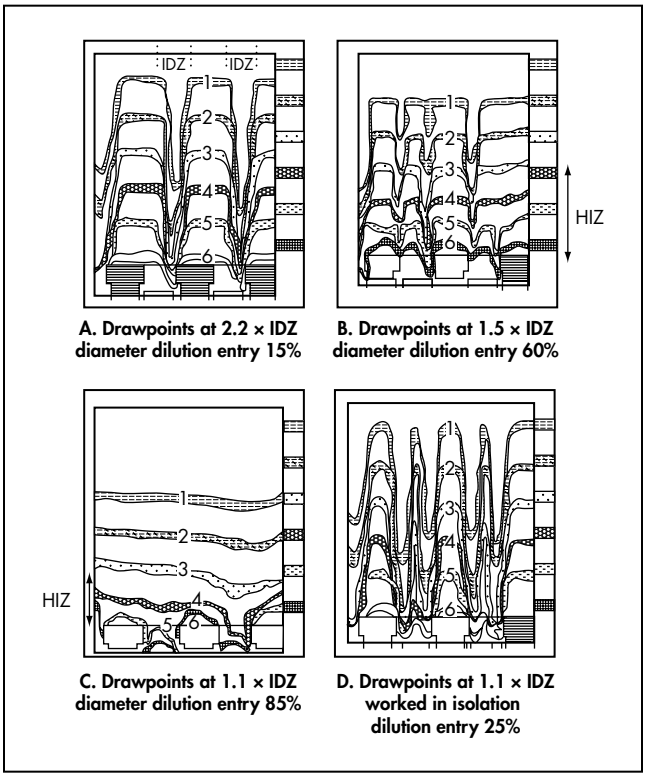


Figure 13.7-8 Results of three-dimensional sand-model experiments

10 m and the production drift spacing to 32 m without affecting the major apex spacing. Interaction in the drawbell will still occur.

Sand model tests have shown a relationship between the spacing of drawpoints and the interaction of drawzones. Widely spaced drawpoints develop isolated drawzones with diameters defined by the fragmentation. When drawpoints are spaced at 1.5 times the diameter of the isolated drawzone (IDZ), interaction occurs. Interaction also improves as drawpoint spacing is decreased, as shown in Figure 13.7-8. The flow lines and the stresses that develop around a drawzone are shown in Figure 13.7-9. The sand model results have been confirmed by observation of the fine material extracted during cave mining and by the behavior of material in bins. The question is whether this theory, which is based on IDZs, can be wholly applied to coarse material, where arching (spans) of 20 m have been observed. The collapse of large arches will affect the large area overlying the drawpoint, as shown in Figure 13.7-10. The formation and failure of arches leads to wide drawzones in coarse material, so that drawzone spacing can be increased to the spacings shown in Figure 13.7-10.

The frictional properties of the caved material must also be recognized. Low-friction material can flow greater distances when under high overburden load, and this can mean wider drawzone spacings. It is therefore logical to expect that when a line of drawbells is drawn, this creates a large low-density zone. Similarly, when lines of drawbells are drawn on alternate shifts, good interaction will result. The draw program at the Henderson mine in Colorado (United States) broadly follows this pattern.

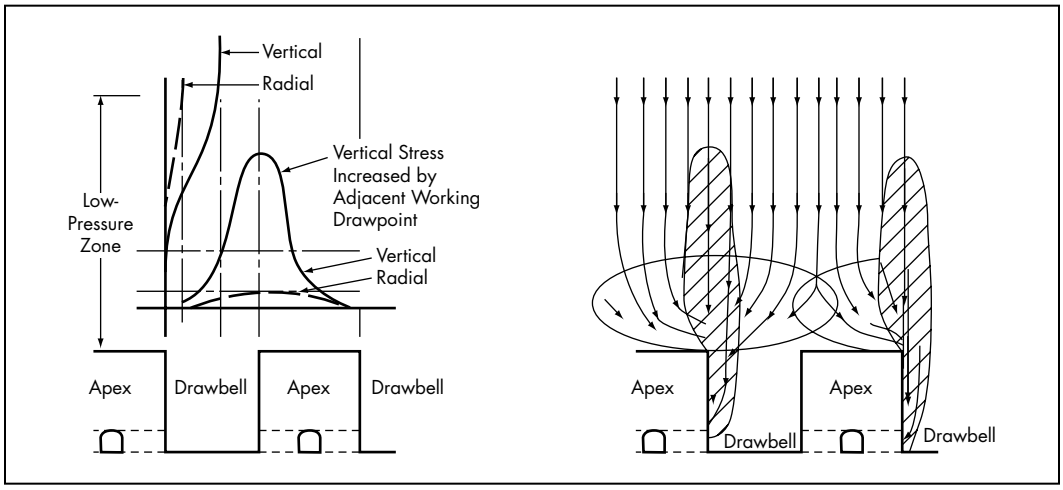


Figure 13.7-9 Flow lines and inferred stresses between adjacent working operations

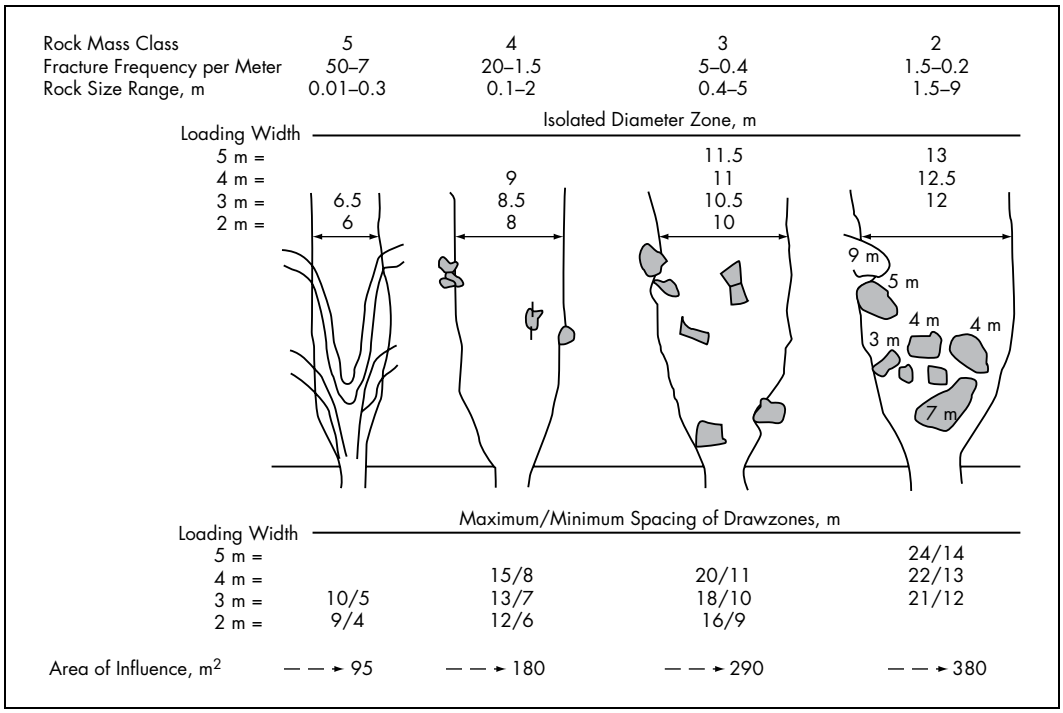


Figure 13.7-10 Maximum/minimum spacing of drawzones based on isolated drawzone diameter

Three-dimensional model tests need to be continued to establish some poorly defined principles, such as the interaction across major apexes when the spacing of groups of interactive drawzones is increased. Numerical modeling could possibly provide the solutions for the draw behavior of coarsely fragmented material.

Draw Control

Draw control requirements and applications are shown in Figure 13.7-11 and Table 13.7-4, respectively. The grade and fragmentation in the dilution zone must be known if sound draw control is to be practiced. Figure 13.7-12 shows the value distribution for columns A and B. Both columns have the same average grade of 1.4%, but the value distribution is different. The high grade at the top of column B means that a larger tonnage of waste can be tolerated before the shutoff grade is reached.

In LHD layouts, a major factor in poor draw control is the drawing of fine material at the expense of coarse material. Strict draw control discipline is required so that the coarse ore is drilled and blasted at the end of the shift in which it reported in the drawpoint.

It has been established that the draw will angle toward less dense areas. This principle can be used to move the material overlying the major apex by creating zones of varying density through the differential draw of lines of drawbells.

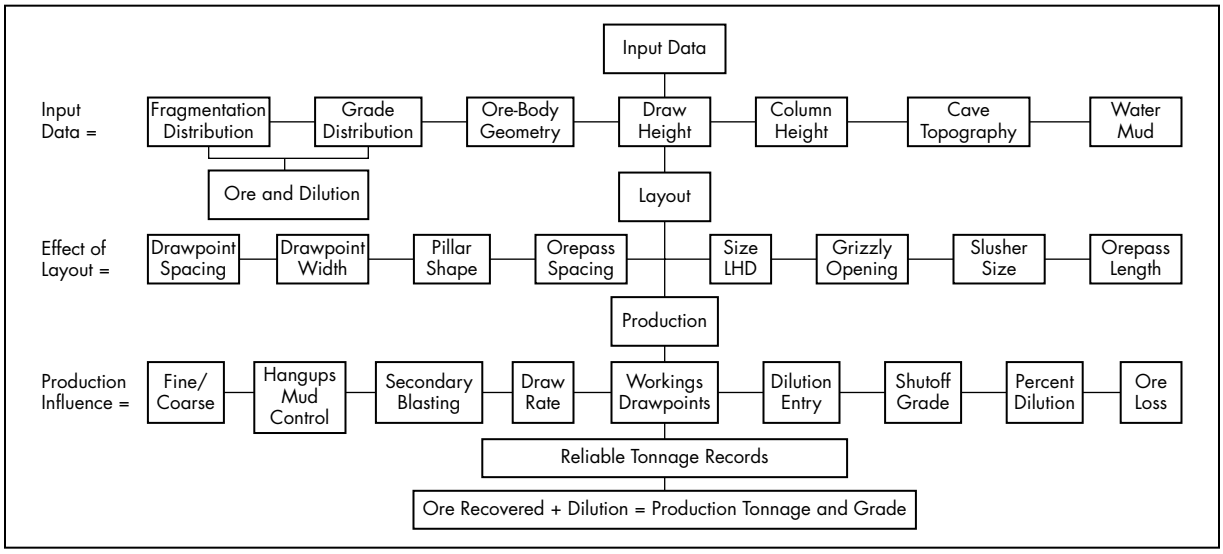


Figure 13.7-11 Draw control requirements

Table 13.7-4 Draw control applications

Objective	Application
Reduce dilution	Calculation of tonnage
Improve ore recovery	Recording of tonnages produced
Avoid damage to pillars	Controlling draw

Dilution

For the purposes of block caving, the *percentage of dilution* is defined as the percentage of the ore column that has been drawn before the waste material appears in the drawpoint. It is a function of the amount of mixing that occurs in the draw columns. Mixing is a function of

- Ore draw height,
- Range in fragmentation of both ore and waste,
- Drawzone spacing, and
- Range in tonnages drawn from drawpoints.

The range in particle size distribution and the minimum drawzone spacing across the major apex will give the height of the interaction zone (HIZ). This is illustrated in Figure 13.7-13. There is a volume increase as the cave propagates so that a certain amount of material must be drawn before the cave reaches the dilution zone. The volume increase, or swell factors, is based on the fragmentation and is applied to column height. Typical swell factors for fine, medium, and coarse fragmentation are 1.16, 1.12, and 1.08, respectively.

A draw control factor is based on the variation in tonnages from working drawpoints (Figure 13.7-14). If production data are not available, the draw control engineer must predict a likely draw pattern. A formula based on the above factors has been developed to determine the dilution entry percentage:

$$\text{dilution entry} = (A - B)/A \times C \times 100$$

where

A = draw-column height (H) swell factor
B = height of interaction
C = draw control factor

The graph for dilution entry was originally drawn as a straight line, but underground observations show that where the dilution occurs early, the rate of influx follows a curved line with a long ore "tail" (Figure 13.7-15). Figure 13.7-16 shows that dilution entry is also affected by the attitude of the drawzone, which can angle toward higher overburden loads.

Layouts

Eight different horizontal LHD layouts and two inclined drawpoint LHD layouts are used at various operating mines. An example of an inclined drawpoint LHD layout is shown in Figure 13.7-17. The El Teniente (Chile) and Henderson mine layouts are shown in Figures 13.7-18 and 13.7-19, respectively. Numerical modeling of different layouts showed that the El Teniente layout was the strongest and had several practical advantages. For example, the drawpoint and drawbell are on a straight line, which results in better drawpoint support and flow of ore. If there is brow wear, the LHD can back into the opposite drawpoint. The only disadvantage is that the layout does not have the flexibility to accommodate the use of electric LHD machines as does the herringbone layout.

A factor that needs to be resolved is the correct shape of the major apex. It is thought that a shaped pillar will assist in the recovery of fine ore. Also, in ore bodies characterized by coarse fragmentation, there is less chance that stacking will occur (Figure 13.7-20). The main area of brow wear is immediately above the drawpoint. If the vertical height of the pillar above the brow is small, failure of the top section will reduce the strength of the lower section and result in aggravated brow wear.

More thought must be given to the design of LHD layouts to provide a maximum amount of maneuver room within a minimum size of drift opening. Thus, larger machines can be used within the optimum drawzone spacings. Another aspect that needs attention is LHD design, that is, to reduce the length and increase their width. Although the use of large machines might be an attraction, it is recommended that caution be exercised and a decision on machine size be based on a correct assessment of the required drawzone spacing in terms of fragmentation. The loss of revenue that can result from dilution

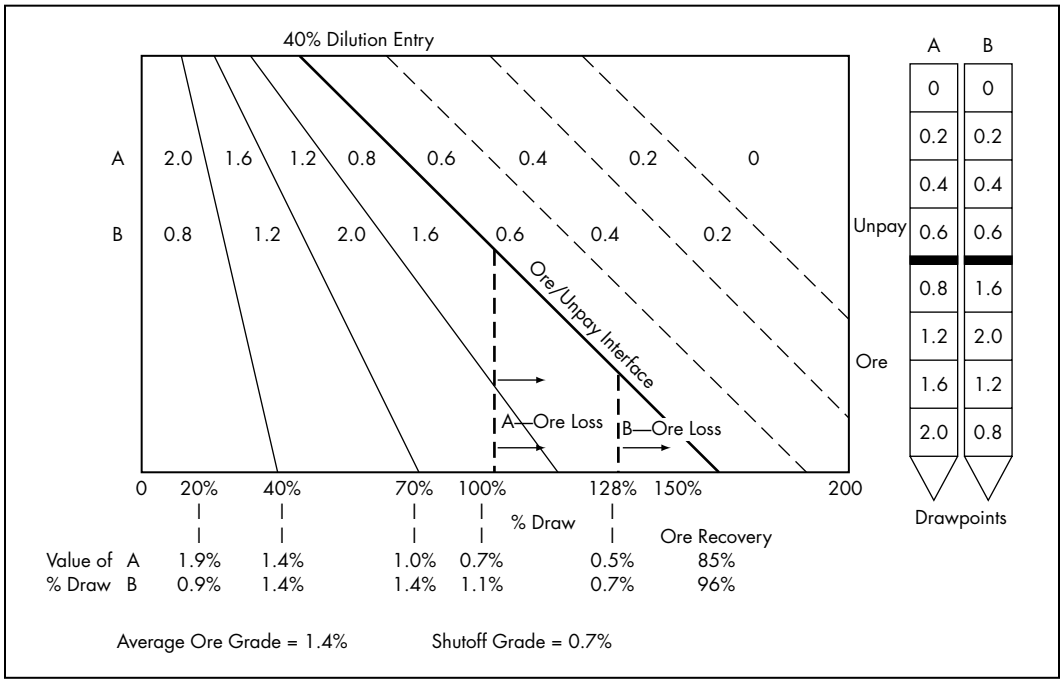
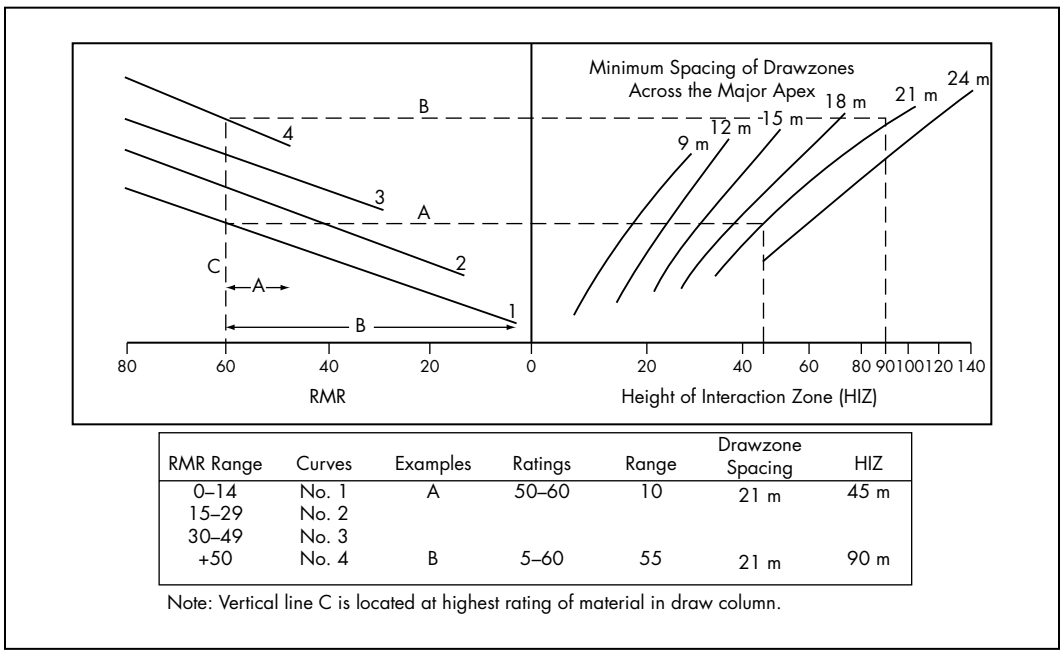


Figure 13.7-12 Grade analysis



RMR Range	Curves	Examples	Ratings	Range	Drawzone Spacing	HIZ
0–14	No. 1	A	50–60	10	21 m	45 m
15–29	No. 2					
30–49	No. 3					
+50	No. 4	B	5–60	55	21 m	90 m

Note: Vertical line C is located at highest rating of material in draw column.

Figure 13.7-13 Height of interaction zone

far exceeds the lower operating costs associated with larger machines.

Undercutting

Undercutting is one of the most important aspects of cave mining because not only is a complete undercut necessary to induce caving, but the undercut method can reduce the damaging effects of induced stresses.

The normal undercutting sequence is to develop the drawbell and then to break the undercut into the drawbell, as shown in Figure 13.7-18. In environments of high stress, the pillars and brows are damaged by the advancing abutment stresses. The Henderson mine technique of developing the drawbell with long holes from the undercut level reduces the time interval and extent of damage associated with post-undercutting. To preserve stability, the Henderson mine has also found it

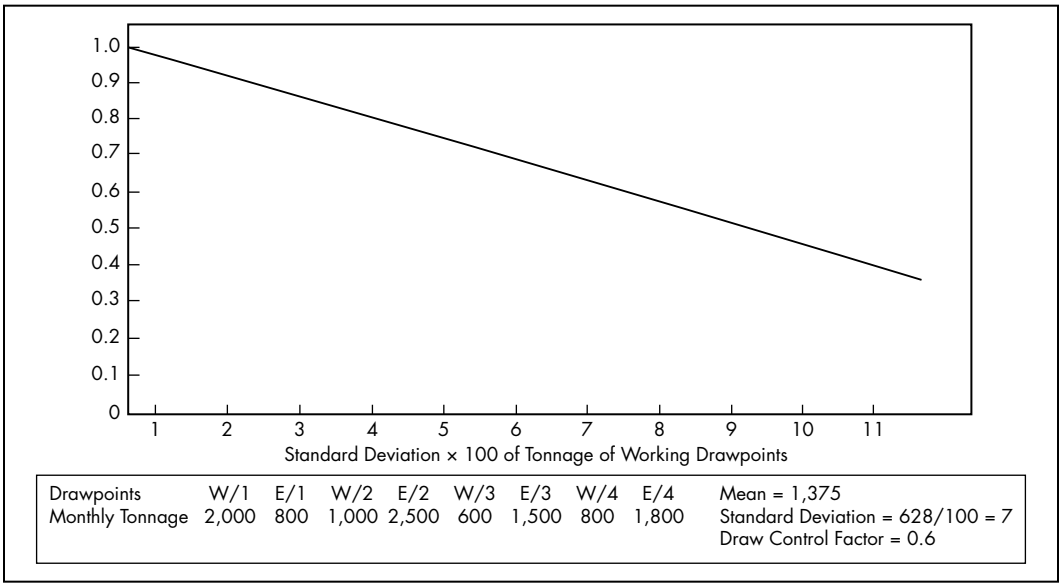


Figure 13.7-14 Draw control factor

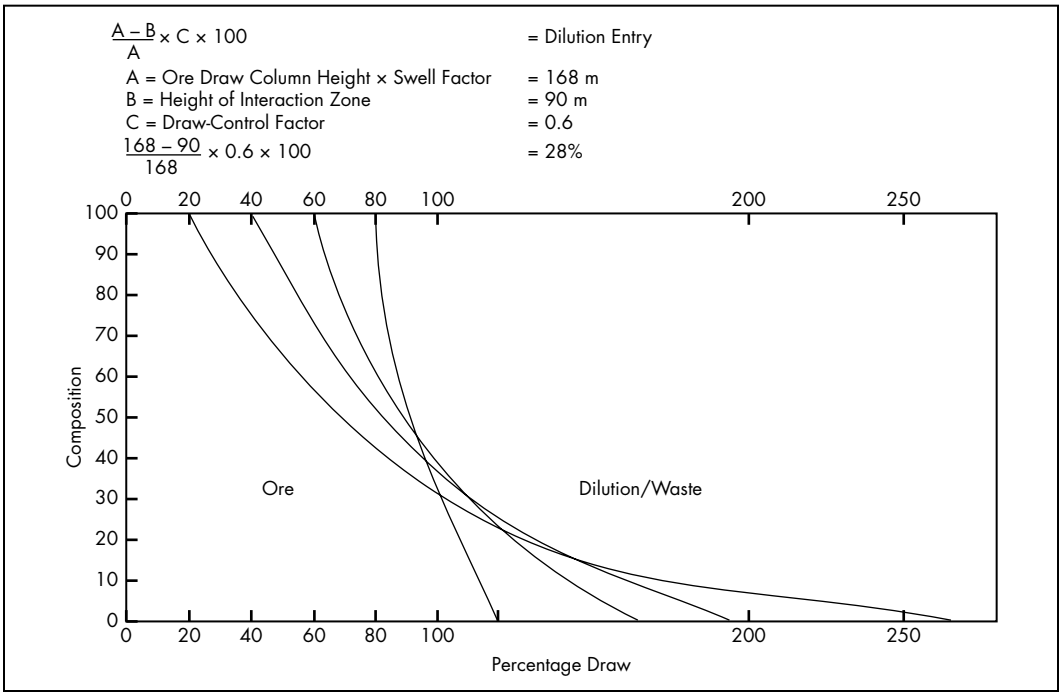


Figure 13.7-15 Calculation of dilution entry

necessary to delay the development of the drawbell drift until the bell must be blasted (Figure 13.7-19).

The damage caused to pillars around drifts and drawbells by abutment stresses is significant and is the major factor in brow wear and excavation collapse. Rock bursts also occur in these areas. The solution is to complete the undercut before development of the drawpoints and drawbells. The "advanced undercut" technique is shown in Figure 13.7-21.

In the past, the height of the undercut was considered to have a significant influence on caving and, possibly, the flow of ore. The asbestos mines in Zimbabwe had undercuts of 30 m with no resulting improvement in caving or fragmentation. The long time involved in completing the undercut often led to ground control problems. Good results are obtained with undercuts of minimum height, provided that complete undercutting is achieved. Where gravity is needed for the flow of blasted undercut ore, the undercut height needs to be only half the width of the major apex. This results in an angle of repose of 45° and allows the ore to flow freely.

Support Requirements

In areas of high stress, weak rock will deform plastically and strong rock will exhibit brittle, often violent, failure. If there is a large difference between the RMR and MRMR values,

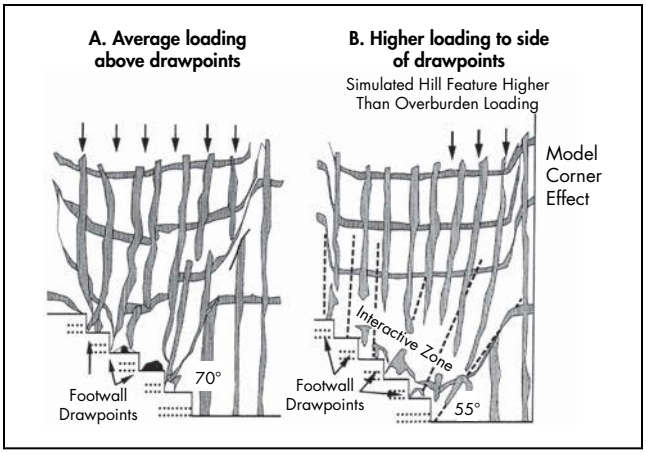


Figure 13.7-16 Inclined drawpoint layout showing effect of different overburden loading (three-dimensional sand-model experiments)

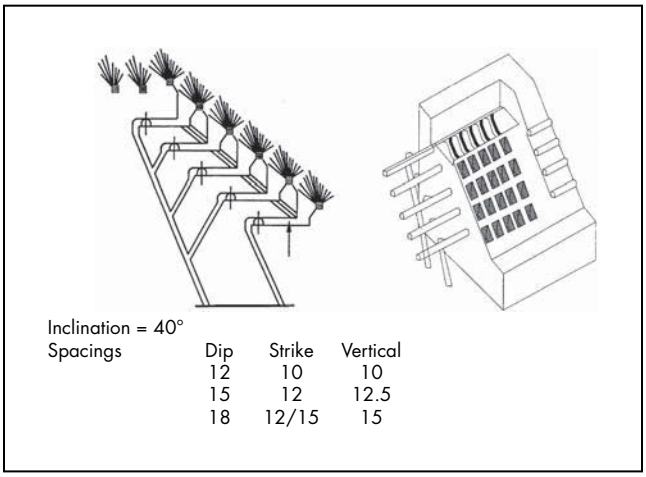


Figure 13.7-17 Inclined drawpoint LHD layout

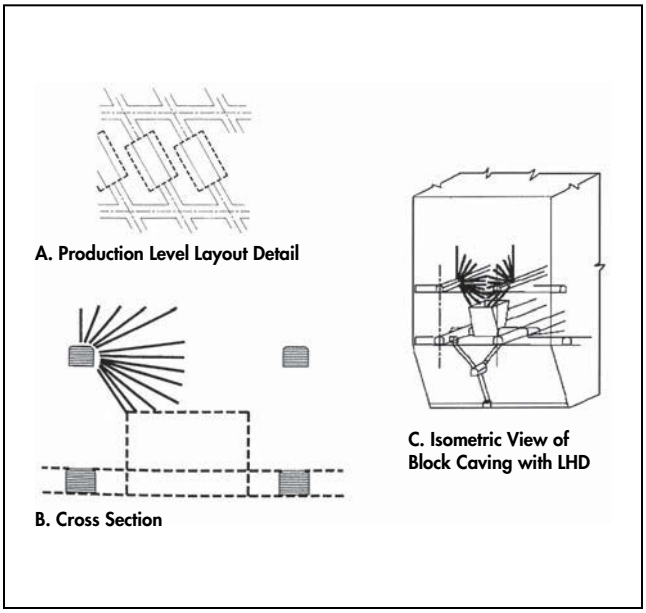


Figure 13.7-18 LHD layout at El Teniente

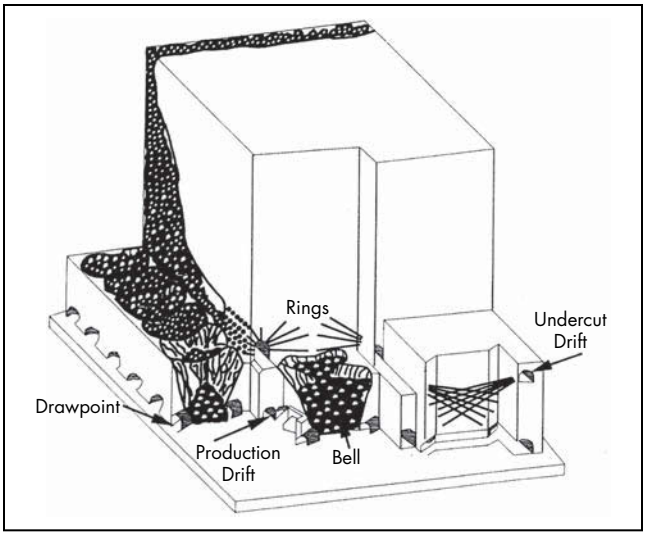


Figure 13.7-19 Isometric view of panel cave operation at the Henderson mine

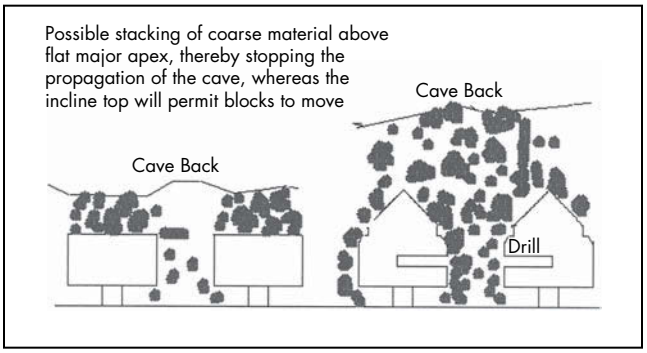


Figure 13.7-20 Effect of shape of major apex

yielding support systems are required. This is explained in Figure 13.7-22.

Prestressed cables have little application in underground situations unless it is to stabilize fractured rock in a low-stress environment. The need to constrain the rock laterally and for lining surfaces such as concrete cannot be too highly emphasized. Support techniques are shown in Table 13.7-5. The use of cable bolts in brows is common practice, but often these bolts are not installed in a pattern that takes into account joint spacing and orientation. Cable bolts should not be installed in highly jointed ground, as the bolts do not apply lateral restraint to the brow and serve only to hold blocks in place.

CONCLUSIONS

Cavability can be assessed provided accurate geotechnical data are available and geological variations are recognized. The MRMR system provides the necessary data for an empirical definition of the undercut dimension in terms of the hydraulic radius. Numerical modeling can assist an engineer in understanding and defining the stress environment.

Fragmentation is a major factor in assessing the feasibility of cave mining in large, competent ore bodies. Programs are being developed for predicting fragmentation, and even the less sophisticated programs provide good design data. The economic viability of caving in competent ore bodies

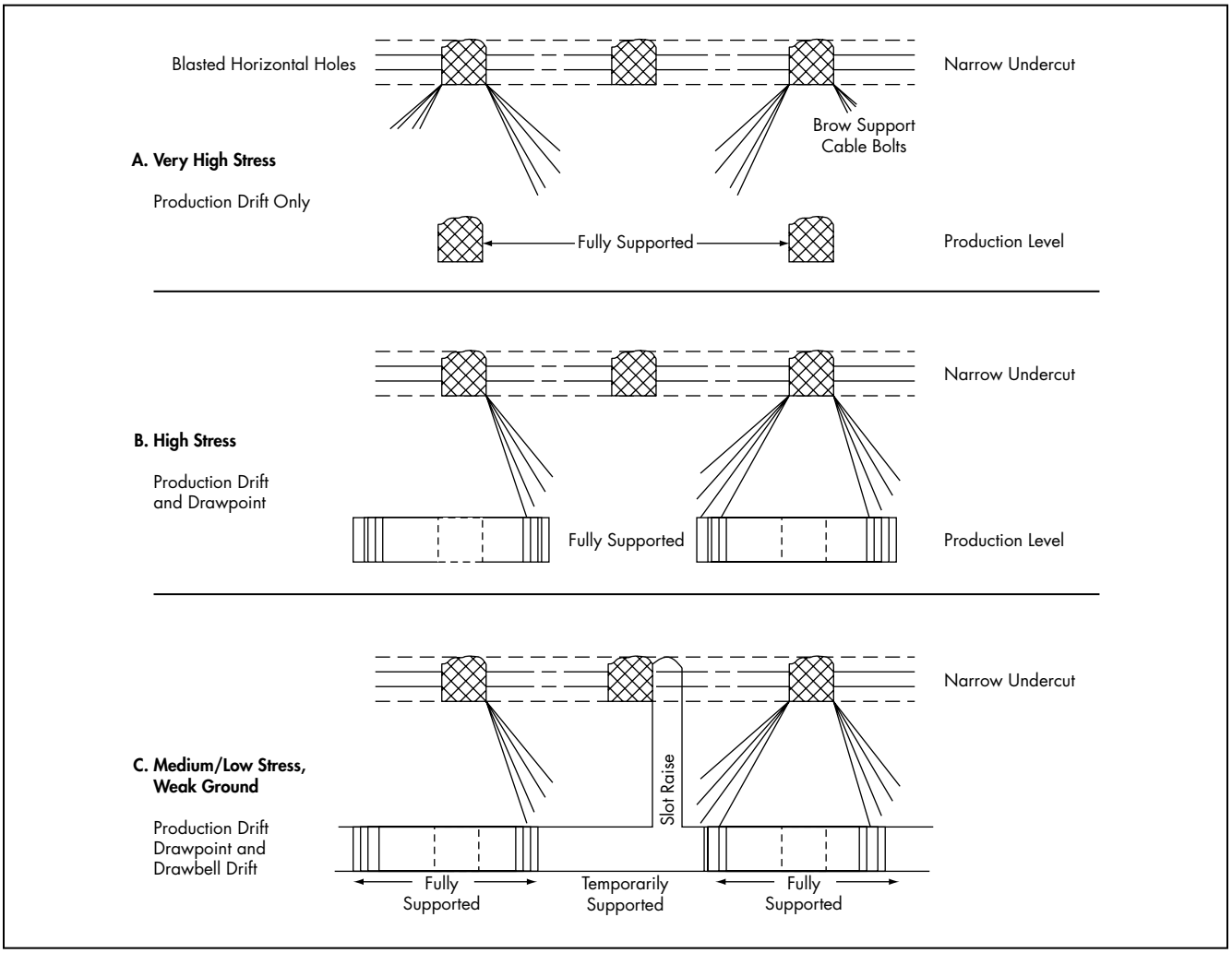


Figure 13.7-21 Different sequences of advanced undercutting RMR

is determined by LHD productivity and the cost of breaking large fragments.

Drawpoint and drawzone spacings for coarser material need to be examined in terms of recovery and improved mining environments. Spacings must not be increased to lower operating costs at the expense of ore recovery. The interactive theory of draw and the diameter of an isolated drawzone can be used in the design of drawzone spacings. Complications occur when drawzone spacings are designed on the basis of primary fragmentation and the secondary fragmentation is significantly different.

The drawpoint is the only control that the operator has once caving has started. The drawpoint should be commissioned as soon as possible after the ore body is undercut. It is therefore essential that every effort is put into ensuring the stability of the drawpoint. The support system must be designed for the rock mass strength and draw height. When it is planned to draw high column heights, the extra expense of ensuring drawpoint integrity is essential. Major drawpoint repairs in a horizontal layout tend to disrupt the draw pattern and can lead to column loading.

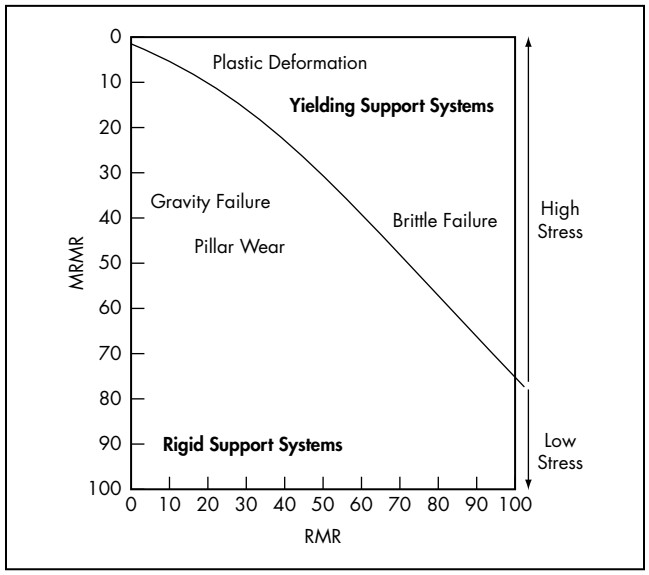


Figure 13.7-22 Support requirements for caving operations

Table 13.7-5 Support techniques

Support Element	Low Stress	High Stress
Bolts		
Length	1 m +(0.33 W × F)	1 m+(0.5 × F)
Spacing	1 m	1 m
Type	Rigid rebar	Yielding (e.g., cones)
Mesh	0.5 mm × 100 mm square	0.5 mm × 75 mm or 50-mm aperture
Deep-seated support	Cables, 1 m + 1.5 W	Steel ropes, 1 m + 1.5 W Long cone bolts
Shotcrete linings	Mesh-reinforced shotcrete	Mesh-reinforced shotcrete
Arches	Rigid steel arches Massive concrete	Yielding steel arches Reinforced concrete
Surface restraint	Large washers (triangles) Tendon traps	Large plate washers Yielding tendon traps
Corners	25-mm rope, cable slings	25-mm rope, cable slings
Brows	Birdcage cables from undercut level Inclined pipes	Birdcage cables from undercut level Inclined pipes
Repair	Grouting Extra bolts, cables, plates, straps, and arches	Grouting Extra bolts, ropes, plates, straps, and yielding arches

Notes: W = span of tunnel. F is based on MRMR = 0–20:F = 1.4; MRMR = 21–30:F = 1.3; MRMR = 31–40:F = 1.2; MRMR = 41–50:F = 1.1; MRMR = 51–60:F = 1.05; MRMR = >61:F = 1.0.

Block caving has advantages over large open pits in environmentally sensitive areas. The crater is much smaller and if necessary can be filled with waste rock even during drawdown. When the operation is completed, the rehabilitation costs will be considerably less than with a large open pit.

ACKNOWLEDGMENTS

This chapter, taken from SME's 2001 publication *Underground Mining Methods*, presents an update of the technology of cave mining. Although it is not possible to quote other references because the bulk of the data supporting the contents of this chapter have not been published, the basic concepts are known to mining engineers. However, it is appropriate to acknowledge the contributions from discussions with the following individuals in Canada, Chile, South Africa, and Zimbabwe: R. Alvarez, P.J. Bartlett, N.J.W. Bell, T. Carew, A.R. Guest, C. Page, D. Stacey, and A. Susaeta. The simulation program for the calculation of primary and secondary fragmentation was written by G.S. Esterhuizen at the University of Pretoria in South Africa.

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CHAPTER 13.8

Longwall Mining

Stephen L. Bessinger

INTRODUCTION

Longwall mining is a technique that evolved from a largely manual method in which rows of individual roof props supported the roof along a long face. The ore was broken from the face, first by drilling and blasting, and later by mechanical means powered by pneumatic, hydraulic, or electrical energy. Ore loading was done by hand shoveling into a conveyor or rail cars at the face. This method was labor intensive, lacked productivity, and exposed workers to significant hazards. Since then, technology evolution has produced spectacular improvements, motivated by the need for improved safety and productivity, as well as reduced labor and production costs. This evolutionary process has resulted in modern longwall systems, which are the focus of this chapter. The configuration and major components of a modern longwall are shown in Figure 13.8-1.

The shearer traverses the face, excavating the ore within a defined extraction height. The mined material is loaded onto the armored face conveyor (AFC) by the shearer and is transported to the main belt conveyor system via the stageloader. The stageloader normally has an integral crusher to provide suitably sized material for conveyor belts. The shields advance sequentially, following the shearer to hold up the roof directly above the face equipment and advance the AFC to repeat the cutting cycle. The excavated area behind the shields is allowed to collapse, which limits the roof load that the shields must support and starts a process that may create surface subsidence. High-capacity longwall mining systems depend on the breaking action of a machine to mine the ore. As such, modern longwalls are used to produce ores that can be mechanically cut or broken without explosives.

High-capacity longwalls mainly produce coal, but examples also exist in trona, potash, and phosphate rock. The potential is being investigated to apply longwall technology to platinum and gold-reef deposits with equipment adapted to the requirements of stronger rocks.

LONGWALL APPLICATIONS

Longwall mining has evolved mostly in response to coal-mining applications. Because many coal mines yield a relatively low-value product, particularly when coal is used for electric power generation, the output of the longwall must be a sustained stream, with low cost on a unit-of-production basis. The desirable image often suggested is that of a *coal faucet*, which can be opened or closed as needed. Embedded in the image is the ability to adjust the flow to desired levels. It should be the goal of those involved in the design, operation, and maintenance of a modern longwall to fulfill the goals of sustainability and productivity suggested by the coal faucet image. It is notable that health and safety considerations are intrinsically interwoven into longwall design and operating concepts because they are fundamental elements of sustainability and success. It is customarily accepted that longwall mining is the safest and most productive mining method that can be applied to soft rock deposits.

Longwall systems are serially dependent processes. This is because all of the associated equipment that supports longwall operation must function simultaneously, at the capacity of the longwall, to permit its operation. This dependence is a particular issue for the outby ore transport method, typically conveyor belts. Frequently shortfalls in outby conveyor belt performance are the most limiting constraint on longwall performance. The important point is that the success of the longwall system depends on the success of each of its constituent components, not just the machinery composing the longwall. This principle extends even to mine development activities that create the panel where the longwall will operate in the future.

Longwalls are typically designed for specific considerations related to the target deposit. Notable among these considerations are those described in the following sections.

Deposit Depth

Longwall mining has been applied to deposits with very shallow depth of cover, perhaps less than 50 m in a few cases and up to 1,600 m at the other extreme. Applications in the 100–600-m depth are common for longwall coal mines, especially in the United States and Australia. The loads on longwall roof supports (shields) are largely independent of depth of cover. Shield loads are more related to the extraction height and the

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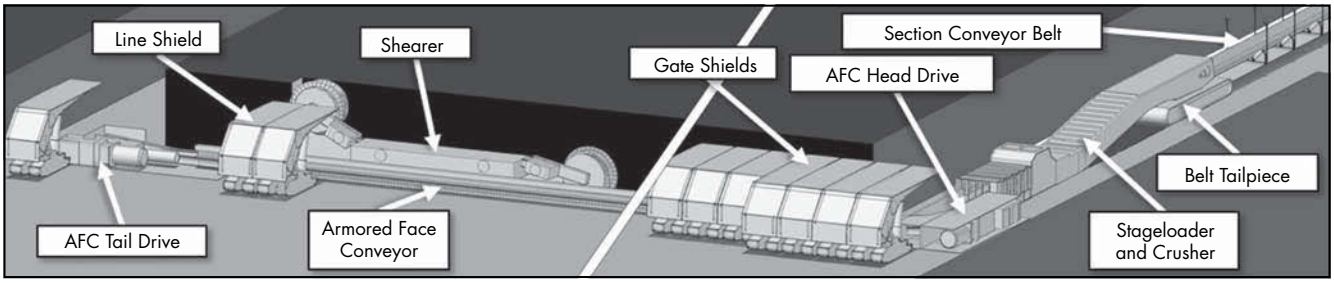


Figure 13.8-1 Major components of a modern longwall

strata overlying the ore bed (seam) than to any other factor. Mining-induced stresses on the face (front-abutment load) and gate road pillars (side-abutment load) are depth related. Not only must gate road pillars demonstrate expected stability, but the roof, in conjunction with the primary and secondary supports, must also be stable. Floor lift (floor heave) is frequently encountered at increased depth or with weak materials in the floor (i.e., under clay, fire clay, or claystone) (Peng 2006).

Deposit Dip

Most longwall mining systems are applied where the inclinations along the longwall face or the panel axis are low, less than 8.5°. Although longwalls are being used at much steeper inclinations, these are not consistent with high productivity. Steeply inclined longwalls also require special adaptations to stabilize the face against downhill creep, shield toppling, and product transport issues. Steep-seam, gate road development is also problematic. Steep, thick seams offer more opportunity for success but are still difficult.

As conveyor belt inclination increases, so do the issues in its operation, up to a practical limit and depending on the material being conveyed. For run-of-mine (ROM) coal, this limit is in the 15° range (CEMA 1997). This value can be even less, possibly 12°, if the material conveyed is wet. The wet condition can arise when production volumes diminish but water application is unchanged. In another case, large volumes of water can be transported by a submerged AFC onto the conveyor belt system. Importantly, operation of either conveyor belts or AFCs is easier when material flow is uphill. Numerous difficulties arise when conveying material down an appreciable grade, which far overwhelms any energy benefits that might be realized (Evans 1993).

The relative layout of the face and progression of panels in the mining sequence with respect to dip is important because of concerns about water migration, methane accumulation, and toppling of face and rib materials.

Deposit Thickness

Longwall systems are typically applied to laterally extensive tabular deposits with limited variation in ore thickness. After the ore thickness and its variation are characterized within the area to be mined, a nominal mining height will be defined. Practical minimum and maximum mining heights are defined by critical equipment sizes and operating envelopes. The longwall has little flexibility in extraction of material within the bounds of a panel and will cut through any area of the panel, extracting at least its minimum operating height. The only alternative is to bypass undesirable areas within the panel by withdrawing the longwall equipment and reinstalling it outside the undesirable area. This is a costly and time-consuming process, chosen only after considerable planning and reconnaissance. Where ore thickness is greater, the longwall can extract up to the maximum cutting height of the shearer or near the fully extended height of the shields. Caution must be used when approaching the fully extended height of the shields to avoid a condition where the shields fail to apply and maintain full loads into the roof without roof failure.

Mining equipment technology has the consequence of defining mining height in the following approximate intervals of deposit thickness (T).

- Thin: $T < 1.75$ m
- Moderate: $1.75 \text{ m} < T < 3.75$ m
- Thick: $3.75 \text{ m} < T < 7.25$ m
- Very thick: $T > 7.25$ m

Thin seam systems have historically applied longwall plow technology but have not achieved the productivity of thicker-seam, shearer-based applications.

Most modern, high-capacity longwalls operate in the moderate height category—from 1.75 to 3.75 m. Optimum height is generally agreed to be in the 2.5–4.0 m range for single-pass longwalls.

Above 3.75 m and up to approximately 7.25 m mining height, the thick category, is the upper limit of single-pass longwall capability. In this range, normal longwall equipment requires numerous adaptations to perform successfully until it reaches a practical limit at approximately 7.25 m.

Above 7.25-m extraction thickness, longwall technology is forced into several alternatives: multilift longwall or top coal caving. On a worldwide basis, as very thick coal seams extracted by surface mining extend to depths beyond economic limits, underground mining methods will be required that offer high resource recovery, such as longwall top coal caving.

Deposit Uniformity

Ideally, a tabular deposit well suited to longwall mining would exhibit uniformity in the ore bed/coal seam as well as the adjacent roof and floor. Unfortunately, examples of such ideal conditions are few. Generally, discontinuities result from structural or depositional origins but may also be the result of post-depositional events.

Structural geologic disturbances normally involve folding and faulting. Gentle inclinations induced by folding can be considered as previously described in the discussion on dip. However, if the gentle dips of a broad syncline, anticline, or monocline go over approximately 8.5° as a trend, this may limit applicability of the longwall. Expectations of high productivity may decline as the inclination steepens from 8.5° up to 15°.

Aside from overall inclination, the rate of change of inclination becomes a constraint, as the AFC, shearer, and shields

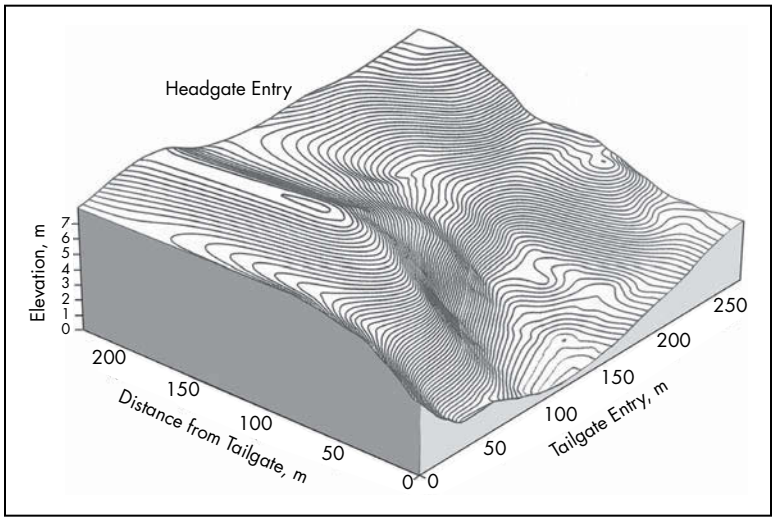


Figure 13.8-2 Disturbed seam topography

all have different concerns in this regard. As an example, if the radius of curvature of seam topography gets short, and the shearer is able to follow it, the AFC may have less vertical articulation than required to follow the changed topography and bridge, forming a span that is not supported by the floor. Thus, the AFC would only be supported by the connections that hold adjacent pan sections together (*dog bones* or *dumb bells*). By the AFC's own weight, or with the additional weight of the shearer, the AFC connections may fail or the pan structure may fail. This can cascade into an AFC system failure, such as a "chain wreck" where AFC chains break, flights are damaged, and main drive sprockets may fail. Such incidents may take hours to days to repair and cost in the hundreds of thousands of dollars in direct and lost-opportunity costs. They also create avoidable safety hazards.

In an alternate case, shields can lift their bases off the floor during cyclic advancement if their canopies become "iron bound" with adjacent shields because of a short radius of curvature in extraction profile. This condition can result in a suspended shield, which can fall to the floor unexpectedly and be a safety hazard. Of course, all of these problems are created when the equipment is asked to do something that was never intended by the original equipment designer or, conversely, was not identified as a performance requirement in the acquisition specifications. Figure 13.8-2. shows a disturbed area that was successfully mined with a longwall but not without impact to productivity.

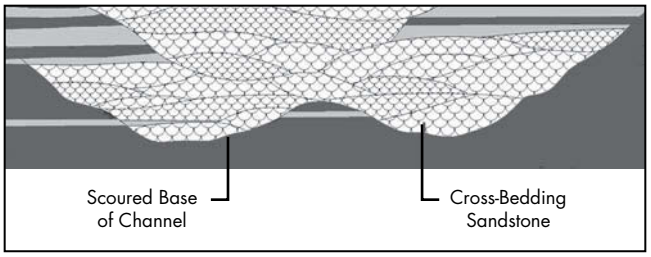
Another structural geology concern is fracturing or faulting. An example might be fracturing along the axis of local bed flexure, which can adversely affect roof control or cause elevated gas or water inflows. Notably, fractures/faults parallel to hillsides exist in some areas and may complicate logical panel layouts beneath a ridge line. Faulting may be present with or without appreciable folding. Strata displacements may be normal faults, and total displacements may be distributed across one or more discrete fault traces within a short distance. This is generally a manageable issue so long as the local displacement is not too large. Because these features are preferably traversed at an acute angle to the face, grading roof, and floor into and out of the feature makes operating through local offsets feasible—up to double the mining height in displacement. This will introduce an adverse impact to ROM product quality and overall productivity. Obviously, proximity of faults on a single face is problematic, as is the case where faults, still at acute angle to the face, exhibit enduring persistence. Both of these cases may require revised ROM quality and productivity estimates.

Then there are those cases where the fault is at a shallow angle with the face line or possibly even parallel to the face at worst case. This scenario can have seriously negative impacts to productivity and may lead to premature face recovery or even threaten the loss of equipment. If such a feature is suspected, it should be approached with caution after careful risk assessment and with a plan of mitigating actions in place. If such a feature is first recognized as it emerges onto the face, it is certainly appropriate to make efforts to evaluate it and implement a mitigation strategy before exposing a large proportion of the face to the hazard. It has frequently been the case that a longwall could "outrun" a local adverse roof condition; however, that is not true for faults parallel or at shallow angle to the face.

In the case of most structural faulting, the ground stresses within one block of the fault may not be equivalent to the stresses in others. Although normal faulting is at some risk in this regard, it is particularly problematic in thrust faults. The stress differential between blocks in a thrust-faulted scenario can result in some blocks having extremely difficult ground control issues compared to others.

Depositional discontinuities develop in response to conditions present at the time that the mineral deposit formed. These discontinuities commonly arise in conjunction with sand channels above or below a coal seam or as scours (also termed *faults* or *wants* in some locales) where they erode and replace the coal seam. Figure 13.8-3 depicts sand channels as they may commonly occur in coal-bearing strata.

In near proximity to these features, it is possible to have elevated stresses and differential compaction surfaces. These glassy smooth discontinuities (slickensides) are weak in tension and can constitute a hazard to roof control. Where large thicknesses of coal occur over a short stratigraphic interval, compactional features can exist as faults, with displacements mostly confined in near proximity to the seam and having normally short lengths, perhaps less than 250 m. Compactional faults are not normally the source of great concern.



Source: Adapted from Papp et al. 1998.
Figure 13.8-3 Sand channel in coal strata

Sand channels above the mining horizon can lead to increased shield set load and yield load requirements and should be evaluated accordingly. Where these features might be encountered at a shallow angle to the face, opportunities to avoid the condition are initially preferred. When near-seam sand bodies are strong and stiff, in conjunction with significant depth, the possibility of sudden strain energy release in response to mining can be a concern.

Another variation of depositional discontinuity occurs when soft sediments are injected into or across the coal seam. These are not usually large or strong, so they represent more of a threat to ROM quality and local ground control as opposed to sustainability (Moore 1940).

In a similar fashion, it is occasionally the case that igneous materials can be injected into coal-bearing strata, forming lenticular or vertically crosscutting dikes. In various locations around the world, including South Africa, Australia, and the western United States, these igneous dikes can be a serious obstacle to longwall operation. Where these bodies are narrow, perhaps less than 1.0 m wide, they are often sufficiently contaminated by entrained debris that they do not achieve strength beyond the limits of mechanical cutting by a shearer or continuous miner. However, when encountered as thicker bodies, typically a stronger core requires other means to penetrate it. In South Africa, this material is identified as dolerite and forms dikes off of sills intruded into the coal-bearing strata. The dolerite dike material can be more than 140 MPa compressive strength (Papp et al. 1998). It is also the case that post-depositional events can lead to the formation of natural coke, or a natural burn can produce coal loss with only residual coal ash.

It is safe to assume that all of forms of rock-mass nonuniformity noted will have a negative impact on longwall operation. As such, it is useful to identify any geologic features at the planning stage. Although reconnaissance drilling and detailed geologic mapping are customary, various geophysical methods can be applied to search for the different discontinuity types. Satellite imagery, magnetic or seismic surveys, and electromagnetic tomography are a few of the methods used. Unfortunately, these methods are not universally useful, and choice of method deserves careful attention. It is common to find that none of the methods are particularly informative for any specific site.

Strata Gases

Characterizing the gases and quantities that might be encountered during longwall development or subsequent longwall mining is important when striving to achieve the coal faucet concept. If acceptable atmospheric conditions cannot be continuously maintained where people work and travel, the longwall is at risk of being idled during periods where excursions beyond acceptable limits occur. In the United States, concern is emerging for atmospheric safety in the mined-out areas (gob or goaf) behind the longwall or in sealed adjacent areas.

Several gases have prominent impacts on longwall design and operation worldwide. These include those described in the following paragraphs.

Methane

Methane (CH_4) is common in coal mines to varying degrees and may be encountered in most sedimentary strata. Coal seams have long been recognized as likely reservoir rocks for methane, but it is increasingly recognized that other porous and permeable near-seam rocks can be significant sources of methane liberation. This is particularly true when the in-situ rock mass is fragmented by mining-induced caving. Higher CH_4 liberation rates in new longwall gobs result from creation of smaller particles than existed in the undisturbed rock mass. Release of lithostatic stress also lends to increased production rates of methane in caved material. Not only does the methane fill the void space of rock porosity at some reservoir pressure, it can be chemically adsorbed by coals. It is also common for fractures or faults to be charged with fluids.

Occasionally the rock stress augmented with interstitial fluid (gas) pressure can be greater than the unconfined rock strength, which results in one mechanism of *outburst* when rock failure occurs. Not only can this create atmospheric inundations from the escaping gas, it can result in violent ground control hazards. Such events are known in a few U.S. and Australian coal mines.

Lastly, the most notable hazard of methane in the mining environment is its explosibility when mixed with atmospheric oxygen in certain ratios. As one step to avoid this hazard, most longwall face equipment is designed, tested, and certified to be explosion proof or intrinsically safe. Explosion-proof equipment will not allow an ignition within the enclosure to propagate outside it. Intrinsically safe components rely on the principle that methane–oxygen mixtures will not ignite below certain initiation energy thresholds, and they ensure that such energy is never available, even during component failure conditions.

Carbon Monoxide

Carbon monoxide (CO) can be adsorbed on coal and naturally occurring or it can be a product of partial combustion of hydrocarbon fuels or coal. This gas primarily represents a health hazard, but can be explosive in relatively high levels when mixed with atmospheric oxygen.

Aside from its human toxicity, it is often an indicator of combustion, spontaneous combustion in some coals, or may result from fire or explosion. Longwall mines with spontaneous combustion propensity may monitor the CO production rate from the longwall as an early indicator of an evolving thermal event.

Carbon Dioxide

Carbon dioxide (CO_2), a common strata gas, may be present along with methane or may be a dominant constituent. In some Australian cases, high-pressure CO_2 resides in localized areas of the coal seam and requires specialized control measures to manage the hazards of outbursts. CO_2 is also a product of full combustion of coal and hydrocarbons. As such, it can be

one of several gases monitored to detect evolving combustion events, spontaneous combustion in particular. Graham ratio, Jones-Trickett ratio, and CO/CO_2 ratio, as well as CO production rate, may be used to detect the onset or monitor progression of a combustion event (SIMTARS 1999).

Hydrogen Sulfide

Hydrogen sulfide (H_2S) is primarily recognized in the mining environment for its toxic effects on humans. It is also a concern in longwall mining for its corrosive effects on metals and its participation in unexpected steel failures.

H_2S can dissolve in still water and be released when the water is disturbed. The aqueous mechanism can bring it in direct contact with metals, such as bronze or steel in longwall components, and result in metal sulfide formation and the liberation of free hydrogen (H_2). Because H_2 is not normally a strata gas but is known to be an indicator gas in combustion events, this can explain its existence without the presence of a thermal event.

H_2S can be a metabolic product of sulfate-reducing bacteria. These bacteria are naturally occurring and can form microscopic colonies beneath which high concentrations of H_2S can exist, perhaps up to 4,000 ppm. This can create overwhelming corrosive or metallurgical effects while atmospheric levels remain low.

Other Gases and Vapors

On occasion, other gases and vapors may naturally occur. Low molecular weight hydrocarbons can exist in gas or liquid phases in sedimentary strata. Their combustibility and quantity may be reason for concern in some instances.

In most cases, the regulatory body that has jurisdiction over mining health and safety for a specific location will have statutory guidelines for safe and legal levels of the noted gases. In order to be successful and sustainable, a longwall system must be designed to accommodate safety and statutory compliance in order to avoid production outages associated with gas level excursions above allowable limits.

Water

In many areas, water is associated with the coal seam or near-seam strata. It may also be associated with overlying or updip workings and fracturing or faults. In any event, its management is reason for careful planning because of its generally negative impacts to longwall development and operation, as well as the typically negative impact it can have on ground control. Where swelling clay minerals are present in the rock, the matrix may decompose when the clay minerals hydrate and swell. The slake-durability test is one measure of a rock's behavior when exposed to moisture. When swelling clay minerals are present, the rock typically performs poorly in slake durability.

Siltstone, mudstone, claystone, and fireclays are often the operating floors for longwall development sections or the longwalls themselves. The often present water, coupled with the duty as a roadway for rubber-tired equipment, turns these floors to mud, sometimes hundreds of millimeters deep. In the immediate headgate or tailgate entries, this can mean that the belt tailpiece and stageloader are resting on mud and may even invite floor heave, which requires clearance before equipment advance. It also can mean that gate shields may unreliably advance or be unable to create the intended load into the roof, leading to a deterioration of roof control in the headgate or tailgate. This same principle can leave standing support such as props, cans, or cribs with an inadequate foundation upon which to create resistance to roof deformation. Water management, or the failure to manage it, is often a reason that longwall productivity falls short of its full potential.

Spontaneous Combustion

Spontaneous combustion is a condition where slow oxidation of a combustible substance leads to heat evolution and increasing temperature until the material reaches a point where combustion starts and spreads to adjacent fuel. Some coals, particularly those with lower rank, exhibit a propensity toward spontaneous combustion (Cliff et al. 2004). It is a particular concern when spontaneous combustion heatings or thermal events take place in a longwall gob (goaf), where they can be difficult to locate and identify, and even more difficult to access and extinguish. Such events typically cease production in the affected portion of the mine until, in the best case, the event ends or, in the worst case, causes loss of life and assets.

As serious as this scenario is, an even worse one is possible where methane is present in combination with a coal prone to spontaneous combustion. In that case, atmospheric oxygen in combination with explosive methane levels can be ignited by a small spontaneous combustion ignition source, resulting in an explosion and fire. In such an event, multiple fatalities and loss of assets are possible.

Because responsible mine operators are committed to a *zero-harm* philosophy, effective controls must be offered to preclude the possibility of a spontaneous combustion event or an explosion and fire. The mitigating strategy for all of these concerns is the long known *fire-triangle* concept for fuel, oxygen, and ignition source. Considering the fire triangle, it becomes immediately evident that in a spontaneous-combustion-prone coal mine, the only component that may be subject to control is oxygen. Thus, the choice of bleederless longwall ventilation, explained later, can effectively exclude oxygen from the gob (goaf) and mitigate the hazard.

As with any underground fire, the issue is only partially getting the fire extinguished, which usually entails oxygen deprivation, but also dissipating the residual heat resulting from combustion so that the event will not simply resume if oxygen becomes available again. This is also why early detection and intervention are key components to a spontaneous combustion control plan. But the most important element is a strong focus on prevention (Mitchell 1996).

Roof and Floor Strata

One of the advantages of a longwall mining system is that it can accommodate a wide range of roof conditions. Gate road stability issues aside, stabilizing the longwall roof is actively done by the shields. The shield must immediately stabilize the near-seam roof when it is initially set against the roof after it is advanced, referred to as *set load*. The set load required to stabilize the roof is a function of site-specific conditions and parameters of the face cross section. The set load produced by the shield is derived from the emulsion system pressure available to the shield during setting, the major stage diameter of the leg, and its resulting area. Set pressure applied to leg area and derated, to account for leg inclination (angle to vertical) functionally defines set load created by the shield. To be successful, set load required to stabilize the roof must be less than the vertical thrust created by the shield at setting. Subsequently, the shield must stabilize the roof throughout the time when the

shearer cuts past it, adding a web depth to the span, and ultimately as adjacent shields are lowered away from the roof as they are advanced. Loads also arise from face deterioration and span enlargement to the degree that it occurs in response to development of front-abutment loads from web to web. If the shield resists this load and the roof remains stable, the result is the end-cycle load required to support the ground. In concept, this should be at or below the yield capacity of the shield.

This yield load is calculated in a manner similar to that of set load, except that instead of system setting pressure, the opening pressure for the leg yield valve must be considered. With set load and yield load independently defined, it is possible to create some of the popularly discussed ratios such as set-to-yield ratio, yield capacity/linear length of face, and set or yield capacity/area of supported roof. Although these terms are frequently discussed as design factors, they are, in fact, only outcome statistics to the actual equilibrium that must be successfully established. They also have limited comparability over a range of possible operating heights, because only the vertical component of the leg load primarily contributes to roof stability. The horizontal force component is mostly resolved within the shield by the lemniscates links.

Because the interaction of the roof and the roof support (shield) has been described, and the need for stability is clear, the factors leading to loads arising from the roof must be characterized. Roof loads on shields increase with increasing distance along the face from either gate road up to a maximum load specific to any site. Distances greater than that effectively only expand the number of mid-face shields exposed to that maximum load. It is not prudent or practical to attempt to select shields in the design process for any load less than the maximum load experienced for the specific installation. Certainly, using outcome statistics or rules-of-thumb to size shield capacity is poor practice and risks the possibility that an entire longwall will fail to perform as intended. Should this be the case, available solutions may require premature equipment replacement or acceptance of undesirable operating performance.

Evaluating the factors that contribute to roof loading on the shields, it is clear that both site-specific and face cross-section factors are contributory. Site-specific factors include

- Seam thickness (extraction height),
- Near-seam strata (type and thickness),
- Seam and nearby strata material properties,
- Caving behavior of strata,
- Seam inclination,
- Natural and mining-induced fractures, and
- Previous mining above or below the seam.

In concept, the overlying rock collapses into the void created by mining. As the intact material fragments and caves, with particles rotating and translating into the void, its volume bulks to fill the void, largely in inverse proportion to particle size. This process continues until little enough void space remains between the rubble and the uncaved rock that further displacement of the overlying rock takes place through particle translation but with little rotation. Eventually, even the downward translation reaches equilibrium with the resistance created by the underlying compacted rubble. Thus, the weight of rock overlying the longwall can be supported across three areas:

1. Unmined face ahead of the longwall (front-abutment load)
2. Longwall roof supports
3. Rubble material of the gob

The area depicted in Figure 13.8-4 graphically shows the region of rock that must be stabilized by the roof supports (shields). In consideration of the loading process described, it is evident that the loading of longwall roof supports has little impact from mining depth. It also becomes evident that overlying roof materials that cave as small particles from thin layers can lead to smaller caving heights before particle rotation is diminished and consequently to lower shield loads and thus lower shield capacity requirements. Conversely, thicker and stronger overlying beds can impose much larger loads on roof supports, especially if they are close to the mining horizon. Another consideration is that stiff roof rocks, typically strong as well, can require high set loads to stabilize them. Further, they can lead to extreme cases of periodic weighting, where strong beds can act as cantilevered loads, imposing successively greater loads on the roof supports until the cantilever fails. Such strong beds can be multiple within the caving horizon and cause superimposed effects of periodic loading. Figure 13.8-5 shows an example of how this variation might look for an example longwall application (Barczak and Garson 1986). Wind blasts originating in the gob often coincide with the caving of thick, strong beds, or the first cave of a new longwall panel.

In some instances a cantilevered bed failure results in the release of stored energy and in a “bump” or “bounce.” This mining-induced seismicity can be damaging to equipment and hazardous to personnel. In general, mining-induced seismicity is often associated with strong rock or coal and significant depth. High overburden stress coupled with mining layouts that may act as stress concentrators can lead to mining-induced seismicity originating at elevations ranging from beneath the mining horizon and extending upward to the surface. High-flow yield valves on shield leg cylinders can afford protection against equipment damage at moderate convergence rates in a periodic weighting event, but only purpose-designed rockburst protection can prevent damage to shields during a bump where a large convergence can occur suddenly.

Face cross-section factors that affect loads imposed on the shields include

- Geometry of the shields, shearer, and AFC;
- Depth of cut (web); and
- Sequence state in cutting cycle.

Because of these factors, coupled with geologic variation that takes place on scales smaller than the area mined by a set of roof supports within their service life or even within the course of a panel, it is not surprising that the loads imposed on shields vary over a range at any installation. Figure 13.8-6 shows the cumulative probability distribution of loads required to stabilize the longwall roof.

These curves can be numerically modeled for a proposed longwall face cross section and strata section, and are site specific for every individual application. Models of this type have shown good correlation with data collected from pressure monitoring on equipment in service. As discussed previously, the set load curve is derived independently of the yield load curve.

It can be observed that in the region above approximately 96% cumulative probability (CP), providing additional support load into the roof offers diminishing returns. Using this method to select roof support capacity leads to selection of

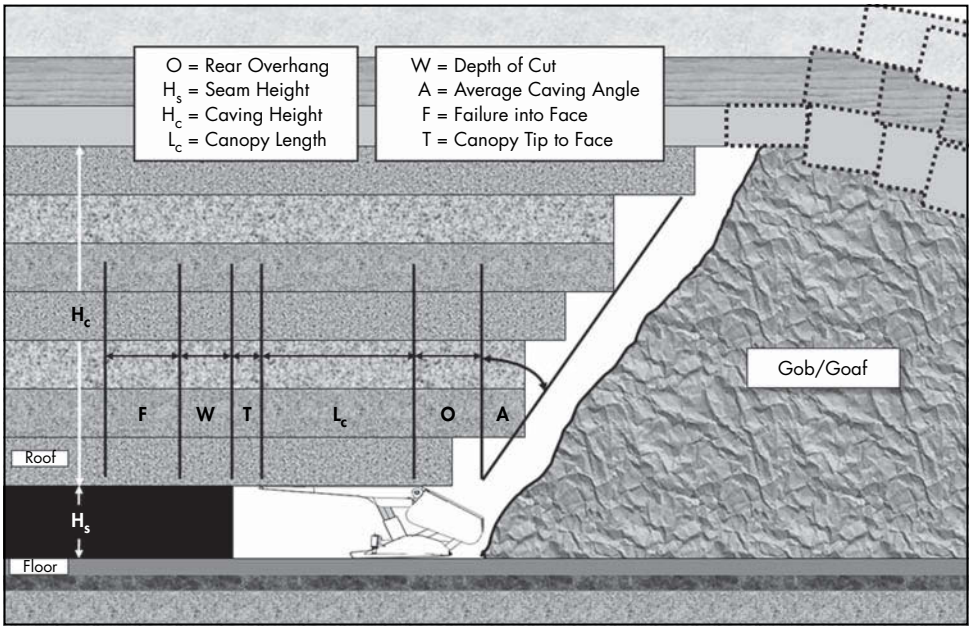


Figure 13.8-4 Conceptual loading model for longwall shield capacity evaluation

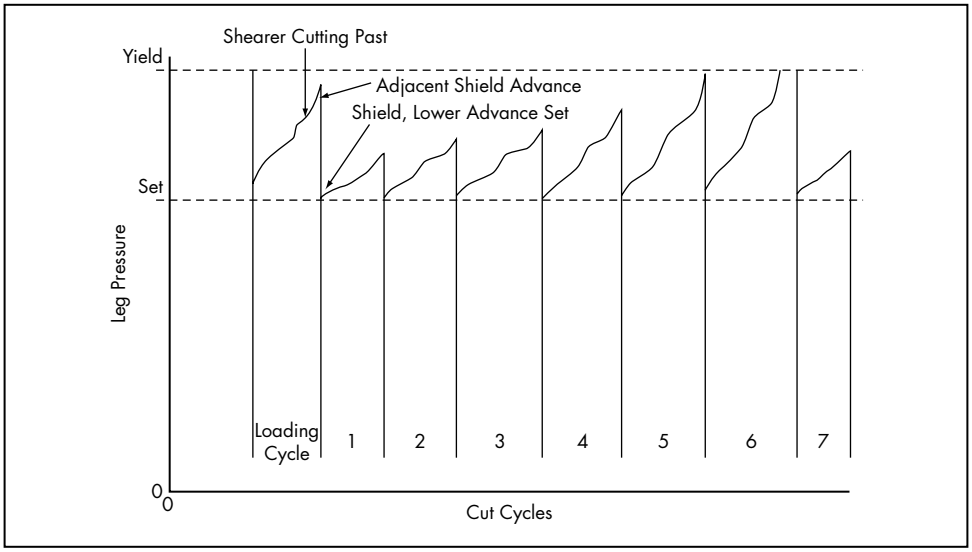


Figure 13.8-5 Periodic weighting behavior of a longwall roof

supports that control loads from the roof in the 91%–96% CP range. This leaves a 4%–9% incidence of yielding. In most cases, yielding events will be of brief duration, and equilibrium will be reached by sharing load onto adjacent shields. Selection of capacity in the higher end of the range, 93%–96% CP, allows for some deterioration in performance, as equipment ages, without creating performance problems.

Choice of supports larger in capacity than 96% CP of the roof loads that will arise offers higher cost, greater weight, reduced travelway dimensions, and elevated emulsion system demands with little perceptible benefit. However, because the probability distribution is nonlinear, selecting capacity much below 91% CP leads to an excessive occurrence of shield yielding and onset of roof damage companion to the more severe yielding events.

Notably, potential shields for any application should be evaluated at different heights (leg cylinder inclinations) to verify that they will be adequate for any mining height condition they may experience in service. Excess debris should not be allowed to accumulate on top of the roof canopy or beneath the base, as this material can be compacted and may allow excess roof convergence and damage to arise before enough support resistance is generated to create equilibrium. This is especially true for strong, stiff roof materials such as competent sandstones near the roof line in some coal mines.

Specific Energy of Cutting

The productivity of a mechanized cutting process for rock is largely a function of the material's specific energy of cutting, which depends on the type of cutting being applied. Shearers

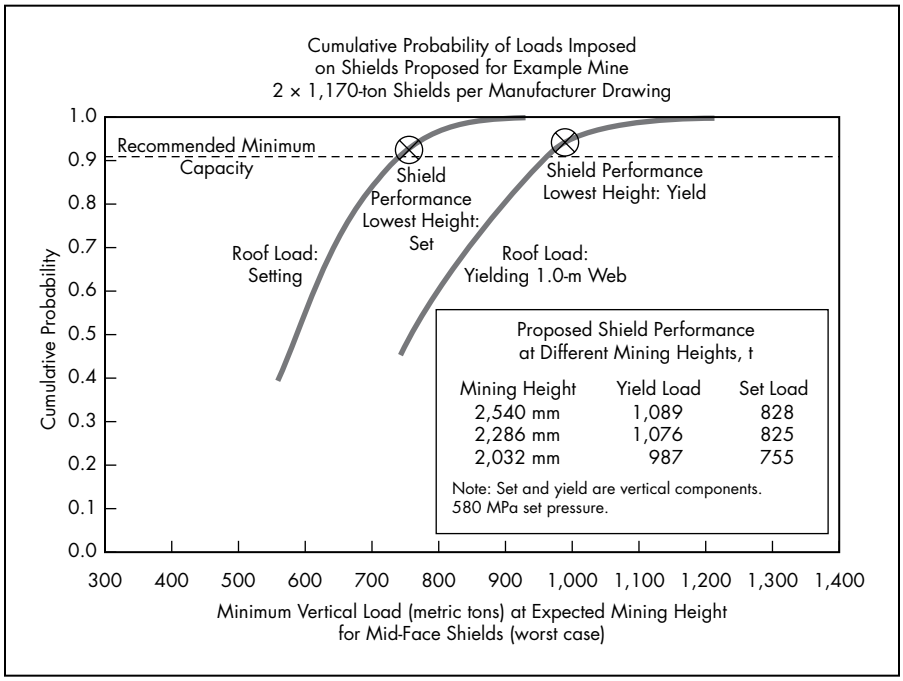


Figure 13.8-6 Example of site-specific stochastic longwall roof-loading evaluation

are different from drum-type continuous miners, which are yet different from longwall plows. While shearers and continuous miners work against forces related to compressive strength of the material being cut, plows primarily work against the tensile strength of the material, which is typically much weaker than the compressive strength. Particularly with respect to shearers, the cutter motor power used to produce a resultant quantity of ROM ore over a period of time defines the specific energy of cutting for the material (kilowatt-hour per metric ton [kW·h/t]). This assumes that the cut material can be cleared so that product accumulation does not impede the process. This measure is useful for the purpose of production process design and management but is not as pure a parameter as the name might suggest.

The load applied to the material to be cut can alter this value and is a benefit for the longwall system. When the front-abutment load has fully evolved, after the start of mining in a panel, this load is usually large enough to "soften" the face and reduce the energy required to produce a quantity of material. Until the front abutment fully develops, cutting will be more difficult than after it exists. The apparent softening can also be noted as abutment load varies in periodic weighting cycles and when production resumes after a stoppage of sufficient duration to allow significant stress redistribution.

The addition of work done by the abutment stress is also advantageous to creation of a larger product size distribution and lower fugitive dust production on a milligram/cubic meter/metric ton basis. Interestingly, the cutting bits deeper in the web do more work and produce a smaller size distribution of product than the bits shallower in the web. This smaller-sized product may be significant in dust production, which can become airborne. The possibility of improved machine utilization and proportionately reduced dust production exists with adoption of increased web depth. In the transition of web depths from 0.76 m to as much as 1.07 m, this appears to have been true.

Specific energy of cutting for various materials being cut by shearing machines ranges from 0.05 kW·h/t for easily cut coals to 1.3 kW·h/t or more for some very hard cutting sandstone and igneous dike materials.

Where very difficult cutting is expected for an extended duration, specially designed *rock drums* are used in lieu of normal production drums. These rock drums are designed with closer bit spacing and more bits per line in the drum lacing to accommodate the heavier cutting demands. The style of bits may also change between radial and conical cutting bits to offer improved longevity and performance.

Typically, high-capacity longwalls operate well when they encounter materials in the 0.05–0.45 kW·h/t range.

Abrasivity

When considering a longwall mining application, the abrasivity of the product to be mined and conveyed is of concern. Most wear on the AFC and beam stage loader (BSL) is related to the tonnage of material that has been conveyed across the systems. As an approximation to the wear that might be expected in a particular application, standardized laboratory tests have been developed that are also used by the electrical generation industry, which has similar concerns as mining about abrasive wear to equipment within its facilities. In high-level summary, sacrificial blades are rotated through a mass of the rock material of interest for a predetermined time. At the end of the test, the weight of material lost by the sacrificial blades is determined and reported. This weight loss ranges from only 8 mg Fe in a trona specimen to more than 1,300 mg Fe in a severely abrasive coal and sandstone ROM product. Typical values range from 200 to 900 mg Fe, which roughly correlates with wear of 0.5 to 2.5 mm of deck-plate thickness reduction per million metric tons of production. Unacknowledged in these values are the corrosion processes, which can certainly influence the outcome in service. Improved results can be achieved by use

of abrasion-resistant steels where the corrosion environment will not provoke premature failures (Ludema 1996).

Mine Layouts

Having identified most of the important site-specific considerations that might have to be addressed by a successful mine design, the general mine layouts can begin to be discussed.

Historically, advancing and retreating longwall layouts have been the subject of much discussion worldwide. In the advancing system, the longwall setup room is close to the main or submain entry set, and the longwall mines away from these entries into undeveloped reserves. In the process, support is constructed to constitute one or more gate roads behind the longwall face. Mining continues until a designated extent is reached or conditions force face recovery. The fact that artificial roadways have to be maintained in the gob adds significant cost and labor to this technique. As cost and productivity benchmarks have evolved across the industry, the elevated costs and lower productivity of advancing faces have generally seen these operations cease or transition to retreating longwall layouts. Only in unusual circumstances or where historic precedent dictates are advancing faces operated. At present, all of the world-class longwall installations are retreating faces. This trend can be expected to continue into the future.

Retreating longwall layouts may be of two types: conventional or punch (highwall) type. Figure 13.8-7 shows a conventional retreating longwall layout with a mined-out panel adjacent to the active longwall. The entries of the previous panel's headgate (main gate) transition into becoming the tailgate for the active face. The active longwall face mines progressively from the setup room at the far (inby) end of the developed panel and is eventually recovered as it enters its intended recovery position. The recovery position of the face is shown by the line that coincides with the near (outby) extent of the mined-out panel. Optional recovery chutes may be used to aid free equipment recovery.

The punch (highwall) longwall is largely similar, except that instead of developing a main or submain entry set from which gate roads are driven, the gate roads are developed directly off a surface-mined box cut or highwall. Punch longwalling is attractive because it eliminates the need to develop and maintain long-term underground workings in the form of mains and submains entries. It also lends itself well to the transition from surface mining to underground mining that confronts many mine operators as the depth and production costs of remaining reserves increase.

Gate roads for retreating longwalls are typically formed with two- to four-entry developments. Although single-entry and dual-entry gate roads are common outside the United States, dual-entry gate roads are an exception in the United States. In the interest of worker safety in a fire or explosion emergency, U.S. mining law generally requires a minimum of three-entry developments in coal mines to accommodate separated intake (fresh air), return (exhaust), and conveyor belt entries.

In cases where it is safer to have a dual-entry development than the statutory minimum of three entries, a variance may be granted by regulatory enforcement authorities based on a demonstrated diminution-of safety-argument. U.S. mine operators have succeeded in the diminution-of-safety argument where adverse geotechnical conditions threaten worse consequences than the reduction of number of entries. Companion to the grant of such a variance are additional controls to safeguard against the hazards envisioned during statute development.

Often, especially in the United States, three- or even four-entry developments are used for gate roads. The motivation for creation of more development length, in total, than is minimally required to sustain longwall operation should be carefully considered. It is ordinarily desirable to minimize the ratio of development costs per longwall metric ton produced. It is also desirable for time value of money considerations to associate as closely in time as possible the expenditure of costs that provide future benefits with realization of those benefits. In the pursuit of these principles, it is common for longwalls to be operated in geometric succession, with the headgate of the prior panel being used as the tailgate of the current panel and so forth, until some predefined extent is reached.

Often this extent is the aggregation of a group of panels to form a *district*. The number of panels in a district is variable, typically based on ventilation or geotechnical considerations. When managing the constraints of a large district eventually becomes difficult enough, mine operators prefer the development of the incrementally additional gate road, which is required to develop the first panel in a new retreating longwall district. Frequently, districts are sealed upon completion, removing them permanently from the worked and traveled portion of the mine. Mine operators are then largely relieved of the burden to maintain access for inspection and ventilation of such a sealed district, but access must be maintained to inspect the final seals where the district meets with active mine workings, and the seals themselves must be maintained in adequate condition.

Accumulations of water, explosive atmospheres, or spontaneous combustion events can be reasons for concern, even in sealed and otherwise abandoned areas. Relatively recent changes to U.S. regulatory requirements for seals and monitoring atmospheres in sealed areas address these concerns.

On occasion, it can be necessary to isolate overburden stress effects between the current longwall panel and the prior panel with a barrier pillar. Where panels are mined at depth with thick, stiff beds in the overlying strata, failure to adequately support the stiff strata can channel vertical stresses toward the current panel. This elevated stress can result in mining-induced seismic activity in the strata section, with possible events in excess of 3.5 on the Richter scale. These events can be damaging to equipment and hazardous to personnel, particularly when seismic activity originates at or near the face location. At a lesser scale, stiff coal subjected to large stresses can fail suddenly and result in the expulsion of material from the face toward the longwall travelway, creating a hazard to personnel.

Longwall gate-road development is characteristically "just in time" to avoid delaying production activity on the longwall. The degree to which the development is ahead of the needs of the longwall in time is often referred to as *float time*.

Some ventilation plans require the next adjacent longwall to be developed in order to implement the ventilation plan. More commonly, this is not the case, and float times are in the 6-to-50-week range. This allows adequate time to withdraw the development equipment and stage longwall-move supplies and equipment prior to the onset of a longwall move. However, it does not impose unnecessary "hang time" on the gate road, as deterioration of conditions can begin soon after development in some settings. It also associates the cost

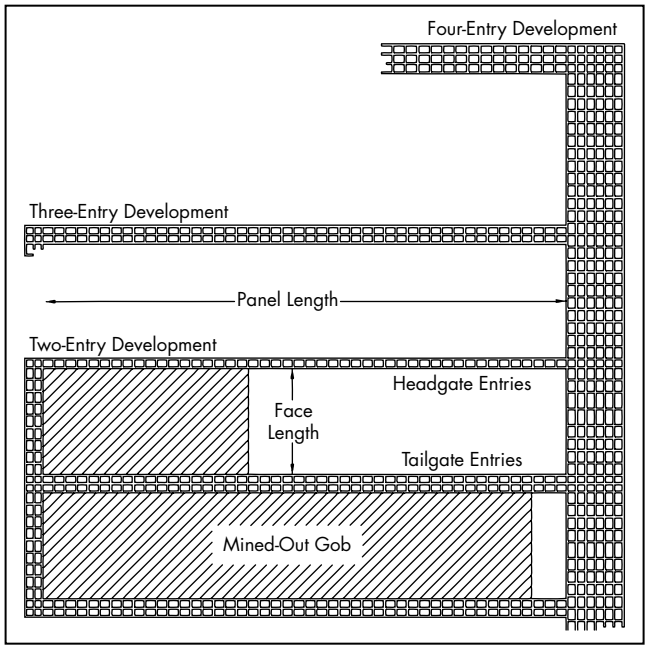


Figure 13.8-7 Fully retreating longwall layout

of such development in time, closer to the future benefits expected to arise from it. For many mine operators, the longwall move actually starts with emplacement of portions of the new face prior to completion of mining in the current panel. Such equipment might include: tailgate gate shields; AFC and BSL; a new or rebuilt shearer; electric power and control equipment; and emulsion-pumping equipment, controls, and tank. The amount of equipment preinstalled depends on arrangements specific to any mine operator. In general, shorter production outages occur with increased amounts of equipment preinstalled in the new longwall panel.

Gate road development itself is done with continuous miners and one of the several face haulage mechanisms available. Figure 13.8-8 shows the general layout of a three-entry gateroad development section with a continuous miner and battery coal haulers. The section depicted uses a place-changing method of operation where a continuous miner mines a cut and then changes places to mine a cut in another entry or crosscut. When the continuous miner has withdrawn, a separate machine installs roof bolts and possibly wire mesh or roof straps to roof and rib or roof trusses, if necessary.

Roof bolting progresses from where support was installed on the prior sequential cut in that working place to the face of the cut. After a cut is fully supported, the roof bolter is withdrawn and the cut is ready for auxiliary activities that may be necessary before the continuous miner returns to make its next cut in that working place. As this process is successively applied to entries and crosscuts, the entry set is advanced. Cuts may range from as little as 3.0 m to more than 12.0 m. In good roof conditions, the cut depth may be limited to the depth attainable without exposing personnel to unsupported roof or airborne contaminants.

Short cuts have a decidedly negative impact on the productivity of place-changing mining methods, as a disproportionately great amount of working time is lost to place changing. As depicted, ventilation is provided by section fans.

The coal haulers transport the mined product to a conveyor belt for ultimate clearance from the working section. It is common for the conveyor belt to be equipped with a feeder breaker. The feeder breaker has a limited amount of surge capacity, allowing coal haulers to unload quickly, but also meters product through a breaker and onto the conveyor belt at rates that optimize belt system performance. This is particularly the case where multiple sections all discharge onto a shared conveyor belt.

Place-changing operations are typically applied in relatively good roof conditions and often have better productivity than the alternative—in-place continuous miners. These in-place continuous miners, also called bolter miners, install roof bolts, mesh, straps, and even roof trusses in parallel with the mining cycle. Thus, roof support is installed close to the face, and the roof has minimal opportunity to deteriorate before additional support is implemented to augment its natural integrity. Also, this method minimizes the number of times that the bolter miner is moved between working places, thereby minimizing damage to potentially weak floors.

Historically, seriously flawed weight distribution on bolter miners led to worse floor damage than place-changing machines, even though machine travel was less. Recognition of this problem has allowed it to be mitigated on modern machines.

Aside from mining and bolting, section operations are much the same between in-place and place-changing systems. Some mine operators use loading machines between bolter miners and the coal haulers. The loading machine allows the bolter miner to discharge product into a small surge pile on the floor, behind the bolter miner. The machine loads from this surge pile into coal haulers as they become available and decouples the continuous miner from producing and loading only when a coal hauler is on station and ready to receive product. The use of a loading machine also caters to an often noted deficit of bolter miners, which is cleanup in-cut.

As a general principle, bolter miner productivity is less than place-changing operations when a reasonably long cut, perhaps greater than 6.0 to 7.5 m, is available. However, if cut depth falls below this range or if ground control is unacceptable in the cut depth available, a bolter miner system may become the preferred option. Current bolter miners have the ability to sump and bolt simultaneously. This has a notably favorable impact on productivity when compared to earlier style bolter miners, which could only cut as much as was available with the machine frame stationary when drilling and bolting activities were ongoing.

Additional concerns for gate road design are entry and crosscut span, crosscut angle, and spacing. Gate road entries must serve several purposes, including the following:

- Provide adequate cross section for ventilation
- Provide access of personnel and materials
- Transport and installation of longwall equipment
- Installation of the longwall section conveyor belt
- Temporary locations for emulsion-pumping equipment and electrical power and controls
- Retreat of the longwall in normal operation

Standing roof support is restricted to only those areas where it does not obstruct necessary machine travel. In the longwall conveyor belt entry, supplemental support may be installed as a precaution because there is little opportunity to add remedial support after the belt is commissioned.

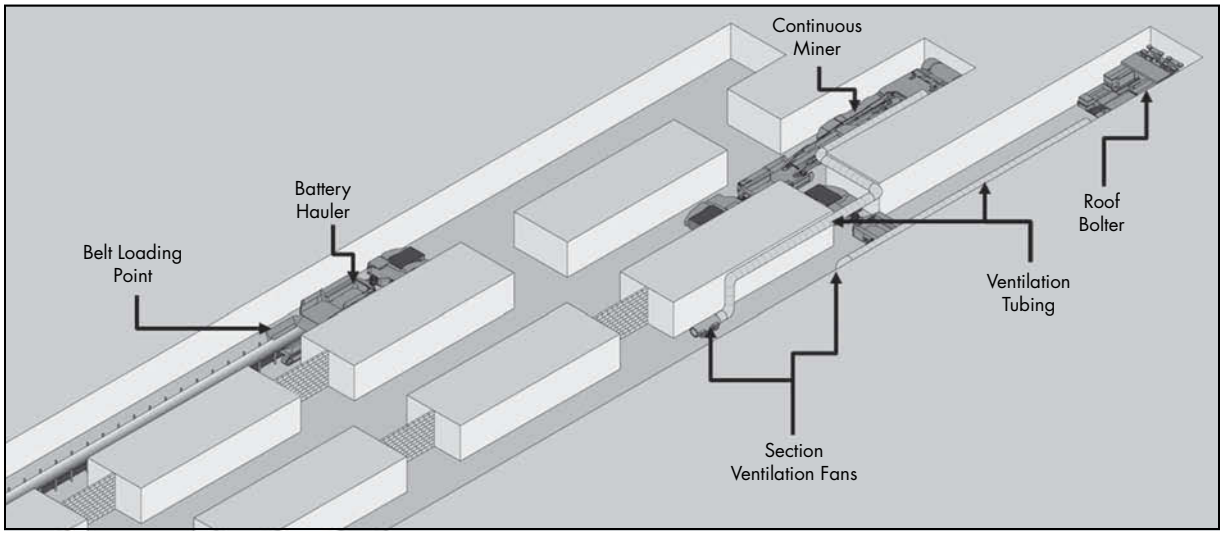


Figure 13.8-8 Three-entry gate road development with place-change mining

Four distinct regimes of increasing loading are applied to gate roads, aside from the first and last panels of a district:

1. Initial development
2. Adjacent to a single longwall gob (headgate)
3. Intermediate to adjacent longwall gobs (active tailgate)
4. Longwall gobs on both sides, well behind (inby) the face

The primary roof-support system controls the ground in the first and second stages, but supplemental support is commonly applied prior to use as a tailgate for the active longwall. This support is typically installed in the immediate tailgate entry and crosscuts before it reaches third-stage loading, and, where practical, it may even be installed at the time of development. Commonly used materials include wooden cribs (4-point or 9-point), cans, steel props, and pumpable cribs.

The alternative of right-angle intersection of crosscuts to main entries versus inclined is a balance of considerations between the convenience of access afforded by inclined intersections and the propensity of acute-angled corners to deteriorate with consequent enlargement of the intersection sum-of-diagonals (SoD). Increasing SoD often correlates with decreased stability of intersections. Floor stability and horizontal stress direction can influence the optimal intersection design.

The result of entry spacing and crosscut spacing is pillar size. The correct selection of pillar size for longwall gate roads is an involved discussion, beyond the scope of this chapter. In general, bigger pillars are more desirable from a stability perspective, but the unrecoverable reserves lost to gate road pillars, the increased development footage, and prolonged development time is not desirable. Long pillars are favorable to gate road development, but wider pillars are not. This is because development along panel length is useful to the purpose of completing panel development, but distance spent on crosscuts is not. Access to the longwall face for heavy maintenance is enhanced by shorter crosscut spacing.

Face Length and Panel Length

The choice of longwall face length is a balance of the initial capital cost committed to acquire longwall equipment versus the benefits of improved productivity, production cost savings, and additional reserve recovery for a wider longwall face. The decision is also limited by the capabilities of existing equipment, although capabilities have steadily improved over time. In subsidence-sensitive areas, wider faces can cause mine plans to be offset further from sensitive surface structures and sterilize additional reserves in the process, a cost weighed against recognized benefits.

As the industry-wide dedication to zero harm redefines acceptable hazards to ever diminishing severity and frequency, a conflicting prospect emerges as mining conditions tend to become increasingly adverse and present increased hazards. The cumulative result is an increased amount, and cost of development footage required to sustain longwall operations. Consequently, increased longwall face length is attractive. For most site-specific circumstances, an optimum longwall face length can be defined. Often there is a broad optimum in net present value terms—between 300 m of installed equipment length and up to perhaps 475 m in other cases. Present technology allows face length up to approximately 500 m with high productivity. Common face lengths range from 250 to 425 m. Achieving maximum lengths implies the use of the largest AFC chains and highest-rated electric motors and AFC gearboxes available.

Longwall panel length is an issue constrained by

- Ventilation system limitations,
- Timing considerations to avoid production outages,
- Longwall equipment longevity,
- Property constraints, and
- Reserve extent.

At some point, excessive ventilation pressure losses and insufficient air quantities dictate a practical limit to the length of a gate road development or may constrain longwall operation. Alternatively, the time required to complete a long development may compare unfavorably with the rate at which an active longwall is retreating. Yet another consideration is whether the longwall equipment can retain the required availability levels as it wears through continued service, although durability of the constituent equipment elements of the longwall has been improving through time. It is not unreasonable to expect shields, AFC drives, and frames to serve through the longest panels. AFC sprockets can be reversed or replaced during planned

maintenance intervals, as can chains, flights, and chain connectors. Shearing machines have longevity on par with the longest panels (highest tonnages) but can also be repaired as subcomponents, or a replacement shearing machine can be implemented during mining of a panel more readily than undertaking a complete longwall move (panel transfer). With this, contemporary longwall panels range from 1,850 to 4,600 m in length, with contained panel tonnages up to 11 million metric tons of coal.

Thin Seams

Most of the discussion so far has been focused on the most common high-productivity longwall variation: single-lift longwall in the 1.75- to 3.75-m extraction height range. However, thin ore deposits, typically with higher value, such as coking coals, are of interest. Thin seams are also a topic where they overlay other coal seams but must be extracted first to prevent their sterilization by undermining. Historically, thin coal seams were extracted with low-productivity shearing machines and longwall plow systems. Low-seam shearing machines of the past are hopelessly obsolete compared to current expectations. Periodic attempts are made to make low-seam shearers practical, as a number of mining projects worldwide reside at the edge of viability and would be successful if only better productivity were available. Shearers in low seams are constrained by factors such as

- Motor size (diameter and power),
- Vane depth of drums and loading performance,
- Clearance (shearer tunnel and ranging arm),
- Length and height limits (traversing undulations), and
- Operator ability to travel at tram speed of the shearer.

While efforts continue to solve the low-seam shearer issues, progress has been made in the modernization of longwall plow systems. Figure 13.8-9 shows a longwall plow. Instead of cutting the coal with a shearer, the coal is cut by a plow body captivated on the AFC and drawn from gate to gate on a dedicated chain driven by motors on each face end. Since the plow cuts only a fraction of the web of a shearing machine, normally 100 to 140 mm but up to 250 mm on the highest-capacity installations, multiple transits of the face are required to achieve equivalent production to one pass with a shearer. In contrast, the plow speed may be more than 10 times that of typical low-seam shearers, so plows offer attractive potential in deposits too low for shearers, provided that conditions are favorable (Gluckauf GmbH 1985).

Favorable conditions for a plow-type longwall include

- Strong, continuous roof and floor rock;
- Ore that plows readily and parts cleanly from roof and floor;
- Ore horizon with little thickness variation and limited undulations; and
- Ore free of intrusions, faults, or dikes.

Roof falls in the tailgate or on the face can be serious obstacles to a plow. Replacement of a significant section of the extracted horizon with sandstone through faulting or washout can be equally problematic.

The history of plow-type longwalls is extensive in Europe, where almost every conceivable condition has been confronted by plowing, but not always to successful outcomes in modern terms. The seam conditions that make plow application attractive in a modern context are limited compared to the totality of potential low-seam reserves worldwide. At present, only a few plow-type longwalls compete with shearer-based longwalls for the same markets.

Thick Seams

Unlike thin seams, thick-seam longwall has great potential worldwide. Single-pass longwall designs up to 7.25 m high have been proposed with installations more than 6.5 m high in service. A variety of issues require adaptation of the designs of these high installations, including

- Shearer steering into/out of the face,
- Structural integrity of AFC to bear heavy shearers,
- Face operator travelways and safety,
- Slab development or persistence,
- Face spalling ahead of the shield tips,
- Shield capacity and resulting weight, and
- Potential worst-case AFC starting capability.

The production history of many of these installations bears testimony to the success of designers and mine operators in overcoming the numerous challenges arising from thick-seam operation. At any period of time, some of these high faces appear in the ranks of the world's most productive longwalls.

Interest is intensifying in longwall extraction of even thicker seams than 7.25 m, but there is an accompanying recognition that single-pass longwall is not the solution. To extract very thick coal seams, the longwall top coal caving (LTCC) technique is being developed. Two major variations of the LTCC method exist. In one, the face cross section is much like a conventional shearer-based system with a single AFC. An original pass is made as customary, at a convenient height, and then a special purpose caving chute integrated into the shield canopy is lowered so that material caving from above the shield is channeled into the AFC. When product quality or caving conditions dictate, the chute is retracted, and successive chutes are opened along the face. The process is repeated until the majority of the available product has been drawn. At this point, additional undercutting is conducted with the shearer in preparation for successive drawing of caved material.

Alternatively, another technique of LTCC uses an AFC, both ahead of the shields as expected but also behind them under a powered flipper used to allow or preclude caved coal introduction onto the rear AFC. Figure 13.8-10 shows a typical installation in cross section.

Potential exists for more new LTCC installations worldwide than the total number of modern high-productivity longwalls currently in service in North America. Today, LTCC installations exist in China and Eastern Europe with considerable expansion of the technique possible there, as well as significant prospects for introduction to Australia and possibly even the Americas. The LTCC technique could require adaptation to conform to statutory requirements in countries where the technique has not previously been practiced.

Multiseam Longwall

On occasion thick seams are extracted by single-lift longwall operations conducted in a coordinated sequence, with the upper longwall progressing ahead of the lower longwall in an interlocked sequence. Though this method has examples in the field, coordinated operation of independent longwalls can be difficult. Thin interburden is more problematic than thicker interburden, as are joints or fractures that transect the interval between the seams. Thin or low-shear-strength interburden, as well as jointed interburden, can transmit stress between

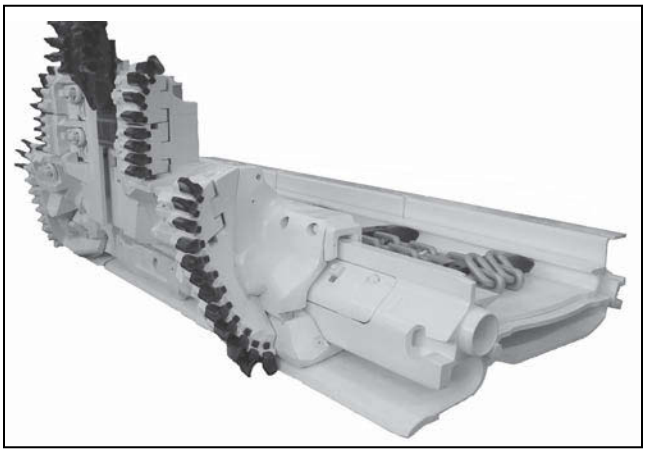
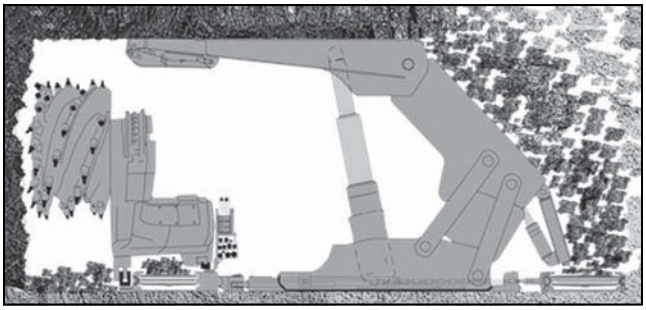


Figure 13.8-9 Longwall plow



Courtesy of Joy Mining Machinery.

Figure 13.8-10 Top coal caving with rear AFC

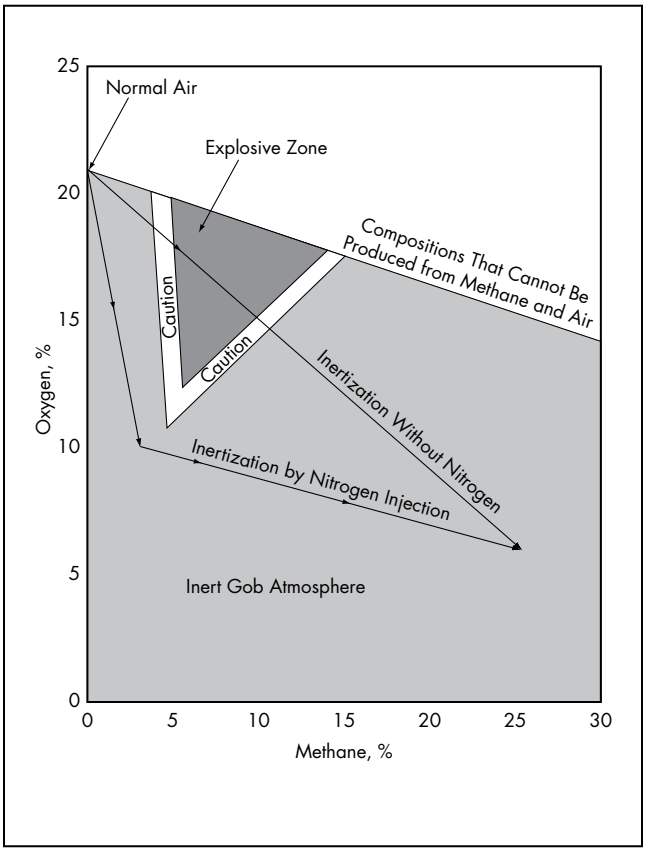


Figure 13.8-11 Coward's triangle

mining horizons more directly than horizons separated by even a modest thickness of strong, stiff interburden. High stresses are often transmitted from one horizon to another where strong structures surrounded by high-extraction workings in one horizon are approached by workings in the other horizon. In general, multiseam operations are more focused on extraction of independent seams at appreciably separated intervals of interburden and time. Often the earlier workings are not executed with anticipation of modern longwall mining in the future, thus making mine planning an exercise in mitigating interseam interactions.

Longwall Ventilation

Ventilation of longwall mining systems seems to exhibit nearly endless variety in the exact details of worldwide plans (AusIMM 1986; Kennedy 1999). Taken at a broader level, longwall ventilation falls into two general categories with very different mechanisms (MSHA 1996). To introduce these principles, some basic attributes of methane and air mixtures should be clearly understood. Figure 13.8-11 shows Coward's triangle (Coward and Jones 1952). Other representations of the explosion hazard are popular in various locations around the world, but the chemistry and physics of the matter are universal. Thus, other representations describe the same physical behavior.

Two principal regions are represented in Figure 13.8-11. The upper portion of the figure describes compositions that cannot be formed from methane and air and is therefore irrelevant. Below these naturally impossible compositions of methane and air is the region of naturally possible mixtures. These possible mixtures include two major subdivisions: inert mixtures and explosive compositions. The inert mixtures will not explode if exposed to a potential ignition source and occur as fuel-lean or fuel-rich compositions with respect to the explosible compositions. The explosible compositions exist within Coward's triangle shown in the approximately 5%–15% methane range and approximately 12%–21% oxygen range. The zone depicted around the triangle is an arbitrary safety margin. This physical behavior of methane and air mixtures being an absolute principal, two approaches to avoid the explosion hazard exist. One is to provide fresh air to dilute methane to a fuel-lean state. If such a state can be maintained without fail, it renders an acceptably safe condition. Paramount to this strategy is that the fuel-lean (oxygen-rich) atmosphere cannot transition from a respirable state, required where people work and travel, to a fuel-rich (oxygen-deficient) state, which is likely to occur in the unventilated gob. Where the respirable mine atmosphere transitions to an unventilated gob atmosphere, particularly after mine workings are sealed, it becomes impossible to guarantee that the reducing oxygen level and increasing methane level will not cross through the region of explosible compositions without active intervention. This case is described by the upper trajectory line in Figure 13.8-11. If even a small energy source above the intrinsic safety threshold is available to ignite the explosive composition, a violent event will result in proportion to the volume of gas mixture ignited (SIMTARS 2001). If coal dust becomes airborne secondary to a typical methane explosion, a much more energetic and violent event can arise (Burrows 1989). This mechanism has repeated itself many times worldwide, often with disastrous outcomes. Application of incombustible dust (rock dust or stone dust) to areas where

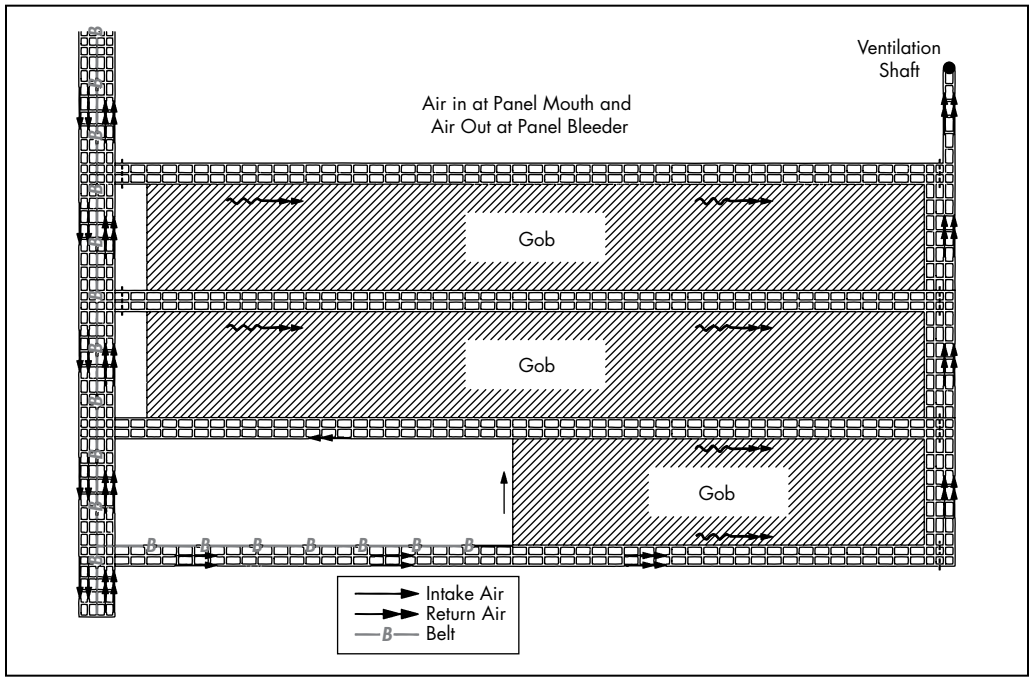


Figure 13.8-12 Typical flow-through bleeder system

coal dust can accumulate is effective in mitigating this dust explosibility hazard.

An alternative mitigation strategy for the explosion hazard involves the deliberate exclusion of oxygen from gob atmospheres, followed with forced inertization if necessary to depress oxygen levels below those where an explosion hazard can exist, regardless of methane concentration. This is represented by the lower trajectory line in Figure 13.8-11, which is actively controlled by inert gas injection and never crosses the region of explosion hazard, Coward's triangle. Periodic atmospheric monitoring, followed by application of the inertization process as necessary, can prevent explosible methane–oxygen compositions from being created by already inert atmospheres where introduction of new oxygen is possible.

With this in mind, the two major strategies of longwall ventilation are bleeder and bleederless ventilation systems. A conceptual flow-through bleeder longwall ventilation system is shown in Figure 13.8-12.

In this system, fresh (intake) air is supplied through the headgate entries and splits at the headgate to ventilate the longwall face. An adequate volume is allowed to flow behind (inby) the face at the headgate to ventilate the perimeter of the gob. Along the face and at the tailgate, air is allowed to "bleed" into the gob and tailgate to ventilate the methane to fuel-lean levels. This is the most common ventilation system used in the United States. It is notable that because atmospheric compositions are uncertain where they cannot be monitored and controlled in the bleeder-ventilated gob inby the longwall and in the adjacent abandoned gobs, any potentially persistent ignition source such as spontaneous combustion or intermittent source of ignition energy such as lightning is reason for concern.

The major alternative to bleeder ventilation of longwalls is bleederless ventilation. The bleederless longwall ventilation system is depicted in Figure 13.8-13.

The notable differences between the bleederless and bleeder ventilation systems are that in the bleederless system seals are built in headgate crosscuts immediately inby the face in order to progressively seal the gob and exclude oxygen as the longwall retreats. If concerns exist about atmospheric compositions in the gob or if oxygen invasion could stimulate spontaneous combustion, the added feature of forced inertization, as discussed previously, can be an effective risk-mitigation strategy. Gases such as nitrogen, CO_2, or even oxygen-depleted air can be used for safely inertizing mine atmospheres. The exhaust gas from combustion processes, such as gas turbine engines, can potentially be a source of oxygen-depleted gas mixtures for inertization.

To prevent oxygen inflow into sensitive gobs, pressure balancing is often applied to minimize pressure differentials across seal lines and minimize introduction of new oxygen to a gob. An alternative technique is the use of *balance chambers*, as depicted in Figure 13.8-14. In this method an initial seal is built to separate the atmospheres, and then another seal is built outby the initial seal, becoming primary. Inertization gas delivery pipes penetrate both seals to allow inert gas delivery to the sealed area and the space between the seals. If the seals have reasonable integrity, the space between the seals can be elevated to a low positive pressure, and the small leakage volume that occurs is either into the gob, the mine airway, or both. The net effect is to prevent oxygen ingress into the sealed area at low total inert gas expenditure. Sudden change in inertization gas consumption can be an early indication of deterioration in seal performance.

A valuable tool to monitor the composition of mine atmospheres in critical areas is a *tube-bundle* system. With this equipment, atmospheric samples are drawn from critical locations underground through dedicated individual tubes to analysis equipment located at the mine surface. This allows sampling of areas where electrical sensors might be prohibited

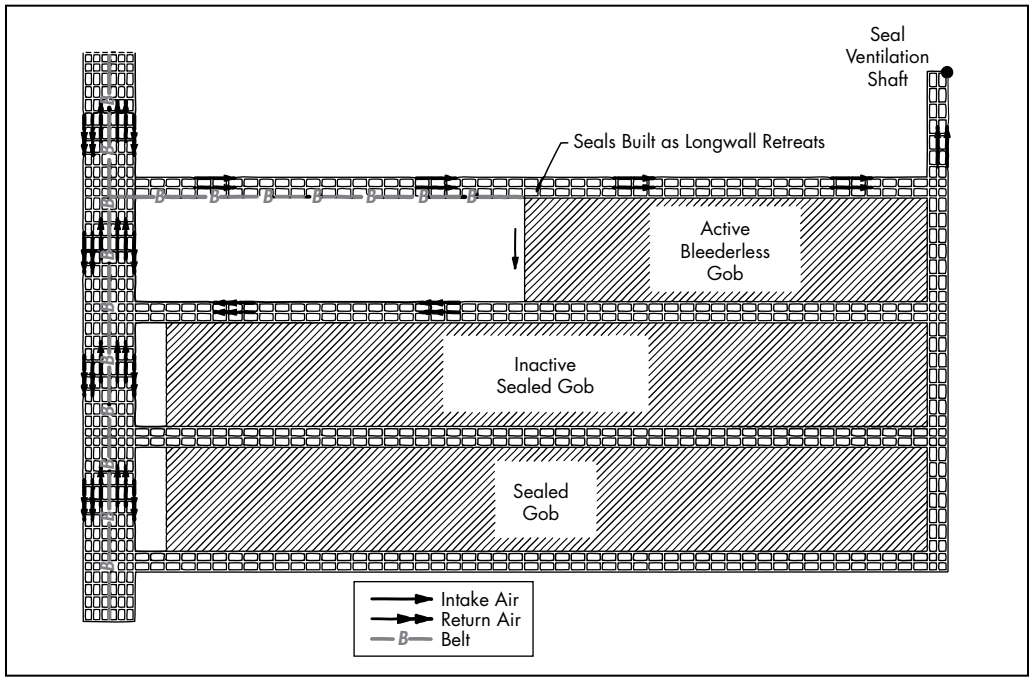


Figure 13.8-13 Bleederless longwall ventilation

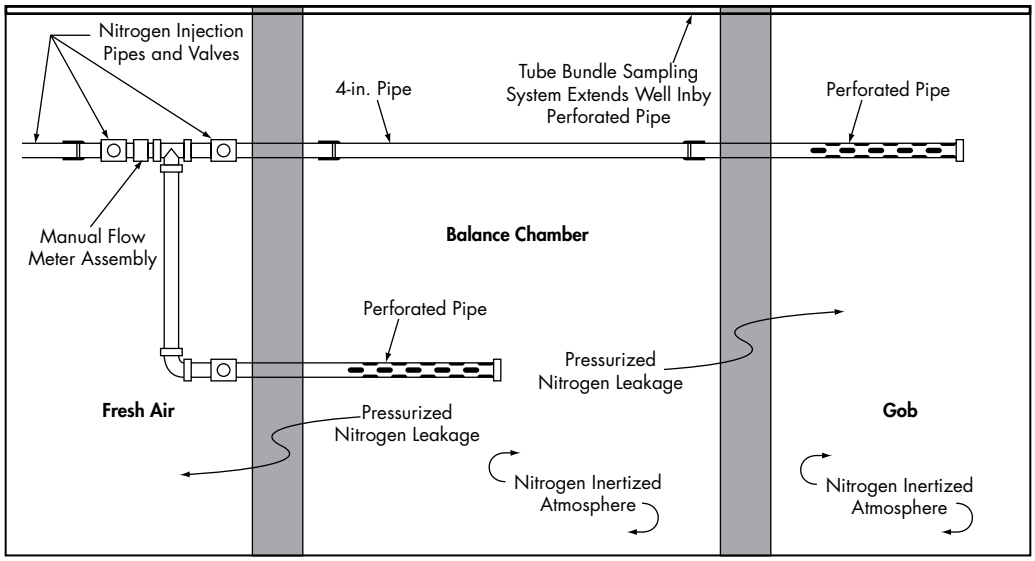


Figure 13.8-14 Inertized balance chamber

and sampling continuation in the event that electrical power is removed, as is the case in many mine emergencies.

An addition to the controls inherent in either of the ventilation systems described, gob ventilation boreholes (GVBs) can be used to assist the ventilation system by extracting gob atmosphere, predominantly methane, before it enters the ventilation system. GVBs are often centrally located in a longwall panel when the intention is to intercept methane evolved from broken strata before it enters a bleeder ventilation system. These holes, drilled from the surface in advance of mining, are often spaced at larger intervals than GVBs intended to control expansion volumes in a bleederless ventilation system. The location of GVBs is typically along the periphery of a panel, where void space is connected to workings at seam level, when the intention is to control barometric expansion volumes, as in bleederless ventilation systems.

Diurnal barometric pressure variation occurs daily, with the barometer falling as the atmosphere warms with duration and intensity of solar exposure and rises as night falls when the atmosphere cools and air density increases. Random barometric pressure variations are related to weather systems. The actual barometric pressure is a function of both effects and can be predicted for a surface location with good overall accuracy. Operation of GVBs in response to barometric pressure variation,

alternating with inert gas injection, can control evolved methane and prevent oxygen invasion into progressively sealed gobs, as in bleederless systems, or other sealed areas.

Although GVBs are typically drilled vertically from the surface, an alternative method applies medium radius drilling technology to postmining methane drainage. The wells start with a vertical collar followed by a build section where the well is steered to a horizontal position. The horizontal lateral is located in a stable horizon above the longwall panel. Holes up to 1.6 km in length, producing approximately 1 m^3/s of methane, have been achieved.

An alternative to GVBs drilled from the surface is an array of cross-measure boreholes drilled at upward inclination from the seam horizon. Such cross-measure holes can drain methane prior to mining and be used later to a similar effect as GVBs when caving behind the retreating longwall opens a communication between the holes and the larger gob. Gas captured by the holes can be collected by an underground piping system and directed to a borehole to the surface without entering the mine ventilation system.

Horizontal boreholes, originating in-seam or turning into the seam from vertical wells to the surface, can be progressed in-seam for more than 1 km to drain methane in advance of mining. These holes have proven to be highly effective for methane drainage but must be planned carefully, as they can later present hazards that must be mitigated before mining progresses through them.

CONTEMPORARY ISSUES

Improving the health and safety of miners by reducing their exposure to hazards has been a major motivation in the steady progress of mining mechanization. Longwall mining as it exists today is the product of this ceaseless effort. Greater productivity and resource recovery are realized, while the incidence of injuries and accidents continues to decline as a trend. Few would dispute that longwall mining is, by far, the safest and most efficient underground soft-rock mining method.

Although many concerns have been addressed with great success, others have persisted. Coal workers' pneumoconiosis, more commonly called "black lung," is a chronic disease which investigators have correlated to respirable dust exposure in the coal industry. Black lung is often debilitating in its advanced phase and may lead to death. Most coal-producing countries worldwide acknowledge this hazard and have occupational health limits for respirable coal dust exposure intended to prevent workers from contracting the disease. Not all dust is deemed hazardous but only the submicrometer (<10-μm size particles) fraction of dust is believed to contribute to black lung. Because longwall mining results in large production, it is not surprising that increased amounts of dust can also result, with respirable dust being a fraction of the total.

Depending on the regulatory strategy applied, this can lead to different conclusions. One strategy is based on environmental exposure. This assumes that a worker will not benefit from any personal protective equipment and that safety is produced only by maintaining environmental levels below disease thresholds. This strategy does not give innovation or technology an opportunity to contribute solutions unless it impacts the environmental exposure as opposed to the respiratory exposure. However, personal protective equipment such as airstream helmets offers demonstrated effectiveness, and automation technology has the potential to remove workers from most dust exposure. Personal dust exposure management is now more feasible than ever, with the demonstration and introduction of personal dust monitors. Seemingly, the optimum solution to the problem of dust exposure would apply the best available technologies, crediting the benefits in realistic terms, to create incentive for continued progress in longwall technology.

Another issue of modern underground mining is related to surface subsidence. The most damaging aspect of mining-induced subsidence is the zone that connects undisturbed surface with areas subjected to near maximum vertical displacement. This boundary varies from a few meters to a few hundred meters wide and commonly wraps around the perimeter of the subsided area. The condition is exaggerated when subsidence takes place years after active mining and may continue for a protracted period of time. It is also worsened when small land areas are subsided, a few hectares per instance, leading to a greater proportion of subsided land area being subjected to the most damaging end outcome (Kratzsch 1983; SAIMM 1982). These situations, often acting in combination, make compensation for damage to surface assets difficult, because the mine operators may no longer have a corporate presence or the ability to pay for reparations years or even decades after the fact (Peng 1992).

The development of longwall mining has brought relief to these concerns. As modern longwall panels have gotten wider and longer, the proportion of subsided land area permanently subjected to the most damaging boundary conditions has decreased. Also, unlike the subsidence of smaller individual areas, the subsidence develops to its maximum level in close alignment with predictions and generally does so in a period of weeks to months after active mining by longwall methods. Thus, all stakeholders are in a better position, both in terms of damage and timing, to make such reparations as may be appropriate.

Longwall mining has been extensively applied to urban and suburban areas in Germany, Poland, and the United Kingdom to mine under a tremendous variety of structures, including public buildings and residences, industrial facilities, roads, bridges, railroads, pipelines, and power lines, to successful outcomes. Longwall mining has also been used worldwide to mine under rivers, lakes, and oceans successfully. With the worldwide knowledge base for mitigation of subsidence damage to surface structures, many alternatives can often be applied to minimize damage to surface assets or repair them effectively.

As longwall mining transitions from rural to suburban areas in the United States and Australia, it has vocal critics. The arguments put forward derive from many origins but generally do not suggest a better method to extract the resource but rather oppose the extraction all together or dispute the value of the damages that might be created or the ability to bring about acceptable reparations.

CONCLUSION

When longwall systems are properly selected and operated, the results can be spectacular and sustainable. These installations make longwall mining look easy. In reality, successful selection and operation of longwall systems are based on attention to detail and a commitment to continuous improvement. These pursuits require much effort and rely on a capable staff as much as the equipment composing a longwall mining system.

Success also depends on a close collaboration between mine operators and technology suppliers, often as a strategic

alliance. Both parties must begin with the end goal in mind, which has to be attainment of safe production and profitability for both partners. Mine operators and technology suppliers must recognize their stake in the shortfalls as much as the successes, constantly being prepared to contribute investments in time and money in pursuit of continuous improvement. This is especially important in consideration of the long-term nature of the relationship created by long equipment service life and large capital investments.

Continuous improvement is necessary to meet the evolving expectations of stakeholders to the overall production process. These stakeholders expect the result of zero harm as it applies to health and safety of employees, local communities, and the environment. They also expect improving productivity and diminishing costs, even in consideration of future mining conditions that typically are more adverse than past or present conditions.

For all of these reasons, success at longwall mining is exclusively a result of organizations with aligned purpose, detailed planning, and controlled execution leading to sustainable results on the part of a wide diversity of involved stakeholders.

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CHAPTER 13.9

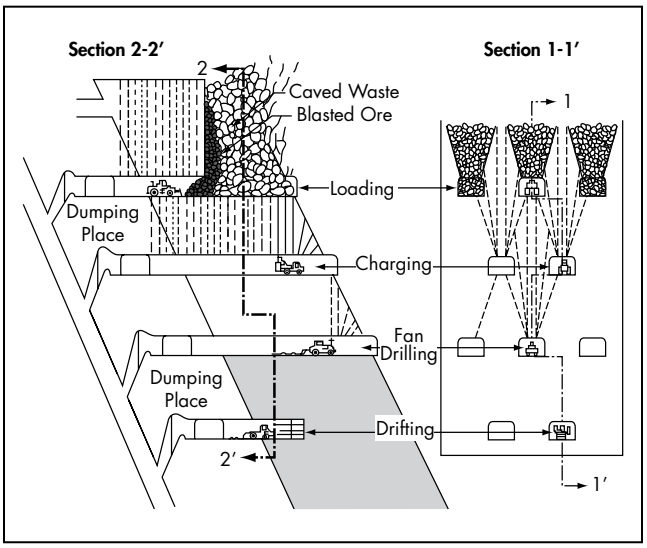
Sublevel Caving

Geoff Dunstan and Gavin Power

Sublevel caving is a top-down mining method allowing earlier production than sublevel stoping with less upfront development than traditional block caving. In large-scale operations it has the potential to be a *factory method* (meaning highly repetitive) with the cost and capacity benefits associated with high production, moderate development requirements, and maximum use of automated equipment (Bull and Page 2000). In smaller mines it can be used as a selective method with lower production rates, where the capacity benefits are less evident. Horizontal slices of in-situ ore are progressively blasted and extracted, with blasted ore and caved rock filling the void created by ore extraction (Figure 13.9-1). Dilution can be managed and minimized by disciplined draw control. As this process progresses, caving propagates upward to create surface subsidence.

Originally, sublevel caving was used in weak ground that would collapse when the timber supports were removed. After a supported drift was pushed across the ore body from an access or perimeter drift, the supports would be individually removed in a retreating fashion. As each support was removed, the ore would cave—thus the name sublevel cave—and be slushed out to the perimeter drift. The next support would not be removed until the amount of diluting material became excessive. Overall, the method was slow and combined poor ore recovery with high dilution.

Later applications of the method saw it applied in relatively competent ore bodies with weak hanging walls in order to facilitate the caving process and avoid the creation of large voids. The stronger ground required drill-and-blast to take the role of the support removal, so in this version of the method it is no longer the ore that caves but the host rock above and immediately adjacent to the ore. With the introduction of drill-and-blast, sublevel caving could be utilized in ore bodies with footprints too small for natural cave development and/or dipping geometry unfavorable for block caving. In recent times, the method has been applied to ore bodies with strong hanging walls where cave assistance techniques have been used as required.



Source: Kvapil 1992.

Figure 13.9-1 Schematic of sublevel caving mining

Current applications of the method are accessed by decline(s) and have sublevels established on approximately 20- to 35-m vertical intervals, far greater than the early applications of the method, due largely to advances in drilling and blasting technologies. A perimeter drift is driven along (transverse layout) or perpendicular (longitudinal layout) to the strike of the ore body and is generally offset approximately 20 m from the ore–waste contact. Some mines develop the perimeter drift in ore and recover it after production is completed from the drifts. The perimeter drift is developed to access the production drifts for ore transportation, services, and ventilation. The production drifts are spaced between 12 and 30 m on horizontal centers. To enable optimal coverage for drilling and to allow for the downward flow of caved material, they are staggered between the levels (Figure 13.9-2).

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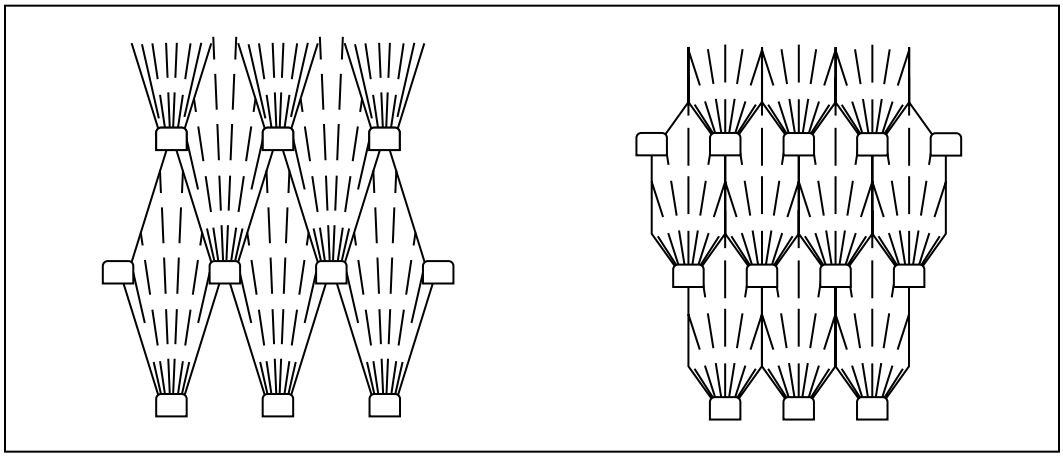


Figure 13.9-2 Sectional view of sublevel caving ring options

Over the years, the ratio of development-to-production ore has increased from approximately 1:3 to as high as 1:10.

The production ore extraction process commences by placing either a slot drift on the far ore–waste contact or placing individual slot raises at the end of each production drift. The slot drift will have a slot raise at one end and a series of parallel rings, generally on 2- to 3-m burden, the width of drift reaching up to the level above. Once the slot has been opened, to create adequate expansion room for the first production rings, some *slashing* rings will start the production rings in the drifts until the production rings reach full height. The production rings will usually be angled forward by 10° to 20° from vertical to assist the breakage and flow of the ore in the confined blasting environment and to help maintain geotechnical stability at the brow. The production ring burden is generally between 2 and 3 m. The production rings drill up through the pillar between the production drifts on the level above, as shown in Figure 13.9-2.

The layout of the method requires secondary ventilation to be forced into the production drifts throughout both the development and production phases of the level. Load-haul-dump (LHD) machines are used to extract the ore from the fired rings and transport it back to either stockpiles or ore-passes located on the perimeter drift.

As the initial level is opened, only the swell material from the blast (approximately 40%) is excavated, because caving will likely not be initiated until production on the level is well advanced. The extraction ratio of ore increases to a nominal value of approximately 80% to 100% over the subsequent two or three levels. The reduction in the early draw rates is designed to minimize the likelihood of air blast until the overlying cap rock and/or hanging wall starts to break up and flow. In addition, the reduced draw builds up a blanket of ore above the active drifts, which can assist in minimizing the impacts of dilution in a blind cave, as well as protecting the drifts when the sublevel caving is commenced directly below an open pit.

Operational sublevel caving practice varies widely, and consequently tonnage and mineral recoveries are also variable from operation to operation. Overall tonnage extraction rates for large-scale sublevel caving mines will be approximately 80% to 120% of the blasted tonnage for approximately 65% to 95% of the total ore-body mineral content. Of the rock drawn from each fired ring, approximately 15% to 25% will be dilution. More dilution comes at the end of each individual ring's life, and varying practices mean that dilution results also vary from site to site (Power and Just 2008).

When extracting the ore body, strict operational rules are required to ensure the method's success. Only after the upper level has been retreated a safe distance, somewhere between one and two times the sublevel interval, can the level below be commenced. Within the level, the production rings need to be brought back in a planned sequence (managing the lead–lag between adjacent production drifts) to minimize the impact of stress and explosive damage on neighboring drifts.

APPLICATION OF SUBLEVEL CAVING

The suitable ore-body characteristics that lead to the successful implementation of the sublevel caving method include

- Cavable overlying host rock,
- Relatively large ore-body footprint or a relatively weak hanging wall to initiate and propagate caving,
- Steeply dipping and relatively uniform ore body,
- Reasonably uniform grade distribution,
- Disseminated ore body with a low-grade zone around the sublevel caving production area to minimize the impact of waste dilution,
- Competent ore body to minimize the excavation support requirements and improve drift availability to enable continuous production, and
- Visually different ore and waste properties.

Although sublevel caving can be successfully applied to ore bodies missing some of these characteristics, any such omissions generally lead to increased operational complexity (Power and Just 2008).

GRAVITY FLOW

The sublevel caving method relies on the gravity flow process to get the ore to the drawpoint. As a result, gravity flow has been a focus of sublevel caving research for many years.

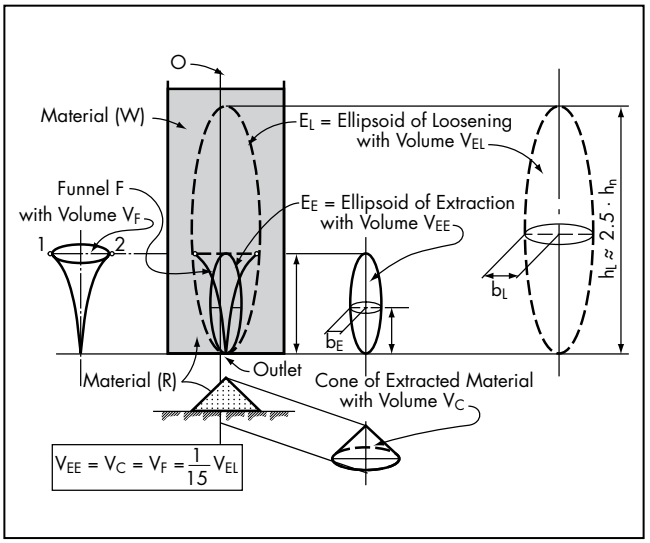
Gravity flow in the sublevel caving environment is unique in a number of ways:

- The effects of blasting produce differential fragmentation within the ring, both vertically and laterally.
- Draw occurs from drawpoints on multiple levels, laterally offset, resulting in multiple stages of flow.



Source: Power 2004b.

Figure 13.9-3 Large 3-D physical model used to study sublevel caving flow at the University of Queensland, Australia



Source: Kvapil 1992.

Figure 13.9-4 Example of idealistic flow analysis derived from analysis of physical models

- Each individual ring is subject to unique factors, including geological structure, drill-and-blast inconsistencies, and differing draw patterns.

As in other fields of research, the gathering of information on flow has been a gradual process, with successive workers building on the data and theories of past workers. Early studies of gravity flow focused on small-scale, glass-fronted physical models and over time expanded to larger three-dimensional (3-D) models such as shown in Figure 13.9-3. These models involved the drawing of broken rock from geometrically scaled and well-controlled environments, so that the resulting movement and extraction zones could be studied in detail and accurately measured.

Research using these physical models resulted in the promulgation of influential theories and relatively precise mathematical formulas to describe concepts such as particle velocity profiles, the ellipsoid of loosening and of draw, and the relationship between these bodies (e.g., Figure 13.9-4). It was later determined from full-scale marker trials that the shapes of the flow and movement envelopes were not exactly elliptical (Janelid 1975); however, they are close enough for the ellipse to be used as a basic design concept in sublevel caves (Power 2004b).

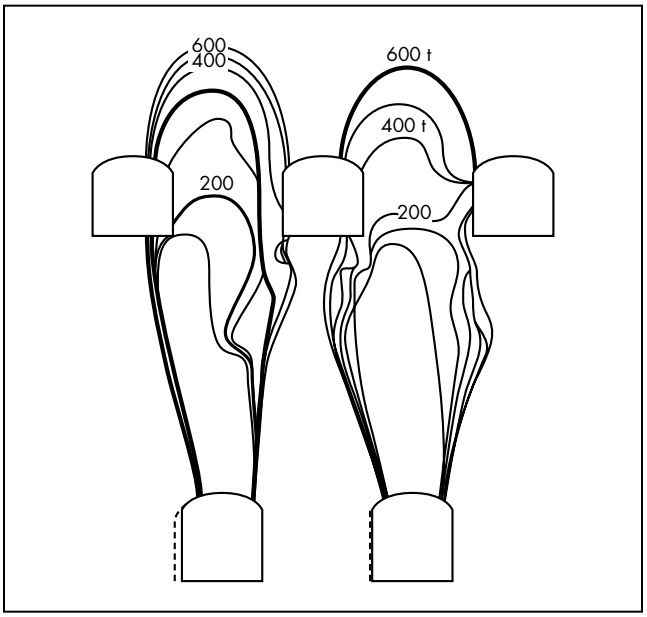
In practice, these theoretical explanations of flow in bins and hoppers serve mainly to give a broad understanding of flow in a highly controlled environment. Taken in isolation, they comprise a highly idealized, single-level description of granular flow with only a general relationship to granular flow in real sublevel caving environments. However, results from these models were useful in guiding design of early sublevel caves and provided important insights into the mechanisms controlling flow. They also led to the development of some of the fundamental concepts underpinning the method, which are well established today (e.g., drawing drawpoints interactively and loading from side to side in drawpoints).

Although model studies provided a basic understanding of gravity flow mechanisms, full-scale field trials were needed to obtain flow data in operational sublevel caves. Marker trials involve the placement of markers (usually short lengths of steel piping or cable) into internal marker rings before the experimental production ring is blasted. These markers are collected as the ring is mucked, and the resulting data is analyzed to assess the flow patterns and ultimate recovery of the ring.

The earliest comprehensive trials were those carried out by Janelid at Grängesberg mine in Sweden (Janelid 1972). Janelid's marker trials showed gravity flow in the full-scale environment but only with reference to recovery from the level on which the rings were blasted. They were carried out on relatively small sublevel caving layouts (approximately 3-m-wide crosscuts on 7-m crosscut centers and 13-m sublevel heights). For the relatively small ring tonnages drawn, the trials showed high angles of draw and irregular draw patterns not quite elliptical in shape (Figure 13.9-5).

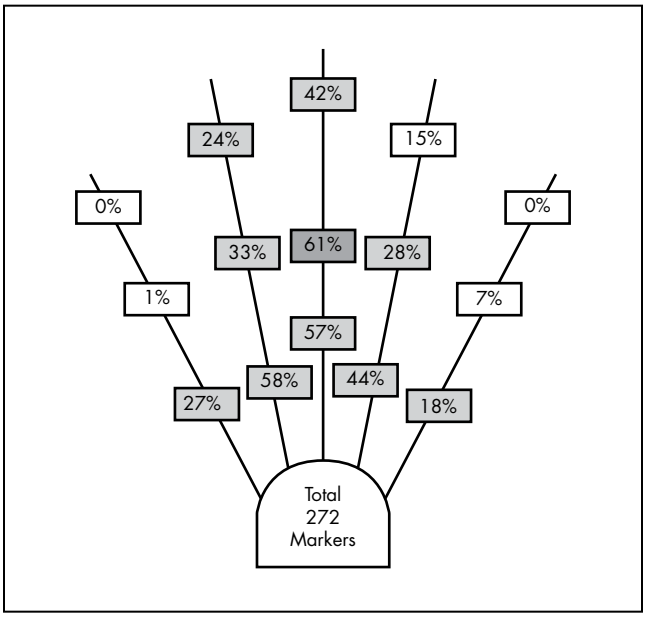
In recent times, marker trials have become more popular in response to significant changes in sublevel caving geometry, which are partly due to improvements in drilling and blasting technology. Figure 13.9-6 shows a comparison of changes in sublevel caving geometry at the Kiruna mine in Sweden between 1983 and 2008.

The Kiruna 2000 program of marker trials was also single-level trials, although they did not have the same density of markers as the Grängesberg trials and relied on visual identification of markers at the drawpoint for recovery of experiment information. This resulted in only about a third of the installed markers being recovered (Gustafsson 1998). A



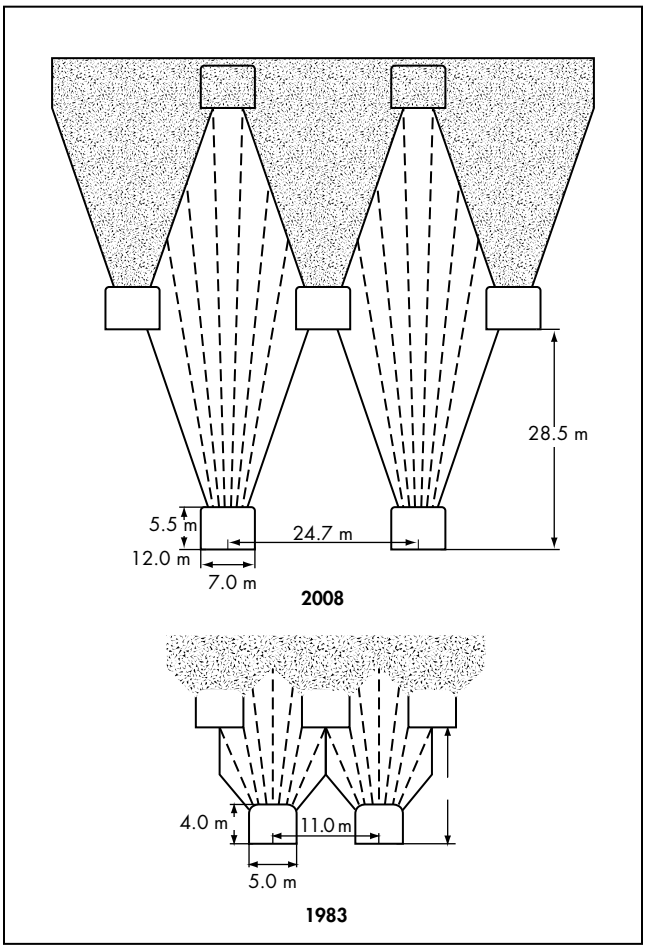
Source: Janelid 1975.

Figure 13.9-5 Example of analysis from Grängesberg marker trials



Source: Quinteiro et al. 2001.

Figure 13.9-7 Summary results from the Kiruna 2000 marker trial program, indicating percentage of markers placed in each location recovered



Source: Power and Just 2008.

Figure 13.9-6 Comparison of sublevel caving geometry at Kiruna mine in 1983 and 2008

diagram summarizing the results of these trials (Figure 13.9-7) indicates a total of 272 markers recovered and shows the percentage of markers recovered from various locations in the marker rings. This analysis indicates that most markers recovered were almost directly above the drawpoint. One important advantage of the Kiruna trials was that they gave an indication of recovery from greatly upsized sublevel caving rings, with crosscut widths, spacings, and sublevel heights all approximately double those used at Grängesberg.

The Kiruna marker trials showed that, although mining scale had dramatically increased because of improvements in equipment technology and a focus on cost reduction, granular flow properties had not changed, leaving large parts of the fired production rings unrecovered (if viewed only from a single-level perspective).

Marker trials carried out at the Ridgeway mine in New South Wales, Australia, were the first comprehensive multilevel marker trials concentrating on granular flow through an entire sublevel caving system (Power 2004a, 2004b). Carried out between 2002 and 2006, they involved more than 20,000 markers in more than 70 trial rings, with data on marker recovery collected up to six levels below the level on which the marker rings were fired. These trials involved the placement of markers in three planes of markers, evenly spaced within the burden of the production rings (nominally 0.65 m, 1.3 m, and 1.95 m from the next production ring to be fired) (Figure 13.9-8).

An important component of these trials was the confidence level gained from the systematic magnet-based marker recovery procedure, which removed some of the uncertainty associated with the need to visually identify markers at the drawpoint and allowed analysts to more confidently assess areas of recovery and nonrecovery in individual rings.

Analysis of data from these trials first introduced the concepts of primary, secondary, and tertiary recovery for the

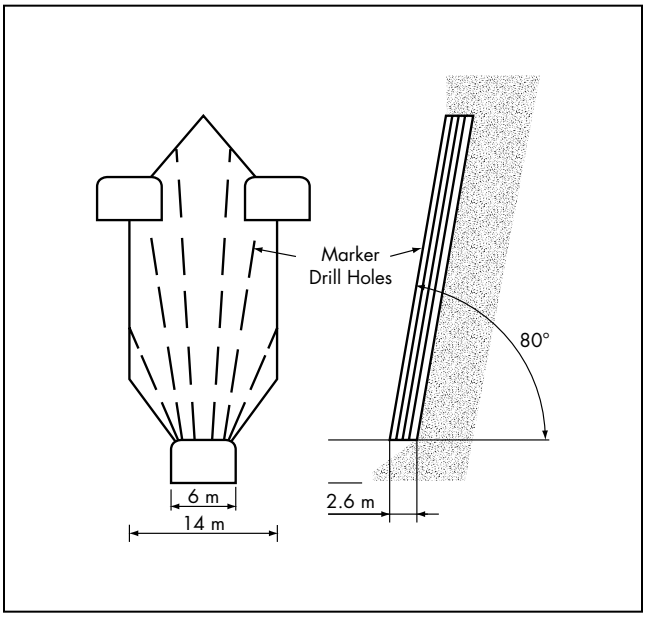


Figure 13.9-8 Layout of marker rings for Ridgeway marker trials. Drill holes in succeeding marker ring planes were offset to increase spatial coverage of markers. Markers were placed at 1-m intervals within marker drill holes.

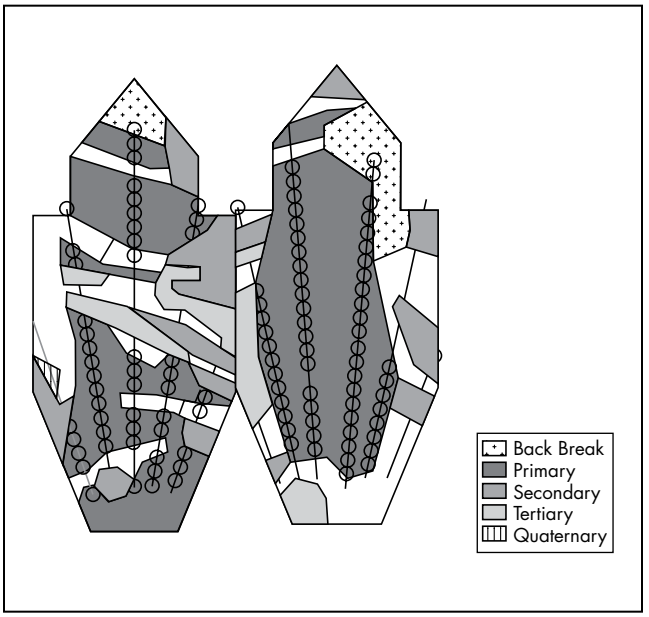


Figure 13.9-9 Marker trial results from Ridgeway showing typical width of draw for flow in adjacent crosscuts (indicating noninteraction between primary drawzones)

three main levels on which sublevel caving ore is recovered. *Primary recovery* is defined as ore reporting to the drawpoint on the level from which its sublevel caving ring was fired, whereas *secondary* and *tertiary* refer to ore that reports on successive levels below this. These trials also allowed testing of a number of theories that had gained traction in the absence of solid data and allowed other advances in understanding of granular flow in real sublevel caving environments.

Similar trials were later carried out at Perseverance mine in Western Australia (Hollins and Tucker 2004; Szwedzicki and Cooper 2007). These trials aimed to determine the precise time of marker arrival at drawpoints and therefore required manual collection of markers in order to determine exactly when, in the draw cycle, each marker appeared at the drawpoint. Although this made multilevel analysis and completion of a large, statistically significant trial program more difficult, important data largely comparable to previous field trial programs was collected.

More recently, research has led to the development of *smart* markers, which can transmit a radio frequency signal to receivers that record their extraction location and time (University of Queensland 2007).

The following sections discuss some of the new understanding that has been gained from full-scale marker trial programs carried out since 2000.

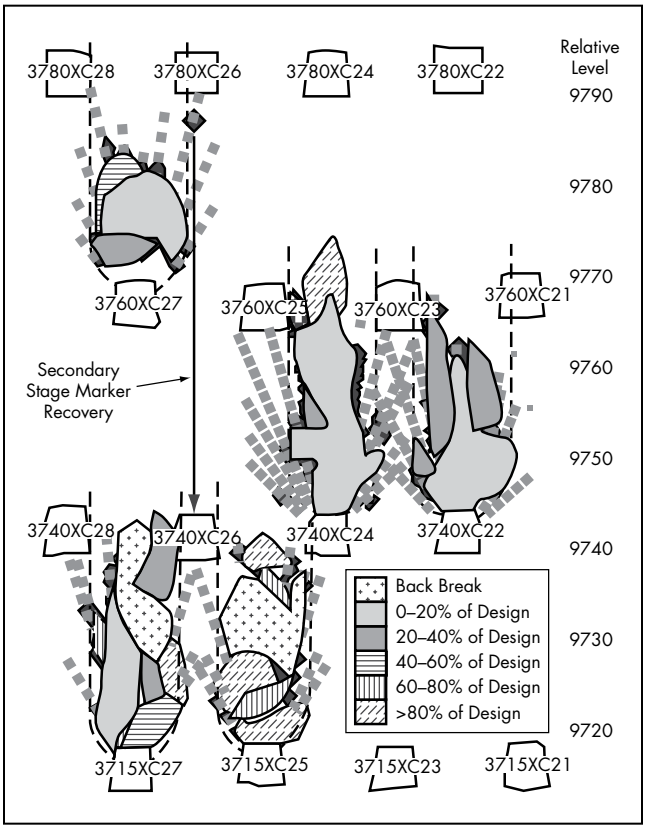
Width of Draw

Width of draw has consistently shown to be, on average, less than the width of the fired ring. The Ridgeway layout comprised 5–6-m-wide crosscuts on 14-m crosscut centers, as this mine was designed using a recovery rather than cost-reduction focus (Power and Just 2008). Here, a 6-m-wide drawpoint generally produced a primary recovery zone with a maximum width of between 8 m and 12 m. Therefore, despite the relatively close crosscut spacing, interaction between primary drawzones rarely occurred. Figure 13.9-9 illustrates this situation for sections taken through two adjacent crosscuts. All the different stages of recovery were recorded and are represented, and it is clear that the primary recovery zones do not interact.

Prior to the completion of these trials, a theory known as interactive draw (as proposed for sublevel caving) had been put forward (Bull and Page 2000) and influenced the design of some sublevel caving mines. This theory suggests that by drawing from adjacent sublevel caving crosscuts interactively or in close sequence, zones of low pressure can be created between the drawpoints, allowing the rock to flow at lower angles than in drawpoints that are drawn in isolation. Prior to this, physical modeling results had also been used to suggest that drawing from an interactive front, rather than in isolation, would decrease dilution ingress from the sides of the rings and reduce hang-ups (Janelid 1974).

In reality, interactive draw did not result in primary drawzones flowing to the full width of the ring and interaction with adjacent crosscuts (Power 2004a). Similar trials completed at Perseverance mine also indicated no evidence of interactive draw in any of the trial areas (Hollins and Tucker 2004). Summary results from these trials clearly show the episodic nature of flow from individual rings; those at relative narrow draw widths were similar to ones seen at Ridgeway (Figure 13.9-10).

Figure 13.9-5 also shows (albeit for reduced production tonnages) that recovery zones for rings spaced on 7–8-m centers at Grängesberg are not interacting, even though strict interactive draw controls were used in these experiments. Flow here was only allowed to develop to half the height seen at later mines (given the smaller sublevel spacing). Figure 13.9-7 also suggests no interaction between adjacent crosscuts at Kiruna, even though these results summarize the marker recovery results of all the trials completed in its program, rather than



Source: Hollins and Tucker 2004.

Figure 13.9-10 Summary of the Perseverance marker trial results

of an individual marker trial. This is not surprising given the relatively large crosscut spacing at Kiruna mine.

In practice, every mine will have a range of differing input conditions that will affect marker trial outcomes, added to which are the significant differences seen between trials at individual sites. Even so, the results on width of draw at different sites are relatively similar.

Depth of Draw

In approximately 20% of cases, draw at Ridgeway never extended to the full 2.6-m depth of the ring. In more than 50% of cases, the primary flow zone only reached full ring depth late in the draw cycle for a small percentage of the ring height. Figure 13.9-11 shows an example of the primary draw, which is both narrow and shallow.

There is some conjecture about the cause for shallow draw as seen at Ridgeway. Visual observations indicated that unrecovered rock at the back of the ring sometimes retained the original rock-mass texture (Figure 13.9-12). However, if a drawpoint opened up, the less mobile ore at the back of the ring often collapsed into the drawpoint. This indicates that the rock was being fractured but the fragments were not being effectively separated from one another. However when hang-ups occurred and the LHD was used to dig farther into the base of the ring, greater depth of draw was sometimes achieved, with corresponding higher recoveries.

The shallow draw at Ridgeway shows a variation from the argument that early dilution entry originates behind the ring (from the waste side of the ring). Like other mines (Gustafsson 1998; Hollins and Tucker 2004), Ridgeway commonly measures early entry of rock from outside the ring (at approximately 20% draw). However, this rock has been shown to originate from the level above the ring, due to flow piping preferentially up the solid face of the ring.

One of the assumptions underlying the sublevel caving method is that the entire ring is fractured effectively, compacting the waste ahead of it and allowing the ore to be drawn relatively undiluted from the drawpoint. Some, however, have shown that the space available to allow adequate swell (and particle disassociation) for all portions of modern sublevel caving rings is insufficient (Hustrulid 2000). The Ridgeway results may support this hypothesis, indicating that the rock at the back of the ring is not being mobilized because of the combination of lack of opportunity to swell and cushioning of the ore at the free face of the ring. It is also possible that differing modes of dilution entry are occurring at different mines, where different blasting and draw control practices are used. Marker trials allow the opportunity to learn more about the sources of dilution entry and for mine operators to refine drill-and-blast and draw control practice to postpone and reduce dilution entry.

Ore Recovery over Multiple Levels

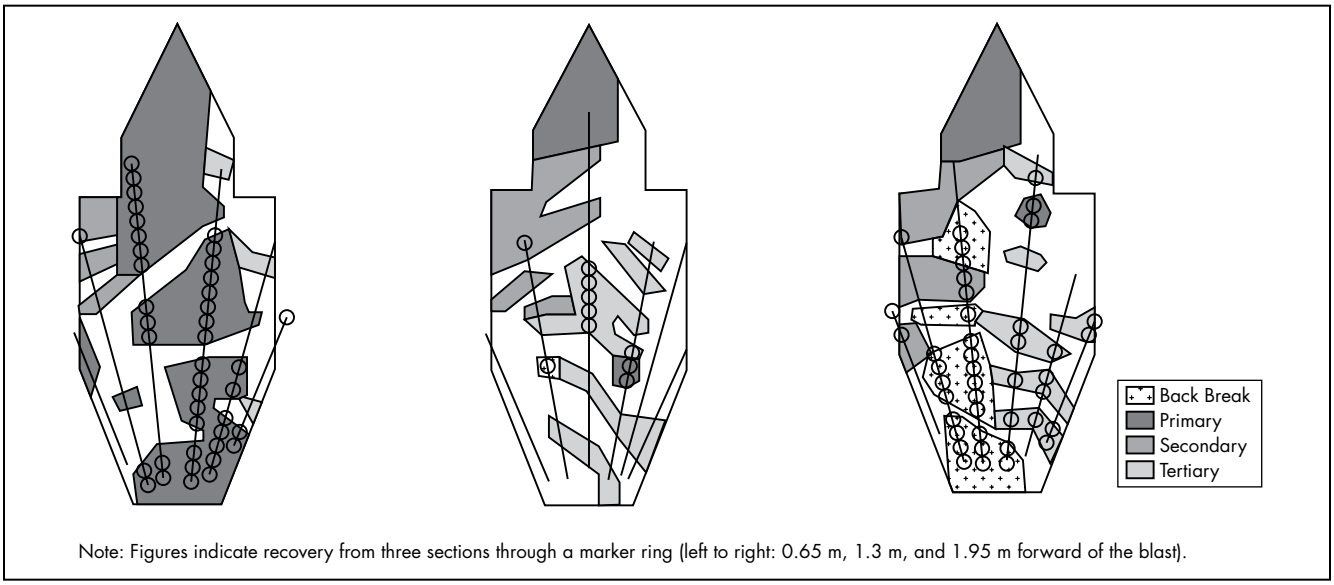
Before completion of the Ridgeway marker trials, assumptions about levels of primary recovery were significantly higher than those actually measured. However, although a lower than expected primary recovery is achieved on average (approximately 50% for 120% ring tonnage extracted), much of the remaining ore is recovered by draw from levels below. Even with this low primary recovery percentage, Ridgeway's closely spaced drawpoints, favorable ore-body geometry, and tightly managed draw control enable it to recover a high proportion of the ore fired, with relatively low levels of dilution (Power and Just 2008).

This indicates the importance of viewing the sublevel caving system as a multiple-level system, rather than making design decisions based purely on recovery from the primary level alone. Figure 13.9-13 illustrates how ring recovery is spread over many levels at Ridgeway. Figure 13.9-14 indicates that an equation can be fitted to this data with a good fit.

It is this general sense of order within an apparently chaotic system that has enabled the development of a number of forecasting tools that can be used with reasonable accuracy in operational mines for estimation of recovery, development of draw control strategies, and optimization of recovery through the targeting of residual grade.

Back Break

Back break is a phenomenon that is difficult to simulate effectively in physical models but can be seen in many marker trials and generally in sublevel caving production drifts. It occurs when a ring blast breaks rock away from the next ring to be fired in the crosscut. This back break often occurs at the brow but can also be present along the full height of the production ring, depending on ground conditions. This was noted in marker trials carried out by Janelid at the Grängesberg mine (Janelid 1972) and has also been seen at Ridgeway (Power 2004a) and Perseverance (Hollins and Tucker 2004). Back break is still ore recovered on the primary level and thus can be included, from a grade forecasting perspective, with primary recovery. Its effect on blasting and flow, however, can be significant. If a blast



Note: Figures indicate recovery from three sections through a marker ring (left to right: 0.65 m, 1.3 m, and 1.95 m forward of the blast).

Figure 13.9-11 Example of shallow primary draw

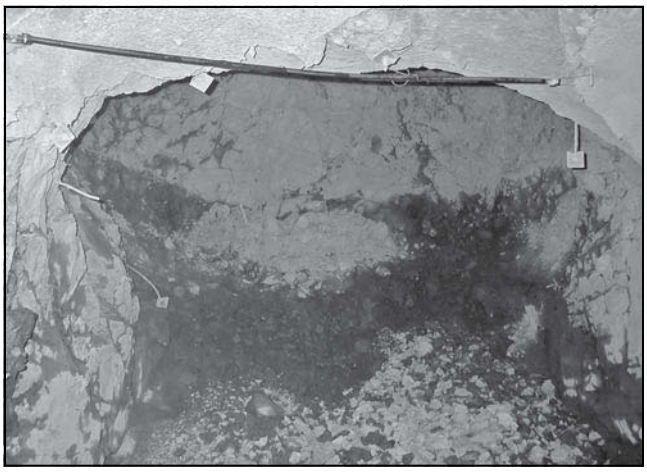


Figure 13.9-12 Intact rock-mass structure visible at the back of a ring in which shallow draw has occurred

produces high amounts of back break, rock flows through this broken area, effectively changing the brow position of the blasted ring and reducing the likelihood of drawing to the full ring depth (Figure 13.9-15).

Figure 13.9-16 shows an example of a ring that has suffered significant back break, indicating that back break from the previous ring has broken almost to the full burden of the next ring to be fired. It is probable that this back break influenced the performance of the blast ring and possibly recovery. In this example, back break extends to the base of the ring; however, primary flow still does not reach the central marker plane for the lower half of the ring. This indicates that, even when ore is likely to have been well fragmented due to a high relative powder factor, shallow flow can still occur.

Parts of a ring that have been damaged by the previous blast can be significantly thinner than the rest of the ring. They will be blasted with a greater effective power factor, causing the blast to behave differently than intended. Generally, the zone of greatest explosive concentration produces most back break. Because this is also the area of greatest explosive concentration in the next ring, this accentuates changes in the actual blasting powder factor distribution for this ring.

Chaotic and Variable Flow

Figure 13.9-16 shows how back break in one ring can affect blasting and flow performance of the next ring to be blasted. Because the blasting and flow behavior of the vast majority of sublevel caving rings cannot be practically measured, it is not known what effects are being generated to other rings. If it is accepted that performance in one ring will affect performance of rings around it, and that these events cannot be predicted, it follows that sublevel caving flow should be treated as a chaotic system. Some of the parameters producing significant variability in sublevel caving systems include

- Rock mass properties,
- Blast design,
- Ability of charging crew to implement blast design (blocked holes, etc.),
- Hole deviation,
- Blasting performance of rings around the current ring,
- Hang-up occurrence impacting draw, and
- Grade variations impacting draw.

Although it is impossible to predict the flow outcome from a particular ring, the research program carried out at Ridgeway identified five common primary flow behaviors by which most of the trial results could be classified. These are shown in Figures 13.9-17 and 13.9-18. Although general primary recovery zones are shaded in these figures, it should be assumed that the rippling flow effects (discussed later) would result in portions of these drawzones being left for recovery on subsequent levels or not being recovered at all.

Figure 13.9-19 shows the relative frequency of these draw behaviors and that the standard and standard/shallow draw classes produce the highest ring recoveries (calculated after the secondary level). As depth of draw is reduced, width of draw increases. If apex mobility is low, width of draw is also likely to be low, and recovery is also likely to decrease.

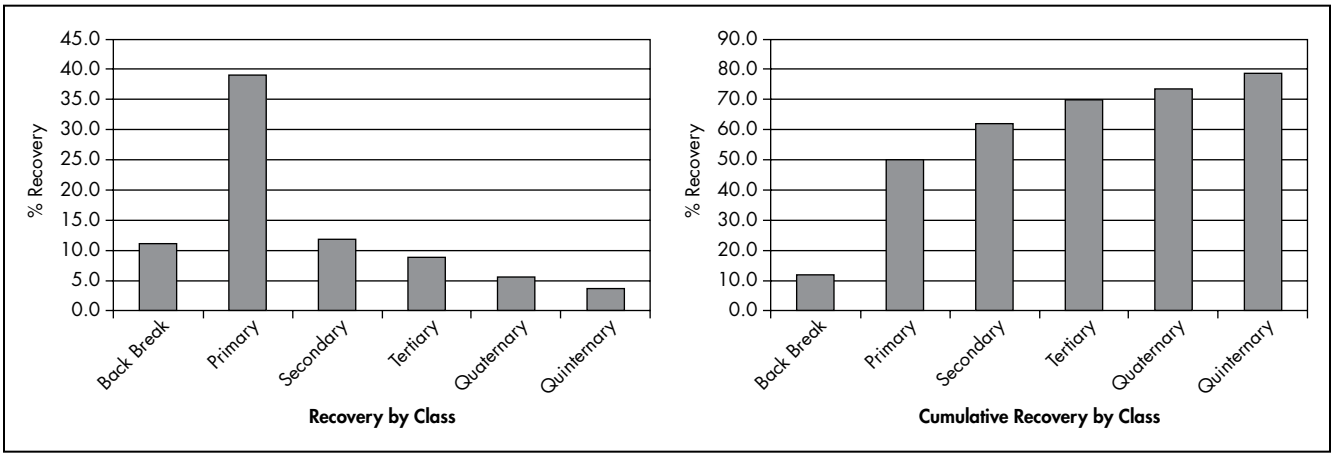


Figure 13.9-13 Recovery by class at Ridgeway

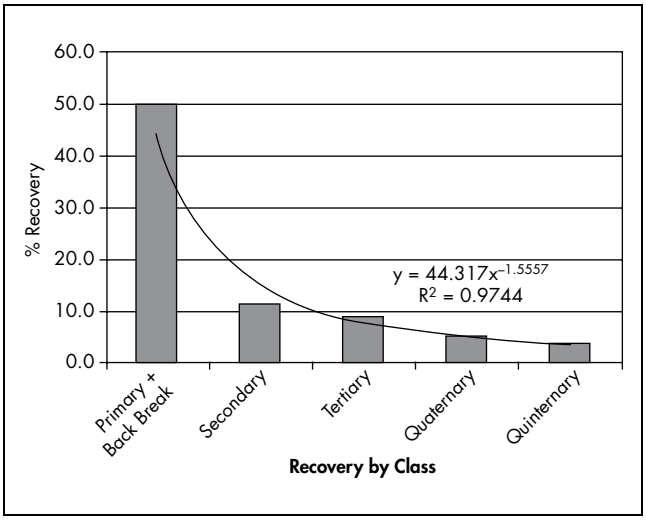


Figure 13.9-14 Recovery formula

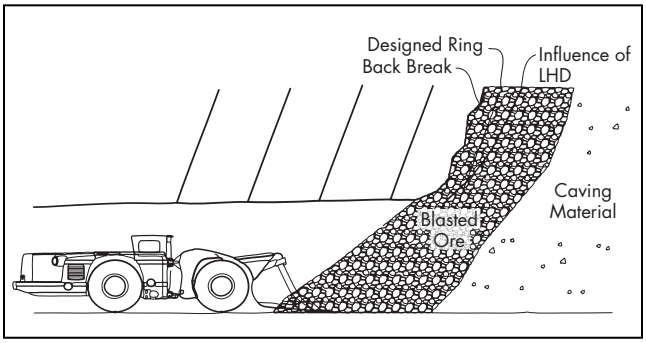


Figure 13.9-15 Impact of back break on ore mucking

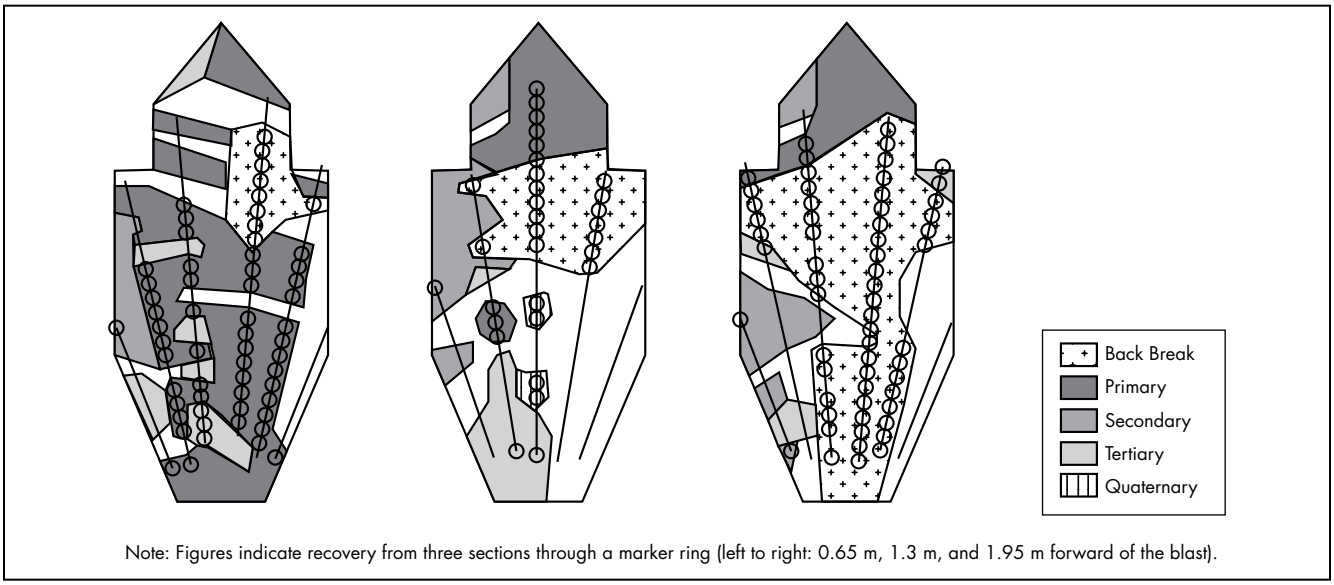
Although not discussed in detail here, coupling full-scale flow assessment with analysis of the drill and blast factors producing these flow types is also important. Marker trials and other drill-and-blast research programs can be used to develop a greater understanding of drill-and-blast, and its effect on flow can be used to optimize operational drill-and-blast design. This research is also used as the basis for layout, blasting, and draw control design for new sublevel caving mines.

Many of the figures included here show examples of ripple patterns in ore recovery. These patterns produce regions of recovery interspersed with sections of nonrecovery, resembling ripple patterns produced in sand due to water flowing normal to the ripples. Researchers suggest a process whereby rock flowing toward the drawpoint encounters zones of lower mobility and continues around these on its original course. These ripple patterns indicate that, although a flow zone may progress through a certain region in the ring, parts of this region are often left unrecovered. Ripple patterns are often at slightly different orientations for different draw classes, suggesting that flow has progressed at different orientations for different recovery classes. The relatively narrow draw pipes progress subvertically from individual drawpoints to heights of at least 55 m, as illustrated in Figure 13.9-20, which shows the results of a secondary draw trial.

In this trial, no primary draw was taken in the rings on the upper level, allowing secondary draw attributable to the rings below to be studied. The primary recovery zone in the lower ring is produced by the same flow event as the secondary recovery zone on the level above. Flow from the ring below appears to have progressed into the center of the ring above (left). The secondary flow zone in the upper ring seems to have been preferentially broken in the blast, possibly due to more concentrated explosive density in the center of the ring. It is likely that low mobility in the apex of the ring below has caused this flow outcome (also resulting in very little secondary recovery from the upper ring on the right).

Episodic flow patterns from individual rings were also identified in the Perseverance marker trials (Hollins and Tucker 2004). Similarly, field experiments conducted at the Kiruna mine showed pulsations of waste entering the drawpoint, with increasing levels of waste appearing with tonnage drawn (Figure 13.9-21). These pulsations were believed to be related to differential mobility in the blasted ore from different parts of the ring and waste rock.

The results of the early model-based research into granular flow in sublevel caving led to advances in design and draw control practice and to a general understanding of granular flow. In many cases, the data recovered from marker trials now supplements or has replaced information previously gained from physical models. Such trials allow testing in theoretical thinking on sublevel caving granular flow in full-scale operational environments but must be designed and analyzed as part



Note: Figures indicate recovery from three sections through a marker ring (left to right: 0.65 m, 1.3 m, and 1.95 m forward of the blast).

Figure 13.9-16 Example of significant back break

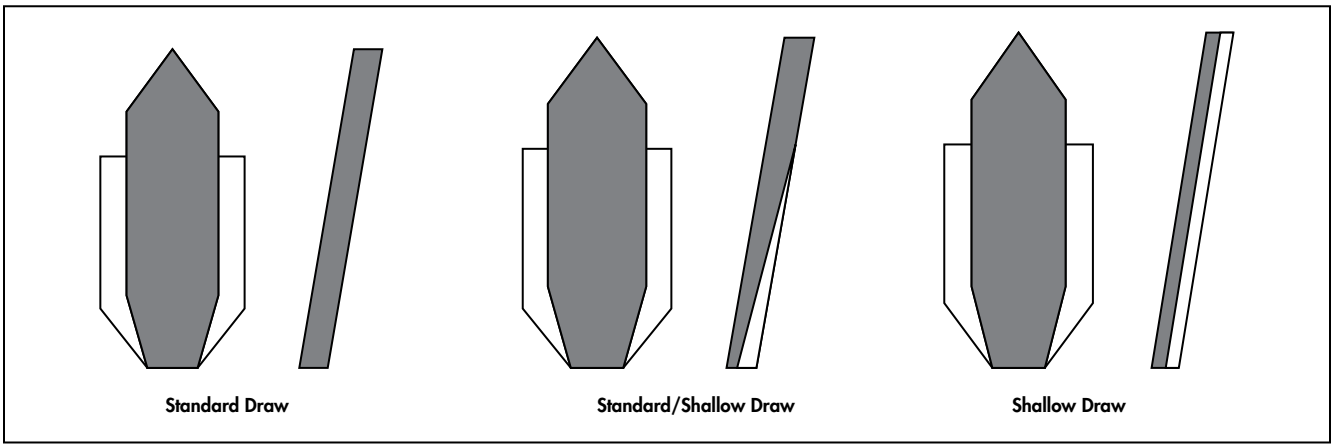


Figure 13.9-17 Primary flow behaviors at Ridgeway

of a systematic scientific program to be completely effective. They often produce results that lead researchers in totally new directions of thought.

Other results through full-scale research of the more recent understanding of flow (particularly in its multilevel nature) are improvements in computer-based forecasting and optimization capabilities. Several different methods of numerical flow modeling specifically designed for sublevel caving environments are now available. Some of these methods not only accurately forecast monthly production grade but also the targeting of high-grade zones and regions of the ore body where extraction has been low. In some cases, they allow shutoff simulation to produce detailed drawpoint-by-drawpoint production targets in situations where visual or weight-based draw control systems are ineffective.

With the continued development of new technology targeted at sublevel caving (e.g., smart markers and improved computing capacity for research into drill-and-blast effects and numerical modeling of granular flow), it is likely that sublevel caving will continue to be a productive and economically viable mass mining method.

MINE DESIGN

Layout selection is a critical success factor in all aspects of a sublevel caving operation. The layout directly relates to the ore recovery, drill-and-blast effectiveness, excavation stability, and the cost of mine development. Because it is a fundamental success factor, considerable attention needs to be given to selecting the layout prior to commencing mine establishment and determining the critical success factors for the mine. A mine focused on low-cost production will implement a different layout than a mine focused on maximizing recovery. Mine layouts will probably change over the operating life of the mine, so the original layout is often modified as a result of lessons learned under operational conditions.

Layout Orientation

Layout orientations are generally classed as either transverse or longitudinal. The transverse layout is generally applied in wider ore bodies, greater than approximately 50 m, where the parallel production drifts are driven across the ore body, according to Figure 13.9-22A. For widths below this, the production drifts are generally driven along the strike of the ore

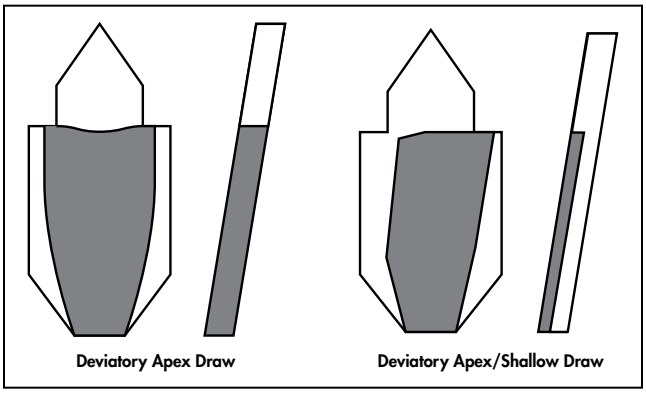


Figure 13.9-18 Primary flow behaviors at Ridgeway

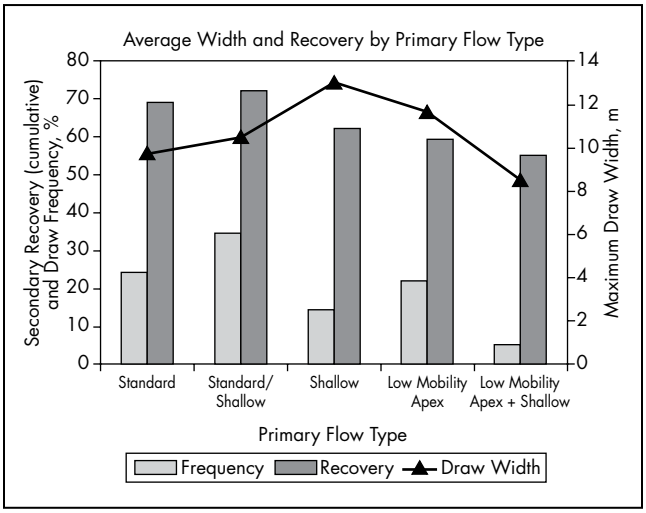


Figure 13.9-19 Frequency of primary draw behaviors and their effect on width of draw and recovery

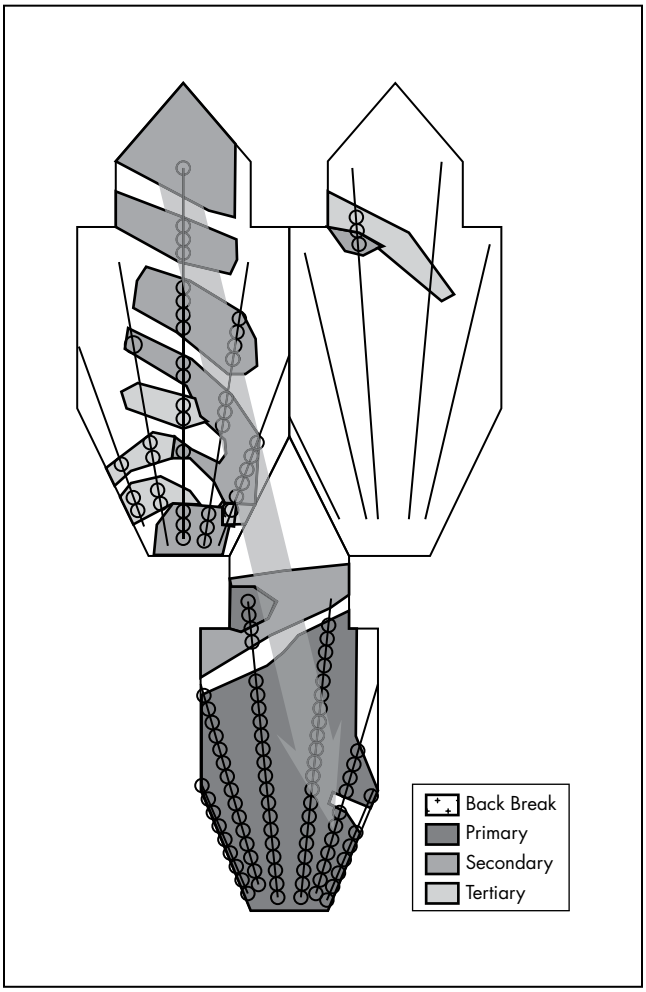


Figure 13.9-20 Example of height of draw and deviation due to low mobility in the apex, from secondary recovery trial

body in a longitudinal fashion, according to Figure 13.9-22B. The longitudinal layout may contain as few as one drift on a level depending on ore-body width. The choice of orientation is predominantly determined by the width of the ore body but can also be influenced by geotechnical and economic factors.

The transverse layout is generally more productive than the longitudinal because of the number of active production faces that can be made available. Some mines will operate with a mixture of layouts due to a variable ore-body geometry, although transition between the two styles is difficult in a vertical sense because of the criss-cross nature of the layouts. However, if transition is required between the layouts, it may be prudent to skip a level and recommence sublevel caving operations with the new layout and allow the skipped level to cave in rather than trying to perform complex production ring drilling and blasting. This technique was successfully employed at the Perseverance mine (Wood et al. 2000).

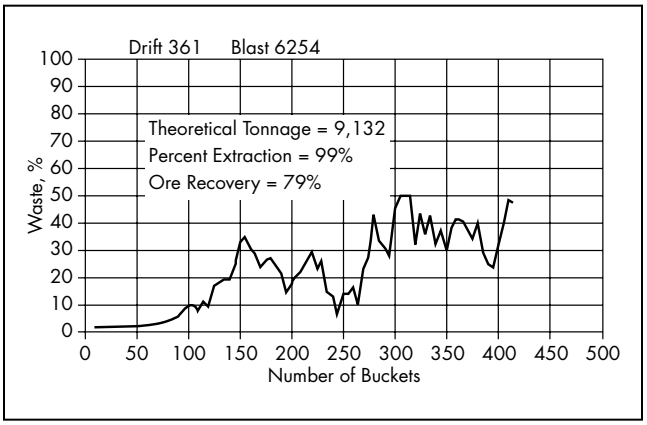
For either layout type, the supporting development is similar with a perimeter drift containing either orepass tipping points or truck-loading stockpiles. The perimeter drifts are also used to provide access to the primary ventilation circuit and enable the installation of secondary fans to allow airflow to be delivered to the production working areas on the level.

Layout Dimensions

Sublevel Height

The sublevel height is generally measured as the vertical floor-to-floor distance between the levels. It is determined by the dip of the ore body and the physical limitations of the production drill-and-blast equipment. If the ore body is vertical, then dip is not a constraint on sublevel spacing. However, as the ore body flattens, the sublevel height needs to be reduced to avoid incurring excessive amounts of hanging-wall dilution, as well as minimizing the amount of unrecoverable ore on the footwall.

Over the years, advances in drill-and-blast equipment have seen the sublevel height increase from less than 12 m to more than 30 m in some cases. The key criterion for drill-and-blast is to ensure that satisfactory fragmentation is achieved through the entire ring. To achieve this, the drill holes must be evenly spaced, straight, and adequately charged with the appropriate type and strength of explosive. The proposed drill pattern is an integral part of selecting the overall sublevel layout and impacts not only the sublevel height but also the drift dimensions and the drift and pillar spacing.



Source: Hustrulid 2000.

Figure 13.9-21 Pulsations of waste entering a ring at the Kiruna mine

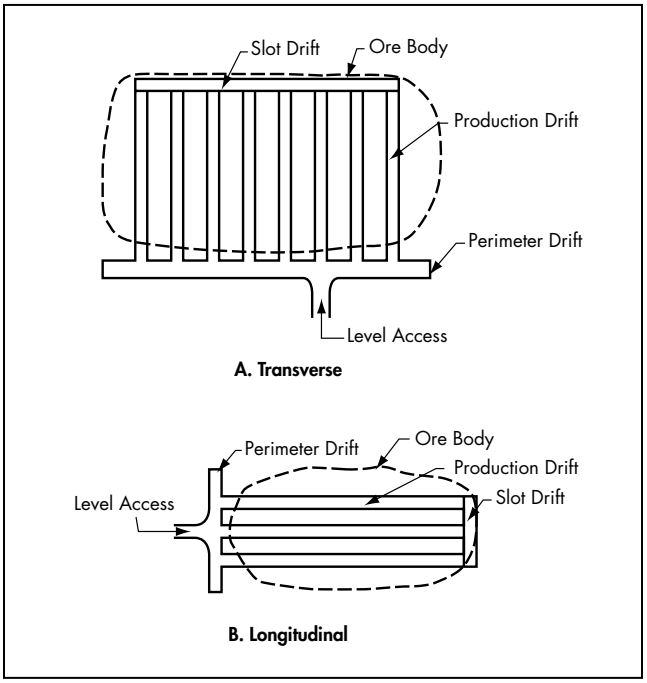


Figure 13.9-22 Transverse and longitudinal layouts

Production Drift Spacing

Important factors in selection of production drift spacing include

- Potential ring shape produced by the combination of sublevel height and production drift spacing (will it produce a ring shape that can be efficiently blasted?);
- Granular flow properties inherent to the mine (based on drill-and-blast procedures and rock-mass characteristics) and the likely recoveries to be achieved;
- Value of the mineral being mined and whether a cost-reduction or recovery-emphasis strategy will be followed;
- Geotechnical factors; and
- Production drilling and charging factors.

For maximum ore recovery, the entire level would theoretically be excavated, similar to a coal longwall operation, but this is impractical. Therefore, a trade-off is required between the amount of ore potentially recovered on its original level and the amount left in the cave for recovery, hopefully, on lower levels (with more associated risk to its ultimate recovery).

Thus, the spacing of the production drifts is strongly influenced by the operating philosophy of the mine. The more cost-focused a mine is, the wider the spacing between the production drifts, because this reduces the amount of expensive development required (Power and Just 2008). Figure 13.9-23 illustrates this for two different layouts, showing theoretical primary recovery zones only. Recovery-focused mines will try to place the production drifts as close together as practical in an attempt to maximize the amount of ore recovered on the primary level to minimize the effects of dilution, as shown on the left side of Figure 13.9-23.

An indicative estimate for the ellipsoid of draw width can be generated by either using a ratio of 1.4:1.7 of the drift width or the drift width plus 1.5 m on each side, according to the following formula:

$$W_{DE} = W_D + (1.5 \times 2)$$

where

W_{DE} = ellipsoid of draw width, m
W_D = drift width, m

Previous estimates for this W_{DE}/W_D ratio for the much smaller sublevel caving layouts also predicted this same ratio range of 1.4:1.7 (Hustrulid and Kvapil 2008).

Because sublevel caving relies on the extraction of ore from not only the level on which it is fired but subsequent lower levels, it is important to overlay the extraction envelopes over all levels to gain an understanding of likely system recovery rather than just recovery on a single production level. In extreme cases, significant overlapping of theoretical draw ellipses may result in excessive extraction of these areas, which can lead to early dilution entry. Conversely, too much of a gap between the extraction envelopes could lead to consolidation of material on the sides of the rings, which may cause poor firing conditions, due to the limited void available. Stress concentration areas may also result, which could lead to damage of the production drifts.

Production Drift Shape

The shape of the production drift impacts the mine's ore recovery and productivity. Ideally, the production drifts are as wide as possible to allow the maximum flow of ore. To further assist with flow, the drift roof should be as flat as practical while still providing adequate support to the roof and brow. A wide and flat roof also assists in improved spacing of the production drill holes, which limits the impacts of blasting on the brow, as well as reducing the amount of back break. If the drift roof is overly arched, the flow can be preferentially funneled to the center of the drift, leading to early dilution. In some cases, a trade-off will be necessary (i.e., an arch may be required to produce a wider stable drawpoint).

Greater drift widths also improve the productivity of the LHD machines. An LHD can tram faster in a drift where it has plenty of clearance. At the muck pile, the width is also an advantage for the LHD, because it is able to load from side to side, ensuring a more even flow of material into the production drift, as illustrated in Figure 13.9-24. In addition, the loader is able to extract material from deeper into the drawpoint,

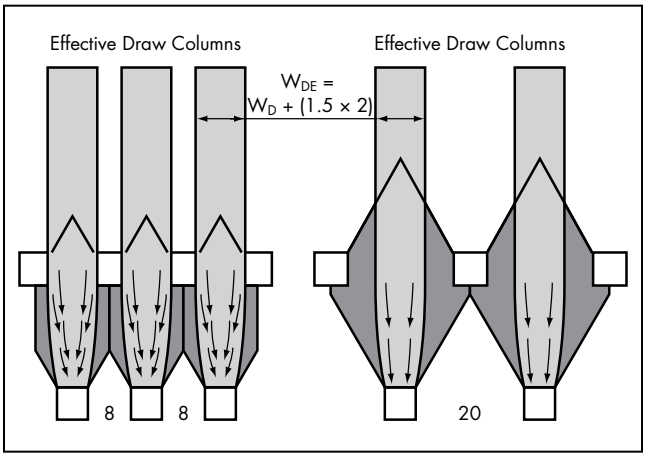


Figure 13.9-23 Effect of drift spacing on primary ore recovery

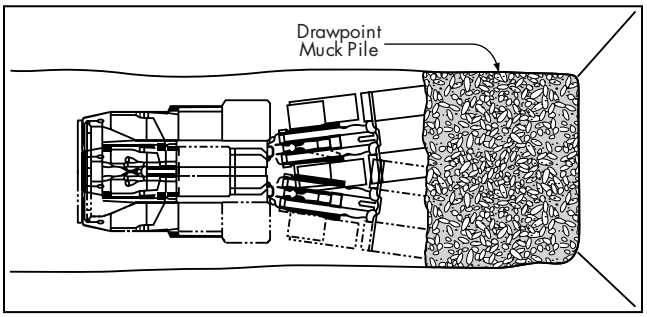


Figure 13.9-24. LHD side-to-side loading in a drawpoint

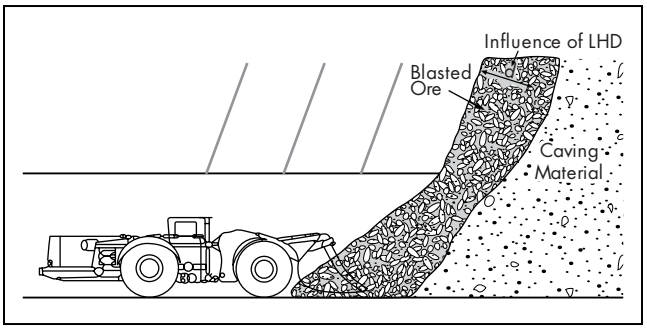


Figure 13.9-25. Influence of LHD on cave material

because it is not extracting the material from the center of the drift, which has rilled the farthest out from the brow, as illustrated in Figure 13.9-25. This further improves ore recovery and influences the burden of the production rings, because the cave material loosens in front of the next ring to be fired.

Minimizing the height of the drift roof restricts the rock's ability to rill too far from the brow. As previously stated, if the depth of material excavation by the LHD can be increased, then the overall ore recovery will be increased. The drift height, however, is usually a trade-off between length of rill and size of machinery required to achieve the desired production rate. Although it is possible to obtain custom-made machinery to fit the optimal drive height, this strategy can have drawbacks if the machinery fails to meet production targets, as machinery replacements are generally not immediately available. In the

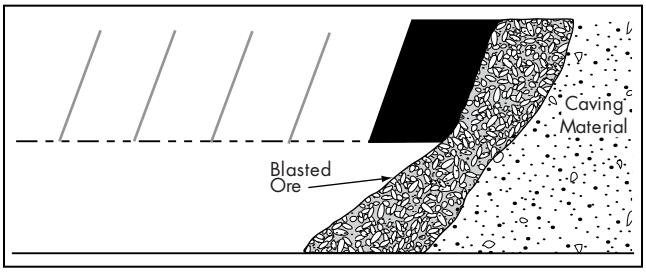


Figure 13.9-26 Unique sublevel cave-blasting conditions

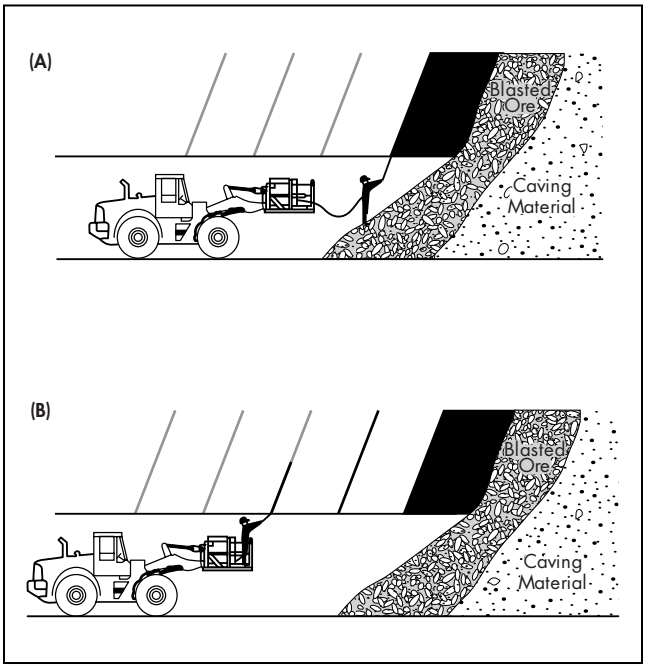


Figure 13.9-27 Charging option set-ups

authors' experience, it is generally the production drill that limits the drift height, and, as the large-scale operations push for longer and longer production holes, this is likely to remain the case. Low-profile LHDs are generally available to achieve the desired production rates, but the desire to drill with longer and longer drill rods for greater hole accuracy is forcing average drift heights higher.

Production Drill-and-Blast

The drill-and-blast requirements of a sublevel caving mine are unique because the blasting environment is always choked. This means that every production ring is blasted against the combination of previously blasted ore and waste sitting in and around the production drift (Figure 13.9-26).

Because of the environment in which the blasting occurs (i.e., the only free face is the roof of the production drift), the blast's function is different from nearly every other production drilling situation. With sublevel caving blasting, the intent is to initially compact the caved material in front of the blast to create a void area for the fired ring to access. After the fired ring is in the void area, the intent is to fragment the ore slice as much as possible so that it will preferentially flow when mucking commences. Although this is the intent, the reality is somewhat different. The nature of the fanned ring pattern used

in sublevel caving and the reducing ratio of development-to-production work against the intended blasting criteria. With the reduction in this ratio, the only true swell available to the blasted rock is reduced and in some cases is <10%, while the blasted rock itself still wants to swell to >30% of its in-situ volume. This lack of initial swell results in the holes fired first, and those sections of the hole close to the drift have the opportunity to swell and move into the drift as desired, while the ends of the holes have limited opportunities to swell and move. In addition, the fan nature of the drill pattern ensures that explosives are concentrated in the center of the ring relative to the outer holes. This creates smaller, more uniform fragmentation in the center, which is thus more mobile than the material at the edges. These fragmentation and flow characteristics affect the gravity flow and ore recovery performance for the mine (Trout 2002).

With the diamond layout style of sublevel caving production drifts, a wide variety of production drilling patterns can be used. With the wider-spaced production drifts, a pattern involving larger and longer holes will generally be used, although, as the production drifts are brought closer together, the drill holes will interact. When this occurs, the production rings often need to be offset by approximately 1 m to reduce the chances of sympathetic detonation between the holes or damage to the neighboring ring when the rings are fired. Steeper side holes will lead to better funneling of ore to the drawpoint and smaller *dead zones* at the sides of the ring but also to higher apexes at the top of the ring (which can be more difficult to blast effectively).

The production drill rings are generally inclined forward by 10° to 20°. This inclination assists in feeding the ore preferentially into the drift by providing a small overhang to the diluting material directly above. The inclination also ensures that the holes are all drilled on a similar plane for improved blasting results and additional protection of the ring brow. Because prevailing structures in the rock mass can sometimes adversely affect rings inclined forward at similar angles, the amount of forward inclination may need to be adjusted in different parts of the mine in order to achieve optimal results.

The sublevel caving production rings can be charged with either ammonium nitrate fuel oil (ANFO) or emulsion products. Prior to any charging activities, the production holes should be *dipped* by the charging crew to ensure that the holes are clear of obstructions and are drilled to the correct depths. Often, charging personnel must stand on the muck pile to charge the ring, as the charging unit frequently cannot get the operators to the production ring because of the rill of broken rock in the drawpoint and the height of the drive (Figure 13.9-27A). This can lead to significant risk of injury and inferior drill-and-blast results. Safety hazards include the muck pile rilling unexpectedly or trip hazards resulting from the uneven nature of the muck pile on which the operators work.

To provide a safer charging environment, the preloading or precharging of multiple rings can be implemented (Figure 13.9-27B), although this is subject to the local regulators and may take time to get approval. *Precharging* relates to fully loading and priming the production rings, whereas *preloading* refers to the practice of only placing the explosive in the hole but not the detonator, which is added immediately prior to blasting. When utilizing precharging, the tails of the detonators are tucked back up the hole so they do not impede the LHD when it is mucking the rings in front; the tails are only pulled out when the ring is ready to be hooked up for blasting.

The implementation of precharging/loading in recent years at a number of large-scale operations has had significant benefits for those operations, including improved safety of personnel around the drawpoint brows, a decoupling of the charging and mucking activities, the removal of production disruptions caused by hole blockage and dislocation, the elimination or significant reduction in the number of production ring redrills, and improvements in draw control (as less mucking is needed to prepare the ring for firing).

Drawbacks to precharging/loading include the need to have a number of production rings predrilled in order to allow the charging and drilling processes to be independent. From a safety perspective, when a precharged/loaded ring is behind a misfire, the scope is limited to get a drill rig back to assist with the misfire unless the drill rig has a remote operation function or the explosives are washed from the precharged/loaded ring. For this reason, precharging is often easier to implement in mines with better geotechnical conditions, where charged holes are less likely to be dislocated or brows lost. If the ore body contains aquifers or seeping water, precharging/loading may be also unsuitable when using ANFO, because the explosive column can be desensitized or washed out by the time the ring is ready to fire (emulsion should overcome this issue).

Because of the choked blasting conditions of sublevel caving rings, a single production ring per drift is generally only fired at any one time. Firing of multiple rings may be undertaken in areas of very poor ground conditions, but this introduces the risk of misfires and leaves pillars within the cave, which can cause problems on the level below. It also leads to lower overall recovery, because most of the ore in the ring farther from the brow cannot flow to the drawpoint and can be occluded by waste flowing from above. Depending on infrastructure constraints and required production rate, some mines fire rings in multiple production drifts at the same time to ensure broken stocks are sufficient for the LHD fleets.

Hang-ups are a normal feature of the sublevel caving mining method, with up to 50% of rings hanging up at some stage of the mucking cycle. In mines with more closely spaced drawpoints, the hang-up can sometimes be cleared by removing additional material from the production drifts on either side of the hung-up drift. This material removal appears to disturb rock in the affected drawpoint by acting to release pressure on either side of the hung-up ring. Other hang-up retrieval methods include water cannons and (as a last resort) explosives, because they often damage the drawpoint brow. If a hang-up cannot be cleared within a specified period, the next ring in the production drift must be fired and the mining process continued.

Although production drill-and-blast design is an input into the overall layout of the sublevel caving mine, it can be readily changed as the characteristics of the ore body become more fully understood. The shape of the production ring may not vary significantly, but the drill-and-blast parameters may. Design parameters, including hole diameter, number of holes in the ring, drill-hole toe spacing, production ring burden and dump angle, and explosive column densities and lengths, can all be modified with relative ease. When adjusting the production drilling parameters, it is important to take a holistic view of the sublevel caving operation in order to ensure that the changes made will not dramatically and negatively affect other parts of the process.

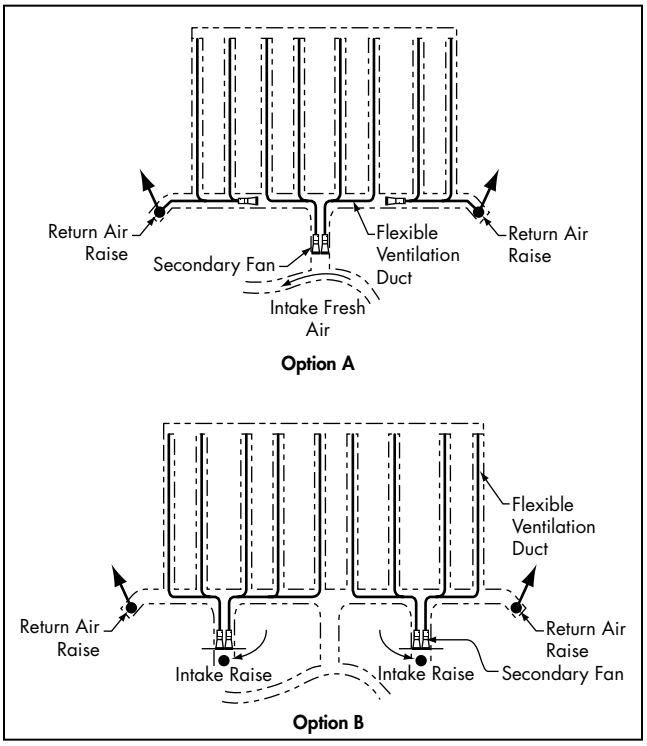


Figure 13.9-28 Variable ventilation layouts for sublevel caving operations

Services

The active ore production areas are all dead-end headings. A single fan may be used to supply up to four production drifts, as the ducting can be tied off in the headings that are not in use. If multiple secondary fans are required on the level, they may need to be set up in a cascading style (Option A, Figure 13.9-28). This means that a fan will be placed downflow of another fan and, in essence, will receive the used air from the upflow fan. This cascade style of secondary ventilation can lead to lost production time if there is a significant amount of dust in one of the drawpoints upflow of the secondary fan, because all the dust will be blown into the headings serviced by the downflow fan. Each level is set up as its own primary circuit so the levels do not interact with each other. Primary intake and return raises with fixed ventilation infrastructure are used to direct the appropriate amount of air throughout the mine. Figure 13.9-28 shows two styles of level ventilation: Option A is for mines using the main access decline as their fresh air source, whereas Option B is for mines that heat or refrigerate their intake air.

The cave itself may provide a path for airflow after it has broken through to the surface, though this will be determined by the porosity of the cave column and the pressure differences between the surface and the cave workings. Generally, movement of air through the caved rock will be in the form of leakage rather than a volume that can support mining activities. Individual mining circumstances will ultimately determine how much air passes through the cave zone area.

Water and compressed air are reticulated to each level to support the mining activities. After development of the level is completed, the compressed air line may be recovered unless required by the production drill. The water line is used for production drilling and to feed the drawpoint sprays mounted on the drift roof to keep the dust levels down during mucking. Drainage on the sublevels is important in ensuring that maximum production rates are achieved. The production drifts are usually driven on a slight incline of between 1 in 50 (2%) and 1 in 40 (2.5%) to remove the pooling of water along the drift or at the face. In addition to the introduced water for the mining process, water is also associated with aquifers that have been breached during mining. Sumps are generally located in the perimeter drift with a drain hole drilled to the level below. The production water can then be collected at a mine dam located lower in the mine and pumped out via the mine dewatering pump system.

Communications systems are run onto the production levels for radio communications, as well as to control any automated or semiautomated mining equipment, including production drill rigs and LHDs in some of the larger-scale sublevel caving operations. Such systems may also be utilized to transmit real-time production data to a central production management area. The larger-scale sublevel operations using automation or the tele-remote operation of equipment are doing so in an attempt to remove workers from hazards associated with drawpoint brows and in the general underground environment. The LKAB mines in Sweden are leading the way in this field. They use the automation of machinery to provide production increases by allowing the machines to work extra hours while the operators are located on the surface or an area of the mine not impacted by the mining process, especially the blasting and clearance cycle.

Geomechanical Factors

With sublevel caving, a number of geomechanical factors must be considered when either laying out the mine or making large-scale operational decisions. These include

- Quality of the ore body and hosting rock mass,
- Orientation and magnitude of the major and minor stress parameters, and
- Presence of large-scale and microstructures.

Prior to establishing a sublevel caving mine, an assessment of the likelihood of natural caving is required. The assessment is a specialized geotechnical task and is beyond the scope of this chapter. However, inputs for these techniques rely on a number of key geotechnical parameters, so it is imperative when diamond drilling into a deposit that may be applicable for caving that all of the relevant information is recorded. If stability assessments indicate that the ore body will not cave naturally, then sublevel caving may not be suitable, unless reliable techniques for cave preconditioning and/or caving assistance are available.

The orientation of the stress field will often determine the retreat direction of the sublevel caving production front. If there are no other influencing factors, it is preferable to retreat perpendicular to the major principal stress direction. By retreating in this manner, the effects of the stress field can be "shadowed out" after production has been initiated on a sublevel. This creates a distressed environment for the majority of production headings in the mine, because the stresses will be forced to wrap around, under, and over the production areas.

Stress that builds up around the periphery of the working areas is called abutment stresses, which can result in rock movement and bursting, depending on the rock strength-to-stress ratio. The ground support requirements are therefore greater in the perimeter drives than in the production drifts.

The bullnose section of the pillars at the turnout points of the production drifts from the perimeter drive are especially susceptible to stress damage, as shown in Figure 13.9-29, because they are reduced in size with the retreating of the cave front.

Production drive stability can also be impacted by the abutment stresses as production on the level above passes over the lower-level production drifts. The preferred ground support system in a sublevel caving operation is one that can yield slightly without failing when the abutment loads come on but is still sufficient for working at the brow in the distressed conditions. Some ground support combination examples include grouted rebar with 50 to 75 mm of fiber-reinforced concrete or split sets with wire mesh.

Identifying mine scale and minor structures is important when laying out the mine, because they can be used to assist in the propagation of the cave or in controlling the effects of the abutment stresses. Should the major structures not be favorable to the cave's progression, then considerable care needs to be taken to monitor caving progress, because it may be necessary to reduce or stop production in certain cave areas to reduce the impacts of these structures. The orientation of the minor structures is also important, because it can influence the inclination of the production rings. Such inclination on a plane parallel to the minor structures can reduce the number of production hole dislocations and therefore improve productivity.

Water-bearing aquifers or surface water courses—that is, streams and dams in the caving area—also need to be identified prior to the commencement of caving activities to ensure that there are no unexpected water or mud inrushes during the operation of the mine. Where an aquifer or water course cannot be dewatered prior to caving, extreme care should be taken and evacuation procedures implemented as the cave encroaches on the water-bearing area. Generally, as the cave propagates, it will ensure that a large number of fractures are created in the cap rock, which will breach the aquifer and potentially drain it before the main cave back moves through the water-bearing zone. However, in extreme cases, uncontrolled mud flows can be generated by excessive and poorly controlled draw in combination with high inflows of water (either from aquifers or channeling of precipitation through subsidence zones) and suitable clay mineralogy. When this is a risk, use of a robust inrush hazard management plan with appropriate triggers and controls is beneficial. Prior to commencing a recent sublevel caving operation at the Kiruna mine, engineers were forced to drain a small lake, while engineers at the Ridgeway mine identified an aquifer system below the surface and monitored it as the cave progressed, noting only minimal water reporting to the production drawpoints, as the majority of the water was absorbed in saturating the cave column.

PRODUCTION DRAW CONTROL

Draw strategy and production management are extremely important in sublevel caving operations to enable the accurate prediction of ore and waste flow from the cave after production commences. The draw strategies and production management plans used at each mine need to be variable and flexible in order to cope with unforeseen circumstances such as the early inflow of waste, hang-ups, geotechnical instability, and arrival of large oversize material at drawpoints.

Draw Strategy

The objective of a draw strategy is to maximize the overall ore recovery while minimizing the impacts of dilution. To achieve this, an effective sublevel caving mine relies heavily on a draw control system that is variable and adaptable to the constantly changing operational situations, including unscheduled equipment breakdowns and production incidents. Effective draw control practices also rely on an understanding of the cave's flow characteristics and the grade distribution in and above the ore body being mined.

Any adopted draw control system needs to include guidelines to allow the frontline management to make tactical decisions. Such guidelines will be grouped according to authority levels—ranging from the LHD operator to the general manager of the mine. In mines with a distinct visual or physically distinguishable difference between the ore and waste rock, guidelines will dictate the percentage of ore-to-waste in the drawpoint when production must stop for a particular drawpoint. To allow decisions to be made, mines without this distinction must rely solely on the tonnage drawn from the drawpoint followed up with a sampling system.

Regular monitoring of the drawpoint status is imperative to achieve overall efficient draw strategy and to ensure that underperforming or troublesome drawpoints are identified early so their potential impact can be minimized. Draw strategies can be classed as either *independent* or *interactive* and are determined by the mine layout and overall mine objective. These terms refer to drawpoints where tactical extraction targets are linked to those of surrounding drawpoints (interactive) and those where tactical production targets can be generated without reference to surrounding drawpoints (independent). Theories discussed previously concerning the widening of draw columns due to interaction through interactive draw do not have bearing on these draw strategies.

Independent draw is often used by operations with a limited number of production drifts available for mucking. Under these circumstances, each individual drawpoint is often depleted before moving onto the next one. The penalty for this type of draw strategy is a greater risk of dilution piping in from above and impacting adjacent production drifts. Interactive draw style operations rotate the LHD between a number of neighboring drawpoints in a fixed sequence, which is repeated until all of the drawpoints have been completed and the next set of rings fired. The interactive approach enables the blasted and caved material to be extracted over a larger lateral area, thus reducing the opportunity for dilution piping. Depending on the ore flow characteristics of the mine and the mine layout selected, either approach may maximize the overall mine performance.

As previously identified, drawpoints can be operated individually or in groups. In mines with substantial numbers of drifts on each level, the practice of grouping a number of drifts together and treating them as a single entity can substantially reduce the complexities of managing the operation on a short-term basis. The orientation and shape of the production front will be influenced by the stress field and major structures. No standard exists for the shape of the cave front, because all mines will have varying conditions and operating requirements. The three most widely used cave-front layouts are the flat, en echelon, and stepped, as shown in Figure 13.9-30.

When selecting a cave front, thorough consideration needs to be given to the lead–lag rules that will be used to operate the sublevel caving. The term *lead–lag* refers to the maximum and minimum spacing between either individual or groups of drawpoints being retreated across the levels. These rules are 3-D, because they also include the level above and the level below



Figure 13.9-29 Example of perimeter drive damage in sublevel caving operation

the specific level. When defining the lead–lag rules, the main influencing factors are stress field constraints, which relate to the minimum distance required between production fronts on adjoining levels to preclude stress damage. Damage can also occur when drawpoints retreat past their neighbors on the same level. The impacts of stress can be mitigated by increasing the amount of ground support in the sections of the drift, though additional support will be limited by the ability to cost-effectively change the lead–lag distances. Another operational limit to lead–lag distances is the interaction of drilling and charging activities within adjoining drifts. Most operations will have an exclusion limit around a charged hole, which must be considered when establishing lead–lag rules.

Compaction of the cave column is also another potential factor. If a level retreats too close to the one above, stability of the cave in front of the upper level may be compromised, and the ore blasted on that level may be more difficult to recover. Conversely, if the distance between the vertical levels is too great, it may take too much time for the lower level to pass under the upper level, and the cave may recompact or reconsolidate, depending on its characteristics, causing considerable production and ore losses.

Production Management

A number of factors impact the theoretical production rate of a sublevel caving operation, including

- Ore-body geometry;
- Cave fragmentation of both ore and waste;
- Mine layout and excavation size;
- Equipment employed, fleet size, and number; and
- Materials-handling capacity

As the sublevel caving method is now often used in more competent ore bodies, opportunities for larger, stable excavations exist. This is transforming production rates for sublevel caving operations as larger equipment is used. In most sublevel caving mines, the majority of activity occurs within the confines of the level with little activity in the access declines. This makes the method efficient, with some operations achieving >35,000 t/operator per annum, as relatively few operators are required underground once the mine is operational.

Overall, the aim of sublevel caving mining is to become as factory-like as possible. This approach is achieved by taking a holistic view of the mining method from the face through to the surface in the understanding that overall mine productivity is governed by the lowest-producing component. With the increase in equipment size and technology, larger fragmentation can be accommodated at the drawpoint, but this may cause downstream issues for the shaft/conveyor or truck fleet used to haul the material out of the mine. As such, it is imperative that the mine production system is integrated with the materials-handling system, as well as with the mill, to ensure overall optimum mine performance.

Recovery and Extraction

Recovery factors relate to the percentage of mineral recovered from the ore body relative to the amount blasted. Mineral recovery from sublevel caving operations can be above 85% of the quantity of in-situ mineral blasted. However, this recovery can be inflated by inflow of low-grade mineral contained in diluting rock from outside the mining boundary.

Extraction or tonnage factors relate to the percentage of tons drawn relative to the tons fired. This percentage varies from site to site but is generally between 80% and 125%. When the extraction rate is more than 100%, this is termed *overdraw*. Depending on the draw control strategies and grade distributions within the ore body and host rock, individual drawpoints may be drawn in excess of 1,000% and still be returning material above the shutoff grade (the grade at which drawpoints should theoretically be shut down). This generally occurs when previous levels of the mine have been significantly underdrawn because of geotechnical or mine design factors.

Some ore bodies will have physical differences between the ore and waste, which are readily identifiable—for example, color or weight. This allows mine personnel to assess in real time whether or not a drawpoint is still running at the required grade. In these situations the mines are able to more selectively recover ore left behind on previous levels. However, they must be careful to ensure that their grade control methods are accurately calibrated in order for their shutoff targets to be met.

Other operations with no readily identifiable physical difference between the ore and waste will often run to a tonnage limit and stop draw at a point independent of the drawpoint grade. This tonnage strategy can lead to a considerable amount of mineral being left in the cave, which these operations will then try to recover over subsequent levels by increasing their planned draw rates using a broader, less selective strategy. Under these systems, total tonnage factors for the mine can be slightly higher than situations where selective grade control can be applied.

Fines migration can also influence extraction strategies, especially when sublevel caving is applied beneath open pits with significant exposures of weak, weathered materials or in ore bodies with weak hanging walls. The cave column applies a significant amount of comminution to the rock as it flows, so weak material is often reduced to a small fragmentation size referred to as fines. Fines can also be created through the overcharging of the blastholes. Due to the properties of gravity flow, fines material will descend through the cave column faster than the coarser rock fragments, sometimes prematurely closing drawpoints, affecting ore-handling system efficiency or creating the potential for mud rush when combined with the introduction of large amounts of water to the cave column.

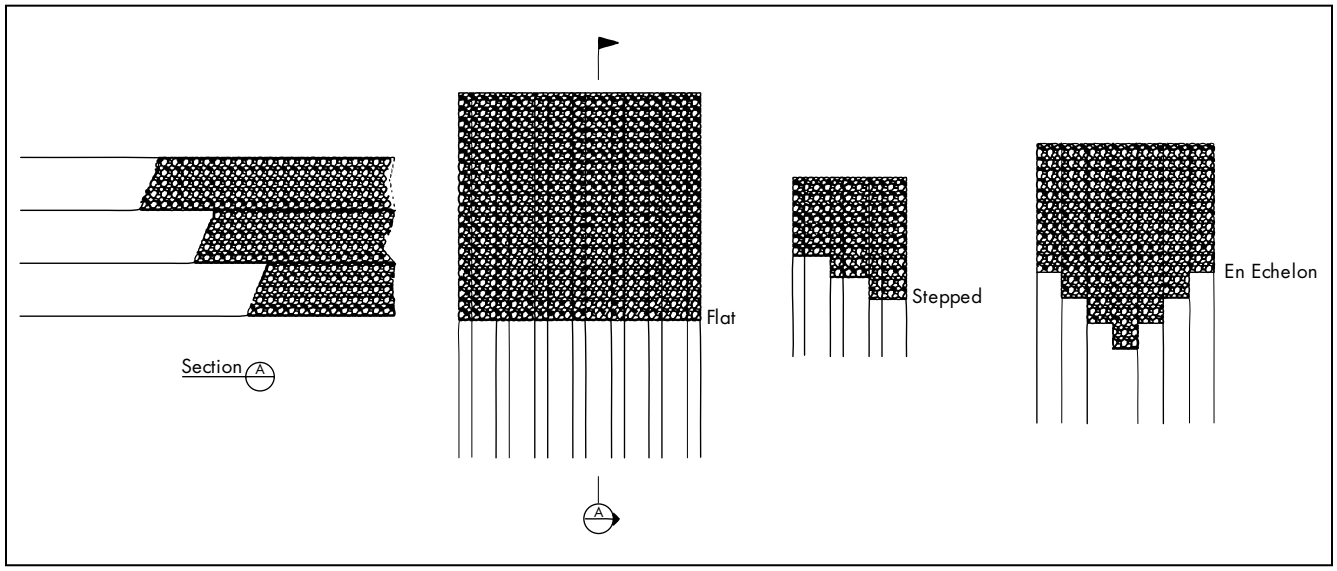


Figure 13.9-30 Examples of cave-front geometries

TECHNICAL COMPONENTS

Cavability

The caving of the host (or *cap*) rock above the sublevel caving operation is crucial to the success of the mine. Mine-scale structures and the general geotechnical conditions of the host rock influence the rate of cave propagation and the size of the footprint required to ensure caving.

There are two distinct phases to the caving of the cap rock: (1) initiation of caving and (2) the continuation of caving as the mine continues deeper (Figure 13.9-31). Should either of these stages not occur and the cave stalls or forms a stable arch, there are considerable safety risks relating to the creation of voids and the potential for air blasts. Cap rock failure may occur as either a time- or production-dependent failure. Time-dependent failure occurs when the cap rock yields and fails only at intermittent periods—a mode that can lead to significant cap rock failures depending on how much swell room is available. Production-dependent failure occurs when the cap rock unravels almost immediately after an air gap becomes available, largely controlled by extraction of the blasted material. Time-dependent failures are often accompanied by a small number of significant seismic events, whereas production-dependent failures tend to have a significant number of seismic events of a small magnitude. The mode of cap rock failure will determine the production ramp-up rate of the sublevel caving mine, because a number of safety hazards can be created if the initial production rates are not matched to the cave propagation rate.

The most significant safety risk is the creation of an excessive air gap, which can lead to a dangerous air inrush/air blast when the cave back yields. In order for rock to cave, some form of air gap has to be present. The geomechanical properties of the rock mass, the footprint of the production level, and the draw control strategy of the mine determine the size of the air gap that can form. Sublevel caving operators must manage their operations to ensure that the size of any air gap that forms is limited and does not constitute a hazard to mining personnel.

Void monitoring is a critical process when establishing a cave under any conditions. The results from the cave-monitoring system need to be cross-checked with the production figures to determine the type of cap rock failure mode that is in place and to ensure that there is a suitable coverage of blasted and caved material above the active production drawpoints to prevent a dangerous inrush of air into operational areas. It may be prudent to reduce or even stop production for a period of time in the sublevel caving to allow the cap rock failure a chance to catch up and return to the safety operating guidelines. Continuing to produce while creating a significant air void can have catastrophic results when the cap rock ultimately fails.

To maintain an understanding of how the cave is progressing, cave monitoring throughout the life of the operations is important. A cave-monitoring system should consist of a number of complementary measurement techniques to allow for the cross-referencing of results. Microseismic systems placed outside the expected cave column give an indication of the seismogenic zone, which exists just above the cave back (Duplancic and Brady 1999) and can be used not only until the cave breaks through to the surface but also as it relaxes over the life of the mine. Open holes into the cave back area can be used until the cave reaches a predefined distance from the surface where it is no longer safe for people to travel above the cave back. The open holes can be plumbed for depth using weighted wire rope and in some cases also provide access for borehole cameras. As the cave approaches the surface, the open holes can be used to place extensometers and time-domain reflectometers, which allow for remote monitoring of the failure of the cap rock.

Subsidence

As a caving method, sublevel caving will create a surface expression and a crater that deepens with time. This surface movement zone is referred to as the subsidence area. Understanding the likely size of this area is imperative when locating major infrastructure both on the surface and underground. Motion

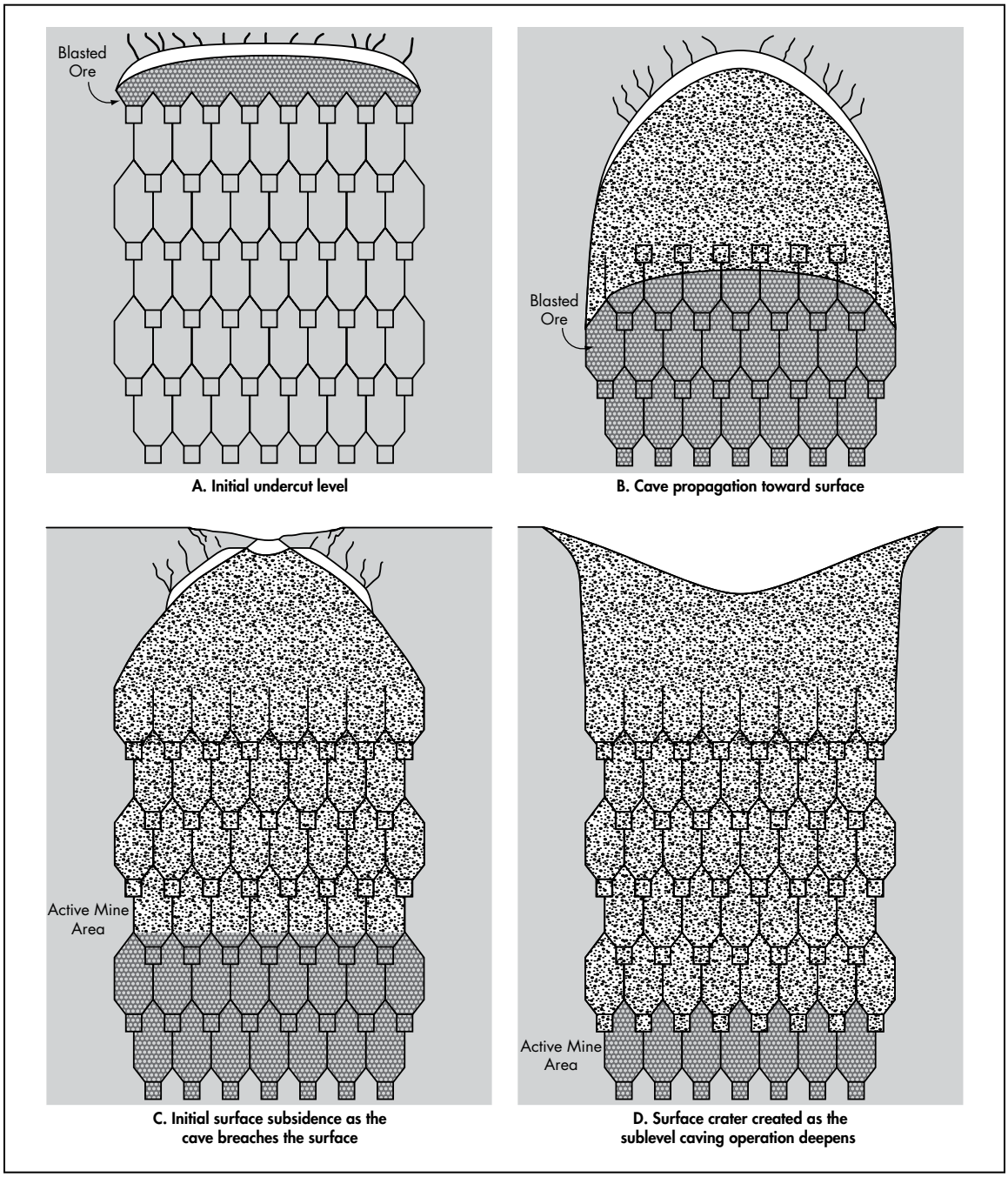


Figure 13.9-31 Stages of cave propagation

sensors and aerial surveys (Figure 13.9-32) can be used to monitor the growth of the subsidence area to determine if it correlates closely with the production being drawn from the mine. Should a discrepancy occur between the surface growth and mine production, a review of the potential void creation should be undertaken to ensure there is no risk to the work force.

As the subsidence zone creates a direct link from the surface to the underground workings, it is imperative that water inflow restrictions are placed around the subsidence zone where practical. For sublevel caving mines below an open pit or at the bottom of a valley, this can be a significant issue as these areas are catchments for rainfall. In these situations, monitoring of the rainfall and the saturation of the cave column may be utilized to determine the appropriate operational response. The flow of water into the subsidence area may not only result in additional pumping from the workings but can create a mud-rush issue when combined with the fines located within the cave column.

In the simplest case of a uniform vertical ore body, the initial cave crater will generally have vertical walls until the crater reaches a sufficient depth for slumping failure to occur. The surface expression will gradually expand as the crater deepens and an exclusion limit needs to be implemented. The

size of the subsidence and fractured zone around a cave will vary due to the host rock conditions, major structures, and so forth, but a rule of thumb estimate is 65° from the lowest level, as shown in Figure 13.9-33.

A variety of methods can be used to estimate subsidence effects, wherein the reader is referred to other sections in the handbook.

ACKNOWLEDGMENTS

The authors acknowledge the efforts of all involved in advancing the understanding of sublevel caving mining, because this chapter is a synopsis of many presentations, research programs, and personal discussions.



Figure 13.9-32 Surface subsidence at Ridgeway gold mine

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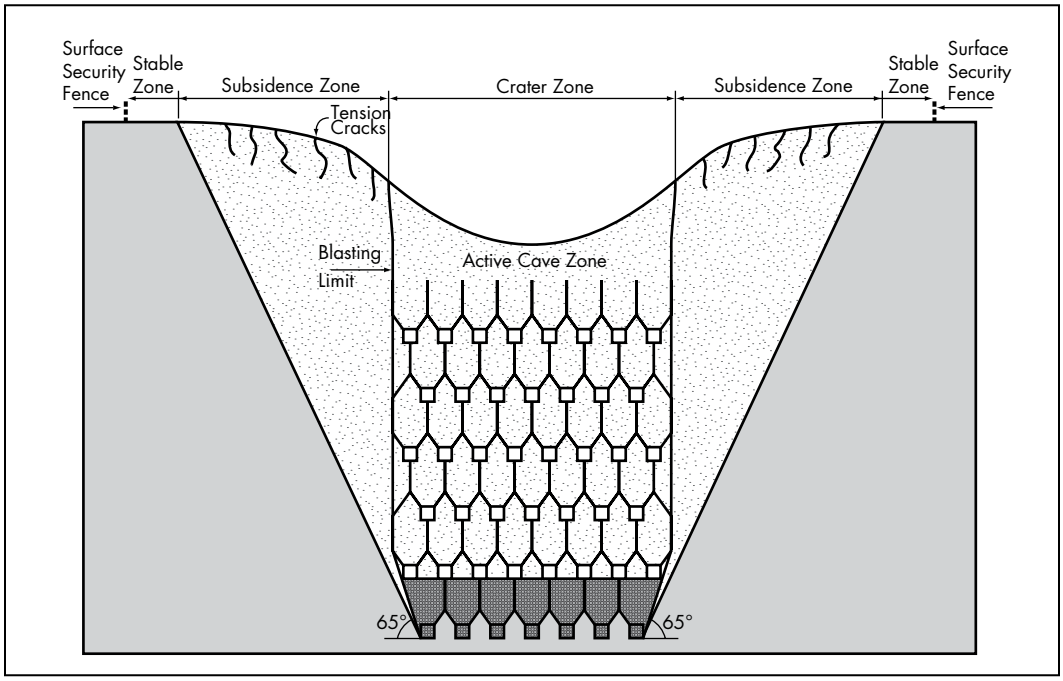


Figure 13.9-33 Subsidence terminology

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CHAPTER 13.10

Block Caving

Charles A. Brannon, Gordon K. Carlson, and Timothy P. Casten

Block caving is a mass mining system that uses the action of gravity to fracture a block of unsupported ore, allowing it to be extracted through preconstructed drawpoints. By removing a relatively thin horizontal layer at the base of the ore columns using standard mining methods, the vertical support of the ore column above is removed and the ore then caves by gravity. As broken ore is removed from the ore column, the overlying ore continues to break and cave by gravity.

Although some relatively smaller block cave ore bodies are caved and mined as a single production block, most existing and planned block cave mines use either of the following:

- An extended block caving system that divides the deposit into discrete production blocks; or
- A single cave front (or series of fronts, or "panels") advancing forward through the ore body, continually opening up new production areas as the earlier caved areas become exhausted.

The block caving method typically allows for relatively large volumes of production after the mine has been developed and production ramp-up has been achieved. The preproduction development period can be significant (typically 5–10 years, depending on the length of time to achieve the initial access). The up-front capital required prior to any return on investment is very high because much of the production levels and infrastructure must be in place before caving can begin. After the mine has reached its sustained maximum production rate, the operating cost tends to be very low with minimal additional infrastructure required to maintain the high production volumes. Block caving is generally the least expensive of all underground mining methods, and can in some cases compete with open-pit mining in cost.

The proceedings from the MassMin conferences, held every 4 years, are outstanding sources of case histories as well as theoretical discussions of block caving. Many of the discussion points in this chapter are drawn at least in part from the many excellent papers in those proceedings. The two latest proceedings are from 2004 (Karzulovic and Alfaro) and 2008 (Schunnesson and Nordlund). An excellent reference book on the geotechnical design aspects of block caving is *Block Caving Geomechanics* (Brown 2003).

CURRENT AND PLANNED CAVING MINES

Figure 13.10-1 shows the worldwide distribution of active and planned block and panel caving mines. A number of the planned mines will be developed under very large open pits that will be exhausted in the next 10–15 years. Thus, many of the planned caving mines will be very large deposits, mined at greater depths.

As technology improves, the trend is for higher production rates in these deposits. The largest caving mines in the world in 2009 were producing in the range of 75,000 t/d (metric tons per day) from a single footprint. At the same time, mining complexes such as El Teniente in Chile were producing 130,000 t/d from a number of production blocks. Planned production rates from future mines are as high as 160,000 t/d, indicative of the evolution of caving mines into "rock factories" capable of these high production rates.

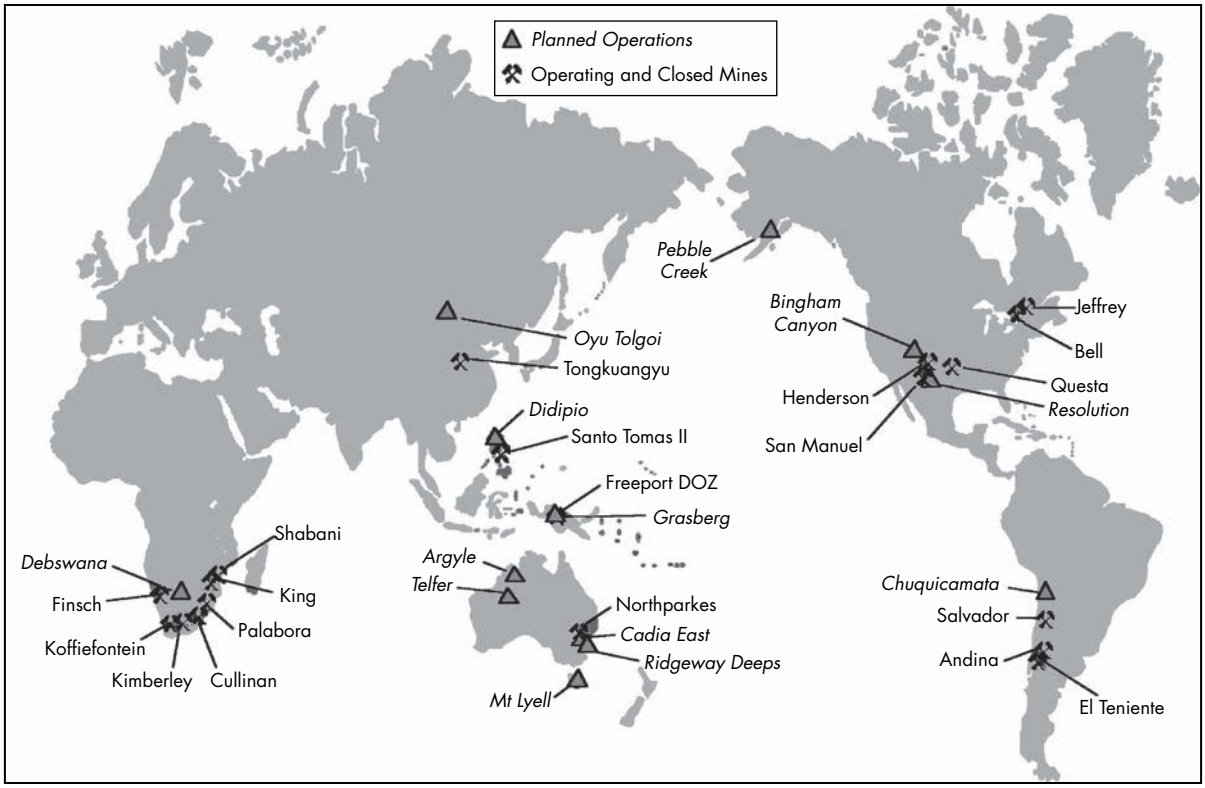
Another trend in the future is for increasing mining depths, as the new "super caves" are developed beneath the open-pit mines and other undeveloped deep deposits begin to come on-line. As technology improves our ability to manage issues in deep mines such as in-situ stress, temperature gradient, and more competent ore bodies, this trend is expected to continue. Currently, caving mines are being contemplated in excess of 2,000 m deep.

ORE-BODY CHARACTERISTICS

As for any mining method, determining the applicability of the caving method must consider many different aspects, including the location, type, size, geotechnical characteristics, and value of the ore body. No single formula can be readily applied to determine the suitability of an ore body to a particular caving method.

The caving method is typically applied to large, fairly flat-dipping ore bodies, or porphyry-type ore bodies, with rock

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Source: Brown 2003.

Figure 13.10-1 Known operating and planned block and panel caving mines

mass characteristics that are amenable to sustainable, large-scale, rock mass failure (caving). The lateral extents of the ore body must be sufficient to both establish and sustain the cave. The required dimensions to achieve this will vary and depend on key characteristics such as rock strengths, fracture frequency and orientation (fragmentation), and in-situ stress conditions. The height of the ore columns must be sufficient to pay for the costs of developing and maintaining the production levels from which the ore is extracted. At the same time, the ground conditions must allow for development of the mine openings required to exploit the ore body, as well as for proximal location of required infrastructure.

GEOTECHNICAL CHARACTERISTICS

The geotechnical character of the rock mass to be caved is a key factor in the ability to effectively initiate and propagate the cave and successfully and efficiently produce ore for life of mine. An understanding of cavability, fragmentation, and stress are primary elements in designing and operating a caving mine.

Rock Mass Characterization

During the process of mine method selection, a thorough understanding of the rock mass behavior is required in order to assess cavability, fragmentation, mine production level design, mine infrastructure design, and surface impacts. For this purpose, topics that require a thorough understanding include

- Geology, including geometry, lithology, and structural features of the ore body as well as the surrounding country rock;
- Hydrology, including surface features and groundwater flow models;
- Geotechnical studies;
- Rock mass classifications;
- In-situ stresses; and
- Discontinuities of the rock mass (faults, joints, bedding, etc.).

Cavability

The cavability of the deposit is a fundamental feature of mine design. Initiating the cave, and then sustaining the cave throughout the mine life, is critical for achieving the desired productivity and thus directly impacts the economics of the project. The cavability may be established through empirical comparisons of the rock mass to other known caving operations, numerical modeling, or both.

A key aspect of the cave design is the hydraulic radius (HR), which is a calculation of the area of the caved zone divided by the length of the perimeter. Geotechnical studies will determine the required HR to initiate the cave, and also the HR required to sustain the cave. This ties directly into the mine design, as the production blocks must honor the required HR for that deposit. In general, caves tend to grow laterally in a circular fashion trying to reach equilibrium. In contrast, long laterals with narrow leads (extended rectangles) may satisfy the HR requirement but may not cave until more area has been opened.

Fragmentation

The fragmentation characteristics of the deposit are perhaps the single most important geotechnical feature. Fragmentation

impacts most aspects of mine design and operation, including drawpoint size and spacing, equipment selection, production rates, draw control requirements, dilution, hang-ups, and mill throughput. Thus an accurate prediction of fragmentation is a key to a successful mine design, and yet this prediction is one of the more difficult to produce with confidence.

The fragmentation of a deposit changes over time, and all of the stages below must be considered in the mine design:

- *In-situ fragmentation* refers to the natural, unaltered blocks before any caving takes place.
- *Primary fragmentation* is the distribution of block sizes during initiation of the cave.
- *Secondary fragmentation* is the fragmentation that occurs as the blocks move through the draw column during mining.

Monitoring

Monitoring of the progress of the cave development is important for many purposes, including

- Safety by maintaining a reasonable air gap between the cave back and the cave muck pile;
- Initiation and development of caving in order to validate the conceptual models, but also to allow modifications to the design as required by that information;
- Stability of the production and infrastructure areas in order to react to issues such as areas of excessive convergence; and
- Surface subsidence monitoring.

BLOCK CAVE MINE DESCRIPTION

Following is a discussion of block cave mine planning that offers additional details about the layout of a block caving mine.

Mine Layout

A block caving mine consists of several horizontal operating levels—the undercut level, extraction level, service/ventilation level, and haulage level. An additional level for drainage may be required, depending on water conditions in the mine.

Undercut Level

Undercut level designs depend on the method of undercutting chosen. Undercut methods are described in more detail later in this chapter. In the cases of pre- or advance undercutting, the undercut level is much closer to the extraction level and the major apex pillar is formed during the undercutting process. Post-undercutting usually places the undercut level drifts directly over the extraction level drifts and at the top of the major apex. Long-hole drilling and blasting is required for the formation of the drawbell in all cases but is sequenced differently according to method. Pre- and advance undercutting start with the extraction of the rings across the drawbell and create the angles of the major and minor apexes. This is followed by a final blast from the drawpoint to create the drawbell connection between the undercut and extraction levels. Post-undercutting has the opposite sequence, with the initial blast from the drawpoint, followed by a series of bell formation shots from the undercut level to create the drawbell. The final sequence of shot or shots will be the undercut rings, which will remove the remaining pillar material at the undercut inducing caving.

Extraction Level

Although there are a variety of possible extraction level layouts, there are two main drawpoint layouts employed in the majority of current and planned caving mines: the offset herringbone and the El Teniente (or "straight-through"). Both methods have advantages and disadvantages that must be considered when selecting the layout.

Herringbone layout. Figure 13.10-2 shows an example of an offset herringbone extraction level layout. Herringbone designs are best suited when electric load-haul-dumps (LHDs) are to be employed because the mining direction is the same on both sides of a production drift. Electric LHDs also help minimize the ventilation requirements. This option tends to be used in smaller-footprint operations such as Northparkes (Australia) and Palabora (South Africa) that do not require the ore body to be split into multiple production blocks. The drawbell of the herringbone layout is rectangular and lends itself well to single-shot drawbell blasting.

El Teniente (or "straight-through") layout. In the El Teniente layout, the drawpoint drifts are developed in straight lines oriented at about 56°–60° to the production drifts and the major apexes. Figure 13.10-3 is an example of the straight-through layout at the Henderson (Colorado, United States) mine. The minor apexes are short and inclined to the major apexes. The El Teniente layout is generally considered to be more robust than comparable herringbone layouts because it has a better pillar shape.

From the development perspective, the El Teniente layout has advantages over the herringbone layout, especially in a panel caving operation. The El Teniente straight-through design allows for the drawpoints to act as temporary crosscuts and accesses or "pseudo fringes," well before final boundary drifts must be completed. The straight line drifts in the El Teniente layout make development more efficient and cost-effective, and potentially create less overbreak, especially around the nose pillar.

Temporary fringes can be achieved with the herringbone layout, but this is more difficult as the drawpoints are staggered and therefore the temporary fringe drifts are not straight. In addition, the boundaries of the panels in a large deposit become problematic. Such problems occur because if a fringe drift is established between panels, the width of the pillar between the herringbone layouts on either side of the fringe drift is excessive, and merging the caves across that pillar is very difficult and likely to cause excessive weight problems.

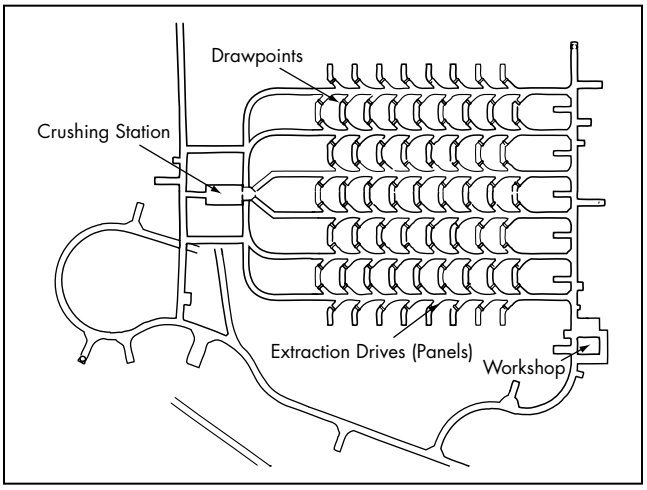
From a scheduling perspective, the El Teniente layout also has advantages over the herringbone layout. This is because a line of drawpoints outside of the planned production block can be used as a "temporary or pseudo fringe," and in this way development meters can be deferred. This also postpones construction of drawpoints, grizzlies, and chutes. Overall this would delay capital spending as well as reduce the risk of slipping on the planned schedule.

A significant disadvantage of the El Teniente layout is that it is impractical for electric LHDs using trailing cables. With a herringbone layout, the LHD can produce from all drawpoints in a production drift while facing the same direction. But with an El Teniente layout, the LHD will have to change direction for either side of the production drift. This will require additional lost productive time to disconnect and reconnect the trailing cable at the other end of the production

drift. The flexibility of having diesel equipment that can easily move to other areas or function in the mine is an advantage for El Teniente layouts. Of course, diesel equipment requires a higher ventilation airflow rate.

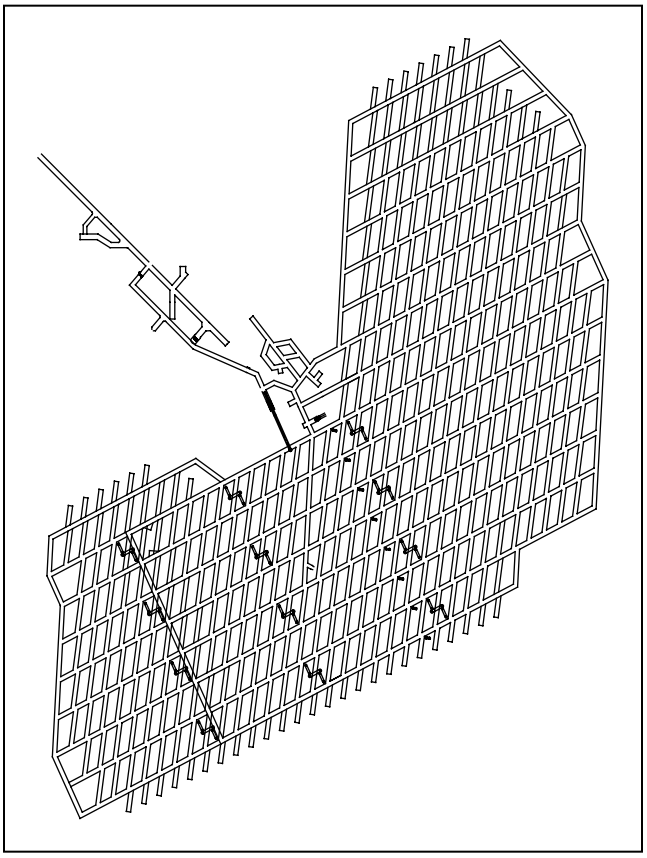
A safety concern in the El Teniente layout is with tramming the loaded LHD bucket-first. This restricts the operator's visibility 50% of the time because he or she has to operate the LHD in both bucket-first and engine-first positions in order to pull from all drawpoints. Secondary breaking units and pedestrians who access operating panels are examples of an increased risk that has to be managed in this case. Procedures are typically in place to protect pedestrians in a working panel and include lights, reflective personal protective equipment, and worker/machine proximity detection systems. A positive aspect of the El Teniente design is that the LHD operator always enters the drawpoints from his side of the LHD, giving him better visibility for safety and operational efficiency.

Another safety issue is the concern over the impact of a wet muck spill in an El Teniente layout. During a significant wet muck spill the straight-through configuration of the El Teniente layout can result in the LHD being pushed back into the adjacent drawpoint by the wet muck and being completely covered. This results in greater damage to remote-controlled LHDs and an extended time required for recovery of the gear from the muck spill. With the herringbone layout, the LHD is typically pushed down the drift and remains exposed. In either case, areas of wet muck would need to be worked by remote-controlled or automated LHDs, thereby eliminating any risk to the operators.



Source: Ross 2004.

Figure 13.10-2 Herringbone extraction level layout at the Northparkes mine



Source: Rech et al. 2000.

Figure 13.10-3 Extraction level plan at the Henderson mine

Service/Ventilation Level

The service/ventilation level is typical of a larger-footprint mine and is primarily a way to provide intake and exhaust ventilation across the production footprint for the undercut, extraction, and haulage levels. It also provides an access to these levels that is independent of production accesses and activities. Smaller-footprint mines will typically not require a separate ventilation level and will simply use the existing levels (extraction and haulage) to provide the required ventilation.

A large caving footprint is a very porous system from a ventilation point of view, with many accesses and cross connections in the preproduction and production areas. In order to provide reliable fresh air splits and exhaust connections, a series of ventilation plenums are required. This is usually via two ventilation levels (intake and exhaust) located between the extraction level horizon and the haulage level. This series of plenums are connected to vent raises that intersect the production levels in key points. The ventilation levels cover the extent of the footprint and are usually quite large-diameter drifts, which makes them good candidates for a service level. The service level can be used to access the haulage level in key areas as required for development as well as for construction and repair of haulage level infrastructure. This is especially important in a rail haulage system where independent rubber-tired access to the track is required for cleanup, construction, or maintenance work. The service level can also be used to assist with level drainage and waste haulage, and act as a conduit for services such as water, compressed air, and power.

Haulage Level

The haulage level design will be determined by the type of transportation method required—rail haulage, trucks, conveyors, or a combination of these. Each method has its advantages and disadvantages, which need to be carefully weighed in the context of that particular deposit before development begins because conversion after the fact is almost impossible.

The greatest advantage of rail haulage is its ability to move large tonnages in a single movement. This is especially

important if the material requires long-distance travel to either crusher systems or to a surface milling facility. Some of the drawbacks to this system include large capacity bins to accommodate train sizes, grade and curve radii restrictions, increased power consumption (for electric locomotives), track maintenance, electrical capital and maintenance, and potential production shortfalls as a result of disabled trains in the loading drifts or main haulage lines.

The greatest advantage of truck haulage is its flexibility. Trucks have the ability to maneuver in smaller areas and can be reversed in traffic flow if the need arises. Road maintenance is continual but not labor intensive, and generally a single disabled truck can be removed from the traffic flow fairly easily, thereby reducing production shortfalls. Another advantage of a truck haulage level is the ability to modify designs easily over the course of time if new reserves or discoveries come into play. The main disadvantages of truck haulage include significantly smaller payloads, which require larger fleet sizing, larger ventilation requirements, increased maintenance costs, more labor, and the inability to efficiently move large daily tonnages long distances. In the case of the Henderson mine, for example, a fairly confined truck haulage route to a centralized crusher feeds a 24-km conveyor system to the mill and has proven to be very reliable and productive.

Several small-footprint caving operations (e.g., Northparkes and Palabora) use a system of conveyors to gather the production material from multiple crushing stations onto a central conveyor that feeds production shaft hoisting.

MINING METHODS

The selection of the methods used to undercut and extract the ore from a caving operation are fundamental to the success and efficiency of the operation. Selecting the appropriate methods at the design phase is key, as making major changes in methodology after the mine is in production can be difficult and costly.

General Mining Systems

The key to designing a suitable mining system is to have a good understanding of the geotechnical characteristics of the deposit. In particular, the fragmentation of the deposit will control the width of the "draw cone" or area of influence of the drawpoint as the material is mined. The area of influence then determines the required spacing of the drawpoints.

The three principal methods of block caving are the gravity, slusher, and LHD systems. Historically, the gravity system was commonly selected for ore bodies consisting of finer material essentially free-flowing to the drawpoint. For somewhat coarser material, the slusher system was employed, while the LHD system was employed for relatively coarsely fragmented deposits. Today mines are moving toward higher production rates with greater reliance on mechanization and automation, and moving away from the labor-intensive gravity and slusher mucking systems. The majority of existing and planned block caving operations now use some variant of the LHD (rubber-tired) system of mucking. Of course, this assumes that the ground conditions allow for development of stable production levels that can handle the size of the equipment. The majority of this section is therefore devoted to the LHD mining system.

Full-Gravity (Grizzly) System

The grizzly system consists of the haulage level, transfer raises, grizzly level, finger raises, and undercut level. The haulage and grizzly levels are driven across the ore block to be mined. This work can be done simultaneously because they are on different levels. Finger raises are driven from the grizzly level to the undercut elevation, and then short horizontal connections are driven between the tops of the finger raises forming pillars that will later be long-holed and blasted to initiate the caving action. Transfer raises are driven between the haulage and the grizzly level with a loading chute at the haulage level and grizzly rails for sizing at the grizzly level. After the undercut has been blasted, broken ore flows down the finger raises and is sized by the grizzly rail spacing, passes into the transfer chutes, and is loaded into rail cars. Sledgehammers or semi-mobile rock breakers are used to break up oversized pieces at the grizzly. Some large pieces may hang up in the finger raises, and these are usually broken up by secondary blasting with packaged explosives placed strategically against the oversized rocks. The Andina copper mine in Chile is an excellent example of the grizzly system.

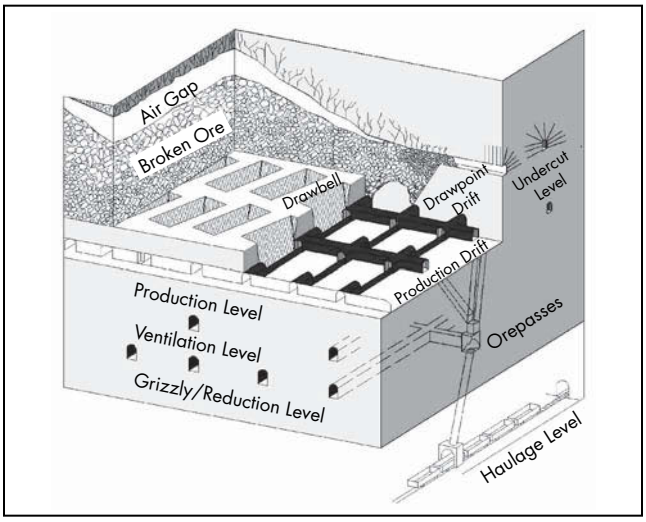
LHD (Rubber-Tired) System

The use of LHD equipment usually requires that the drawpoints be spaced at greater intervals to allow room for the equipment to operate. The larger equipment requires larger drift/drawpoint excavations, so larger spacing is also needed for pillar strength. The wider spacing is suitable for coarser rock; finer fragmented deposits may require tighter drawpoint spacing and therefore smaller equipment. This system consists of the haulage level, ore-transfer raises, the production level, drawpoint entries, draw cones to the undercut level, and undercut drifts. Long-holing and blasting is used to form the undercut that promotes caving. The large draw cone will allow larger pieces of ore to move down near the drawpoint where small drills and explosives can be used to break up the larger pieces; high hang-ups practically never occur with this system. The haulage, production, and undercut levels can be developed simultaneously using separate accesses. The haulage and production levels should be separated by a substantial distance to provide adequate storage in the transfer raises, so that the loading of the trains is not delayed waiting for the LHDs. When the haulage and production drifts are complete, the transfer raises and drawpoint entries can be driven. Drilling of the draw cones can be done from the drawpoint entries or from the undercut level. Freeport-McMoRan's Henderson molybdenum mine in Colorado, the DOZ copper-gold mine in Indonesia, and CODELCO's El Teniente copper mine in Chile are examples of mines that use the LHD system.

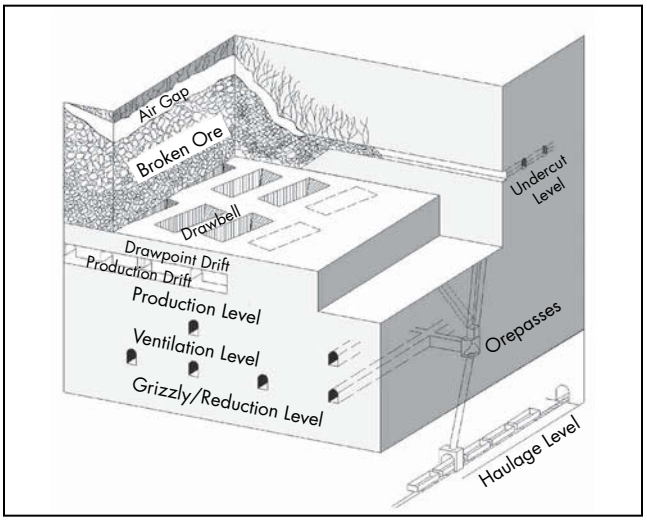
Undercutting Methods and Issues

The undercutting strategy, design, and management are critical elements to a successful block cave mine. Some of the aspects of undercutting to consider include the following:

- Undercutting strategy (post-, pre-, or advance undercutting methods)
- Undercut design (initiation, direction, and shape of the undercut face, rate of undercut advance, and undercut heights)
- Shape of the undercut and extraction method (flat versus "crinkle" undercutting)
- Impact of the undercut design on stresses in the undercut and extraction levels
- Drill-and-blast designs for the undercut and drawbell



Courtesy of G. Flores.
Figure 13.10-4 Post- or conventional undercutting



Courtesy of G. Flores.
Figure 13.10-5 Pre-undercutting

Post-Undercutting

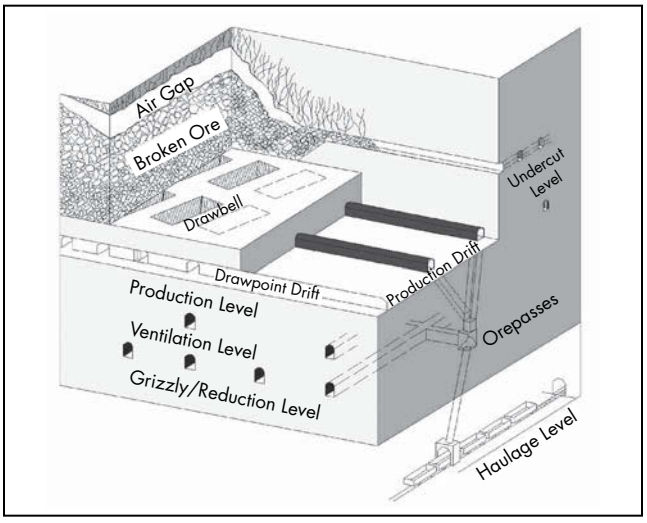
Post- or conventional undercutting refers to undercut drilling-and-blasting after development of the underlying extraction level has been completed (Figure 13.10-4). Drawbells are also prepared ahead of the undercut blasting. Production blocks can be brought into production more quickly than with other undercutting methods, and this method has been most successfully used on smaller mining footprints. Other advantages are that this system requires about half of the development on the undercutting level as the other undercutting systems. In addition, no separate ore-handling facility is required on the undercut level, because the ore is transferred through the already-developed drawbells.

The primary disadvantage is that the mine openings are all subjected to the abutment stresses when the cave front passes through during undercutting. The resulting damage to the major and minor apex can be substantial and reduces the effective life of the drawpoints and production drifts (or at least often requires repair work to maintain drawpoint integrity). Selection of this method requires careful consideration of ground conditions and stress issues. Drawbell blasting can be easier to recover from because the undercut is not yet blasted, but conversely, undercut remnant pillars (stumps) can be difficult to deal with. The drawbell can be mucked clean and inspected prior to undercutting.

Pre-Undercutting

In pre-undercutting, the undercut is mined ahead of the extraction level development (Figure 13.10-5). This allows the development of the extraction level in a destressed environment. This method is typically seen in small-footprint mines and in mines where ground compaction is not deemed an issue.

The pre-undercut can occur quite some distance ahead of the last drawbell, with a considerable amount of development and construction work carried out in the cave shadow area. The footprint has to be small enough that the pre-undercut does not sit idle too long, allowing the abutment stress to refocus on the undercut level or the extraction level through compaction. In addition, facilities must be developed to handle the swell muck from the undercut level.



Courtesy of G. Flores.
Figure 13.10-6 Advance undercutting

Advance Undercutting

Advance undercutting is increasingly popular and many future caving mines are planning to use this technique. In this method, the undercutting is completed above a partially developed extraction level (Figure 13.10-6). The extraction drifts, and in some cases all or part of the drawbell drifts, are completed before the undercut passes. When the undercut has passed, the drawbelling is completed. The crucial drawbell blast is thus completed in a destressed condition. Because significantly less of the extraction level excavation is completed in this method, damage to the pillars can be greatly reduced. The cave is brought into production more quickly than for the pre-undercut method, and the probability of damage from compacted undercut material is reduced.

Typically an advance undercut only extends ahead by the length of one drawbell, thereby keeping the active cave close to the leading edge of the cave front. If the caved undercut is left to sit for too long before taking the drawbell beneath, it

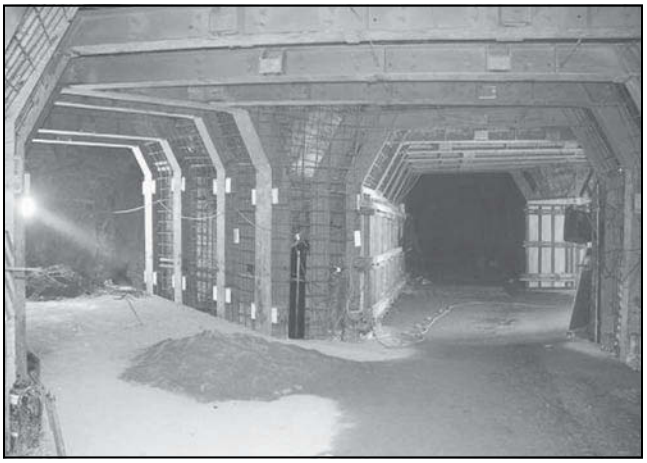


Figure 13.10-7 Extraction level under construction using an advanced undercut, DOZ mine, Indonesia

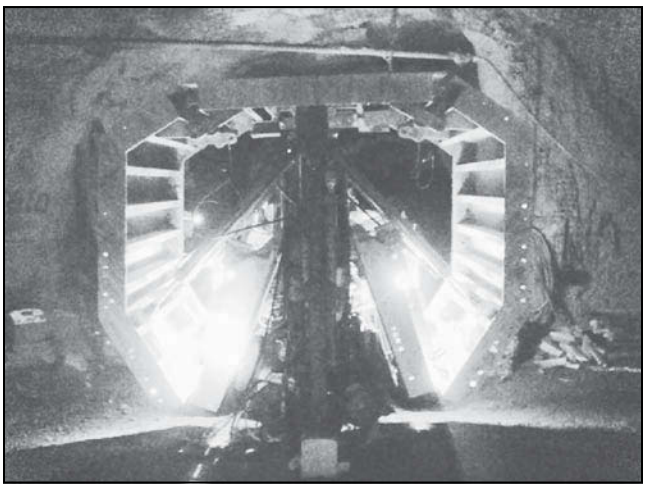


Figure 13.10-8 Drawpoint construction-forming jumbo, Henderson mine

can begin to compact and "accept" the weight of the ground above. The abutment stress is redirected to the undercut level and far less damage is seen on the extraction level.

A separate level for undercutting is required and typically has twice the development as the other undercutting methods. Because development and construction occur ahead of the undercutting and drawbelling, this method requires relatively greater up-front capital and use of development resources. It can be difficult to recover a drawbell if it is not blasted properly. If this becomes necessary, a hung drawbell needs to be drilled from the adjacent drawbell to the unblasted drawbell through the minor apex.

PREPRODUCTION DEVELOPMENT AND CONSTRUCTION

When initial mine development is complete, and depending on the geotechnical requirements for the area, a significant amount of preproduction ground support may be required in the extraction level to maintain panel and drawpoint integrity for the duration of the caving and production periods. The production period can last from 3 to 5 or more years depending on the drawdown rate and column heights. The ground support may, for example, consist of grouted threadbar (or cable) bolts, wire mesh, and shotcrete throughout the extraction area and will vary depending on ground conditions.

Preproduction work is followed by drawpoint construction, which typically consists of two lintel steel sets placed at the mucking point of the drawpoint to protect the brow and pillar from erosion by the caving material. The goal is to maintain the opening for the life of the ore column with a minimum of repair. Figures 13.10-7 and 13.10-8 show typical drawpoint construction. The lintel sets are often concreted and bolted in place, with the posts being set into concrete in the floor. The drawpoint and panel floors are typically concreted. Some operations choose to add surface hardening products and steel wear bars (often this is simply used rail) to prevent early floor wear from both mucking and tire wear by the LHDs. If the area has particularly poor ground, a full series of steel sets and concrete may be applied to the drawpoint and panel section in addition to the two lintel sets.

A typical drawpoint and corresponding section of panel may cost in the range of US$250,000 to US$500,000, depending on the level of support required. Designing for adequate drainage in the panels is key as water can play a major role in early floor cracking and wear. A good general rule is to keep panel roadways at a minimum 3% gradient to ensure adequate drainage.

Some mines are small enough that a separate ore handling level is not required (e.g., Palabora and Northparkes). Where the footprint is larger and the haul distance to the crusher is prohibitively high, a separate haulage level is employed. As the panel and drawpoints are being constructed, the relevant orepasses, ore handling systems, and ventilation infrastructure must keep pace. Orepasses can be drop raised, raise climbed (mechanically or manually), or raise bored down to a handling level. The top of the orepass is usually fitted with a grizzly and often a rock breaker for oversize material. Grizzly dimensions vary depending on the orepass size and handling method, and can range from 400 $\times$ 400 mm up to 1,000 $\times$ 1,000 mm openings for mines with the larger equipment that can handle material of that size. Grizzlies should be sized to the estimated fragmentation parameters; otherwise, excessive choking can occur and clearing the obstructing material will decrease the LHD efficiencies. Ideally, as much of the rock reduction or breaking as possible should be done at the primary crusher to reduce secondary breaking requirements on the extraction level. This requires the largest-sized grizzly openings that can be delivered to by the LHD without causing hang-ups in the orepass or other downstream systems. With a 1,000 $\times$ 1,000 mm grizzly the orepasses would need to be at least 4,000 mm in diameter, with chutes and transport systems sized accordingly. The DOZ mine in Indonesia initially used 1,000 $\times$ 1,000 mm grizzlies on a 4,100-mm-diameter orepass. These fed into large chutes and 55-t-capacity haul trucks that dumped at a 1,370 $\times$ 1,955 mm gyratory crusher.

Figure 13.10-9 shows a typical grizzly installation from the DOZ mine. The grizzly openings are 800 mm, feeding into a 4.1-m-diameter bored orepass. The bumper is 300 mm high.

The orepasses can vary in length and angle depending on the haulage level elevation. Orepass diameter is a function of grizzly size and expected material types. A good general rule is to keep the orepass diameter greater than three to four times the size of a grizzly opening to minimize the potential for hang-ups. The base of the orepass is fitted with a typical lip and gate chute arrangement that is used to load either trucks

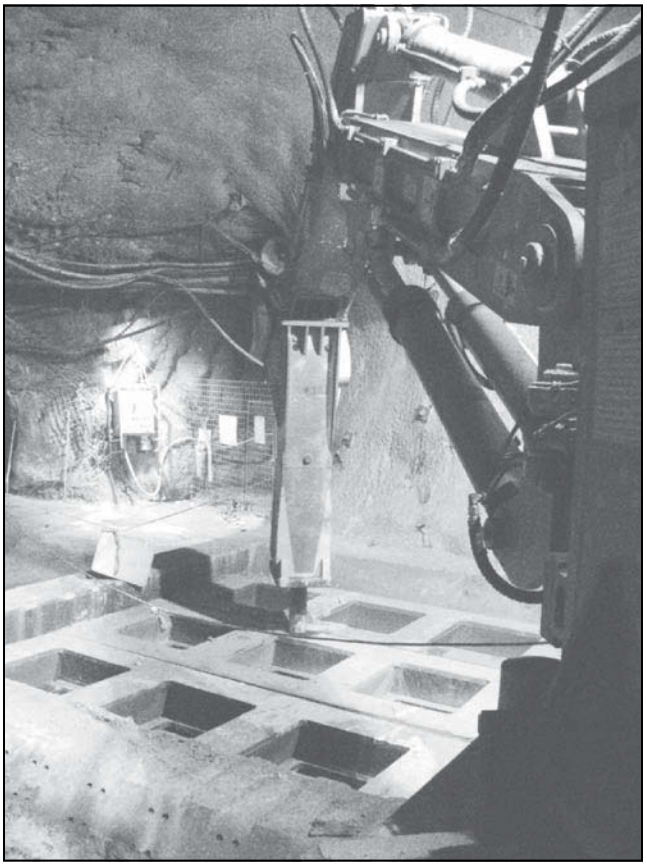


Figure 13.10-9 Grizzly installation example at the DOZ mine

or rail cars with ore. Where the broken ore is fine enough, a feeder may be used to load onto coarse ore belts. The haulage level delivers the ore to a primary crusher from which it is transferred to the surface by means of conveyor, hoists, or rail (as surface topography permits).

PRODUCTION

The production rate of a caving mine plays a major role in the required infrastructure, the time to ramp up to full production, and the drawpoint opening requirements. Selecting the production rate is one of the primary mine design variables required early in the process.

Capacity

A primary benefit of block cave mining is the high production capacity relative to other underground mining methods. Some of the larger caving mines in the world are producing up to 75,000 t/d from single mining blocks (the DOZ mine in Indonesia), and mines operating in multiple mining blocks are producing greater than 130,000 t/d (El Teniente, Chile).

Those upper limits will continue to grow as more and more of the large open-pit mines in the world transition to underground mining and the technology for large-scale underground production improves. Mines are being planned today that will exceed 150,000 t/d when at full production.

Setting the sustained maximum production rate for a caving operation is a key variable in the mine design process. The production rate is driven by several criteria: the rate of drawpoint opening, the rate of draw (t/d per drawpoint), the height of draw (HOD), the life of each drawpoint (calculated from the previous two variables), and ventilation and ore-handling constraints. The latter two variables typically act as a step function in production capacity, as the addition of either ventilation capacity or ore-handling infrastructure results in a significant step-up in production capacity for a mine. The goal is to ramp up production to achieve a production rate that is sustainable over a period of time, in the order of 7 to 10 years. That may be less than a potential maximum production rate, because it would not make economic sense to ramp up to some maximum rate only to have production immediately fall back as the mining block begins to become depleted. Commonly, the construction of drawpoints proves to be a primary limiting factor in the production ramp-up.

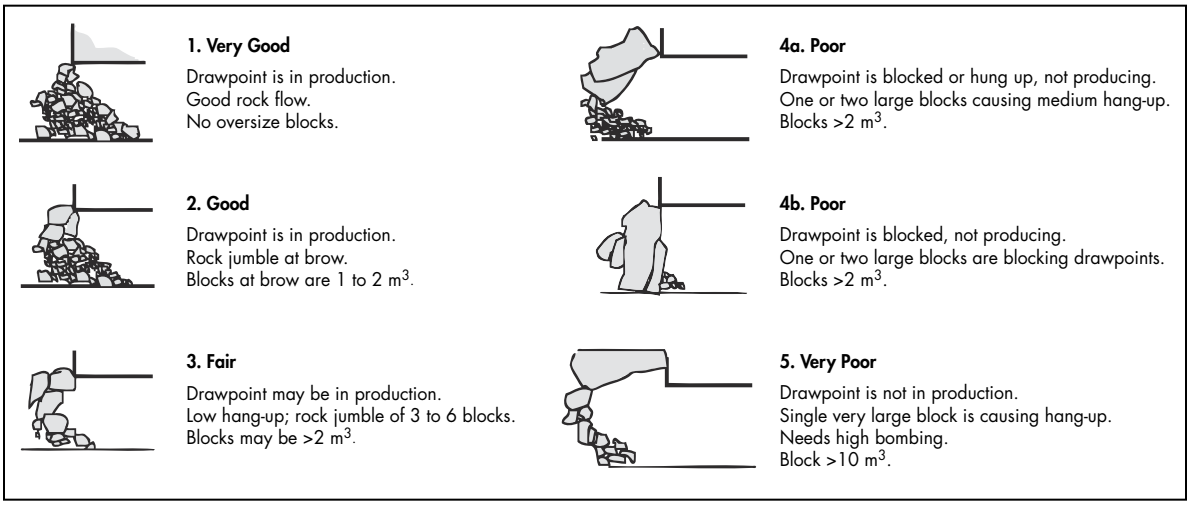
The maximum possible production may not be the economically best option, and an analysis of the economics of the mine plan needs to be considered. For example, should an inordinate amount of incremental capital be required to achieve a modestly higher production rate, cash-flow analyses would indicate that the optimal plan would be for the lower production rate.

Daily draw rates commonly range from about 150 to 600 mm. The allowable safe rate of draw depends on the rate at which the ore will cave. Drawing faster than the cave rate can result in an "air gap" that can produce conditions for a dangerous air blast should the cave fail suddenly. Coarsely fragmented material typically caves more slowly than weak, more finely fragmented material. In addition, coarsely fragmented ore will require extra effort in secondary blasting of oversize material and hang-ups, affecting the average rate of draw for those drawpoints.

Drawpoints at the leading edge of the cave are commonly mined at significantly lower rates to allow for new cave growth and to minimize opening of an air gap at the leading edge. In addition, draw rates can change over time for a given drawpoint; as the draw column is pulled, the material commonly becomes finer-fragmented through comminution in the column. Typically, early in the life of the drawpoint, very slow average draw is achieved due to coarse fragmentation and the resulting oversize and hang-ups that need to be dealt with. Most block cave planning tools use some sort of production rate curve that takes the change over time of the draw rate into account, so that the production capacity of the mine is not overestimated.

Another major consideration in determining the optimal production rate for a caving operation is the manner in which the cave is advanced in the ore body. Large deposits cannot be caved in a single pass because the advance of the cave front is slowed by the limits of the drawpoint opening rate, and the cave fronts themselves get excessively long. Long, slow-moving cave fronts generate a range of geotechnical issues that can be very damaging to the mine (Araneda and Sougarret 2007). Thus, the larger footprints must be divided into smaller mining blocks that can be caved and managed effectively, which limits the risk of excessive geotechnical issues. Larger deposits will require multiple caves that at times will be adjacent to or merged with other blocks. The issue then becomes one of production block design and drawpoint opening sequences that allow for an overall sustained production rate and at the same time are geotechnically achievable.

The maximum HOD varies from deposit to deposit and commonly ranges from 200 to 400 m, although column heights of >500 m have been achieved. The limit depends largely on



Source: Srikant et al. 2004.

Figure 13.10-10 Drawpoint rating system at DOZ mine

the ability to maintain the drawpoint and production drifts in good and safe operating condition. If a drawpoint is well constructed, excessive stress conditions avoided, and an even draw maintained, the life of the drawpoint can be maximized. It may be necessary to repair or rebuild a drawpoint for higher draw columns more than once, which is a common occurrence in mines with high column heights.

With a good estimate of the maximum sustained production rate and the life-of-mine production profile, the size and quantity of the major ore-handling and ventilation infrastructure can be designed.

Drawpoint Opening Rates

The rate at which an operation can open new drawpoints controls the production ramp-up as well as the full production rate that can be sustained. If the number of new drawpoints does not equal or exceed the number of drawpoints closed for depletion or due to damage, then production cannot be sustained. Commonly the development and construction of the drawpoint is the critical path to drawpoint opening rates. Undercutting and drawbelling can generally keep up with the pace of drawpoint construction.

For example, a typical maximum sustained rate of drawpoint opening for the DOZ mine is 10 drawpoints per month (usually referred to as five drawbells). The Henderson mine typically opens four to six drawbells a month, although the mine has peaked at higher rates.

Drawpoint Opening Sequences

Few aspects of the design of the block cave mine are more critical than developing a workable drawpoint opening sequence. The drawpoint opening sequence determines the order in which the ore is extracted. Where possible, preference is commonly given to the higher-value zones of the deposit. However, equally importantly, the sequence determines the shape and direction of the advancing cave and, for larger mines, determines how the production blocks interact with each other as the mine life progresses. It is critical to devise an opening sequence that is geotechnically sound; otherwise, the production goals, and perhaps even the mine itself, can be put at risk.

The mining sequence on a large-footprint cave will also drive the development and construction schedule. With a smaller footprint (e.g., Northparkes), the whole mine can be developed and constructed prior to caving. On larger footprints (e.g., DOZ, El Teniente, or Henderson) the development and construction of the levels takes place on an as-needed basis ahead of the caving front. This reduces early capital expenditures and smoothes development and construction activities. Overextending development in a large, active cave can often cause ground control problems in the open areas, especially if they sit for extended periods of time in stressed ground ahead of the cave front.

Production Issues

Cave fragmentation is a key issue, as the amount of ore that requires secondary breaking is a major factor in the ability of the mine to reach and sustain its maximum production level. Coarse fragmentation can give rise to high, medium, and low hang-ups in the drawpoints as well as excessive secondary breaking at the grizzly. Figure 13.10-10 shows the types of hang-ups that may occur.

Low hang-ups are dealt with using smaller drills and can be easily reached by the operators. Often nonexplosive breaking techniques are used to prevent damage to the drawpoint and lintel set area. The same mechanism can be used to break oversize material in the drawpoint or material too large to go through the grizzly. Mobile rock breakers and fixed rock breakers at the grizzly areas are also employed for secondary breaking.

When a drawpoint has been caved, accessing the drawpoint past the lintel set is too dangerous as it exposes personnel to falling rock. Longer-reach equipment is required to deal with intermediate and high hang-ups past the lintel set. An intermediate step is the use of a water cannon to “sound’ the hang-up and often bring it down by dislodging key sections. These units consist of mobile water tanks with a high-pressure water nozzle fitted to a robotic boom.

Medium-height hang-ups are dealt with using single-boom jumbos and harness remotes to drill holes in the key rocks. In some cases high bombing will be used without drilling. A bundle of explosives is tied to the end of a series of

blasting sticks and placed at key intersections of the hang-up to dislodge and bring the arch down. This process is high risk, however, as personnel spend considerable time around the front of the drawpoint and are exposed to potential falling and bouncing rocks.

In a high hang-up, the coarser fragmentation "jumbles" together inside the drawbell at a height above about 8 m. These are very rare occurrences and are handled using remote long-boomed drilling and charging rigs or by drilling through the major apex and blasting the larger rocks.

Ventilation

There are several components to ventilating a caving operation that bear particular attention. Planning these components into a general scheme early on will reduce operational trouble at a later stage. Because multiple LHDs will be operating in parallel galleries, sufficient air must be provided to support this. Regulation and control of the airflows becomes critical (especially with diesel equipment) in order to maintain appropriate levels for emissions and heat. Additional volume is also required in panel caving operations to support the development, drawpoint construction, and shotcrete/concrete operations that will be in the same panels, albeit upstream from the production.

Depending on the form of undercutting, more parallel paths on the undercut level will need to be planned for. This needs to be done years ahead of the actual caving. Henderson mine designs, for example, take several paths across the panel at set intervals to allow for the changing of the main intake as the caving advances. The intake level uses loop connections so that the airflow is equalized across the width of the panel and will not starve the last intakes to the production level at the end of a drive. Likewise, as the cave progresses over a long length, proper design can allow for the conversion of the intake paths to an exhaust path at the back end, thereby reducing the overall requirements to a boundary drift.

Another aspect that deserves attention is the orepass connections to the exhaust for dust mitigation. Every time a bucket of muck is dumped, dust will congregate at the orepass and potentially move downstream in the main drift. This not only causes visibility issues but also compromises the filtering and air conditioning systems on the LHDs. By connecting to the exhaust level, either at intercepts or through other channels, regulation of air can be accommodated to cause a slight drawdown, pulling the dust out of the production drift without compromising the overall airflow downstream of the orepass. In small-footprint caves, orepasses can be designed in the middle of the production drifts and, if connected to the main exhaust, will allow two LHDs to operate in the same panel from opposite ends without interference.

A final prudent design consideration is to have direct exhaust connections around any and all underground crusher stations. Often multiple accesses for high production rates are designed at these stations without proper assessment of the ventilation requirements to support them.

Automation

Caving mines lend themselves to automation due to the regular drawpoint layouts and fairly consistent nature of the production operations. Several caving mines employ automated LHDs on the extraction level (e.g., the DOZ and El Teniente mines) and in some cases automated trucks (e.g., the Finsch mine in South Africa) and rail (e.g., El Teniente) on the haulage level. Many aspects of rail haulage systems are automated and can typically be managed from a remote control center. The automated equipment operates in isolated areas to prevent worker/machine interface issues. Automation is employed to improve productivities and reduce exposure of operators to injury. In certain mines the risk of rock bursting or wet muck rushes is great enough to warrant the removal of the operator from the panel area. These areas have been typically operated using remote control stations, but automated technology provides better guidance for tramming than manual remote control. The automation uses scanning lasers mounted fore and aft on the LHD to scan and remember the tunnel profile on a single horizontal section. This allows for greater LHD speeds and less damage from side wall collisions. The buckets still require manual remote loading using cameras, as the muck type and size have been shown to be too variable to allow automation of the bucket-loading process. The isolation of areas due to remote and automated loading can cause productivity issues as all automated equipment is shut down when personnel are present for drawpoint sampling, inspection, secondary breaking, and so on.

Typical Equipment

Production in rubber-tired operations today is fairly standardized around the LHD equipment. These may be electric- or diesel-driven, both having their advantages and disadvantages. Small footprints and herringbone drawpoint designs favor the electric LHD, which also requires significantly less ventilation support. Diesel-driven LHDs have more versatility to move anywhere and can be used for more than production mucking. However, they do require significantly more ventilation. Newer regulations for dust, noise, and diesel exposure are also driving operators to use enclosed, air-conditioned cabs, which reduce some visibility but definitely give the operator a much-improved working environment. Capacities will vary, depending on ore characteristics and drift sizing, but generally range from 8 to 14 t in load capacity for most caving operations. Some of the super caves plan to use 17-t LHDs.

Production haulage is typically by truck or by train. In the case of trucks, most are end-dump designs in the 40–60 t range. These smaller trucks are designed more for development haulage but are being used in a production mode in many mines. PT Freeport Indonesia's DOZ operation employs 55-t-capacity trucks. The Henderson and Andina mines operate 80-t side-dump trucks, which have good haulage characteristics, but only on level ground. They are designed for production haulage only and cannot easily negotiate ramps when loaded.

Trains will be sized according to production requirements. The larger block cave mines using rail will typically use full-size rail equipment with a rail gauge of 1.4 m. For example, the El Teniente operation produced more than 130,000 t/d in 2008 using electric locomotives ranging from 72 to 130 t for ore haulage. Each of the five coarse ore trains hauls 18 bottom-dump railcars at 100 t per car. Another large caving operation plans for trains with two 40-t locomotives and 24 cars at 30 t per car. Rail systems lend themselves very well to automation and, in the right environment, can have much lower operating costs than haul trucks.

Several varieties of long-hole drills exist, and newer computerized versions are available. As with most other equipment, new regulations for operator exposures are pushing operators to use enclosed-cab drills. Undercut drilling requires

these drills to have the capability of 360° rotation. Some drills have the enhanced ability to drill at forward angles from the perpendicular, so that the same machine is able to drill the V-cuts as well as undercut rings. Depending on the method of undercutting (explained in previous sections), other drilling equipment such as blind-bore machines or raise machines will be required for the bell relief slot raise.

In addition to the primary production equipment, a wide range of support equipment is used to develop and support production needs. In direct support are the secondary drills and rock breakers used to deal with hang-ups and oversize. Although these types of equipment may seem of lesser importance at first glance, they are probably among the most valuable tools for maintaining high production. The ability to efficiently clear choked or hung drawpoints has a huge impact on the ability of LHDs to meet draw-control parameters and production requirements. Storage of boulders for secondary blasting not only detracts from a LHD's primary function of production but requires removal of the LHD from mucking operations to load and blast this oversize. A variety of smaller drills and breakers are on the market, but no standardized methodology has arisen as yet to combat these issues. Some particular examples of this style of equipment were described in a previous section.

Also to be considered is the variety of equipment that supports the development and general operation of the mine. In particular, the concrete/shotcrete delivery systems are an integral part of maintaining appropriate development leads and ground support. Shotcrete placers, concrete pumps, drawpoint-forming equipment, and concrete transporters are critical components of the mobile fleet. Concrete/shotcrete batching systems, whether surface or underground, and the support needed for material storage are major considerations in infrastructure design to ensure a smooth operation.

Miscellaneous support equipment for moving workers, materials, and utilities throughout the mine should also be given careful thought. Forklifts, lube/fuel trucks, high-lift trucks for hanging utilities and ventilation fans/ventlines, water trucks for dust suppression, cable trucks for power runs, and service trucks for localized construction should all be understood and planned for. These types of units are as critical as production equipment for keeping the operation working efficiently and minimizing disruption to production.

Support equipment for rail haulage operations typically includes smaller (12-t) locomotives, often diesel-powered so as to be independent from issues with the electrical power supply. Functions handled include shift change, track repair, cleanup under ore chutes, switching and marshalling, electrical and control maintenance, and development support. In addition to the locomotives, a variety of flat cars, man-haul cars, and muck cars are used.

CAVE MANAGEMENT

The ultimate goal of the cave management functions is to safely maximize the recovery of the ore reserves. An effective cave management system is critical to the block cave mine operation and has an active role in virtually all phases of the mine. From the calculation of mining reserves and development of the production forecast, the drawpoint opening schedule is developed. The development drifting, construction, and undercutting plan are then based on that schedule. As caving proceeds, draw orders are developed that will pull the cave evenly to avoid early dilution, while managing issues such as drift convergence, rock bursts, areas of high stress, excessive dilution, water inflows, and the like. The cave must be monitored and plans altered to account for any issues that arise.

Reserves, Reporting, and Forecasts

The initial ore reserves are established during the feasibility and detailed design period, which sets the target production rate and guides the mine layout, design, and development and production schedules. Typically, mine planning and scheduling software is used.

Production forecasting is a critical process in cave management. The metal to be produced from the mine is the ultimate goal for the caving operation, and so as accurate a prediction as possible is required. Forecasting is an ever-evolving process. As mining proceeds and more is learned about how the cave is progressing, it is necessary to modify forecasting assumptions to match predicted to actual metal recovery as closely as possible.

As the ore body is mined, production data is tabulated and reported. This information must be reconciled back to the predicted tons and grade, as well as to the actual tons and grade received by the concentrator facility. These reconciliations provide key data on the modeling of the draw behavior of the cave, which in turn feeds into both the forecasting and reserve models.

Tracking the Progress of the Cave

How the cave progresses has a major impact on what can be expected from the drawpoints and so provides a constantly changing input into the forecasting process. Geotechnical monitoring provides input on the overall geometry of the cave. Geological observations at the drawpoints are reconciled back to the in-situ geological models that determine key parameters for modeling the draw such as the dilution rates and draw cone interaction.

Managing the Cave

When issues such as excessive weight problems or wet muck develop, modification of the draw in the problem areas can often alleviate or eliminate those issues. It is therefore important for cave management to understand and react to those issues. Special draw orders for specific drawpoints might be issued to address those issues. For example, a hung-up drawpoint might be causing excessive weight in a production drift, so orders are implemented to prioritize blasting of that hang-up. Extra tons might be pulled from that drawpoint and surrounding drawpoints for a period of time until the condition is stabilized. For a drawpoint with wet muck, a priority will be placed on pulling ore from that drawpoint, which helps to pull the water from the cave and prevents the water from spreading to additional drawpoints.

Draw Control

Draw control is an essential component in the caving process. Controlling the shape of the cave back and keeping the drawdown of the cave even and controlled will minimize cave issues such as early dilution, accelerated breakthrough to older workings, and excessive air gaps. Proper cave draw will improve the prediction of ore grades for the diluted block model. Draw control is managed by setting daily draw orders for each drawpoint in the mine. A record of draw is held in a database and regularly compared to the plan. Corrections in the short-term draw order are made to keep the draw control

in line with the long-term plan. Systems to monitor draw from each drawpoint can range from a manual form filled out by the operator on an honor system or an electronic tagging system where every drawpoint has an individual tag that registers with a reader on the LHD every time a bucket is taken.

Drawpoint sampling is performed based on tonnage drawn and the HOD. Frequency of sampling is typically increased as a drawpoint nears the end of its life so the drawpoint can be closed as required to prevent overdilution. Sampling involves personnel manually sampling each drawpoint using some form of a sampling grid. Although this statistically tends to be quite a poor sample, it is a good indicator of drawpoint grades and can be compared back to the model to ensure a reasonable correlation.

CAPITAL COST CONSIDERATIONS

Caving operations typically require a significant commitment in up-front capital expenditures, and capital development for a caving mine requires a long lead time. The mine accesses and infrastructure, as well as sufficient portions of the operating levels, have to be in place before caving begins. This is particularly true for the current and planned super caves exceeding 100,000 t/d, Some of the deep deposits may take several years just to establish access and ventilation to the deposit, followed by multiple additional years developing the mine infrastructure. Once production is initiated, the time required to ramp up to full production is significant. As an example, a mine with a planned full production of 100,000 t/d may require 5 years after caving is initiated to reach that objective. Thus for a large caving operation, the lead time from initial access development to full production can be 10–15 years.

Capital estimates are typically divided into the following:

- *Capital development*, which includes all preproduction and production buildup development and infrastructure required to bring the mine to its sustainable full production rate; and
- *Sustaining capital*, work required on long-term development, facilities, or systems to sustain full production operations, but not direct production costs.

A typical breakdown of capital development and sustaining capital components follows. A number of the categories are the same as for the operating cost breakdown; these costs are classified as capital or operating, based on whether or not full production has been achieved.

- Drifting and excavating:
  - Rail haulage, conveyor drift, material handling excavation
  - Ramps and accesses, fixed facilities (shops, lunchroom, etc.)
  - Extraction and undercut fringe drift; production drift, drawpoint drift, drawbell drift development
  - Production and drilling drift on extraction and undercut levels
  - Ventilation drift and accesses, mass excavation, dewatering drifts
  - Shafts, raises, ventilation raises, orepasses, and waste passes
- Preproduction:
  - Ground support
  - Drawbelling and caving
- Ground repair (drawpoint rebuild and repair, and production drift repair)
- Indirects (costs not directly allocated to production, such as ventilation, electrical, engineering, water control, and maintenance)
- Equipment and infrastructure:
  - Rail or truck haulage, or other ore-flow infrastructure
  - Drawpoint construction
  - Mobile equipment
  - Power generation
  - Fixed facilities (ventilation, pumping, batch plant, shops, offices, warehouses, compressors, etc.)

Table 13.10-1 Development capital requirements for an 80-kt/d block cave mine (in 2009 U.S. dollars)

Capital Outflows	$, in millions
Drifting and excavating	260
Preproduction	10
Ground repair	10
Indirects	
Technical services and management	50
Electrical (including power cost)	130
Safety and miscellaneous	30
Mechanical shop (including fuel cost and rebuild)	170
Equipment and infrastructure	
Haulage and ore-flow infrastructure	190
Mobile equipment	250
Power generation and distribution	100
Fixed facilities infrastructure	100
Total development capital cash outflows	1,300

Table 13.10-1 presents a generic example of development capital costs for an 80,000-t/d block caving operation to illustrate the distribution of costs by major category. These expenditures occur over about a 10-year period from initial access development through to full production.

OPERATING COST CONSIDERATIONS

Block caving is typically the lowest-cost underground mining method. This is principally because the amount of drilling and blasting per metric ton of ore is much less than with any other system. Once the cave is progressing, blasting is limited to secondary breakage of oversize material within a drawpoint. In addition, because permanent production levels are established from which all ore tonnage is mined, total development drifting per metric ton of ore is substantially less than for other methods. Block caving lends itself to a high degree of mechanization, both in the production areas and in the ore-flow infrastructure. The overall result is lower costs due to reduced requirements for labor, mobile equipment, and consumables.

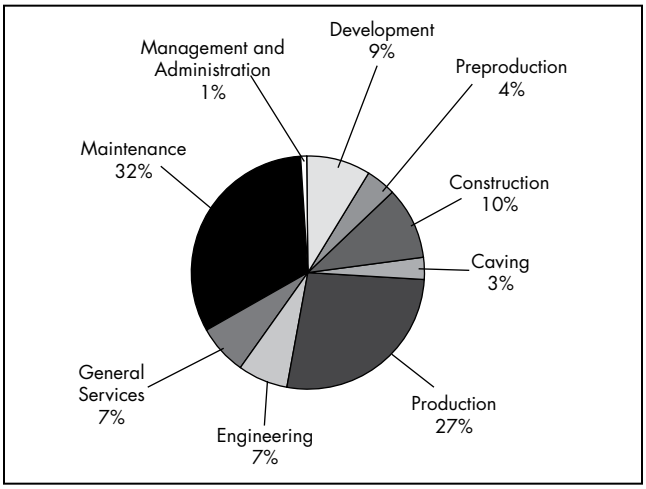
Table 13.10-2 is an example from an 80,000-t/d block cave mine operating in 2009, to illustrate the general distribution of costs by category. Of course, costs can vary widely from mine to mine because of many factors such as country of operation, ground conditions, infrastructure requirements, maturity of the mine, depth, and so on.

LABOR

Block caving mines can be highly mechanized and efficient operations, commonly achieving more tonnage per worker-shift than any other underground method, and potentially producing many times more tonnage per worker-shift. The initial labor required to develop a caving mine can be relatively high

Table 13.10-2 Annual operating costs breakdown for an 80-kt/d mine (in 2009 U.S. dollars)

Category	Direct Cost, $	
	in millions	per metric ton
Production	10	0.36
Development	16	0.59
Raises	5	0.17
Preproduction	12	0.43
Ground repair	3	0.11
Mine support services		
Drilling and water control	2	0.09
Underground technical services	9	0.31
Underground electrical	5	0.18
Management and administration	2	0.07
General services	3	0.09
Maintenance	49	1.80
Underground safety	5	0.17
Other	3	0.13
Construction	22	0.80
Total direct cost	145	5.30



Source: Casten 2005.

Figure 13.10-11 Distribution of labor by category at the DOZ mine producing 50 kt/d

and is a factor in the early, high-capital expenditures typically required to bring a block cave into production. As major infrastructure items such as crushers, fixed facilities, ventilation, and ore handling systems are completed, labor levels will reduce down to a set of relatively specialized crews that focus on the repetitive activities required to achieve and maintain full production. These crews typically consist of (a) level development, (b) preproduction support, (c) construction, (d) caving, and (e) production. In addition, required support groups include management, engineering, geology, geotechnical, mine services, logistics, maintenance, and electrical crews. Most of these skill sets are common to most underground mining methods.

In a caving operation, the control and interaction of these different crews in a fairly constrained area can be a challenging task requiring clear management, good engineering, and seasoned supervision. Each of those key activities listed above requires careful scheduling to prevent the various crews from obstructing each other. In addition, the crews must work around interruptions such as scheduled and unscheduled blasting and shifting priorities to ensure that the cave is propagating well and being drawn correctly.

As the mine life progresses, activities such as development, construction, and undercutting will be completed before the mine is closed. Thus the number of different crews begins to decrease until finally the mine will be left with only production activities. In the case of small-footprint caves, all of the preproduction activities are managed by contractors, and production crews are quickly the only set of crews that remain. Northparkes Lift 1 was a good example of this with only 63 employees involved in production at 22,000 t/d as the mine had been fully constructed. In the case of the much larger-footprint DOZ mine, where development and preproduction activities continue on an actively producing and expanding cave front, labor levels remain higher for longer periods with approximately 1,800 personnel planned to produce at 80,000 t/d with only about one-half of the ore body developed. Figure 13.10-11 shows the labor distribution by category for the DOZ mine in 2005, when the mine was ramping up to a production rate of 50,000 t/d.

RISK CONSIDERATIONS

This section presents a review of typical risks in the block caving process. The first part discusses risks that present major operational hazards, and the second part discusses risks to consider when designing or operating a caving mine. A detailed discussion of the risk assessment process is beyond the scope of this chapter, but an informative section on operational hazards and risk assessment for block caving can be found in Chapters 10 and 11 of *Block Caving Geomechanics* (Brown 2003).

As with any mining project, a detailed risk assessment is required. Typically a *qualitative* risk analysis is initially completed to identify potential risks to the project, assess the likelihood and consequences of those risks, and develop an action plan to mitigate or eliminate the identified risks. A *quantitative* risk assessment, where a cost is calculated for each risk should its consequences be realized, could then be completed.

Major Operational Hazards

Major operational hazards that could result in loss of life or premature mine closure and are associated with caving methods include (1) issues with "inrushes" such as air blasts, mud rushes, or water; and (2) issues with stability of the underground workings, such as rock bursts or major excavation collapses.

Inrushes

An *air blast* may occur when the draw rates exceed the cave propagation rate, so that an air gap develops in the cave (prior to propagation of the cave to surface). Should the cave back then experience a large enough failure into the air gap, the compression of the air in that confined space can produce a rapid flow of air through exposed underground openings and can produce a violent and damaging air shock wave.

A *mud rush* is a sudden inflow of saturated fines from the drawpoint or other underground openings. A mud rush can occur when water builds up within caved material that contains sufficient potentially mud-producing materials. The inrush is then triggered by a mechanism such as production from the drawpoint, blasting, or seismicity. The sudden inflow puts at risk the personnel in the path of the mud rush. Mud rushes are a fact of life in caves with the appropriate conditions and must

be managed to mitigate the risk to the people and to the operation. For example, automated or remote-controlled loaders can be used to produce from the wet areas. A key is to attempt to keep the material moving in the wet areas to minimize the buildup of the wet material and prevent the event from spreading further into the cave.

An external mud rush might occur when the cave breaks up into a saturated zone of material such as a tailings or slimes retention area, or perhaps a backfilled zone from an upper lift of the mine. Tailings or slimes ponds should never be sited over caving areas as catastrophic failures and inrushes can and have occurred that can claim hundreds of lives (e.g., West Driefontein in South Africa and Mufulira in Zambia).

Underground Stability Issues

A *rock burst* is an underground seismic event that, depending on its scale, can cause violent and significant damage to excavations in the mine. Rock bursts are the result of stored strain energy that is released suddenly into the rock mass. The induced stresses are high enough to overcome the strength of the rock mass and the released energy cannot be absorbed in slips or fractures. Rock bursts will usually occur only in strong, brittle rocks such as quartzite or granite. Effects include violent fracture and expulsion of rock from and near the boundaries of excavations, and the full or partial closure of mining excavations. Damage can range from localized spalling in a drift to complete destruction and closure of a part of the mine.

Current and planned caving mines are tending toward deeper, higher stress conditions, and in stronger and more brittle rock. Thus it is expected that the risk of rock bursts in future caving mines is an issue that will have to be addressed.

An additional type of collapse is stress-induced, but unlike a traditional collapse, it is generally progressive rather than sudden and violent. Collapses of this type are typically caused by the stress abutment produced ahead of the undercut; this is sometimes referred to as "weight" problems. This type of collapse involves failure of the pillars left on and above the extraction level. It may take weeks or months to develop and once initiated is difficult to arrest, although some mines have had success by overpulling from drawpoints within the stressed areas. Such collapses are exacerbated by the presence of faults or other discontinuities in the rock mass, by a change in rock type, or by incomplete blasting in the undercut level that leaves a "stump" or remnant that puts excessive stress (point loading) on the major or minor apex.

Major Risk Categories

The following subsections address major risk categories that must be considered during the planning and operation phases of a caving mine.

Geotechnical Issues

Most block caving operations, which tend to be large in size and occur at considerable depths below the surface, are difficult to explore completely in advance of having to make a development decision. Virtually all aspects of the mine design—from level designs, ground support criteria, ore-flow systems, equipment fleets, and so on—depend on a design that correctly incorporates the geological and geotechnical aspects of that deposit. Therefore, a considerable number of risks are present in the category of geotechnical information. A few of the potential risks include

- Cavability—An assumption that the ore will cave completely and in a predictable manner.
- Fragmentation—Underestimation of the fineness of the fragmentation that causes the drawpoint spacing to be too large, leading to isolated draw and loss of reserves. Underestimating the coarseness of the fragmentation can cause excessive hang-ups in the drawpoint, leading to excessive secondary blasting, and creating a bottleneck in an ore-flow system that was designed for fine fragmentation.
- Caving performance
  - **In-situ stress:** Underestimation of the stress levels encountered leads to inadequacies in ground support, mass excavation designs, and undercut sequencing.
  - **Undercut sequence design:** The geometry of the panels and the sequence of the undercutting are key components to the caving performance. Should the chosen design prove to be unstable, large portions of the production area could be lost and ground repair would be extensive, leading to lost or delayed production.
  - **Caving rate:** If lower than expected, ramp-up will be delayed.
  - **Cave shape:** If the developing cave fails to propagate as expected, excessive dilution could be introduced earlier than anticipated.
- Excavation stability
  - Panel excavations may be designed too large in areas of unanticipated poor ground conditions, resulting in too large a drawcone spacing and loss of reserves. The poor ground also leads to weaker pillars and excessive ground repair costs.
  - Location of the major underground fixed facilities relies on identifying adequate areas of good ground conditions in which to locate those facilities.
  - Orepasses are at risk as the cave front passes and must be designed to handle the abutment stresses.

Other Issues

The following are additional risk categories to be considered during planning and operation of a caving mine:

- **Fixed facilities design.** Infrastructure required to deliver workers and materials to the mine must be adequate for the production rates required. The ore-flow system must be capable of adequate ore delivery to the mill.
- **Environmental impacts and permitting.** If surface impacts exceed those anticipated, it could have an impact on the mine life or production capacity. For example, larger-than-expected subsidence areas that affect the surrounding area, or development of acid rock drainage exceeding that permitted, and so on.
- **Procurement.** Lead times for procurement of major components can be lengthy, leading to delays in mine development and production if not managed properly.
- **Labor.** Many future major block cave mines are scheduled to be developed and operated in similar time frames in the next few decades. Experienced labor, both in engineering and operations, may be limited when needed.
- **Safety.** For new mines, various aspects of the operation will comprise new types of facilities unfamiliar to the personnel, and will require training, familiarization, and supervision to minimize hazard exposure.

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PART 14

Mineral Processing

CHAPTER 14.1

Introduction to Mineral Processing

Maurice C. Fuerstenau and Kenneth N. Han

The term *mineral processing* is used in a broad sense throughout this chapter to analyze and describe the unit operations involved in upgrading and recovering minerals or metals from ores. The field of mineral processing is based on many fields of science and engineering. In addition, environmental science and engineering have become inseparable components; the steps involved in mineral processing have to be founded not only on sound scientific and technological bases, but on environmentally acceptable grounds as well.

GOALS AND BASICS OF MINERAL PROCESSING

In the traditional sense, mineral processing is regarded as the processing of ores or other materials to yield concentrated products. Most of the processes involve physical concentration procedures during which the chemical nature of the mineral(s) in question does not change. In hydrometallurgical processing, however, chemical reactions invariably occur; these systems are operated at ambient or elevated temperatures depending on the kinetics of the processes.

The ultimate goal in the production of metals is to yield metals in their purest form. Mineral processing plays an integral part in achieving this objective. Figure 14.1-1 shows a generalized flow diagram for metals extraction from mining (step 1) through chemical processing. Steps 2 and 3 involve physical processing, and steps 5 and 7 involve low-temperature chemical processing (hydrometallurgy). All four steps are considered part of mineral processing. High-temperature smelting and refining (pyrometallurgy), steps 4 and 6, are not included under the heading of mineral processing.

Table 14.1-1 specifies processing routes from ore to pure metal for a number of metals. Processing routes can be quite different, and more than one route may be possible for many of these metals. For example, in the extraction of copper or gold from low-grade ores, dump or heap leaching is commonly practiced. The choice of this leaching practice is frequently driven by the overall economics of the operation. Because crushing and grinding of ores are quite expensive, leaching of ores in large sizes is attractive compared to the leaching of finely ground ores, even though the overall recovery of metals from the leaching of fine particles is, in general, much greater than that obtained with large particles. The introduction of this innovative leaching process has made feasible the mining of many mineral deposits that could not be processed economically through conventional technologies.

METALLURGICAL EFFICIENCY

One of the most important and basic concepts in mineral processing is metallurgical efficiency. Two terms are commonly used to describe the efficiency of metallurgical processes: *recovery* and *grade*. These phenomena are illustrated in the generalized process presented in Figure 14.1-2. In this example, 100 t/h (metric tons per hour) of ore are being fed into a concentration operation that produces 4.5 t/h of concentrate and 95.5 t/h of tailings. In upgrading this process, then, 1.0 t/h of the desired material, A, is introduced into the unit operation and 0.9 t/h (4.5×0.2) of this material reports to the concentrate, resulting in 90% recovery ($0.9/1.0 \times 100$). The grade of the mineral, A, has been improved from 1% to 20%. The term *percent recovery* refers to the percentage of the valuable material reporting to the concentrate with reference to the amount of this material in the feed. Sometimes obtaining the highest possible recovery is not necessarily the best approach in a concentration process. High recovery without acceptable grade will lead to an unsalable product and is therefore unsatisfactory.

Mineral processing engineers are responsible for optimizing processes to yield the highest possible recovery with acceptable purity (grade) for the buyers or engineers who will treat this concentrate further to extract the metal values. To achieve this goal, economic assessments of all possible technological alternatives must be conducted.

ECONOMIC CONCERNS

Table 14.1-2 summarizes the total U.S. supply and recycled supply of selected metals in 1996. Although the data shown here are historic (reprinted from an earlier publication), the trends are still current. The total supply of iron and steel includes supply from primary and secondary sources as well as imports; these two metals represent by far the largest of commodities produced and consumed, followed by aluminum,

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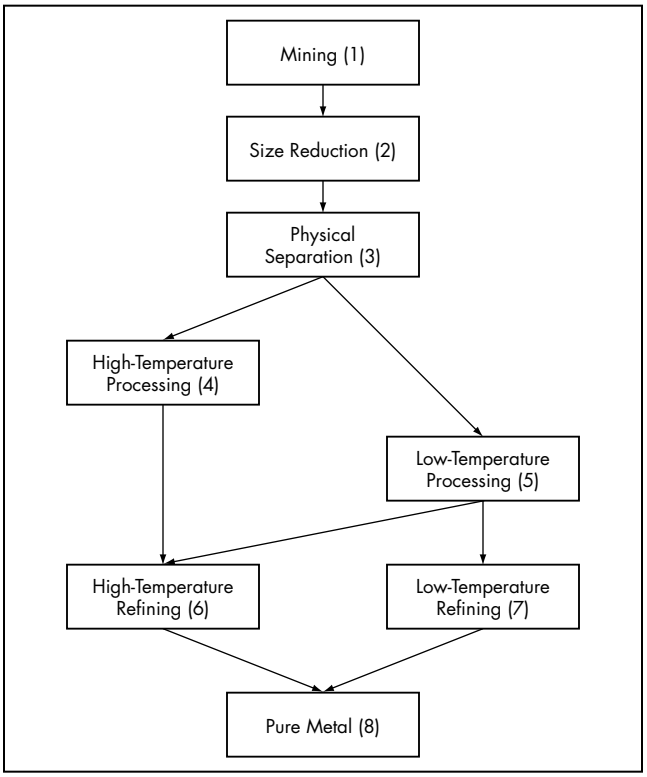


Figure 14.1-1 Generalized flow chart of extraction of metals

Table 14.1-1 Processing sequences for selected metals

Metal	Associated Major Minerals	Steps Involved in the Processing Route (Figure 14.1-1)							
		1	2	3	4	5	6	7	8
Iron	Hematite, Fe_2O_3; magnetite, Fe_3O_4	x	x	x	x		x		x
Aluminum	Gibbsite, Al_2O_3-$3H_2O$; diaspore, $Al2_2O_3$×H_2O	x	x			x	x		x
Copper	Chalcopyrite, $CuFeS_2$; chalcocite, Cu_2S	x	x	x	x	x		x	x
Zinc	Sphalerite, ZnS	x	x	x	x		x		x
		x	x	x	x	x		x	x
Lead	Galena, PbS	x	x	x	x		x		x
Gold	Native gold, Au	x	x	x		x	x		x
		x	x*			x	x		x
Platinum	Native platinum, Pt; platinum sulfides	x	x	x		x		x	x
Silver	Native silver, Ag	x	x	x		x	x		x

*Only crushing is practiced; grinding is usually omitted.

copper, and lead. The recycled supply of these metals from processing scrap is strikingly high. In addition, the tonnage of precious metals consumed is rather small. However, because of the high prices of precious metals, their monetary value is substantial. For example, the monetary value of 516 t (metric tons) of gold was $12.8 billion in 1996, compared to $10.7 billion for 5.3 million t of copper and lead.

Table 14.1-3 lists the relative abundance of various metals in the earth's crust. Most metals are present in extremely small concentrations in nature, and none of these metals can

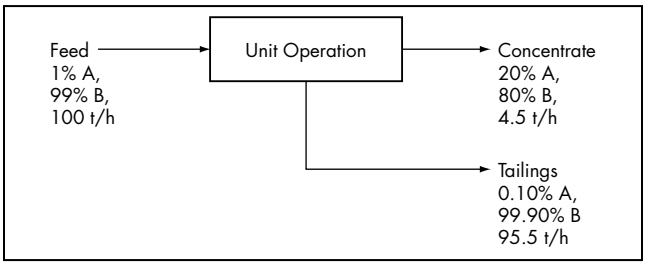


Figure 14.1-2 Simple material balance for a unit operation

Table 14.1-2 U.S. total and recycled supply of selected metals in 1996

Metal	Total Supply, million t metal content	Recycled Supply, million t metal content	% Recycled
Iron and steel	183	72	39
Aluminum	8.34	3.29	39
Copper	3.70	1.30	35.1
Lead	1.63	1.09	66.8
Zinc	1.45	0.379	26.1
Chromium	0.48	0.098	20.5
Magnesium	0.205	0.0709	35
Gold	516 t*	150 t*	29

Source: USBM 1997.
*Value for 1995.

Table 14.1-3 Abundance of various elements in the earth's crust compared to annual U.S. consumption

Element	Relative Abundance, %	U.S. Consumption, stpy
Fe	5.00	1.28×10^8
Al	8.13	5.4×10^6
Cu	7×10^{-3}	2.3×10^6
Zn	8×10^{-3}	1.0×10^5
Pb	1.5×10^{-3}	1.2×10^6
Au	1.0×10^{-7}	113
Ag	2.0×10^{-6}	4.52×10^3

Source: USBM 1990.

be recovered economically at these concentrations. Rock that contains metals at these concentrations is not ore; ore is rock that can be processed at a profit. An average copper ore, for example, may contain 0.3% to 0.5% copper. Even this material cannot be treated economically at high temperature without prior concentration. There is no way that rock containing 4.5 kg (10 lb) of copper and 903 kg (1,990 lb) of valueless material can be heated to 1,300°C (2,372°F) and treated to recover this quantity of metal economically. Concentrating the ore by froth flotation to approximately 25% or more copper results in a product that can be smelted and refined profitably.

UNIT OPERATIONS

Numerous steps, called unit operations, are involved in achieving the goal of extracting minerals and metals from ores in their purest possible form. These steps include size reduction, size separation, concentration (froth flotation, gravity concentration, and magnetic and electrostatic concentration), dewatering, and aqueous dissolution.

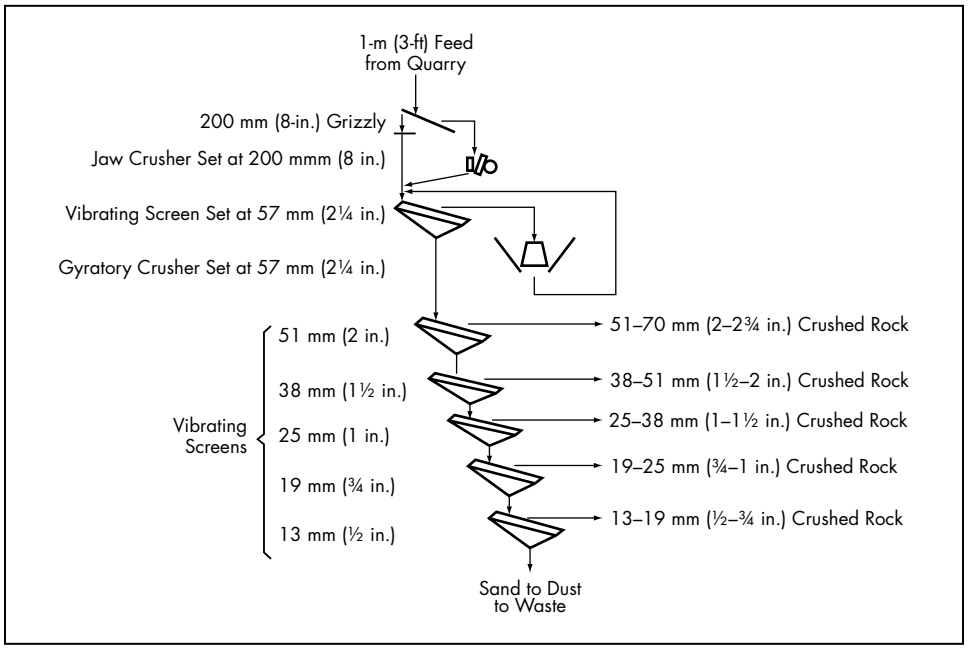


Figure 14.1-3 Flow sheet for crushing and grading rock

1. **Size Reduction.** The process of crushing and grinding ores is known as *comminution*. The purpose of the comminution process is threefold: to liberate valuable minerals from the ore matrix, to increase surface area for high reactivity, and to facilitate the transport of ore particles between unit operations.
2. **Size separation.** Crushed and ground products generally require classification by particle size. Sizing can be accomplished by using classifiers, screens, or water elutriators. Screens are used for coarse particulate sizing; cyclones are used with fine particulates.
3. **Concentration.** Physicochemical properties of minerals and other solids are used in concentration operations. Froth flotation, gravity concentration, and magnetic and electrostatic concentration are used extensively in the industry.
   - **Froth flotation.** The surface properties of minerals (composition and electrical charge) are used in combination with collectors, which are heterogeneous compounds containing a polar component and a nonpolar component for selective separations of minerals. The nonpolar hydrocarbon chain provides hydrophobicity to the mineral after adsorption of the polar portion of the collector on the surface.
   - **Gravity concentration.** Differences in the density of minerals are used to effect separations of one mineral from another. Equipment available includes jigs, shaking tables, and spirals. Heavy medium is also used to facilitate separation of heavy minerals from light minerals.
   - **Magnetic and electrostatic concentration.** Differences in magnetic susceptibility and electrical conductivity of minerals are utilized in processing operations when applicable.
4. **Dewatering.** Most mineral processing operations are conducted in the presence of water. Solids must be separated from water for metal production. This is accomplished with thickeners and filters.
5. **Aqueous dissolution.** Many metals are recovered from ores by dissolving the desired metal(s)—in a process termed *leaching*—with various lixiviants in the presence of oxygen. Following leaching, the dissolved metals can be concentrated by carbon adsorption, ion exchange, or solvent extraction. Purified and concentrated metals may be recovered from solution with a number of reduction techniques, including cementation and electrowinning.

EXAMPLES OF MINERAL PROCESSING OPERATIONS

Figure 14.1-3 shows a typical flow sheet for crushing and sizing rock in a quarrying operation. Run-of-mine ore can be present as lumps as large as 1.5 m (5 ft) in diameter. In this figure's example, 1-m (3-ft) lumps of rock are fed to a crusher that reduces the material to 200 mm (8 in.) or less in diameter. After screening to remove rock that is less than 57 mm (2¼ in.) in size, rock between the sizes of 57 mm (2¼ in.) and 200 mm (8 in.) is further reduced in size by a gyratory crusher. The product from this step is then classified by screening to the desired product for sale.

Figure 14.1-4 shows an integrated circuit demonstrating crushing, grinding, size separation, and gravity concentration of a tin ore. Initial size separation is effected with a grizzly set at 38 mm (1½-in.). Oversize material is fed to a jaw crusher set at 38 mm (1½-in.), and the crushed product is then further reduced in size to 20 mesh by ball milling. The minus-20-mesh material is classified by hydrocyclones set at 150 mesh, and the minus-150-mesh material is sent to shaking tables to concentrate the heavy tin mineral, cassiterite. The middlings in this process receive additional treatment. The concentrate from this operation is reground and sized at 200 mesh. Two-stage vanning is used to produce a fine tin concentrate.

The flow sheet describing the flotation processing of a copper ore containing chalcopyrite and molybdenite is shown

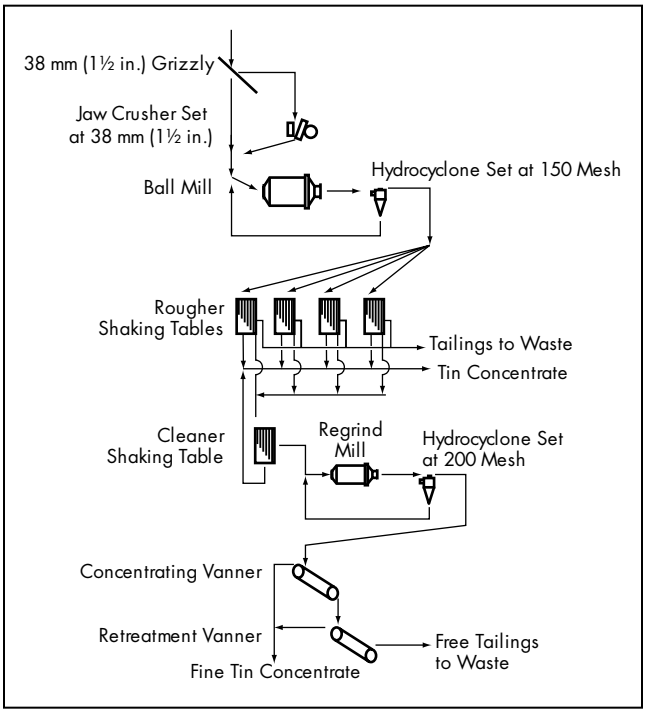


Figure 14.1-4 Flow sheet for the gravity concentration of a tin ore

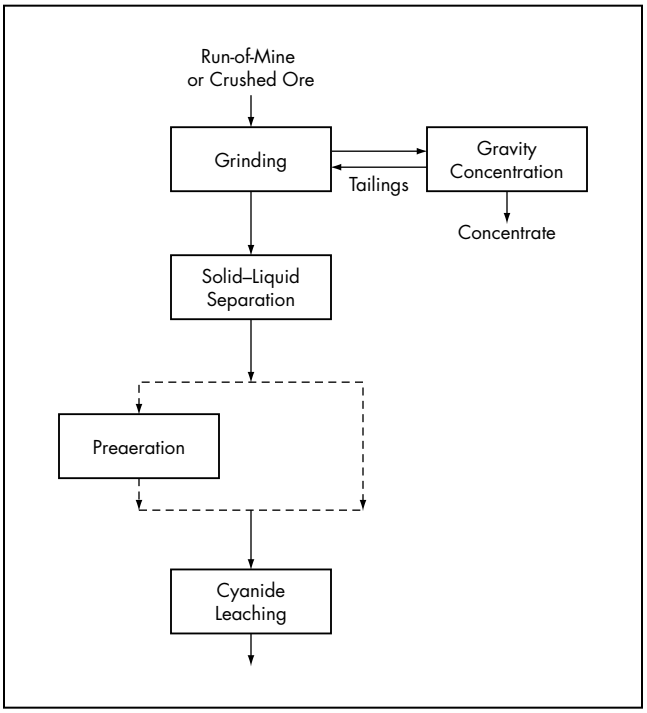


Figure 14.1-6 Flow sheet options for grinding and agitated leaching of free-milling oxidized gold ores

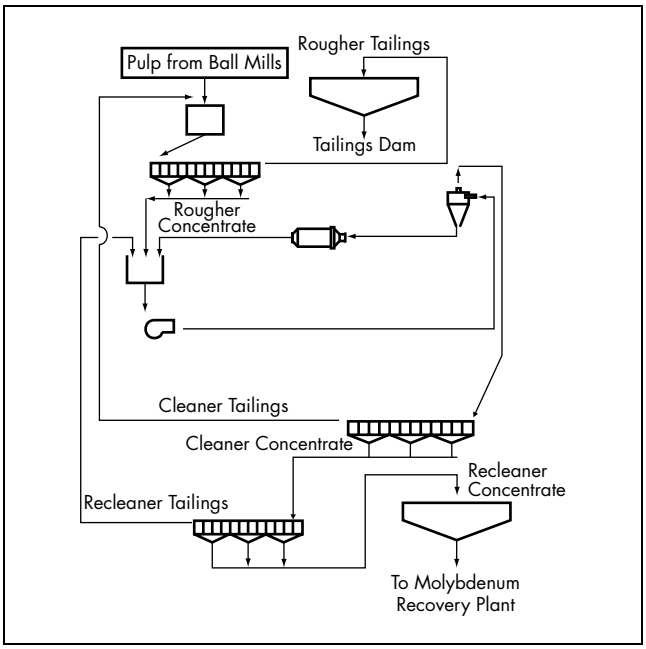


Figure 14.1-5 Flow sheet for the flotation of copper sulfide ore

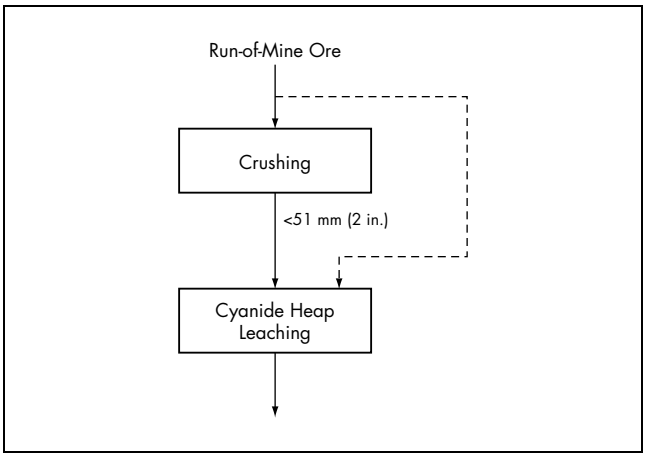


Figure 14.1-7 Flow sheet for heap leaching of oxidized gold ores

in Figure 14.1-5. After grinding and classification, pulp is fed to rougher flotation. The rougher tailings are thickened and sent to a tailings dam. The rougher concentrate is classified, and the oversize is reground. Cyclone overflow is fed to cleaner flotation, and the cleaner concentrate is recleaned. Cleaner tailings are recycled back to rougher flotation, and the recleaner concentrate is thickened and sent to the molybdenum recovery plant for further processing. In this operation, the feed contains 0.32% Cu and 0.03% Mo. Rougher concentrate, cleaner concentrate, and recleaner concentrate contain 7% to 9% Cu, 18% Cu, and 25% Cu, respectively. Recleaner concentrate also contains 2% to 3% Mo.

Figure 14.1-6 depicts a flow sheet for processing free-milling oxidized gold ore. The kinetics of gold leaching is slow, and gold ores are frequently ground to less than about 76 mm (3 in.) before leaching. Even then, one day is usually required in the leaching step. In this process, run-of-mine ore is crushed and ground. The ball mill discharge is subjected to gravity concentration to recover the larger particles of free gold. The tailings from this operation are thickened, and the underflow from the thickeners is then subjected to cyanide

leaching. In some instances, ores may contain oxygen-consuming minerals, such as pyrrhotite and marcasite, and a preaeration step may be conducted ahead of cyanide leaching.

Heap leaching has revolutionized the gold mining industry. Low-grade oxidized ores containing approximately 0.933 g gold per metric ton (0.03 oz gold per short ton) of ore can be processed with this technology, whereas they could not be processed by the higher cost of the grinding/agitation leaching (milling) process. Figure 14.1-7 presents a simplified flow sheet of heap leaching. As the figure shows, run-of-mine ore may or may not be crushed. If crushing is done, the ore is generally crushed to <51 mm (<2 in.) in diameter.

ACKNOWLEDGMENTS

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CHAPTER 14.2

Crushing, Milling, and Grinding

John Mosher

Comminution is the collective term used to describe the progressive reduction in size of run-of-mine (ROM) ore. Size reduction is required either to prepare the ore for market (for iron ore and coal, for example) or, as in the case of base and precious metals, to allow liberation and separation of the valuable constituents. For ore requiring beneficiation, liberation and separation are the essential steps in mineral processing. The degree of liberation attained by comminution defines the grade–recovery curve for a given beneficiation process, as well as typically representing the largest category of mineral processing capital and operating expenses. As such, the comminution process has a large impact on an operation's bottom line. Proper circuit design and operation are critical to overall project success.

Crushing and grinding have been regarded as separate processes, with each step performed on a certain size range of material. This concept is rooted in the historical conventional circuit that entailed staged crushing, followed by open-circuit rod milling and closed-circuit ball milling. Modern equipment has blurred this concept, with autogenous grinding/semiautogenous grinding (AG/SAG) mills and high-pressure grinding rolls (HPGRs), in particular, spanning a broad range of comminution applications and particle size ranges.

This chapter addresses in general terms comminution circuit configuration, equipment types, and operation. Stages of comminution are discussed from ROM through fine grinding, with sections devoted to special grinding applications, ore characterization and circuit design, and mill liners and media. No attempt is made to cover all comminution machines or flow sheets; however, methods processing the largest tonnages of ore in the metals mining segment are summarized. Selected unit operations in broad use for each stage of comminution are presented in Table 14.2-1, with size ranges for the most commonly used unit operations depicted in Figure 14.2-1. Unit operations are discussed in more detail in relevant sections.

Examples of selected circuit configurations are presented in Table 14.2-2. This selection is not exhaustive; the summary presents both a range of grinding applications and demonstrates examples of selected circuit configurations. Throughout this chapter, commonly used references to describe feed and product sizes are employed; F and P represent feed and product, respectively, with the subscript indicating the percentage of the material passing a given size. For example, a P_{80} of 13 mm describes a size distribution for a material that has 80% of the material finer than 13 mm.

Individual comminution unit operations are well described in contemporary mineral processing textbooks (Wills 2006; Kelly and Spottiswood 1995). Classic works such as Taggart (1945) are still useful and provide perspective, and a number of comminution monographs provide detailed references (including Mular and Jergensen 1982; Napier-Munn et al. 1996). To keep up with changing practices, various forums addressing academic, engineering, and operational aspects of comminution were developed. Perhaps the most focused of these forums have been the International Autogenous and Semiautogenous Grinding Technology Conferences, hosted by the University of British Columbia. Proceedings from these conferences (1989, 1996, 2001, and 2006) are the broadest reference body for case studies on contemporary milling practices. Despite the name, a broad spectrum of mineral processing comminution circuits is addressed.

MINING AND PRIMARY CRUSHING

Because the process of comminution starts with mining, the degree of comminution achieved should be used as a performance indicator for the mining process. Correspondingly, the comminution circuit should also be developed in consideration of the selected mining process.

For surface mines that produce broken ore or muck via drilling and blasting, there are numerous case studies of optimizing drilling and blasting in conjunction with the overall economics of shovel productivity, primary crusher throughput, and downstream comminution efficiency. Efficient blasting is effective at both decreasing top size as well as improved fines generation. In general, the following conclusions can be drawn:

- Staggered patterns (versus square patterns) reduce the maximum distance between drill holes and typically reduces the ROM top size (boulders).

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Table 14.2-1 Unit operations for stages of comminution

Milling Stage	Typical Top Size	Common Unit Operations	Alternatives
ROM	F_{100}* = 1 m	• Primary crusher (gyratory/jaw) • Secondary pre-crush	• MMD Sizer • McLanahan log washer
Primary	F_{100} = 250 mm F_{80} = 50–100 mm	• AG/SAG/pebble crusher circuits • 2º/3º/4º crushing circuits • Crusher/HPGR circuits	• Rod milling • Impact mills • Dry AG/SAG
Secondary	F_{100} = 13 mm	• Ball milling • Pebble mills	• Attrition scrubbing • Dry ball mills
Tertiary (concentrate regrind, handling, pelletizing)	Nominally –0.2 mm	• Ball mill • HPGR • Vertical shaft mills (screw and pin) • Horizontal shaft mills (IsaMill)	• Roller mills • Vibratory ball mills • Centrifugal mills • Jet mills

*F = feed; subscript represents the percentage of the material passing a given size.

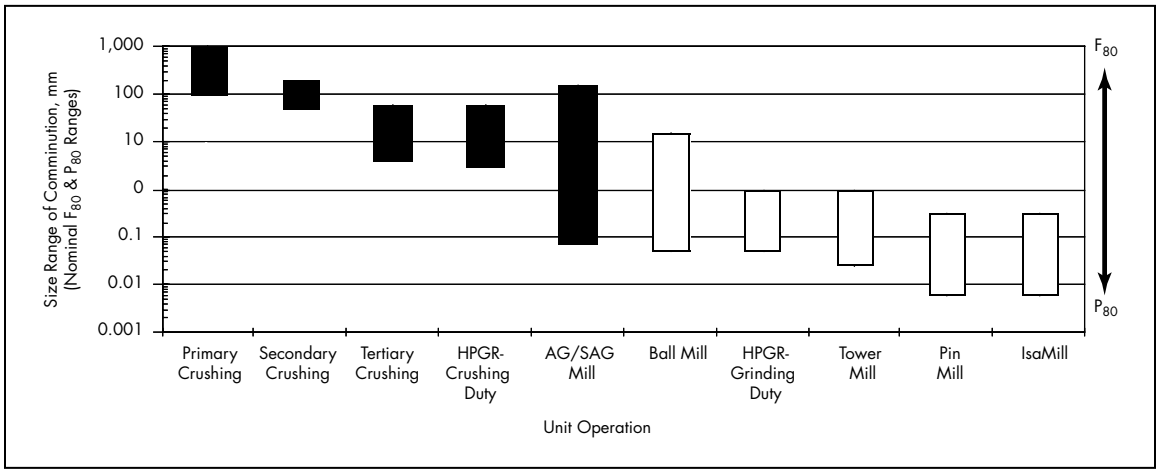


Figure 14.2-1 Size range of common comminution machines

- Quality of drilling in terms of the accuracy of holes in x-, y-, and z-axes improves fragmentation as well as shovel productivity.
- Customized blast designs by material type (burden and spacing, explosive type and amount, and initiation timings) can improve fragmentation and offer cost benefits.
- The effect of blast sequence (including presplitting, production, and trim blasting, where required for wall control), blast size, and pattern geometry should be considered for the effect on comminution.

Additional considerations for underground mines apply. As a consequence of mining through areas previously subjected to ground control, many underground mines produce extraneous metal in the mill feed. Additionally, block cave mines typically produce coarser material at drawpoints early in their development. With higher draw columns, greater AG breakage occurs, and drawn material becomes finer. Block cave mines usually employ secondary breakage to control material hang-ups in drawpoints, as well as mechanical rock breakers and grizzlies to ensure top size control prior to truck loading, before primary crushing is undertaken.

Typically, ROM material must undergo a stage of size reduction to facilitate conveyor transport. Questions to be addressed in circuit development include the size, number, and type of crusher used to produce suitable mill feed, where to place the crushers, and how to feed them. The choice of primary crushing equipment is typically between jaw and gyratory crushers. Jaw crushers can be selected for applications with low throughput, while throughputs greater than 1,000 t/h (1,100 stph) favor gyratory crushers. Gyratory crushers of up to 1.9 × 3.2 m (75 × 124 in.) and installed power of 1 MW or more are currently being marketed. A modern gyratory crusher is depicted in Figure 14.2-2.

The location of primary crushing stations is a critical question. Primary crushers can be permanently located, semimobile, or mobile. Even permanently located crushers can be disassembled and relocated during a mine's life, so the question as to which of the three options to select is one of tradeoffs between the capital and operating cost of the incremental truck haulage fleet required versus mobile or semimobile crushing stations.

Fixed plants simply rely on the haulage fleet to bring ore to the crusher. Mobile and semimobile plants often used tracked crawlers, which have capacities of more than 1,000 t (1,100 st), to periodically relocate the crushing unit. In conjunction with movable conveyors, such systems are placed to shift the haulage burden from a mobile equipment fleet to conveyors and more economically transport material mined to either ore or overburden stockpiles. The evolution of larger truck sizes, along with a general trend of steeper and deeper pits, have tended to disfavor in-pit crushing and conveying. However, these can offer advantages in terms of reduced operating costs, increased availability, and reduced labor requirements compared to a truck fleet; a thorough review of the topic is provided by Boyd and Utley (2002).

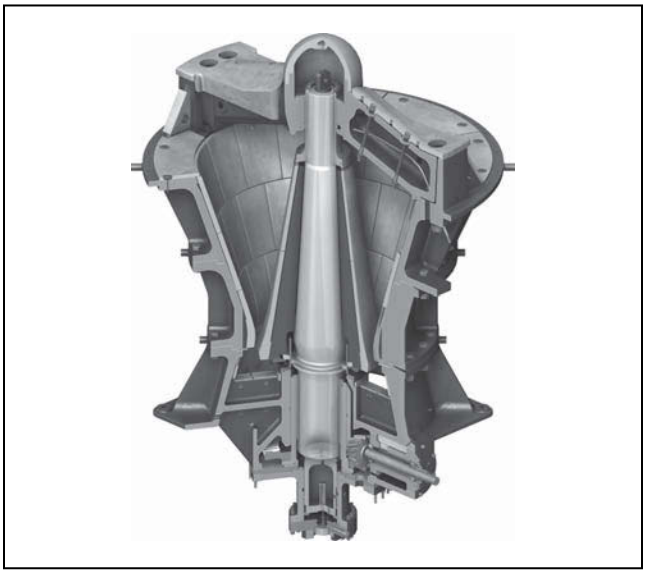
Table 14.2-2 Selected contemporary circuit configurations (all circuits milling ROM ore)

Configuration	Operation*	Major Equipment, Dimensions and Installed Power, kW	Nominal Capacity, t/d	Total Installed Power, MW	P_{80}, μm
Crushing–milling	C1-2, PT Freeport Indonesia; Cu-Au (Indonesia)	Secondary crushing: 2 × 750; Tertiary crushing: 8 × 600; Quaternary crushing, HPGR: 2.0 × 1.5 m, 2 × 3,600/each; ball mills, 8 × 1,950	75,000	29	210
Crushing–dry milling†	Goldstrike roaster; Au (Nevada, USA)	Secondary crushing: 2 × 600; Double-rotator mills: 2 × 7,500	11,000	15.6 (excluding 5-MW air sweep fans)	74
Single-stage autogenous milling‡	Iron Ore Company of Canada, Carol Concentrator; Fe (Labrador, Canada)	One grinding line: AG mill, 9.8 × 4.2 m, 1 × 5,225 (closed with screens)	37,000	5.2	1,000
Single-stage SAG§	Henderson; Mo (Colorado, USA)	SAG mill, 8.5 × 4.3 m, 1 × 5,225 (closed with hydrocyclones)	7,500	5.2	105
AG pebble milling**	Empire IV; Fe (Michigan, USA)	AG mill, 9.8 × 5.0 m, 3 × 6,720; pebble crusher, 1 × 300; HPGR, 1.4 × 0.8, 1,400; pebble mills, 4.7 m × 9.9 m, 6 × 1,975	26,000	34	20
SAG ball milling	C4, PT Freeport; Cu-Au (Indonesia)	SAG mill, 11.1 × 6.1 m, 1 × 20,000; pebble crusher, 2 × 750; ball mills, 7.3 × 9.3 m, 4 × 10,400	120,000	64	175
HPGR ball milling††	Cerro Verde; Cu-Mo (Peru)	HPGRs, 2.4 × 1.6 m, 4 × 5000; secondary crushers, 4 × 750; ball mills, 4 × 7.3 × 11.4 m, 12,000, ABB VS drive	107,000	80	140

*For plants with multiple lines; data is by plant, concentrator, or individual grinding line (as indicated).
†Thomas et al. (2001).
‡Chung et al. (2001).
§Wood (1996).
**Greenwood and Rajala (1992).
††J. Monteith (personal communication).

Aside from selection of crusher and crusher location and mobility, design considerations include methods to feed the crusher and convey crusher product to the mill stockpile. Crusher feeding considerations include the number of truck dump locations and the size of the crusher feed hopper (and appropriate design for dead beds). Primary crushers can either be fed with direct truck dumping (as in Figure 14.2-3) or by using ore pockets and apron feeders. Direct dumping is the simplest, but it can be more difficult to sustain peak crusher utilization, and truck wait times can be incurred. Apron feeders offer the ability to partially disconnect the crusher from truck dumping (with the idea of both reducing truck queues and improving crusher performance) but introduce additional capital costs, as well as the substantial maintenance costs of operating apron feeders in the demanding duty. Crusher installations should also consider the requirement for mobile equipment to clear dump pockets in case of blockages, rock breaker installations to handle oversize inadvertently fed to the crusher, overhead craneage (fixed or mobile) for maintenance, and overall elevation required for an installation (controlled by crusher dump pocket and product bin dimensions).

Crusher product handling using a bin and apron feeders to place the material on a conveyor belt is effective and allows matching the operation of the semicontinuous, variable capacity crusher (depending on material type) with a conveying system of fixed capacity. Product bin sizing should consider peak flow-through rates observed when crushing fine ore to avoid bin overfilling. In cases with sufficient belt capacity relative to crusher throughput, product can be fed directly to belts via appropriate chutes.



Courtesy of FLSmidth.
Figure 14.2-2 Primary gyratory crusher

Alternatives to conventional crushing prior to further processing include mineral sizers (whose manufacturers include Krupp and MMD). These machines operate dry, employ horizontal counter-rotating shafts equipped with wear tips, and are alternatives for size reduction of low- to moderate-strength materials with low-head-height devices. A somewhat similar



Courtesy of PT Freeport Indonesia.
Figure 14.2-3 Direct truck dumping to a gyratory crusher

device, the McLanahan log washer, operates wet and serves to deagglomerate coarse from fines rather than affect true comminution. These devices are well proven and work effectively in appropriate applications. There can be substantial capital cost savings with reduced civil and structural work for these units.

All of these machines are vulnerable to damage through introduction of incompressible foreign objects (notably ground engagement tools and tramp metal). In cases where direct dumping is used, protection is limited to tools such as cameras and image analysis software (useful for detection of the loss of shovel teeth), spotters, and the installation of safety clutches to limit damage for unavoidable events. Where material is reclaimed or dumped to feeders, metal detectors or magnets can be used in some applications.

PRIMARY MILLING

A critical linkage point between the mining and milling operations is the mill stockpile. Most comminution plants reclaim from a stockpile of primary crushed material, and stockpile management is essential to maintain consistent throughput. There is generally some degree of stockpile segregation, with coarse material preferentially accumulating toward the outside of stockpiles. Maintaining a live stockpile and balancing multiple reclaim feeders will result in the highest average (and most stable) throughput. Feed size to a primary milling circuit can have a substantial impact on throughput, particularly for AG and SAG circuits. This underscores the importance of evaluating mining technique and practice, primary crushing, and the downstream comminution circuit. It is also important in terms of nonsteady-state operation, such as stockpile drawdown conditions. As the proportion of material drawn down from the stockpile increases, the feed to the comminution circuit typically becomes coarser, correspondingly requiring a higher unit energy input for comminution. The often observed effect was well quantified by Morrell and Valery (2001); that paper also discusses broadly the effect of SAG feed on circuit performance.

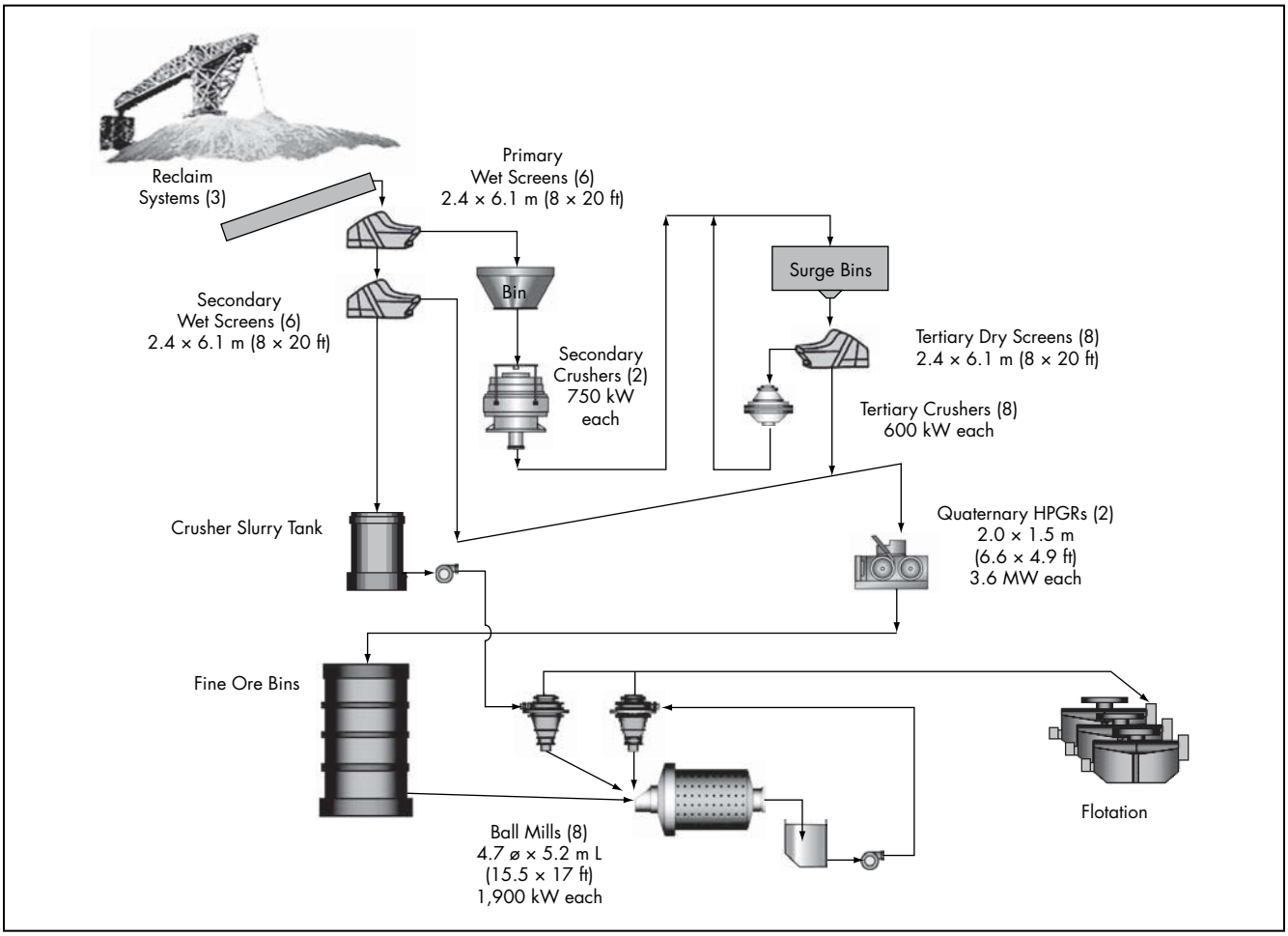
Maintaining reclaim stockpiles at reasonable levels minimizes the effect of load and haul equipment shift changes on downstream operations and decouples the mine and mill from maintenance activities associated with shovel and mill maintenance periods. Stockpile size should be based on anticipated interruptions due to primary crusher maintenance, load-and-haul asset maintenance, mill maintenance downtime requirements, and normal fluctuations due to mine sequencing. Depending on climate, ore moisture, surrounding facilities, dusting characteristics, and regulatory requirements, covering stockpiles may be desired or necessary.

The following sections outline three circuit configurations in broad use as the first stage in a comminution plant (primary milling): (1) standard crushing plants, (2) AG/SAG mills, and (3) the combined application of cone crushers and HPGRs. The latter circuit is now broadly viewed as an alternative to SAG circuits.

Crushing Circuits

The classic crushing circuit prepares primary crushed ore for secondary milling through two stages of crushing. Most typically, stockpiled primary crusher product is reclaimed and fed to a secondary crusher. In turn, the product of a secondary crusher operating in open circuit is screened, with oversize fed to a tertiary crushing circuit operating in closed circuit with screens. The product of a standard tertiary crushing plant is nominally 100% passing 13 mm. The basic design of a tertiary crushing plant (secondary crushers in open circuit with tertiary crushers in closed circuit with screens), along with variations, has been widely used. Such a circuit, inclusive of HPGRs in a quaternary role and follow-on single-stage ball milling, is depicted in Figure 14.2-4; the crushing plant and concentrator depicted has a nominal capacity of 75,000 t/d (83,000 stpd). The plant depicted shows wet primary screening followed by dry tertiary screening. In some applications, this is reversed, and some circuits (particularly for heap leaching) are fully dry. Selection of wet or dry screening is a function of material type, ROM moisture, efficiency of classification required, and the type of follow-on material handling or processing.

The product of such a typical crushing plant can be directly fed to a fine-crush heap leach operation. In milling operations, the product of a crushing plant was historically further reduced in an open-circuit rod mill prior to comminution in a ball mill. However, rod mills are limited to nominally less than 5 m (16.4 ft) in diameter and 6 m (19.7 ft) in length due to practical constraints on mill size before rod tangling occurs—such a mill has a power draw of less than 2.0 MW. As such, economies of scale have largely led mill circuit designers to send crushing plant product directly to ball mill circuits. Doing so may introduce a degree of grinding inefficiency (based on the coarseness of feed to a ball mill or a high reduction ratio in one stage of grinding), but this has been widely accepted as being favorable to a requirement



Courtesy of PT Freeport Indonesia.

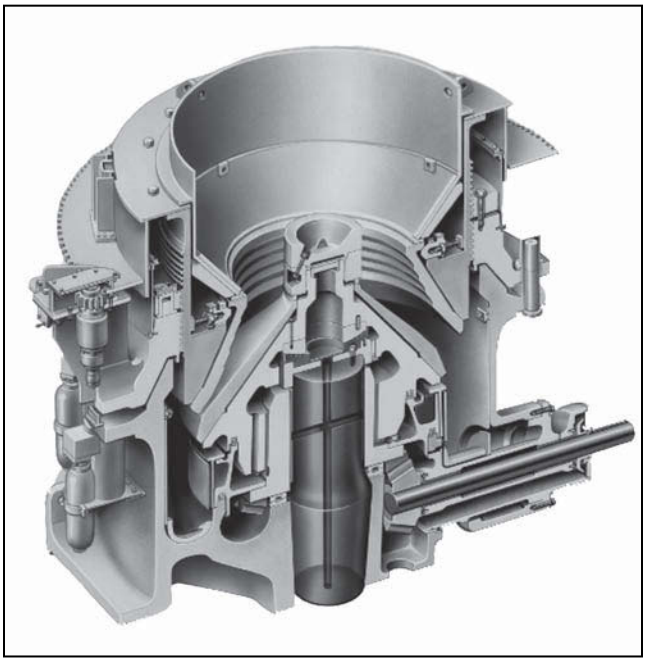
Figure 14.2-4 Crushing ball mill plant

for a large number of rod mill lines and an additional unit operation. As will be discussed in a following section, HPGRs in a quaternary role perform top-size reduction admirably, as well as doing an excellent job of fines generation. Depending on circuit configuration, large single-line secondary/tertiary crushing lines have a nominal installed power of 3 MW and the capacity to crush 25 to 40 kt/d (28,000 to 44,000 st).

Crushers are generally of two types of mechanical design. In the Symons design (Figure 14.2-5), a threaded bowl liner is adjusted to modify the crusher gap. The bowl is threaded to the adjustment ring, which is secured flush against the mainframe through hydraulic accumulators that release in the event of a tramp metal event. In the Allis design (Figure 14.2-6), the crusher mantle is shaft supported against a fixed bowl liner (analogous to a gyratory crusher's concaves) to maintain either a fixed gap or a fixed pressure setting. In the event of tramp metal (or other uncrushable item) entering the crushing chamber, the hydraulically supported shaft drops, allowing the material to pass.

Because crushing plants are well understood, the skill sets required for plant operation and maintenance can be readily sourced or developed. Throughput for the machines used is largely volumetrically controlled; as such, plant throughput is relatively insensitive to changes in feed hardness or feed size distribution (though the amount of comminution work and power drawn by the machines is clearly affected). Power input is a direct result of comminution effort required to effect breakage, unlike tumbling mills, where it is the outcome of the potential energy transmitted to the mill charge.

Crushing plants have some drawbacks relative to other plant configurations. First, particularly for larger plants, they normally require many more units of equipment (bins, crushers, screens, conveyors, chutes, etc.) for an equivalent throughput than other major circuit types. This is due to the fact that the current upper limit for installed power for a conventional crusher is 750 to 1,000 kW, smaller than the average unit installed power for mills. Partially as a result of the number of units in series, and partially due to liner change-outs, crushing plant run time is lower than that of rotating mills. Crushing plants do not produce fines to the same degree as other primary milling options, which increases the amount of comminution effort required in a second stage of milling. The mode of breakage is dominated by compressive fracture, which produces fewer fines, and screening (wet or dry) at finer sizes is not practical at large tonnages; screen area required escalates nonlinearly with decreasing cut size. Finally, crushing plants have material-handling difficulties with sticky ores, significant downtime due to metal detector trips, and potentially a resulting loss in production with ores that contain a high degree of tramp metal or magnetite.



Courtesy of Metso.
Figure 14.2-5 Symons-style crusher (Metso MP-series)



Courtesy of Sandvik.
Figure 14.2-6 Shaft supported crusher (Sandvik CH-870)

AG/SAG Circuits

Since their early development, AG and SAG mills have become a mainstay of comminution in mineral processing applications. Although crushing-rod mill-ball mill circuits were referred to as "conventional" plants until relatively recently, AG/SAG mills are now the foundation unit operation for most large grinding circuits. The variety of AG/SAG circuits is substantial. The following duties are all well represented:

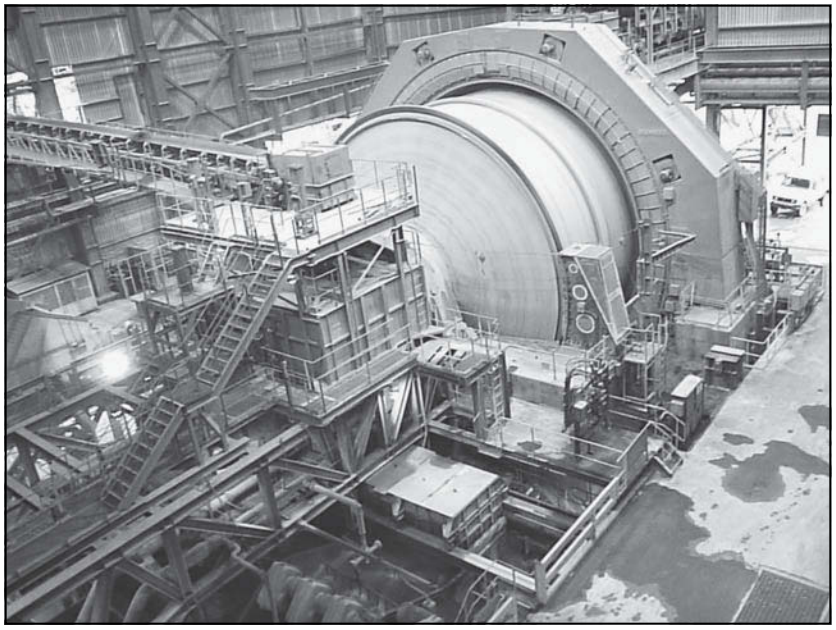
- Single-stage autogenous/semiautogenous milling
- AG/SAG mills as one stage in a larger comminution circuit
- Inclusion of pebble crushing circuits in the AG/SAG circuit
- Employment of a crushing stage after primary crushing (such that SAG feed has been through two stages of crushing)

Single-stage AG circuits have been installed in relatively high-tonnage, coarse-grind applications such as Iron Ore Company of Canada's (IOC's) Carol Concentrator, and in lower-tonnage, finer-grind applications (closed with hydrocyclones) like Noranda's Brunswick operation in Quebec, Canada. Very high single-line capacities have been demonstrated, with some circuits having up to 70 MW of installed grinding power in a single plant. PT Freeport Indonesia's 11.6-m- (38-ft-) diameter SAG mill in Concentrator 4 (Papua, Indonesia), presented in Figure 14.2-7, has registered monthly tonnages averaging more than 125,000 t/d (138,000 stpd), with single-day performances in excess of 150,000 t/d (165,000 stpd).

Although common convention generally refers to high-aspect-ratio mills as SAG mills (with diameter to effective grinding length ratios of 3:1 to 1:1), low-aspect-ratio mills (generally, a mill with a significantly longer length than diameter) can also grind autogenously or semiautogenously. Such mills are common in South African operations and are sometimes referred to as tube mills or ROM ball mills. Many of these mills operate at higher mill speeds (nominally 90% of critical speed) and often use grid liners to form an AG liner surface. These mills typically grind ROM ore in a single stage, either autogenously or semiautogenously.

The broad application of AG/SAG mills lies in the large single-line capacity of such plants, the ability of circuits to mill a broad range of feed types, and the variety of circuit configurations. The application of gearless drives has allowed mills to grow beyond sizes allowed by twin-pinion drives; a nominal limitation of 12–16 MW (based on a limit of 6–8 MW per pinion on a twin-pinion drive) results in a restriction of nominally an 11-m (36-ft) diameter SAG mill (based on typical diameter/length ratios) if using geared drives. Many large mills around the world (including Cadia, Collahuasi [New South Wales, Australia] with 12.2-m (40-ft) mills, and Antamina [Peru], Escondida Ph IV [Antofagasta, Chile], PT Freeport Indonesia, and others with 11.6-m [38-ft] mills) have installed SAG mills of nominally 20 MW with gearless drives. A number of mills with 25 MW or higher gearless drives are either under construction or in engineering. Some of these grinding lines have design capacities exceeding 100,000 t/d (110,000 stpd). The process flow sheet for the large SAG installation (with pebble crusher product combining with SAG discharge and feeding screens) depicted in Figure 14.2-7 is presented in Figure 14.2-8.

AG/SAG mills comminute ore through impact breakage, attrition breakage, and abrasion of ore serving as media. AG circuits require an ore of suitable competency (or fractions within the ore of suitable competency) to serve as media. SAG circuits may employ low to relatively high ball charges (ranging from 2% to 22%, expressed as volumetric mill filling) to augment AG media. Higher ball charges shift the breakage mode away from attrition and abrasion breakage toward impact breakage; as a result, autogenous milling produces a



Courtesy of PT Freeport Indonesia.
Figure 14.2-7 Large SAG installation (20-MW gearless drive)

finer grind than semiautogenous milling for a given ore and otherwise equal operating conditions.

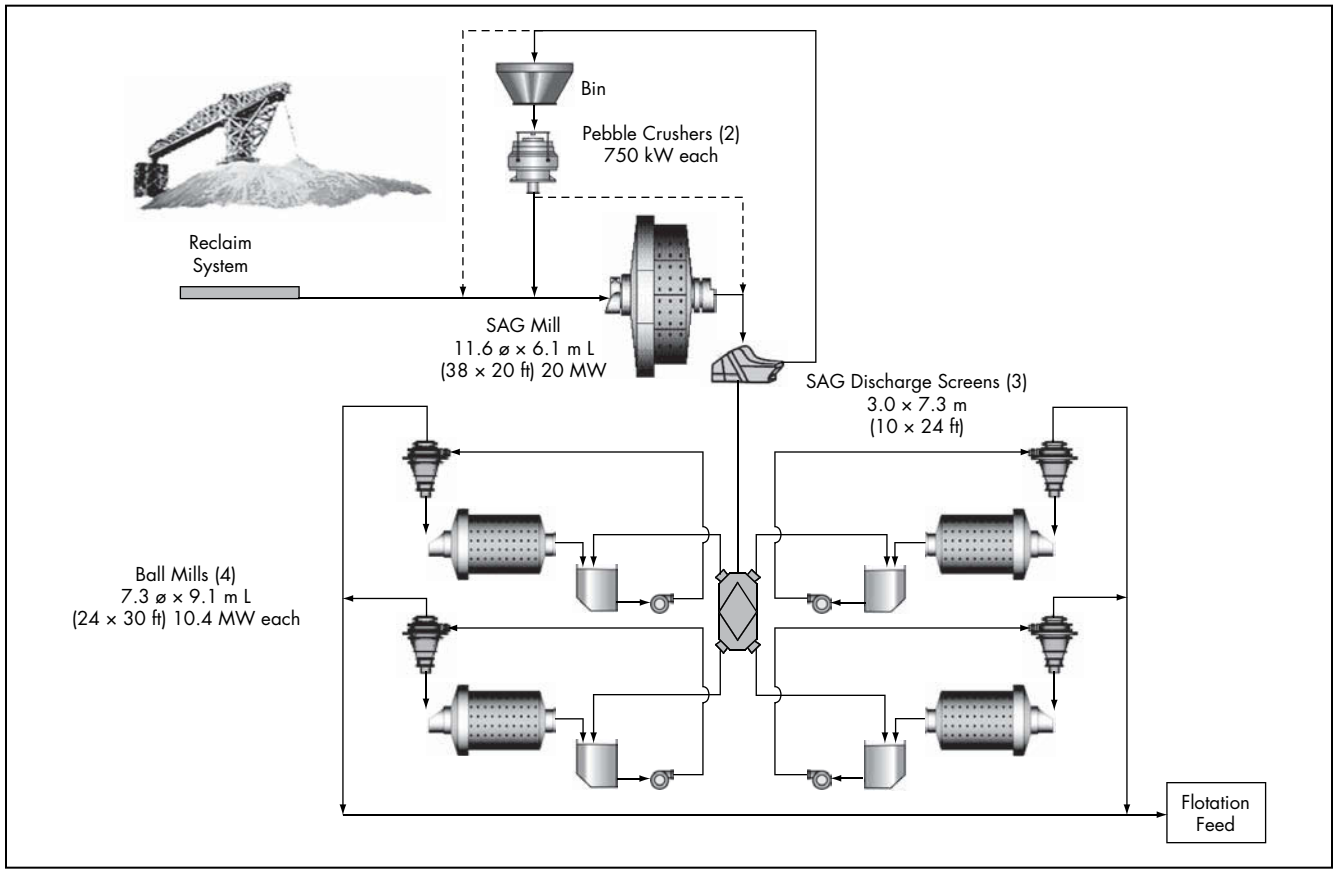
With a higher-density mill charge, SAG mills have a higher installed power density for a given plant footprint relative to AG mills. With the combination of finer grind and a lower installed power density (based on the lower density of the mill charge), a typical AG mill has a lower throughput, a lower power draw, produces a finer grind, and has a higher unit power input (kilowatt-hours per ton) than a SAG circuit milling the same ore. In the presence of suitable ore, an AG circuit can provide substantial operating cost savings due to a reduction in grinding media expenditure and liner wear. In broad terms, this makes SAG mills less expensive to build (in terms of unit capital cost per metric ton of throughput) than AG mills but more expensive to operate (as a result of increased grinding media and liner costs). SAG circuits tend to be less susceptible to substantial fluctuations, because of feed variation, than AG mills and are more stable to operate. AG circuits are more frequently (but not exclusively) installed in circuits with high ore density, with iron ore being the classic example. A small steel charge addition to an AG mill can boost throughput and result in more stable operations, typically at the consequence of a coarser grind and higher operating costs.

AG or SAG mills with low ball charges are often used in single-stage grinding applications. Conversely, SAG mills produce coarser grinds than AG mills in equivalent applications. Based on their higher throughput and coarser grind relative to AG mills, SAG mills are more likely to be used as the primary stage of grinding, followed by a second stage of milling. AG/SAG circuits producing a fine grind (particularly single-stage grinding applications) are often closed with hydrocyclones. Circuits producing coarser grinds often classify mill discharge with screens. For circuits classifying mill discharge at a coarse size (coarser than approximately 10 mm), trommels can also be considered to classify mill discharge. Trommels, however, are less favorable in applications requiring high classification efficiencies and can be constrained by available surface area for high-throughput mills. Regardless of classification (hydrocyclone, screen, or trommel), oversize material can be returned to the mill or directed to a separate comminution stage.

The addition of pebble crushing is the most common variant to closed-circuit autogenous/semiautogenous milling (instead of direct recycle of oversize material). The potential efficiency benefits, both in terms of grinding efficiency and in capital efficiency through incremental throughput, has long been recognized, but the challenges of metal removal (primarily grinding media passing mill grates) were perceived to be a substantial hurdle. With time, and after successful pioneer operations, operators and circuit designers became comfortable with the required magnetic separation and metal detectors/bypass chutes required to protect pebble crushers from tramp steel.

Even after steel ball removal had proved to be reliable, pebble crushers installations were still thoroughly scrutinized at the design stage because of the additional cost and circuit complexity. Instead of foregoing pebble crushing completely, though, many operations chose instead to leave provisions in the design for future pebble crushing expansions. Today, every AG/SAG flow sheet conceived is likely to seriously consider the inclusion of a pebble crusher circuit. For certain ore types (particularly those with a chert, andesite, or other hard component that develops a critically sized material that constrains milling rates), a pebble crushing circuit is almost an imperative for efficient circuit operation. For other cases, it is simply a technique to improve circuit capacity. Construction of an AG/SAG circuit without provision for later pebble crusher installation should have a persuasive economic justification. Important aspects of pebble crusher circuit design include

- Preparation of a clean, well-sized, and dry feed;
- Metal removal (with additional protection via metal detectors and bypass);
- Surge capacity (through bins or, more costly, a pebble stockpile);



Courtesy of PT Freeport Indonesia.
Figure 14.2-8 SAG pebble crusher–ball mill circuit

- Sufficient capacity (primarily a concern of large circuits where multiple pebble crushers are required to serve one grinding line);
- Design for bypassing crusher(s) during maintenance; and
- Evaluation of where to reintroduce the crushed pebbles back to the grinding circuit.

The standard destination for crushed pebbles has been to return them to SAG feed. Doing so is often convenient based on conventional plant layouts. However, open circuiting the SAG mill by feeding crushed pebbles directly to a ball mill circuit is often considered a technique to increase SAG throughput. PT Freeport Indonesia's operation has the ability to return crushed pebbles to SAG feed per a conventional flow sheet, or to SAG discharge. By combining with SAG discharge and screening on the SAG discharge screens, top size control to the ball mill circuit feed is maintained while still unloading the SAG circuit (Mosher et al. 2006).

Aside from parameters fixed at design (mill dimensions, installed power, and circuit type), the major variables affecting AG/SAG mill circuit performance (throughput and grind attained) include

- Feed characteristics in terms of ore hardness or competency;
- Feed size distribution;
- Selection of circuit configuration in terms of liner and grate selection and closing size (screen apertures or hydrocyclone operating conditions);
- Ball charge (fraction of volumetric loading and ball size); and
- Mill operating conditions including mill speed (for circuits with variable-speed drives), density, and total mill load.

The effect of feed hardness is the most significant driver for AG/SAG performance, which is the case to a much greater degree than for (compressive) crushing plants; variations in ore hardness result directly in variations in circuit throughput. With AG circuits, the variations can be substantial and occur suddenly. The effect of feed size is also marked, with both larger and finer feed sizes having a significant effect on throughput. A number of case histories of AG mills have failed to consistently meet throughput targets because of a lack of coarse media. Compounding the challenge of feed size is that, for many ores, the overall coarseness of the primary crusher product is correlated to feed hardness. Larger, more competent material consumes mill volume and limits throughput.

To counteract this effect, many circuits (including Troilus [Quebec], Kidston [Queensland, Australia], Ray [Arizona, United States], Porgera [Papua New Guinea], Granny Smith [Western Australia], Geita Gold [Tanzania], St. Ives [Western Australia], and KCGM [Western Australia]) currently use or have used a secondary SAG precrush circuit to further comminute primary crusher product prior to feeding to a SAG mill. The objective of SAG precrush is to increase SAG throughput. Occasionally, secondary crushing is included in the original design but more often is introduced as an additional circuit to

account for harder ore (that proves to be harder than expected or becomes harder as the deposit is developed) or as a capital efficient mechanism to boost throughput in an existing circuit. Such a flow sheet is not without drawbacks. Not surprisingly, some of the advantages of semiautogenous milling are reduced in terms of increased liner wear and increased maintenance costs. Also, precrushing can lead to an increase in midsize material, thereby overloading pebble circuits, and in challenges controlling recycle loads. In certain circuits, the loss of top size material can lead to decreased throughput. It is now widespread enough to be considered a standard circuit variant and is often considered an option in trade-off studies. The concept of feeding AG mills with as coarse a primary crusher product as possible in order to maximize AG mill throughput is obviously at the other end of the spectrum.

To a more significant degree than in other comminution devices, liner design and configuration can have a substantial effect on mill performance. In general terms, lifter spacing and angle, grate open area and aperture size, and pulp lifter design and capacity must be considered. Mill liner designs have moved toward more open shell lifter spacing, increased pulp lifter volumetric capacity, and tailored grate design to facilitate maximizing both pebble crushing circuit utilization and SAG mill capacity (Mosher 2005).

AG/SAG circuits can handle a broad range of feed sizes, as well as sticky, clayey ores (which challenge crushing plants). A range of circuit configurations have been developed, and large single-line plants can be designed with significant economies of scale and streamlined maintenance shutdowns and practices. Compared to crushing plants, wear media usage is reduced, and plants run at higher availabilities. Circuits, however, are more sensitive to variations in circuit feed characteristics of hardness and size distribution. Unlike crushing plants, for which throughput is largely volumetrically controlled, AG/SAG throughput is defined by the unit power required to comminute the ore to the closing size desired. A higher degree of operator skill is typically required of AG/SAG circuit operation, as is more advanced process control to maintain steady-state operation, with different operator/advanced process control regimes based on different ore types. Additionally, very hard ores can severely constrain AG/SAG mill throughput. In such cases, the circuits can become capital inefficient (in terms of the size and number of primary milling units required) and, in some cases, require more total power input compared to alternative comminution flow sheets.

Crusher–HPGR Circuits

Recently, HPGRs have made broad advances into the nonferrous metals mining. Long established in the cement industry, HPGRs first made inroads into diamond processing (where rock fracture along grain lines favored a reduction in diamond breakage during comminution) and in the iron ore industry. Penetration to the hard-rock mining industry was slow and hampered by high maintenance requirements both for wear surfaces in general and high wear on the edge of rolls. The intensity of the maintenance effort (relative to other comminution machines) required for a trial unit at the (then Phelps Dodge) Sierrita operation (Arizona) in 1996 was high.

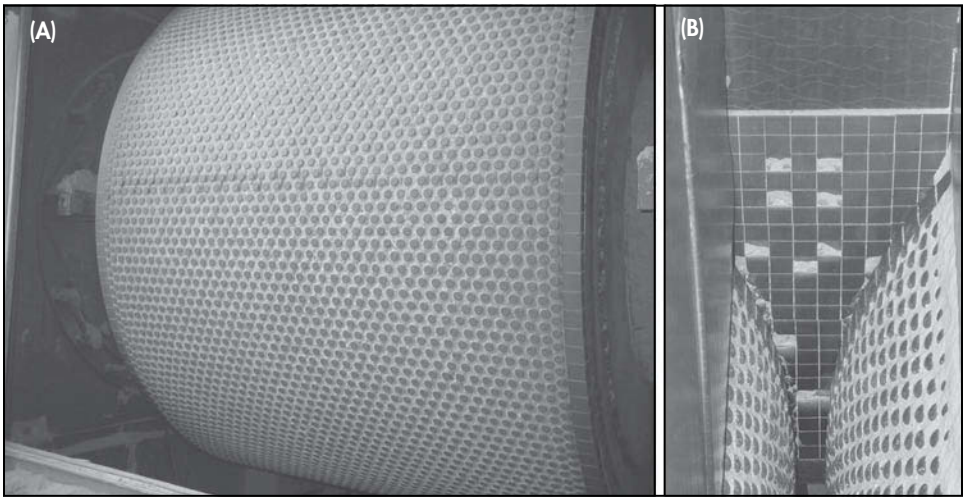
Significant product improvements since then, however, have increased wear life, improved availability, and decreased the overall maintenance effort required. Incorporation of studs on the surface of rolls to allow formation of AG wear surfaces, as well as implementing edge blocks of a long-wearing material for edge protection, have allowed HPGRs to break into the mainstream of mineral processing. These wear-retarding innovations were the focal point of a full-scale trial at Lone Tree in Nevada, United States (Seidel et al. 2006). Successful completion of this trial marked somewhat of a turning point for interest in HPGRs for hard-rock applications. Manufacturers have also paid special attention to the bearings, wear surfaces, and the handling of tramp metal through the rolls to improve operational reliability, reduce maintenance, and obtain longer service lives. Availabilities are now such that aside from rolls change-outs (akin to a mill liner change), the unit rarely controls circuit availability.

The wear on a roll's surface is a function of the ore's abrasiveness. Increasing roll speed or pressure increases wear with a given material. Studs allowing the formation of an AG wear layer, edge blocks, and cheek plates are depicted in Figure 14.2-9. Development in each of these areas continues, including profiling of stud hardness to minimize the "bathtub" effect (wear of the center of the rolls more rapidly than the outer areas), low-profile edge blocks for installation on worn tires, and improvements in both design and wear materials for cheek plates.

In hard-rock-metals mining applications, HPGRs are currently used in tertiary and quaternary crushing applications, as well as in secondary pebble crushing. In some respects, HPGRs replace crushers as a unit operation. However, from a process standpoint, HPGRs produce a product with substantially more fines (for a given P_{80}) than a crushing circuit. In this regard, the size distribution of an HPGR circuit is much more similar to the product of a SAG circuit than a conventional crushing circuit, reducing the required amount of power in the ball mill circuit (relative to a crushing circuit). Also, an HPGR represents a much larger installation of power in a given footprint relative to tertiary crushers: An HPGR has up to a 5-MW power installation per unit compared to 750 kW for a single large cone crusher. HPGRs have also been cited as crushing more efficiently than rotating mills, as well as generating residual cracks that improve efficiency in subsequent milling operations. This effect is material- and operating-condition specific.

As a result, the number of material handling units (feeders, conveyors, screens, chutes) can be sharply reduced compared to a conventional crushing plant, and plant layouts are also much more straightforward. Freeport-McMoRan's (Phelps Dodge at the time of construction) Cerro Verde operation in Peru was a groundbreaking installation because the combination of secondary crushing (using MP1000s), tertiary crushing using HPGRs, and screens replaced what would have been more typically a large SAG mill feeding a multiple ball mill circuit (Vanderbeek et al. 2006). This was a significant step, as it presented an alternative to conventional crushing plants or autogenous/semiautogenous milling for primary milling applications (see Figure 14.2-10). Newmont Mining Corporation's consideration and ultimate selection of HPGR comminution for its Boddington project in Western Australia, widely documented in the literature, has now been commissioned with a similar comminution flow sheet to the Cerro Verde flow sheet (and also employing four units measuring 2.4×1.7 m).

Although the primary focus of this section is to discuss HPGRs (as they have been combined with crushers and screens) in primary milling roles, the versatility of the HPGRs in grinding applications must be noted. Though limited in top size they can accept (based on nip angle and resultant stud



Courtesy of PT Freeport Indonesia.

Figure 14.2-9 HPGR roll (2.0 × 1.5 m diameter) with (A) studs and edge blocks and (B) cheek plates

breakage, leading to selection in tertiary rather than secondary roles), the machines are quite versatile because they have been installed in tertiary, quaternary, second-stage pebble crushing roles, and a number of finer grinding applications. In the fine-grinding role, HPGRs are widely used in pelletizing operations. One area not currently applicable to HPGRs is secondary milling in a wet process. In the cement industry, it is common practice to use HPGRs in closed circuits with a dry air classifier for finish grinding (S. Kirsch, personal communication). However, wet classification and subsequent dewatering does not compare favorably to the use of ball mills for wet processes most commonly used in mineral processing. A summary of the broad range of comminution applications of HPGRs is presented in Table 14.2-3.

SECONDARY MILLING

For those circuits requiring secondary grinding (most circuits aside from heap leach operations and single-stage AG/SAG circuits), the use of wet ball mills has remained the standard unit operation. Most such circuits use ball mills in closed circuit with hydrocyclones. However, closing circuits with fine screens is becoming more common, particularly in the iron ore industry (Valine et al. 2009). The majority of applications grind from a typical range of F_{80} values of 1–13 mm to P_{80} values of 300–37 μm. Pebble mills are also used in a number of applications.

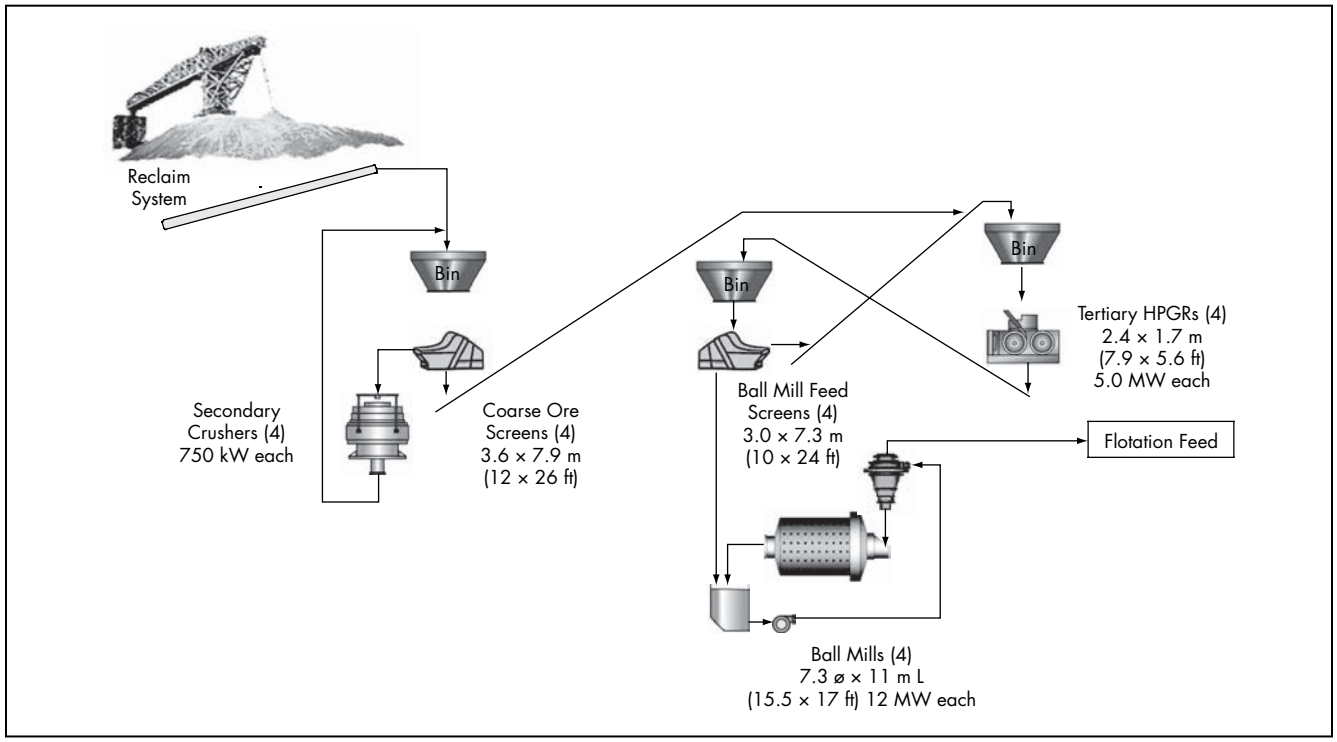
Ball mills as large as 8.5 m (28 ft) in diameter are now being constructed. As with SAG mills, twin pinion and girth gears can be considered for mills with power input up to at least 15 MW (generally mills with a diameter of 7.3 m [24 ft] or less). Wraparound drives are the standard for larger mills.

In general, given a mill of suitable power and operated with an appropriate feed size, reduction ratio, media size selection, and the use of efficient classifiers, accurately predicting grinding circuit response is relatively straightforward. These principles apply for either wet or dry milling.

Aside from the widely used Bond equation (discussed later in the "Circuit Design" section), Bond used eight factors to "correct" for the power required for grinding in different operating scenarios (Rowland 2002). Bond's work to develop criteria for use in designing grinding circuits has proved to be enduring and useful, even with the contemporary availability of powerful simulation tools. Although the utility of correction factors for quantitative contemporary use in formal design is debatable, using the factors for qualitative evaluation of aspects affecting grinding efficiency can have value. Two of the eight correction factors apply strictly to rod milling, with the remaining six to ball mill grinding. Of greatest interest for current applications are corrections for coarse feed (with the definition of what is excessively coarse being a function of ore hardness), open-circuit milling (vs. closed-circuit), grinding to a P_{80} finer than 75 μm, and excessively low reduction ratios. One correction factor was developed to correct for inefficiency of large ball mills; many mill designers today believe that this factor had more to do with inefficiencies of classification in large hydrocyclones in early mill installations than in any actual milling inefficiency. The last factor for ball milling applies to dry grinding. In summary, although using these factors as originally developed for design may not be relevant, using them qualitatively to assess efficient operating conditions can be a useful tool.

By far the largest tonnages processed through secondary milling today are classified using hydrocyclones. Specialized low-tonnage operations may use screens, selected contemporary operations still employ screw classifiers, and dry operations generally use cyclones. A detailed discussion of the appropriate design for recirculating load is beyond the scope of this chapter. Milling efficiency typically increases with closed-circuit operation and will often increase to a certain level with increasing circulating load. After a certain point, the grinding efficiency declines with increasing recirculating load. Also, at a certain point, the power required for pumping for increased recirculation is not offset by improved grinding efficiency.

Some circuits perform grinding in one step, others in multiple stages (though rarely exceeding two), and occasionally with beneficiation integrated into the grinding circuit. The use of gravity concentration integrated into the mill recirculating load is common for the recovery of gold, flash flotation is often seen in sulfide mills, and recovery efforts after each stage of grinding are common for recovering platinum group metals and certain iron ore operations relying on magnetic separation.



Courtesy of Minera Cerro Verde.

Figure 14.2-10 Crusher–HPGR–ball mill circuit

Table 14.2-3 Summary of range of HPGR applications

		Size				
Operation	**Feed Application**	**Tire Diameter, m**	**Tire Width, m**	**Power, MW**	**Tons per Hour**	**Top Size, mm**
Cerro Verde, Peru	Cu/Mo, tertiary, closed circuit	2.4	1.7	5.0	1,250	<50
CMH, Chile	Iron ore, tertiary, closed circuit	1.7	1.8	3.7	1,800	<60
Premier, South Africa	Diamond, tertiary, open circuit	2.85	0.5	1.2	330	<32
Empire, Michigan (USA)	Iron ore, second-stage pebble, open circuit	1.4	0.8	1.4	440	<30
Argyle, Australia	Diamond, quaternary, closed circuit	1.7	1.4	1.9	800	<15
PT Freeport Indonesia	Cu/Au, quaternary, open circuit	2.0	1.5	3.6	1,300	<10
Samarco, Brazil	Iron ore, pellet preparation regrind, open circuit	2.0	1.5	2.9	1,200	<0.2
Vale Tubarao, Brazil	Iron ore, pellet preparation regrind, open circuit	1.4	1.6	3.5	715	<0.2
Vale Sao Luis, Brazil	Iron ore, pellet preparation regrind, open circuit	1.7–2.4	1.2–1.4	2.0–3.6	650–1,200	<0.2

Source: Adapted from Mular and Mosher 2006.

TERTIARY MILLING

In the context of this paper, the term *tertiary milling* is used to describe concentrate regrind, grinding conducted for materials handling (iron ore pelletizing and pumping of mineral concentrates, for example), and fine grinding for specialized metallurgical applications. The border between a typical tertiary milling application and fine grinding is not distinct. However, as the methodologies for sizing change substantially at nominally 37 μm (400 mesh, and switching from conventional sieving practice toward subsieve analysis and alternate techniques at finer sizes), designating grinding to a P_{80} <37 μm as fine grinding is a useful designation.

Most grinding circuits are developed with the concept that the mill feed will be ground to an economic optimum such that further grinding expense (in terms of energy and media wear) is not justified based on the additional liberation and recovery of additional value units. In many cases, grinding 100% of the bulk mill feed to this size represents the optimum in terms of overall project economics and simplicity. In some circuits, it is desirable to have multiple stages of liberation and beneficiation. Since physical mineral separation processes have an optimum size range for efficiency, and essentially all lose efficiency at very fine particle sizes, this sequence can allow recovery of value minerals as they are liberated and before they are overground. Particularly in flotation, value minerals are often recovered to rougher concentrates when only partially liberated. Further grinding of such concentrates is employed to increase liberation and facilitate production of concentrate at a salable grade. Aside from liberation and recovery considerations, material handling of concentrates may dictate further size reduction. Examples include grinding of concentrates to allow pumping without pipeline wear or grinding of iron ore concentrates prior to pelletizing operations.

The field of options for fine grinding is diverse, with many originally developed to fulfill a specific need and subsequently marketed. Although not all operations require this stage of grinding, for those that do it is often a critical unit operation. Although the mills are smaller in size than mills for primary and secondary grinding, unit energy input (ranging from 5 to 40 kW·h/t) for tertiary mill feed often exceeds the total unit energy requirement for the bulk feed. In broad terms, the equipment field can be classified into rotating mills, stirred mills, and a general category of specialized mills. In general, the stirred mill category has the ability to grind finer with a higher efficiency than rotating mills. Specialized mills can cover a broad range of sizes but tend to be more focused to specific applications. Options for fine grinding are summarized:

- Conventional rotating mills (ball mills)
- Stirred mills
  - Vertical shaft mill (screw type). Also referred to generically as tower mills, after an early manufacturer, the Japan Tower Mill Company. Similar mills are now made by several manufacturers, with perhaps the most common being the descendant of the original mill design (by Nippon Eirich Co. Ltd.) by the trade name of Vertimill (marketed by Metso) (Lynch and Rowland 2005).
  - Vertical shaft mill (pin type). One trade name is the Metprotech mill (marketed by Bradken); another with a higher tip speed is the stirred media detritor (SMD) (marketed by Metso).
  - Horizontal shaft mill. The IsaMill has made broad inroads into fine grinding.
- Other specialized mills
  - Jet mills
  - Vibratory ball mills
  - Roller mills
  - Centrifugal mills

These machines are classified primarily by their mechanical configuration. Lichter and Davy (2002) provide another useful distinguishing feature for stirred mills—the charge motion. They cite a distinct difference between mills in which the charge is stirred (tower mills and some pin mills) and those in which the charge is fluidized (some pin mills and IsaMills), indicating that stirred mills are more efficient with coarse, hard feeds and that fluidized mills are more efficient with relatively finer, softer feeds.

Conventional rotating (tumbling) ball mills are often used to grind concentrates down to sizes of nominally –37 μm. Common applications include tertiary milling with multiple stages of grinding, in regrind applications, and in preparation of iron ore concentrates for pelletizing.

Stirred mills are often used for fine grinding. Mills in this category are sometimes more commonly referred to by their trade names. Stirred mills offer a higher power density (kilowatts per cubic meter), which, combined with the fact that they are filled with media (as opposed to tumbling mills, where the charge volume is nominally 35% of the total mill volume), means that these mills are compact in terms of installed power. Tower mills are occasionally used as way to retrofit additional grinding power with a small footprint and low capital for civil works; in this capacity they are often used for additional secondary or for tertiary grinding. They are widely used in lime slaking and in concentrate regrind applications. Pin mills (of either nonfluidizing or fluidizing tip speed) are generally the next mill considered at progressively finer grinds to a nominal grind size with a P_{80} of 5 μm. IsaMills are established as a versatile fine grinding mill, having been commercialized after being developed at Mount Isa Mines (Queensland, Australia). The IsaMill has demonstrated an ability to grind sizes with a nominal P_{80} of 5 μm. IsaMill models with up to 3 MW of installed power have broadened the mill's usage to regrind applications where higher tonnages dictate larger mill sizes (Burford and Niva 2008).

Grinding energies required (sometimes judged by the calculation of an operating work index, Wi_o, at coarser sizes, and at finer sizes by the specific energy input required to attain the target grind) often increase substantially as grinds get progressively finer. This can be due to a combination of factors, including higher energy required as the mineral grain size is approached, the number of micro-imperfections in the particle structure to exploit decreases, and, not insignificantly, the efficiency of classification is reduced. The required energy inputs can vary substantially based on mill type, media type, and the efficiency of classification employed. Tests should be conducted with a variety of machines and with a variety of media types to measure the specific energy input required to attain the target grind. Media types and sizes are summarized in a subsequent section ("Media and Liners"), but it is worth noting that the energy efficiency of fine grinding is strongly influenced by the media type selected (Gao et al. 2001). Additionally, based on the high unit power input, the integrity of media is also critically important. At very fine grind sizes, and depending on the material ground, the energy input can modify the reactivity of the material, and this can be a consideration in the chosen grinding methodology.

There are number of specialized fine-grinding applications, particularly in the fields of industrial minerals, powders, and the like. Fine grinding in specialized applications is often conducted dry and using either ceramic media or autogenously to avoid iron contamination. Examples include jet milling, centrifugal mills (including the nutating mill), vibratory ball mills, and roller mills. The reader is directed toward manufacturers and the body of literature for detailed coverage of these applications.

No discussion of ultrafine grinding would be complete without touching on the method of particle size analysis. At extremely fine sizes, particle size determination is not always straightforward. The measurements used are often indirect measurements from which particle size is estimated, with different techniques resulting in a different size being inferred. Additionally, the intense energy input can change the surface chemistry of the ground material, which can have effects including agglomeration or increased galvanic activity. For example, the use of specialized electroformed nickel subsieves (which offer precisely sized apertures for sieves at sizes less than 20 μm) is restricted for copper concentrates because of rapid galvanic action with the sieve itself.

Techniques often used include conventional sieves, specialized subsieves (generally available at 15, 10, and 5 μm) made of textile or electroformed and photo-etched nickel, and indirect methods, which employ material properties. Techniques rely on measurements of settling velocity (by means of elutriation, sedimentation, or centrifugal sedimentation), light dispersion characteristics, or both. Examples include the Warman Cyclosizer, the Malvern Mastersizer laser analysis, Microtrac's laser unit, Quantachrome's Microscan

units, as well as others including the Coulter counter and surface area techniques. Regardless of technique, it is important to use the same one throughout a study. It can also be worthwhile to confirm sizing calculated via a certain technique using microscopy, either direct or in combination with image analysis software. Though this is unlikely to have any benefit for a circuit in operation, it can be useful in comparing size distributions of different machines. Comparing numbers derived using different techniques should be performed only with great caution.

SPECIAL APPLICATIONS

This chapter has focused largely on the comminution means employed by the majority of mineral processing operations (as judged by the tonnages processed). A number of special considerations apply either to specific industries, minerals, or focused applications. Comminution to develop a specific size distribution (rather than the simpler approach of assessing the size distribution through the use of a single data point) and a certain shape of particle (rather than using liberation as the goal), along mineral boundaries, and beneficiation steps integrated in the comminution process are all often considerations.

Beneficiation during comminution is often employed. The concentration of heavier specific gravity minerals in hydrocyclone feed or more often in hydrocyclone underflows is well documented and is used as a preconcentration step in flash flotation and gravity concentration units in a number of grinding circuits. The addition of reagents and lixiviants to grinding mills is commonly practiced, either for conditioning (commonly in flotation) or to increase leach residence time.

In certain applications (typically aggregate or industrial minerals), the particle shape resulting from comminution can be important and is a criterion in the selection of certain types of comminution machines. Examples include vertical shaft impact crushers or Gyradisc (marketed by Metso) crushers, which are examples of machines sometimes selected for comminution to a specific particle shape. Similarly, in certain applications, minimizing fines generation while attaining a grind below a maximum particle size is important. When this requirement is overriding, certain grinding equipment (listed in order of coarser grinds and lower fines generation) can be used: center discharge rod mills, end-peripheral discharge rod mills, overflow rod mills, and grate discharge ball mills. In quarrying and certain other applications, maximizing yield in certain size fractions preferentially over other size fractions can add significant value. For specialized applications such as this, close consultation with vendors and thorough testing are recommended.

Dry grinding is an important, albeit niche, application. For any pyrometallurgical operation, dry grinding is an obvious consideration. Newmont's Carlin mill and Barrick's Goldstrike mill in Nevada both have roaster facilities fed by Krupp double-rotator dry mills. The double-rotator mills consist of a twin chamber mill (with sides for coarse and fine milling), with both sides discharging to a common air classification system. Operation of Goldstrike's system is well described by Thomas et al. (2001). Newmont's Minahasa gold operation in Indonesia employed an Aerofall SAG mill followed by dry ball milling prior to decommissioning. The IOC's Carol Concentrator in Labrador (Canada) formerly milled iron ore for gravity concentration using dry Aerofall AG mills; Chung et al. (2001) demonstrated substantially better energy efficiency in grinding using dry milling. Energy costs for drying, particularly for a wet downstream process, contributed ultimately to the full conversion of the Carol Concentrator to wet AG. Richards Bay Minerals uses an Aerofall mill for dry grinding of titanium slags. Dry grinding with air classification should be considered in applications where the follow-on process is pyrometallurgical and drying would be required anyway, minimization of fines during comminution is critical (with air-sweep and classification assisting in this regard), or an ore's aggressive soluble components make dry grinding more cost-efficient than construction of a wet milling circuit made from corrosion-resistant materials.

A number of other specialized grinding units are referenced in this chapter based on their comminution range (primary, secondary, or tertiary grinding). Media selection to avoid contamination is also a consideration, referenced later in the "Media and Liners" section. Finally, certain applications find that media selection can impact metallurgical performance (through the interaction of redox potential and dissolved oxygen); for certain applications, AG, high chrome, or inert media can be preferred.

ORE CHARACTERIZATION, CIRCUIT DESIGN, AND CIRCUIT SELECTION

Many ore bodies can be successfully processed using a range of comminution circuit configurations. In some cases, however, certain types of circuits are favored over others. Ore characteristics (competency, density, work index, abrasivity, chemical composition), plant capacity, downstream processing requirements, mine life, capital costs, operating costs, delivery time for major components, and plant location are all factors in process selection. Given ore characterization data, alternative circuit designs to accomplish the required comminution can be developed. Capital and operating cost estimates for each circuit alternative allow for trade-off studies. Ore characterization, circuit design, and circuit selection are summarized briefly, with each addressed in depth in the broader literature.

Before an effective study culminating in a circuit design can be commissioned, a sufficient knowledge of the deposit and the mining method and sequence must be defined to allow development of a sampling design. Sampling of a greenfield deposit via drilling can be an expensive undertaking, and collection of a bulk sample for pilot testing (if desired or required) often exceeds the cost of the testing itself. Knowledge of the overall range of comminution characteristics within a deposit can be useful, but, as a guideline, the testing effort based on ore type should be proportional to the representation in the overall reserves. If ore types have divergent characteristics and/or will be mined at different periods in the mine life, it may be prudent to consider construction of a modular plant.

In some cases, blending of samples may also be considered. If so, blending should be done in a fashion consistent with the planned mining sequence. For instance, blending two samples that will be milled together may make sense. However, compositing of an entire drill core consisting of different horizons almost never makes sense and typically results in misleading data. Many designers prefer to blend the results of testing (mathematically) of individual ore types rather than blending a sample for testing. Both approaches have advantages and disadvantages. Obviously, testing for circuit design should be performed on ore and not waste; too many projects have had initial characterization done on waste (with waste samples being more readily available). Efforts in reasonable ore-body definition and the planned mining sequence to allow

development of a rational sampling program is almost always a good investment.

Ore Characterization

Bench-scale breakage characterization tests are the basis for circuit design. Such tests fall into three broad categories: (1) grindability, (2) single-particle breakage, and (3) standard materials. Pilot-scale testing has long been considered the most reliable way to perform test work. However, the cost of testing, the difficulty in getting representative samples, advances in design methodologies, and greater experience with newer comminution circuits have tended to decrease the importance of pilot testing.

Typically, grindability tests rely on (1) a known size distribution of feed, (2) the application of a known or measured amount of energy to (3) a measured feed rate, and (4) a measured product size distribution. Based on these four items, an index is then calculated. The traditional Bond work index is perhaps the best known, but similar indices are calculated by other tests. Grindability tests that reach a locked cycle equilibrium or steady-state continuous operation are a better indicator of actual mill performance than batch tests. The design methodology for HPGRs using a bench-scale test (such as Polysius' LABWAL test) is largely analogous to this approach, except that primary index is a specific throughput, with measurements for specific energy input and the comminution outcome of that energy input also measured.

Single-particle breakage tests typically endeavor to quantify breakage as a function of energy input. Regardless of the test, using a breakage energy regime as close to the actual milling conditions as possible is preferred. Single-particle breakage tests provide fundamental data critical for a simulation effort; traditional grindability tests are unable to generate this data. Unfortunately, tests of this type cannot reliably predict the steady-state mill load; for autogenous/semiautogenous milling, the composition of the mill load defines mill performance.

Some comminution machines use conventional civil engineering measurements of strength as a direct basis for design, such as unconfined compressive strength. This is particularly true for crushers. Aside from the comminution tests presented, other indicative data can be used to map the overall comminution profile of an ore body, particularly rock quality designation and point load testing. Such tests, though not directly usable for design, can prove beneficial in determining the distribution of ore regimes. In some cases, correlations to comminution tests using these (inexpensive and quick) test results can prove constructive.

In addition to breakage tests, because media and liner costs represent significant operating costs, tests to determine wear rates are often conducted. The Pennsylvania abrasion test (often called the Bond abrasion test) is the most widespread. However, plant correlation with test results is often not particularly good. Because the test is conducted dry, it does not measure corrosion, a significant source of media consumption in wet grinding circuits. Abrasion data can also be obtained using specialized equipment, such as Polysius' ATWAL test for HPGRs, or by direct measurements of wear in pilot testing. A good approach is to use these tests to determine the relative wear of an ore type to others, then to benchmark with similar circuits and similar ores in operation.

Pilot testing involves larger-scale, steady-state testing at a range of operating conditions (including a feed size similar to the anticipated design) to evaluate potential design and operating scenarios. The response is then either directly scaled from pilot-sized to full-scale equipment using either a volumetric or a power basis. Pilot testing offers a high degree of confidence but can be inadequate in actual utility because of the limited number of variables that can be reasonably tested, the amount of sample required, and the logistics and expense of sample acquisition.

A number of tests have been developed over the years with various strengths and weaknesses. This is partly a reflection of proprietary design approaches by various companies and designers, and perhaps also a reflection of the never-ending quest for a test that can fully define the comminution response of an ore type quickly, with little sample, and at low cost. A summary of selected ore characterization tests are presented in Table 14.2-4. Most of the tests summarized are conventional comminution tests relevant for primary and secondary grinding operations. Bench-scale and pilot-scale HPGR tests are included, with direct consultation with manufacturers recommended for tertiary grinding tests (most of which rely on direct scaling based on unit power input). All testing (to include the benchmark of pilot testing) presents trade-offs, in terms of data developed, with the time, effort, and cost of sample collection and testing. In many cases, factoring how representative the samples tested are of the ore body presents its own challenges. Fundamentally, a large number of poorly selected samples cannot overcome a lack of sample design. Data developed from testing samples that poorly represent the intended population are at best useless and at worst damaging in that they can contribute to incorrect designs or performance projections.

Ore testing should also consider the methodology of the subsequent circuit design. Traditional power-based design methods are generally based on work indices measured during grindability tests. Then these data are typically scaled based on power input. Simulation circuit design generally relies on the results of single-particle tests. A combination of computational mathematical modeling (circuit simulation) and use of traditional work indices are generally used in current practice. Additionally, to broaden the applicability of certain tests, some designers have developed correlations (some with defined bands of variance) that allow prediction of single-particle test parameters from grindability tests (particularly the SMC Test; Morrell 2006) and vice versa.

A sound approach for ore-body comminution is an integrated test program using both well-designed steady-state tests and single-particle breakage tests. These tests may be complemented with other tests to map a deposit and with equipment-specific tests (i.e., bench-scale tests for specific comminution machines) supplanting or replacing portions of the test program. Simulation of bench- and pilot-scale mill tests via discrete element modeling or other high-definition models is well within the scope of today's technology. Perhaps single-particle ore characterization in conjunction with high-definition modeling and scaling of steady-state bench-scale tests will become the norm for future design work.

Circuit Design

Given a design throughput, grind size target, and ore breakage characteristics for an ore body (most appropriately evaluated as a range and ideally based on a preliminary mine plan), a number of circuit designs can be prepared for evaluation. Based on circuit specifications, ore characteristics, downstream processes, location, operator preferences, and scoping

Table 14.2-4 Summary of ore characterization tests

Test	Mill Diameter, m	Top Size, mm	Core Size, Standard Reference	Peak Energy*, J/kg	Peak Energy, J	Sample Requested, kg	Sample Used†, kg	Closing Size, mm	Type
AG/SAG pilot	1.83	100–150	N/A‡	50	150	~10,000/test	Varies	Varies	Continuous
Media competency	1.83	165	N/A	18	100	750	400	N/A	Batch
Pilot HPGR	0.71‡	32	NQ	N/A	N/A	700	Varies	Varies	Continuous
Bond impact	N/A	75	PQ	500	200	Twenty 50–75 mm rocks	7.5	N/A	Single particle
Drop weight	N/A	64	HQ	1,400	450	75	24	N/A	Single particle and batch
MacPherson autogenous grindability	0.45	32	NQ	70	3.3	135	100	1.2	Continuous
SMC Test	N/A	32 or 22	HQ-NQ	12,600§	500	15	2.5	N/A	Single particle
SAG Power Index	0.305	25, F_{80} = 13	NQ	8	0.2	10	2	N/A, P_{80} = 1.7	Batch
Bench-scale HPGR (LABWAL)	0.25**	12.5	BQ	N/A	N/A	250	25	N/A	Continuous
Bond rod mill	0.305	13	Any	500	1.6	20	10	1.2	Locked cycle
Bond ball mill	0.305	3.3	Any	—	0.6	10	4	0.149	Locked cycle

Source: Adapted from Mosher and Bigg (2002); McKen and Williams (2006).

*Peak energy is estimated as the nominal maximum impact breakage energy input based on the highest energy event on the coarsest size range used in the test.

†Sample is the amount typically used in conducting the test and typically still usable for metallurgical testing if required. The amount is typically less than requested based either on the amount required for sampling purposes or to allow for repeat testing by the laboratory.

‡N/A = not applicable.

§S. Morrell (personal communication).

**Diameter for HPGR tests specifies rolls diameter.

and prefeasibility studies, a short list of comminution circuit types can generally be developed that merit development of a process design.

Fundamental to achieving size reduction is the application of power; applying power at peak efficiency reduces unit operating costs and ensures efficient utilization of capital. The machines most widely used to apply comminution energy have been discussed in previous sections. The machines described apply energy via two mechanisms:

1. Directly through devices that break particles through compressive force (crushers, for example)
2. Initially to impart motion to a grinding charge, a portion of which energy is then translated into particle breakage

For the first case, the machines are generally controlled by volumetric throughput. The amount of power drawn to impart compressive forces is a direct function of the hardness of the material being broken by compression. As a result, the unit power can vary based on material type at a fixed throughput. For a given material type and circuit/machine configuration, the size distribution of the product tends to be fairly consistent.

In contrast, application of power in tumbling mills is somewhat more complex. Once power is drawn in a rotating mill, a portion of the power is applied to the charge, resulting in breakage through the mechanisms of impact, abrasion, and attrition. With a steady-state charge, the power imparted is constant, regardless of material hardness or throughput. In overflow mills, the throughput through the mill can be readily varied, and there is a degree of control over the unit power input. A SAG mill's throughput, though, is volumetrically controlled based on the mill's discharge grates. As throughput decreases in a tumbling mill with a constant power draw, the unit power increases, and a finer grind results.

In both cases, though, the fundamental driver of the amount of breakage achieved is the amount of power delivered to each unit mass of material. Therefore, in designing for volumetric throughput machines, it is essential to have a firm understanding of throughput and how much power will be drawn under the operating conditions used. In tumbling mills, it is essential to understand the power draw that the mill will achieve and the range of throughput that can be expected under such conditions.

Historical articles on comminution circuit design have presented tabulations of power draw estimates for mills and machines of a given size. Indeed, substantial effort by mill designers went toward understanding how the power installed on a mill would be drawn. In numerous case studies, the installed motor power was inadequate for operation at the desired operating conditions. Conversely, motors with rated power outputs larger than could ever be drawn have been installed, leading to confusion on the part of designers and operators who believed that this power should be available for grinding.

Much of the art of power estimation was converted to direct computation through the development of a mathematical model by Morrell (1996). His model is an outcome of modeling the physics of the power required to impart charge motion and lift; fundamentally, a mill's power draw is the sum of the power (applied over time) required to impart kinetic and potential energy to the mill charge. The power draw of a mill is a function of (1) the charge shape (including the volumetric filling of the mill, the charge density, amount of slurry pooling, etc.), (2) charge position (particularly top [or shoulder] and bottom [or toe] position of the charge), and (3) the mill speed. It is evident to practitioners that these three fundamental variables of mill power draw are interrelated—for example, charge shape and position are impacted by changes in mill speed.

The critical question in design is the efficient application of power. As with ore characterization, there are a number of design techniques. The use of power-based, simulation-based, and direct-scaling benchmarking with existing plants

or (preferably) a combination of techniques can be used to develop circuit designs for evaluation. Regardless of methodology, a key metric of the design process is the design power input (kilowatt-hours per ton) for each ore type. Designers often evaluate (and sometimes rank) circuits in terms of unit power input; mill operators more commonly reference circuit performance based simply on throughput at target operating conditions.

In power-based designs, the Bond equation continues to be a fundamental tool. The correlation between the power applied in a grinding operation and the amount of size reduction attained has long been intuitively known, with several relationships between the two proposed through the years. Over decades of application, F.C. Bond's third-theory equation has an established track record for both design and analyses of conventional grinding circuits. The standard Bond equation is presented in Equation 14.2-1 (Bond 1960). The equation can also be used to evaluate the relative power efficiency of simulated and operating circuits. By rearranging the formula to Equation 14.2-2, the operating work index (Wi_o) of an existing grinding circuit can be calculated, given known mill power draw, throughput, feed, and product size distributions. The use of operating work indices can be useful for forecasting future mill performance and as a benchmark to compare against laboratory measured work indices.

$$\text{kW} \cdot \text{h/t} = Wi * \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right) \tag{14.2-1}$$

$$Wi_o = \frac{\text{kW} \cdot \text{h/t}}{\left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}} \right)} \tag{14.2-2}$$

where

kW·h/t = power applied per ton of feed to the grinding unit
Wi = standard Bond work index of the ore
P_{80} = 80% cumulative passing size of the product, in μm
F_{80} = 80% cumulative passing size of the feed, in μm

Care must be employed in using the Bond equation for sizing individual unit operations when the feed and product particle size distributions are not parallel; this challenge is an artifact of describing a particle size distribution through the use of one number (represented by the P or F size). For example, SAG mills and HPGRs both produce substantially more fines than a traditional crushing plant for a given F size. As a result, an operating work index of a ball mill following a SAG mill or an HPGR is substantially lower than the operating work index of a ball mill following a rod mill or crusher. Similarly, the operating work index of the unit preceding the ball mill will be higher for those machines producing substantial fines than for crushers or rod mills. The apparent advantage in ball mill efficiency is a misnomer; the ball mill simply has less work to do with greater proportions of fines in the feed. This effect can be addressed in a number of ways, but recognizing it is essential in design (McKen et al. 2001).

Power-based designs rely on circuit configurations that will deliver the targeted energy input. As such, sound use of computational methods, tables, or benchmarking to ensure that the designed mill/motor configurations will draw and apply the target power is essential. Aside from proper equipment sizing, the most critical pitfall of a power-based design is ensuring that the design is achievable in terms of the assigned power split for each grinding stage. Many designs have relied on arbitrary power allocation between grinding stages. When commissioned, it was found that throughput constraints between units would either prevent the design throughput from being achieved, most often while attaining a finer grind than designed, or vice versa.

Simulation-based designs use a set of mathematical models to calculate the anticipated response of a circuit under a set of operating conditions. A number of commercial and proprietary simulation packages are available. Major equipment manufacturers often use in-house simulation models. Many of these packages are based on a combination of empirical and fundamental models, and correlations with observed plant performance.

Simulation is a powerful tool that offers the ability to evaluate circuit response for a range of operating conditions and of a given circuit for a range of feed types, and readily compare a number of circuit options. It can also provide a great deal of useful process information (in terms of the volumes and size distributions of intermediate flows) that is unavailable from power-based designs. Notably, simulation offers the ability to evaluate the impact of classification on the comminution circuit, something that cannot be done with power-based techniques. In addition to computer simulation to model steady-state comminution circuit performance, discrete event simulation can be useful in evaluating overall circuit response with respect to linkages with discontinuous (truck dumping) or semicontinuous (such as a crushing plant with periodic stops due to metal detects) processes.

When conducting a design exercise, it is critical to validate that the circuit being modeled is within the boundaries of the underlying model. Although this is generally the case for typical processing plants, the use of models outside of the operating window for which the model was developed can be misleading. Calculating the operating work index of circuits developed using circuit simulation can provide a cross-reference (reality check) and can be useful for evaluating the relative efficiency of circuits.

Direct scaling is employed for a number of comminution machines and, depending on the design approach, can be based on achieving a target unit energy input (kilowatt-hours per ton) or on a specific throughput determined in laboratory testing. For single unit operations, and when used in conjunction with a good database of existing plant operations, robust designs can be produced. However, examples frequently emerge of direct scaling and commissioning of plants that could not achieve the design throughput but are capable of grinding finer than target grind, or vice versa—that is, they exceed nameplate throughput but do not achieve the target grind.

Regardless of the design technique, benchmarking is a useful tool. Frequently, comparison of ore breakage characteristics and circuit design parameters can yield a list of relevant similar operations. In fact, although this approach may not be bankable, it is widely used for low-risk, self-financed expansion projects or when scaling the anticipated throughput of a new deposit in an existing mill. Using it as the sole basis for design, however, can introduce a higher degree of uncertainty in the design outcome.

An additional consideration is the question of how to integrate the stages in a comminution circuit. Primary grinding units, whether they are crushers, AG/SAG mills, or HPGRs, arrive at their unit power input largely based on the volumetric throughput at which they are operated. For AG/SAG mills at a given operating condition of speed and load, the power input

is fixed, so the only variable is feed rate (which will change based on ore hardness and size distribution). For crushers or HPGRs, though, the feed rate is fixed based on volumetric throughput at a given aperture, and the power drawn varies as a function of the ore characteristics. Regardless, a design methodology that considers designing the primary milling stage based on the required throughput, with the secondary stage based on the amount of power required to finish grind the design throughput, is more successful than a methodology that counts on a set amount of power input in the primary grinding stage.

For power-based designs, determining the appropriate and efficient way to input the design grinding power is the core question. For simulation-based designs, determining the amount of power drawn under a given set of simulated conditions is required. Cross checking this value with calculated power requirements or benchmarking with operating circuits is wise. For scaled designs, it is essential to ensure that power input scales similarly to volumetric throughput, or the unit power inputs will not match, and consequently the desired size reduction may not be attained at the target throughput.

Most modern circuit designers consider the various design techniques to be complementary. Although various designers may have preferences, a robust design will consider several different approaches. A higher degree of confidence can be assigned to a design that is confirmed based on traditional power-based approaches, with various operating conditions investigated and evaluated by simulation, and which benchmarks and scales favorably with similar operations. Although any of the techniques can be used independently, the most robust design will eventuate through the combination of these techniques.

Circuit Selection

Staged crushing plants are favored in cases of low-tonnage operations where little fines generation is desired and a dry, coarse product is required (heap leaches), or in moderate tonnage operations with hard ore. Plants that will be fed sticky ores, with large amounts of tramp metal, or high-tonnage operations are generally less favorable for a traditional crushing plant. A benchmark for annual run time (including mechanical availability and all other factors) for a well-maintained and run modern crushing plant is 85%. Lower availabilities are typical in more-demanding applications.

Semiautogenous milling is favored for high-capacity plants, soft to moderately hard ore types, sticky or clay-bearing ores, and ores containing significant amounts of tramp metal or magnetite. Ores with a competent fraction, particularly higher-density ores, can be amenable to autogenous grinding. This is particularly true for small to medium throughputs with a long mine life. Because AG mills tend to have a lower unit power input for a given mill size, they can be more expensive to construct but much less expensive to run (when compared on unit cost basis) compared to a SAG mill. Circuits designed for autogenous grinding are often designed to accommodate semiautogenous grinding either for operational flexibility or with the anticipation of milling future ore bodies; a case study of an operation with a versatile single-stage AG/SAG circuit is Brunswick (Larsen et al. 2001).

When using the largest mills available, a single SAG mill often produces feed for secondary grinding two or more ball mills. In a typical SAG–ball mill circuit, the SAG circuits should generally be designed for the target throughput, with the ball mill circuit designed to achieve the target grind. Based on ore breakage characteristics, feed size, grate aperture, and circuit closing size, the ability to shift the transfer size substantially between the SAG and ball mill circuits is often limited. Though exposure to significant downtime for a failure in a single line is often considered a drawback, SAG plants have demonstrated annual run times of 95%.

HPGRs are considered favorably as a circuit alternate to autogenous/semiautogenous milling. Such a circuit must be designed in conjunction with secondary crushing to reduce ROM ore to a size suitable for HPGR feed. HPGRs, in closed circuit with screens, then produce ball mill feed. HPGRs were selected for the Cerro Verde Cu-Mo project commissioned in 2006 and have also been specified for the Boddington project. Particularly in the case of hard and abrasive ores, HPGRs can offer operating cost benefits relative to SAG. Depending on the delivery time for large SAG mills (and often its accompanying wraparound electric motor), delivery and construction of HPGRs may be executed more quickly than for a SAG mill circuit. The largest HPGRs can be reasonably sized with a single large crusher to feed one ball mill for typical circuits. A crusher and closed-circuit HPGR grinding line feeding a ball mill will typically have lower availability and more unit operations than a similar SAG/ball mill line.

Substantial progress has been made in terms of availability and reliability. Preventive maintenance periods are required for cheek plates and inspections, but HPGR maintenance is rarely the controlling factor in overall circuit maintenance. Aside from rolls changes (akin to a mill reline and typically occurring between 2,000 and 15,000 running hours, depending on the material and application), more often circuit availability is controlled by belt, chute, screen, and conventional crusher maintenance. Circuit availabilities between a traditional crushing plant and a SAG mill should be expected for circuits where a secondary crusher/screen/HPGR replaces a SAG mill, with 90% run time a nominal benchmark. Aside from the fact that the degree of ancillary equipment (belts, transfer chutes, additional pumps, etc.) can increase the maintenance demand relative to a SAG plant, the additional unit operations also drive up capital costs.

Circuit selection is typically based on the calculation of the net present value of various circuit options (implicitly incorporating both differences in capital and operating costs and in performance between plant types). Qualitative considerations include location, operator preference, equipment lead times, and the like.

Selection of motor and drive types for tumbling mills is a specialized topic in itself. For larger mills (nominally more than 7.2 m [23.6 ft] in diameter for ball mills and more than 9.8 m [32.2 ft] for SAG mills), gearless (wraparound motor) and geared (pinion/bull gear combinations) drives are normally compared. Gear manufacturers indicate that geared drives with up to 10 MW per pinion are feasible (Hankes 2001). Failures with both large wraparound motors and large gears have occurred in the industry, and engineering trade-off studies will likely continue for those mills where either geared or wraparound motor drives could be used.

MEDIA AND LINERS

Table 14.2-5 summarizes the typical media sizes and types in the broadest use for mills discussed in this chapter. Contemporary practice in tumbling mills is to add a single-size makeup ball mass to maintain constant power draw. Ball

Table 14.2-5 Typical media sizes and types

Media	Size*	Type
SAG mills	127 mm (100–150 mm)	Forged steel
Ball mills	50–65 mm (25–100 mm)	Forged or cast steel
Regrind/fine-grind ball mills	25 mm (9–40 mm)	Forged or cast steel
Tower mills	12 mm (6–25 mm)	Forged, cast, or other (e.g., shot) steel
Pin mills	12 mm (3–12 mm) (Metprotech) 3 mm (SMD)	Steel or ceramic Ceramic or sand
IsaMills	3 mm (1–6 mm)	Sand, slag, ceramic

*Common makeup ball sizes indicated, with a typical range indicated in parentheses; coarser or finer sizes may be found in certain applications.

charging should be done on a daily basis or, more ideally, on a shift-wise basis. Some plants with a mechanized ball charging system (typically for SAG mills) have adopted continuous charging (at a rate matching the mill's steel consumption) to facilitate operating at peak conditions. As steel media most often have a hardness profile with a softer center, once balls reach a certain size, they tend to wear very quickly.

The use of a mixed-size makeup ball charge has been repeatedly demonstrated in laboratory studies to offer incremental efficiency benefits, but the benefits have been difficult to document at plant scale. Selected operations have adopted mixed-sized makeup ball charging for SAG mills, but it remains uncommon for ball mills. There is little reason to believe that the steady-state media size distribution resulting from the wear rate of the makeup ball size corresponds to the optimum ball size based on the mill's feed and target grind.

The concept of selecting a makeup ball size based on the coarseness of feed has long been recognized (Taggart 1945). In general, a mixed makeup ball charging regime improves grinding efficiency, with greatest potential benefit for single-stage milling applications with large size reductions. McIvor and Weldum (2004) suggests that the benefits from graded charges extend to pebble mills and hypothesizes that this is a fundamental reason for improved efficiency (relative to laboratory-measured work indices) at the Empire mill. Nonetheless, most operations tend to use a single-sized makeup ball size for reasons of convenience. In practical terms, for grate discharge mills, there is essentially zero grinding media smaller than the grate aperture at any given time. For overflow mills, the lower media size is constrained by the size at which media flow out of the mill (which, in turn, is effected by volumetric throughput, pulp density, and media size and density).

The use of ore as AG media was reviewed previously in the "AG/SAG Circuits" section. Pebble milling (a form of secondary milling) most often uses oversize material extracted from the primary milling circuit as media. This is most often done in applications with higher ore densities and can represent a substantial savings in milling costs. With ore densities lower than steel densities, larger mills are often required, and some degree of flexibility may be needed when the generation and consumption rates of AG media do not match.

The use of recycled SAG scats (worn and broken media recovered via magnets from SAG mill grate discharge) in ball mills has grown in application. The balls are direct charged to ball mills as a replacement for new grinding balls. The larger soft core of SAG media (as a result of larger size and requirements for ball durability) means that the recycled media wear more quickly than a typical like-sized new ball mill ball. Discussion of the use of alloys, cast/forged or other types of media, or whether to effect improved wear life through surface hardness, hardness profile, or improved resistance to corrosion is outside the scope of this chapter.

Liner innovation has contributed substantially to comminution performance. Nowhere has this been more evident than in semiautogenous milling. After early work that tended to focus more on maximizing life, current best practice is focused on maximizing total productivity over the life of a liner set. Generally this means development of a shell lifter set that facilitates (within the constraints of original mill shell drilling) a quick startup period with little to no lifter packing and for which full power draw can be maintained over the planned life of the liner set. Critical aspects of shell lifter design include lifter spacing, height, and face angle. In general, common practice has evolved toward operation with liner sets that are more open (have greater spacing between lifter bars relative to lifter heights). In general terms, this philosophy leads to less liner packing, the shortening (or elimination) of a break-in period with new liners, and higher mill performance over the liner set. Going too far can lead to short liner life offsetting performance benefits and, in extreme cases, insufficient charge lift to achieve baseline mill performance. Design of pulp lifters and discharge grates has been an area of intense effort by mill operators, particularly large, high-volumetric throughput mills.

Because of the HPGR's role in replacing duties performed by mills, HPGR liner wear is often relevant in comparison with liner and media cost. Cost savings relative to the overall costs of liner and media consumption in a rotating mill can often be realized. Relative to mill liners and grinding balls, HPGR wear surfaces are much lower volume and a more highly engineered product. The unit cost of media wear, generally in terms of dollars per metric ton (at equivalent kilowatt-hours per ton) is a good metric for comparison.

The use of liner handlers to reline mills is now standard practice, and the use of larger machines has facilitated consolidation of liner parts. The world's current largest liner handler is in use at PT Freeport Indonesia; the machine's 7,250-kg capacity has allowed consolidation of parts and reduction in the number of units requiring handling during a reline. Progressive refinements, with each pushing the then-existing technology's capacity in terms of casting integrity and liner handlers, now facilitates the use of 40% fewer liner pieces than the original design in PT Freeport Indonesia's 10.4-m (34-ft) and 11.6-m (38-ft) SAG mills. The use of a sound, integrated, and well-resourced maintenance approach can lead to annual run times in the 95%–96% range in even the most challenging environments. Such techniques can also be applied to ball mill liners or to primary crusher concave relines (e.g., with larger castings, reducing the number of rows from a typical five-row installation to three-row).

LOOKING FORWARD

This chapter has summarized a range of comminution options for mineral processing applications. Although machine design, performance, maintenance, and other aspects have been touched upon, the focus has been on the process. Since the last comminution chapter in the previous edition, the range of equipment options for mineral processing comminution plants

has broadened. AG/SAG circuits as well as circuits incorporating HPGRs are in wide use. In particular, AG/SAG mills and HPGRs have blurred the distinctions of using a certain machine for a certain feed size.

The low fraction of total power input in the crushing and grinding process that actually goes toward size reduction (with the balance being dissipated largely as heat and noise) has long been cited (Kelly and Spottiswood 1995). Despite intense research efforts with both public and private sponsorship, no viable commercial processes with markedly better efficiency are on the horizon. With increasing focus on energy input, from both high energy cost aspects but also considering the embodied energy and greenhouse gas impacts of various products, these efforts will continue. Interestingly in the case of energy input in mineral processing applications, the effects of increased energy input for liberation are offset by the benefits of improved resource recovery.

To the extent that the marketplace can accurately assign a value to the material produced and a cost to the negative impacts, assessing the benefits of increased comminution to affect increased recovery versus the costs of higher energy consumption should be clear. The focus on comminution as often the highest-cost category of mineral processing (both in terms of capital and operating) is likely to increase given continuing demand for resources and increasing awareness and attempts to mitigate the impact of the human carbon footprint. At first glance, decreasing the comminution effort may seem to be an easy target to achieve both cost and carbon footprint benefits. In many cases, though, the incremental comminution effort required for the incremental value recovery is less than the energy requirement to recover a like quantity of values from a freshly mined ton of ore. From an energy perspective, increasing comminution will in many cases be favorable to mining and milling new ore tons. Thorough study and cost quantification are required for a true optimization of the overall process.

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CHAPTER 14.3

Classification by Screens and Cyclones

Brian Flintoff and Ronald Kuehl II

INTRODUCTION

The classification of particles based on size is an important step in almost all mineral processing flow sheets. Although the technical focus in plant design, control, and optimization is most often on comminution (e.g., size control and mineral liberation) and/or separation equipment (e.g., recovery/yield and grade), classification plays a critical role in optimizing process efficiency. This chapter focuses on two major classes of classification unit operations: screens and hydrocyclones.

SCREENS

Matthews (1985) said, "Screening is defined precisely as a mechanical process that accomplishes a separation of particles on the basis of size and their acceptance or rejection by a screening surface. Particles are presented to the apertures in a screening surface and are rejected if larger than the opening or accepted and passed through if smaller." The screening process has been a part of mineral processing flow sheets for a long time, and like many other unit operations, screening has seen some interesting mechanical and process developments over the past few years.

Looking across all of the process industries, there are many different kinds of screens and applications. However, the authors have restricted the discussion to the most common applications in the mining industry. Figure 14.3-1 is a generic graphical representation of the major applications of screening in mining: scalping, size control, and sorting.

Table 14.3-1 provides more amplification on the type of screen one might find in given applications.

Screen types can generally be categorized as follows:

- **Fixed types:** grizzlies, riffles, sieve bends, and so forth
- **Moving (linear) types:** vibrating, reciprocating, and resonance
- **Moving (rotating) types:** cylindrical trommels

Figures 14.3-2 through 14.3-6 are photographs of a grizzly screen, vibrating screen, trommel screen, Stack Sizer, and sieve bend. In general, these are the workhorses of the mining industry, with the vibrating screen standing out as the most widely employed screening unit operation. It is for this reason that most of the chapter is devoted to the vibrating screen.

Vibrating Screen Basics

The vibrating screen is ubiquitous in mineral processing plants. One would imagine that with this breadth and history of application and its apparent simplicity, there would be few remaining unknowns in the screening process. However, the design and operation of screening systems remains a blend of art and science—a statement used by many other authors (e.g., Mathews 1985). It is interesting to note that the screening process is enjoying something of a renaissance today, as new simulation tools offer a closer examination of the physics of the process, providing quantitative insights into mechanisms and their relation to design variables. In addition, new instrumentation offers real-time modulation and measurement of mechanical and process condition. The following section deals with vibrating screen basics and introduces some of the latest tools and techniques in the design, analysis, control, and optimization of vibrating screens.

Principles of Operation

The vibrating screen is used here as a proxy for all other screens, as the basic operating principles are similar or the same. In the screening process, a feed material comprised of particles of varying sizes is presented to the screen media (collectively, the screen deck) in such a way that those particles finer than the screen aperture fall through to the undersize stream, and those that are larger than the screen aperture continue moving along the screen surface to eventually report to the oversize stream.

Screening can be carried out with wet or dry feeds, although coarser particle separation (greater than a 5-mm aperture) is usually performed dry (at the surface moisture of the feed). Wet screening usually involves either sticky/clayey feeds or finer particle sizes where the solid material has been slurried to facilitate transportation and processing. In dry applications, some combination of vibration and gravity are responsible for particle transportation. In wet applications,

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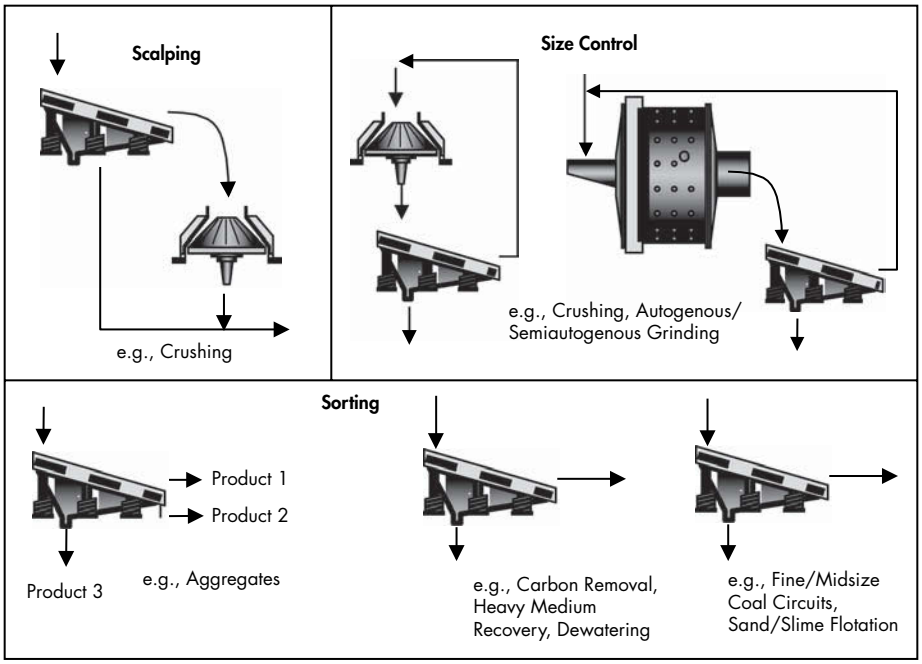


Figure 14.3-1 Examples of the application classes for screening in mining

Table 14.3-1 Types of screening operations

Operation and Description	Type of Screen Commonly Employed
Scalping: Scalping is strictly the removal of a small amount of oversize from a feed consisting predominantly of fines. Scalping typically consists of the removal of oversize from a feed with a maximum of 5% oversize and a minimum of 50% half size. Coarse scalping is typically smaller than 150 mm (6 in.) and larger than 50 mm (2 in.).	Coarse (grizzly), fine (same as fine separation), and ultrafine (same as ultrafine separation). *Example*: Grizzlies are usually used to scalp oversize material from primary crusher feed or are used to scalp fines from the feed to crushers or grinding rolls.
Course separation: Consists of making a size separation smaller than 50 mm (2 in.) and larger than 5 mm	Vibrating screens (banana, incline, and horizontal) and trommel screens. *Examples*: The removal of pebbles from a semiautogenous grinding discharge stream, sizing material for the next crushing stage, and sizing material for leach pile or product pile.
Fine separation: Consists of making a size separation smaller than 5 mm and larger than 0.3 mm (48 mesh)	Vibrating screens (banana, incline, and horizontal), which are typically set up with a high speed and a low amplitude or stroke; sifter screens; static sieves; and centrifugal screens. *Examples:* Sorting coal into ±28 mesh fractions for washing, and iron ore processing
Ultrafine separation: Consists of making a size separation smaller than 0.3 mm (48 mesh)	High-speed, low-amplitude vibrating screens; sifter screens; static sieves; centrifugal screens. *Example:* Size control on grinding circuit product
Dewatering: Consists of the removal of free water from a solid-water mixture and generally limited to 4 mesh and above	Decline vibrating, horizontal vibrating, inclined vibrating (about 10°), and centrifugal screens. *Example:* Wet quarrying operations
Trash removal: Consists of the removal of extraneous foreign matter. This is essentially a form of scalping operation, and the type of screen depends on the size range of the processed material.	Vibrating screens (horizontal or inclined), sifter screens, static sieves, and centrifugal screens. *Example*: The removal of extraneous organic and other matter from the leach feed in a gold plant
Other applications: Other applications include desliming (the removal of extremely fine particles from wet material by passing it over a screening surface), conveying (in some instances, the transport of a material may be as important as the screen operation), dense media recovery, a combination washing and dewatering operation, and concentration.	Vibrating screens (banana, incline, or horizontal), oscillating screens, and centrifugal screens. *Example*: Dense medium recovery in a coal washing plant, a wobbler feeder for conveying, and classifying sticky materials to a crusher

Source: Adapted from Mathews 1985.

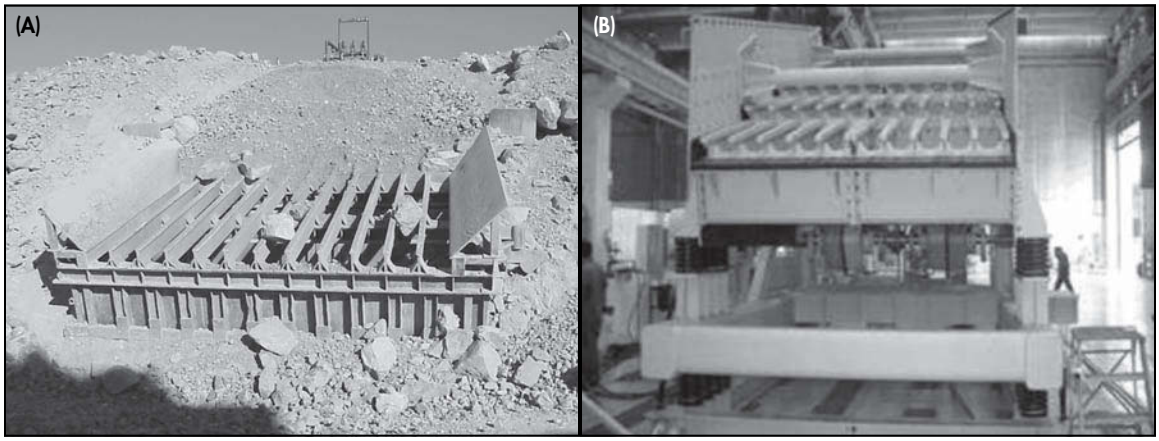


Figure 14.3-2 (A) Static grizzly (scalping oversize from a primary jaw crusher feed) and (B) vibrating grizzly screen

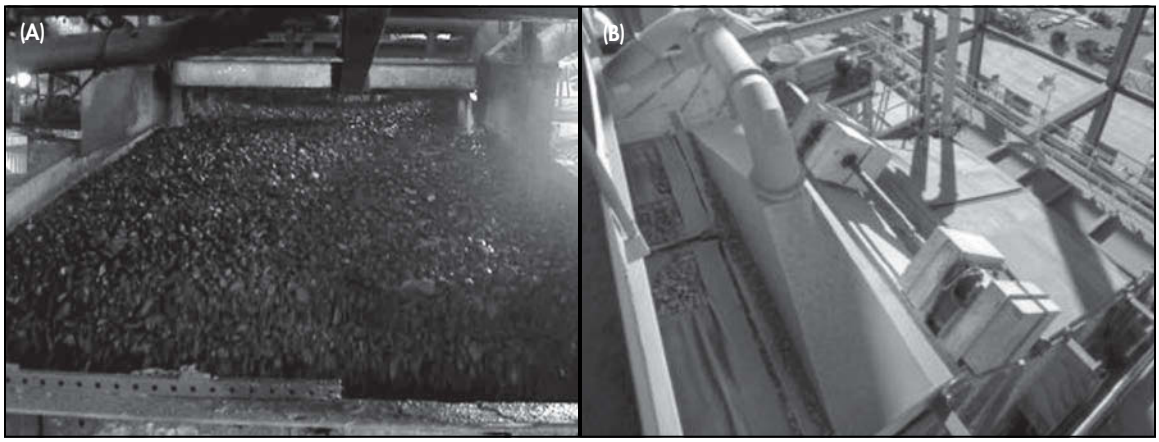


Figure 14.3-3 Vibrating screen with a dust collection system

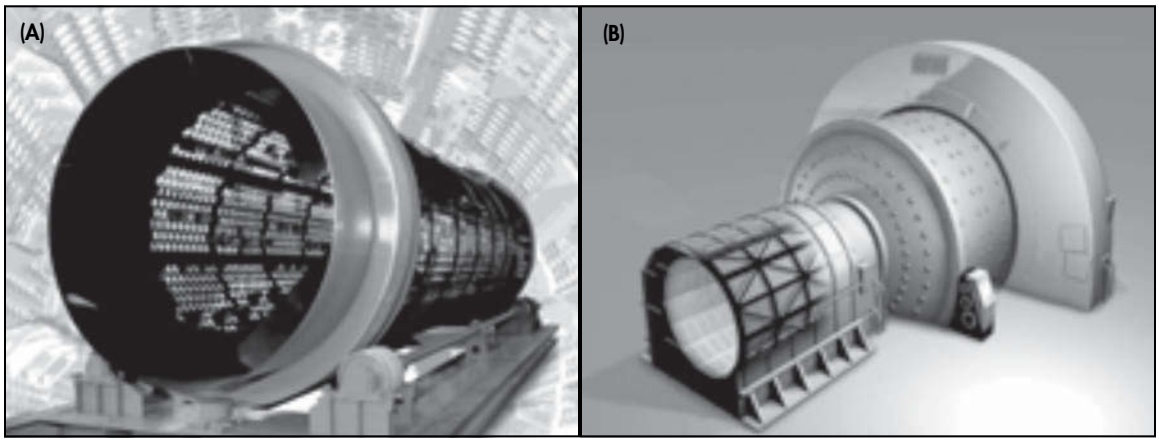
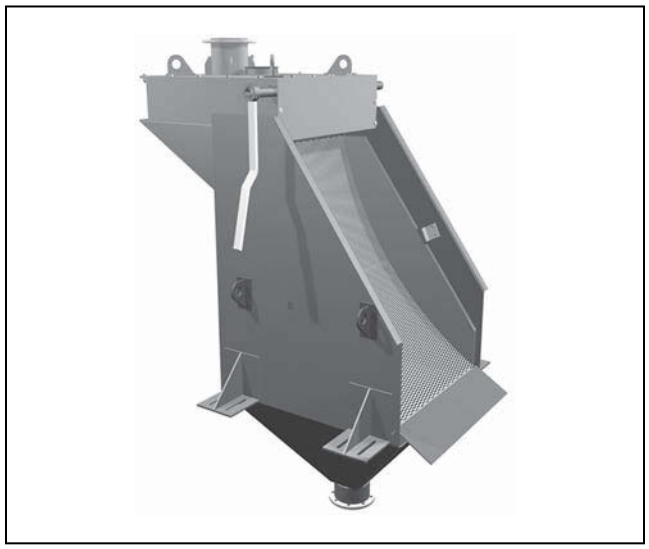


Figure 14.3-4 (A) Trommel screen and (B) mill discharge trommel screen



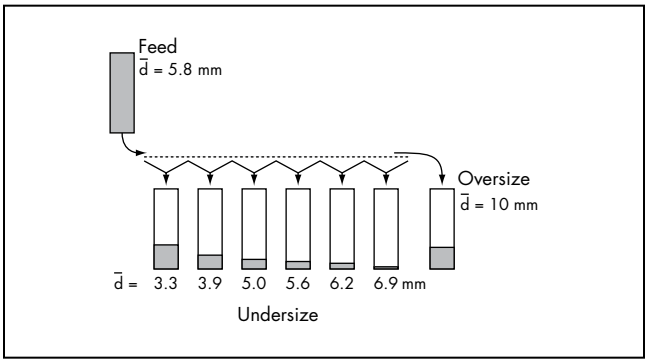
Courtesy of Derrick Corporation.
Figure 14.3-5 Derrick Stack Sizer



Courtesy of Multotec Process Equipment
Figure 14.3-6 Sieve bend

some form of vibrator mechanisms dictates the movement of material. However, hydrodynamic (drag) forces are introduced. In wet applications, deck water sprays or flood boxes are sometimes used to aid in classification. Spray selection is chosen based on two criteria: it must be decided whether the goal is to flush material through the deck or to wash off the fines. A general guideline is that water added to the feed is more effective than water sprayed on the deck in improving the classification.

Figure 14.3-7 is a schematic representation of the results of an experiment with a small horizontal vibrating screen having a screen aperture of 10 mm. (This particular screen and test [from Hilden 2007] is referenced throughout the text.) The shaded bars depict the relative mass flows, and the annotations show the mass mean size for each stream. As the feed is introduced onto the deck, it forms a bed that is usually several times the screen aperture in thickness. The bed moves



Source: Hilden 2007.
Figure 14.3-7 The relative mass flows and mean particle sizes for a horizontal vibrating screen

under the influence of gravitation and vibrational motion, and in the process, the particles stratify as the finer materials move quickly through the interstices of the larger particles and find their way to and then through the screen apertures. In this initial zone on the screen, rapid segregation means there is always sufficient fine material in the layer next to the screen deck that the flow through the screen apertures is more or less constant. This is said to be typical of crowded or saturation screening. However, as the fines are depleted, the bed height is reduced, and at some point the particles begin to act more or less as individual entities. This is the zone of separated or statistical screening, that is, each time the particle approaches the deck it has a chance (or a statistical trial) to pass through the aperture. (This notion of repetitive trials underpins many semi-empirical screen models.)

Figure 14.3-7 shows that the greatest flow of material to the undersize stream occurs closest to the feed end, which is typical of crowded screening. Segregation ensures that these particles presented to the apertures are small enough that the probability of passage to the undersize stream is ~1.0. The mass flow to the undersize stream decreases along the length of the deck, which is typical of the transition to separated screening, more typical of the coarser particles (the so-called "nearsize," as it is near to the size of the aperture) that remain in the bed. For these particles, the probability of passage on any one trial is significantly less than 1.0. The increase in the mass mean size along the deck also reflects the fact that the remaining material on the deck becomes coarser as it moves from the feed end to the discharge. This helps to explain why screen length governs efficiency. On the other hand, screen width is related to the amount of material that can be fed to the screen and be effectively separated, and therefore this is the major factor governing capacity.

On this latter point, a guideline for good dry-screen operation is for the bed depth at the discharge point to be three to four screen apertures in thickness. A guideline for wet screening is to design for a bed depth of four to six times the opening. With anything larger, the efficiency of removal of the particles close to the aperture size is reduced, as segregation is more difficult. This suggests that multiple screen decks probably should be considered, using a larger aperture size on the top deck to scalp out much of the larger oversize and reduce the loading on the lower deck for the final classification. Anything less and these particles will probably bounce excessively, reducing the number of trials and the screen efficiency. (It is

Table 14.3-2 Design and operating variables for vibrating screens

Design Variables	Operating Variables
Screen area and open area	Particle size, shape, and distribution
Aperture size and shape	Solids feed rate, distribution, and bed depth
Slope of screen deck	Feed moisture content
Speed	
Magnitude of stroke	
Type of motion	
Feed arrangement	

Source: Bothwell and Mular 2002.

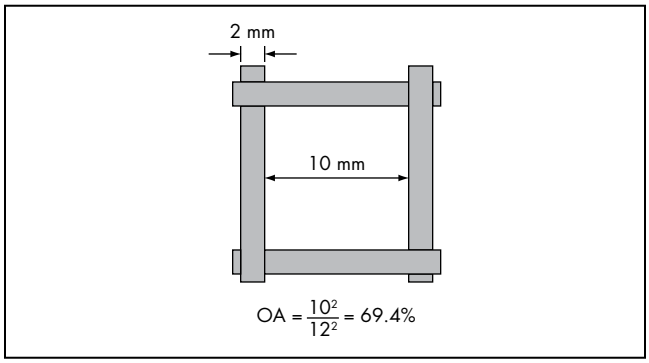


Figure 14.3-8 Computing open area



Figure 14.3-9 Blinding on a screen deck

also sometimes useful to install weirs on the deck's surface to hold back the flow as an aid to sustaining an active bed.)

Screen design and operation is about matching the screen characteristics to the (range of) feed conditions. Table 14.3-2 provides a summary of the important variables for each.

When focusing on design variables, screen area is important because capacity is proportional to width (W) and efficiency is proportional to length (L). It is common for $L \approx 2$ to 3 W. Because of the mechanical fittings on and around the screen decks, the effective area is often approximated as 90% to 95% of the actual area (LW). Screens are standard products, so the manufacturers dictate the actual sizes. It is therefore common to require more than one screen, operating in parallel, to effectively separate a feed.

Open area (OA) is expressed as the ratio of the total area of the apertures over the total active area of the screen deck. Figure 14.3-8 illustrates this for the screen in Figure 14.3-7 with a square aperture of 10 mm in a wire mesh screen with a wire diameter of 2 mm. From the explanations of screening mechanism, it is clear that the greater the OA, the greater the capacity of the screen. In practice, OA is limited by the need for sufficient mechanical strength and wear resistance of the deck and is a function of the media materials of construction, for example, wire mesh versus perforated plate.

This is a good point to introduce some common, and occasionally confusing, jargon associated with phenomena that act to reduce the OA and, hence, the capacity. The first is pegging (also called blinding or plugging), which occurs when a nearsize particle becomes firmly lodged in an aperture effectively eliminating it from the screening process. The second is blinding (also called bridging), which is the phenomena whereby smaller particles aggregate in such a way that they combine to block up the aperture, having the same overall effect as pegging, as shown in Figure 14.3-9. Blinding is more of a problem with moist or sticky materials and small aperture sizes.

The aperture size is, by convention, the minimum linear measurement in an opening; for example, the diameter for a round hole and the width (where width ≤ length) for a slot or a square. (Because screen performance assessments start with sieving samples of process streams and because the sieve has square apertures, the best correlations occur with square apertures in the screen media.) What is perhaps more important is the so-called throughfall size, h_T, which accounts for the change in effective opening as the screen is inclined. This is illustrated in Figure 14.3-10, where the angle of inclination is exaggerated to show the difference between the aperture size, h, and the throughfall size for a wire mesh screen surface. (There is a similar effect for rectangular slots.)

Although square apertures may be the most common, there is a multitude of aperture shapes. Some of the more common shapes are illustrated in Figure 14.3-11. Square or round shapes generally allow for better control of the separation size, although circular apertures are most often used in coarse-screening applications. Longitudinal slotted apertures allow slabby particles into the undersize stream. This increases capacity and reduces pegging and blinding. Transverse slots are mainly used in dewatering applications. (For steel or polymeric screen media, it is possible to design tapers and other forms of relief into the apertures to minimize pegging—see the insert in Figure 14.3-11 for an example.) Finally, the screen deck or panel is designed to withstand the load and maximize wear life. Therefore, in the case of a large spread in feed particle sizes and/or some large particles in the feed, it is common to use multiple screen decks to satisfy both mechanical and process requirements.

Inclining a screen deck increases the slope and causes the feed material to move more quickly on the deck, reducing the residence time and increasing capacity and sometimes efficiency. Vibration motion on a horizontal screen deck usually means that material velocities of 10 to 16 m/min (33 to 52 fpm), with an inclination in the neighborhood of 20°, will increase this into the range of 20 to 28 m/min (66 to 92 fpm). Generally, inclinations range from horizontal to as high as 30°, although ~20° is the usual limit. This led to the evolution of the multislope screen (also commonly known as a banana

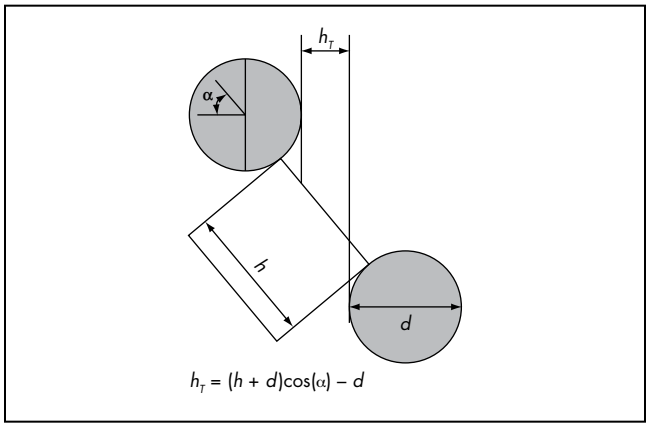


Figure 14.3-10 The relation between the throughfall size (h_T) and the aperture size (h)

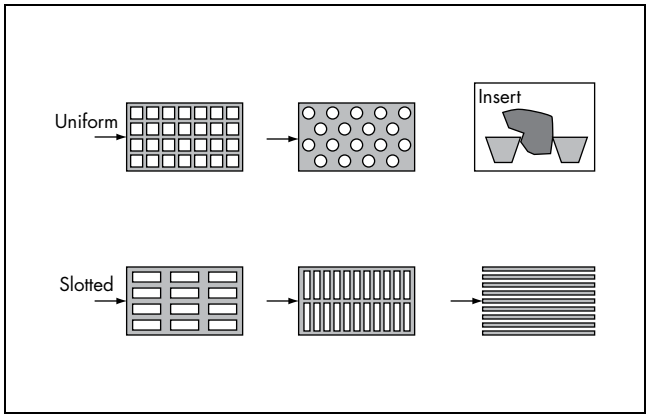


Figure 14.3-11 Common screen aperture shapes

screen). The banana screen (Figure 14.3-12) has a relatively high slope in the initial screening zone, which takes advantage of the fast kinetics of fine particle removal. The slope then decreases, which slows particle movement and offers an additional G-force to accelerate removal of the larger fine material. In the final section, the slope is relatively flat, yielding minimal velocity and maximum residence time, for finer nearsize particle removal to the undersize stream.

Vibrating screen deck motion is circular, elliptical, or straight line, with the vibrating mechanism rotating in the direction of flow or counter to the flow. Strokes are usually in the range of 3 to 20 mm (0.118 to 0.787 in.) and speed ranges from 650 to 950 rpm. A key parameter here is the G-force (G), which is computed by Equation 14.3-1.

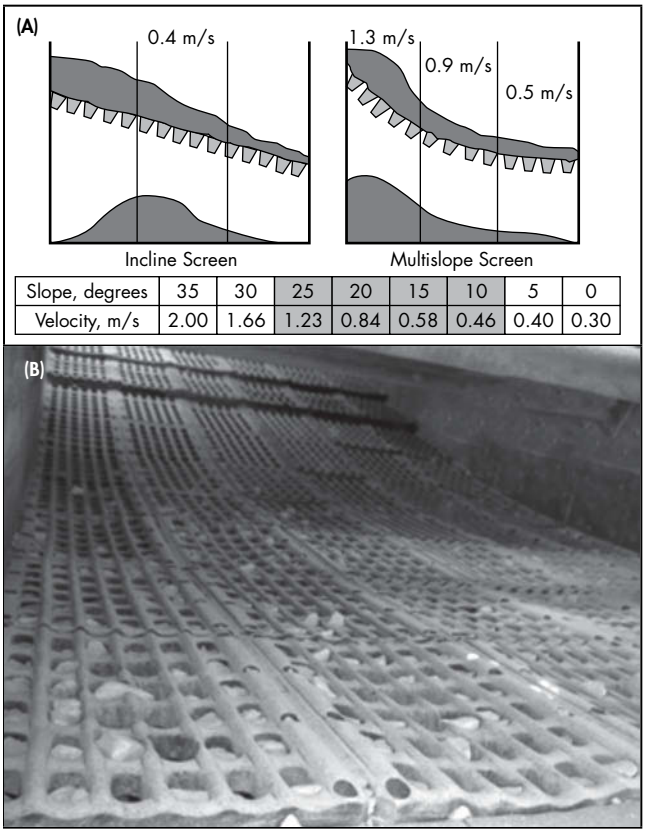
$$G = \frac{N^2 S}{1{,}789{,}129} \qquad (14.3\text{-}1)$$

where

G = the G-force (also known as the throw number)
N = speed (rpm)
S = stroke (mm)

Typical G-forces are in the 3.5-to-5.0 range, tending to higher values with heavy loads, sticky materials, or where pegging and blinding are issues. The G-force is usually higher in a horizontal screen, as it must provide the means for both stratification and material movement. (For example, with the horizontal test screen in Figure 14.3-7, the speed was 981 rpm and the stroke 5 mm [0.197 in.], giving a G of 2.69.) Table 14.3-3 amplifies on the conditions prevalent over a broad range of G-forces.

Figure 14.3-13 is a simplified graphic of particle motion with different trajectories, and Figure 14.3-14 shows how deck motion can be induced using weights on the shaft. Of



Slope, degrees	35	30	25	20	15	10	5	0
Velocity, m/s	2.00	1.66	1.23	0.84	0.58	0.46	0.40	0.30

Figure 14.3-12 Banana screen deck

Table 14.3-3 G-force implications in inclined vibrating screens

G-Force	Comment
<1.5	Practically no movement on the deck, so it is of no use.
1.5–2.5	The stratification is limited and the relative velocity between the particle and the deck is low. These parameters are suitable for soft screening and where size degradation is problematic and there is a low tolerance to impact.
3.0–3.5	The particle is presented to the screen deck at its maximum relative velocity and offers the best screening condition for inclined screens. The stratification is adequate.
4.0–5.0	The stratification is good, but the relative velocity between the deck and particle is low, offering poor screening efficiency. Bearing life is shortened.
5–6	The stratification is good, but the particle projection is in excess of the pitch (particles are in flight too long) and screening opportunities are lost. The relative velocity between the screen and particle is low, combining to give poor parameters for efficient inclined screening. The high acceleration forces can act negatively on the screen structure, and bearing life is further shortened.

Source: Moon 2003.

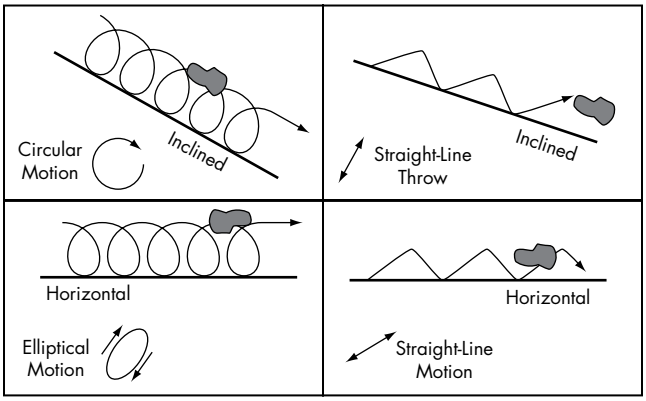


Figure 14.3-13 Particle motion as a function of the deck orbit

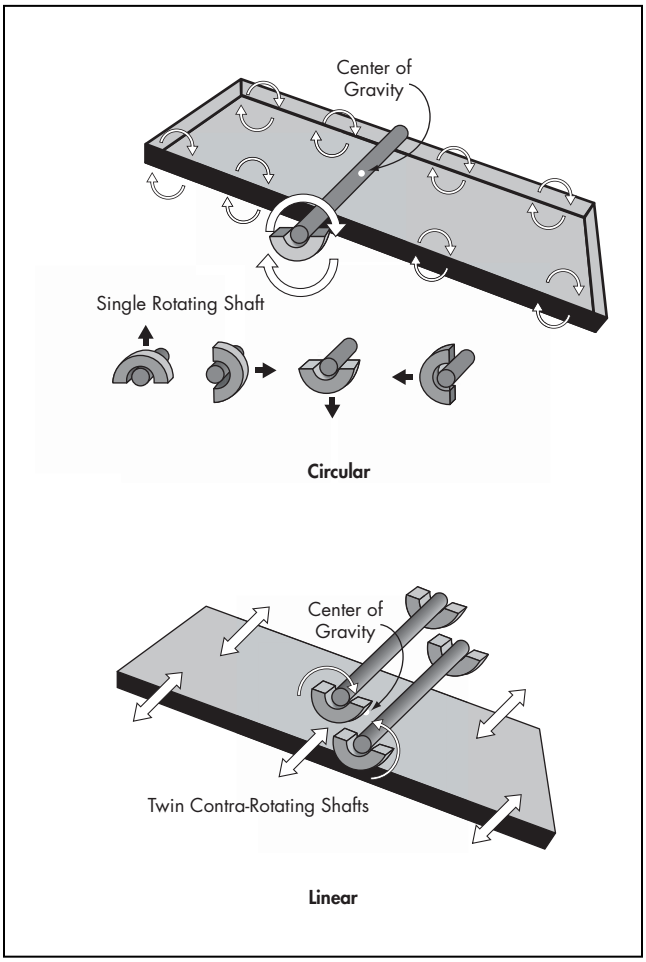


Figure 14.3-14 Using weights to induce specific orbits

course, there are other methods to induce motion and other, more complex orbits.

The stroke angle, measured relative to the length of the deck and in the direction of flow, can affect efficiency and capacity. Very low (~0°) or high (~180°) angles do little other than accelerate wear. For other angles, there is a component of vertical movement (aiding stratification) and a horizontal component aiding (<90°) or retarding (>90°) material flow. (For the screen in Figure 14.3-7, the stroke angle was 50°.)

Table 14.3-4 Process matrix for a vibrating screen*

Design Variable	Screen Capacity	Screen Efficiency	Undersize Mean Size	Pegging/ Blinding
Screen area	+	0	0	0
Open area	+	+ or 0	0	0
Aperture size	+	+	+	–
Aperture shape (increasing nonuniformity)	+	+ or 0	+	–
Deck inclination	+	– or 0	–	– or 0
Speed	+	+	0	–
Stroke angle	–	+	0	– or 0
Stroke	+	+	0	–
Orbit (from linear toward circular)	–	+	0	+

*All other factors being equal, an increase in the design parameter will cause an increase (+), a decrease (–), or no change (0) in the performance metric.

The designer for the screening application chooses the orbit motion (circular, elliptical, or linear) based on a compromise between material movement (capacity), which emphasizes linear motion, and efficiency, which emphasizes circular motion. The ellipse is an optimized solution. All give good stratification with the correct stroke angle. For horizontal and banana screens, linear motion is the defacto standard as material movement is critical. For inclined screens, circular or elliptical orbits are more common.

Finally, there is the matter of feeding the screen. Here, there are the following three general rules:

1. It is important that the feed be uniformly distributed across the width of the screen for the unit to achieve rated capacity.
2. It is important that feed rate be modulated where possible (e.g., using belt feeders) to ensure that the screen works efficiently in the face of changing feed conditions.
3. Where possible, it is good to introduce the feed having a motion opposite to the eventual material flow, which helps in distribution and avoids flooding in the initial zone of the screen deck.

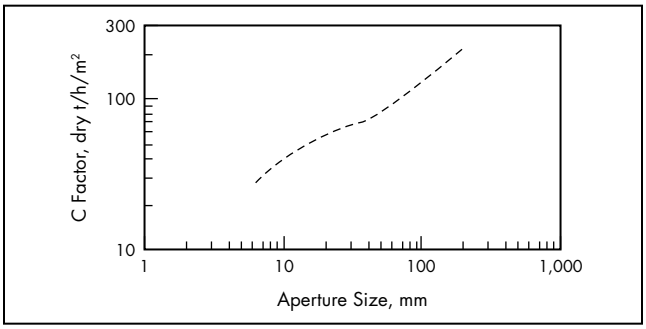
Table 14.3-4 provides a review of the major design variables of a vibrating screen and indicates usual design ranges where appropriate. This "one-variable-at-a-time" approach does not always take into account the myriad interactions among these variables.

Screen Selection

Many opinions and methods of screen selection suggest fertile ground for development and refinement. Three general selection method classes follow:

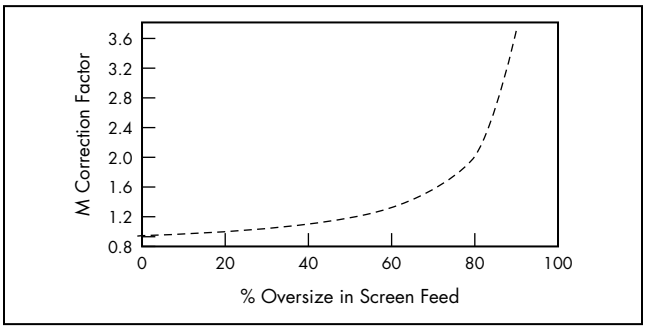
- *Class 1:* Empirical methods based on correlations derived from extensive databases of industrial experimental results
- *Class 2:* Lab screening tests with empirical model-based scale-up procedures
- *Class 3:* Fundamental model-based methods (these are relatively new and are described later in more detail)

Class 2 methods have been used sparingly in the past, but recent work (Hilden 2007) suggests there is good scope for improvement and further suggests there is room to combine Class 2 and Class 3 methods very effectively. Despite the



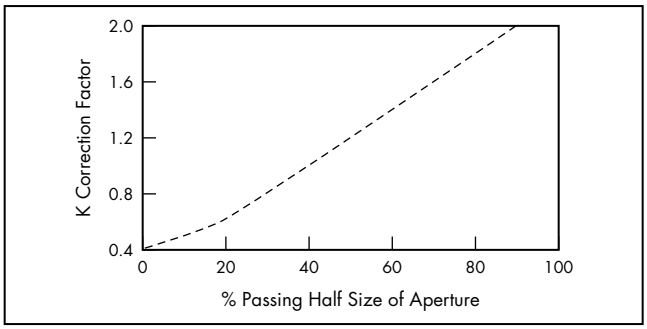
Source: Allis Chalmers, n.d.

Figure 14.3-15 Basic capacity factor (C) as a function of aperture size



Source: Allis Chalmers, n.d.

Figure 14.3-16 Correction factor (M) as a function of percent oversize



Source: Allis Chalmers, n.d.

Figure 14.3-17 Correction factor (K) as a function of percent half size

potential of using Class 2 and Class 3 methods, most industrial vibrating screens are still selected using Class 1 methods. Because in most cases these screens operate more or less as designed, it is understandable why these methods have survived with a few improvements for 70 years.

Class 1 selection methods employ empirical correlations to deduce screen area. The design process can begin once the application (e.g., a pebble screen on a semiautogenous grinding [SAG] mill discharge), the design feed rate of solids, and the typical feed particle size distribution are given. Early decisions on the number of decks, inclination, motion, lo-head or hi-head screens, and so forth, come from the design basis documents and past practices in similar applications. Once these parameters are set, the correlations can be used to compute screen area requirements, and the deck with the largest area determines the screen size. The task is then to match one of the standard products to the design. This often is an iterative design process.

Two usual approaches can be used to solve the Class 1 design problem—one based on the total solids feed rate to the screen and the other based on the solids feed rate of undersize material to the screen. Not surprisingly, this is also an area of debate, as the scientific community tends to prefer the latter method, yet the former is very commonly used. The former method will be described in an abbreviated way simply for illustrative purposes. To provide a real example, the data for the screen shown in Figure 14.3-7 is used.

The fundamental equation for determining the screen area required is given in the Equation 14.3-2 (Allis Chalmers, n.d.)

$$A = \frac{T}{CMKQ_1Q_2Q_3Q_4Q_5Q_6} \tag{14.3-2}$$

where

A = required screen area
T = mass flow of feed to screen
C = basic capacity factor
M = oversize factor (percent larger than aperture size in feed)
K = undersize factor (percent smaller that ½ aperture size in feed)
Q_1 = bulk density factor (usually ~60% of rock density)
Q_2 = aperture shape factor
Q_3 = particle shape factor
Q_4 = OA correction factor
Q_5 = wet or dry screening factor
Q_6 = surface moisture factor

Figures 14.3-15 through 14.3-17 provide the basis for estimating the factors C, M, and K, respectively. Table 14.3-5 provides the means for estimating the Q factors. (OA and aperture shape come from Figure 14.3-8.) Table 14.3-6 presents the summary data for the calculations, showing both the input data required and the empirical corrections, as deduced from the previously mentioned graphs and tables (which are adapted from Allis Chalmers, n.d.). The required area allows for a 6% loss due to mechanical fittings and so forth. Following the rough rule that $L = 3W$, this screen would have the approximate dimensions of 0.34 × 1.03 m (1.12 × 3.38 ft). In fact, the screen in Figure 14.3-7 is 0.34 × 1.67 m (1.03 × 5.48 ft). The width is the same, but the test screen has extra length, so a higher classification efficiency might be expected. This topic is discussed in the following subsection.

Performance Assessment

Mechanical and process performance can be distinguished, but this subsection only focuses on the latter.

A single parameter can be calculated in several ways that will quantify screen performance. Among these, the most common are E_u, which is defined as the efficiency of removal of undersize material from the oversize stream, and R_u, which is defined as the efficiency of undersize removal from the feed stream. Referring to Figure 14.3-18, Equations 14.3-3 and 14.3-4 are used to compute numerical values:

$$E_u = \frac{F(1-\hat{f}_u)}{O} = (1-\hat{o}_u) \tag{14.3-3}$$

Table 14.3-5 Q correction factors

Bulk Density Factor	Value
Bulk density, dry t/m^3	Q_1
0.4	0.25
0.8	0.5
1.6	1
2.08	1.3
Aperture Shape Factor	
Opening shape	Q_2
Square	1
Rounded	0.8
Slotted	
2:1 Slot	1.15
3:1 Slot	1.2
4:1 Slot	1.25
Particle Shape Factor	
Particle shape	Q_3
Cubical	1
Slabby	0.9
Open Area Factor	
Open area	Q_4
Calculation	Q_4 = percent open area ÷ 50%
Wet or Dry Screening Factor	
Opening	Q_5
Dry	1
Wet	
0.8 to 3.2 mm	1.25
4.8 to 6.4 mm	1.4
8 to 12.7 mm	1.2
14.3 to 25.4 mm	1.1
Surface Moisture Factor	
Surface moisture	Q_6
≤3%	1
3% to 6%	0.85
6% to 9%	0.7
Wet screen	1

$$R_u = \frac{U}{F\hat{f}_u} = \frac{(\hat{f}_u - \hat{o}_u)}{\hat{f}_u(1 - \hat{o}_u)} \qquad (14.3\text{-}4)$$

where

F = feed rate
O = oversize mass flow rate
U = undersize mass flow rate
$\hat{f}_u$ = cumulative mass fraction in the feed passing the aperture size
$\hat{o}_u$ = cumulative mass fraction in the oversize stream

To utilize these performance measures, one is required to perform a sampling experiment around the screen while it is running at steady-state conditions. This raises topics beyond the scope of this chapter (e.g., experimental design, sampling theory, sample preparation and analysis, and data massage). Fortunately, for the screen shown in Figure 14.3-7, the balanced data are already available and are summarized in Table 14.3-7.

Using these results, E_u = 77.9% and R_u = 93.5%. The data suggest that there is a considerable amount of finer (less than the aperture size of 10 mm [0.4 in.]) nearsize material in the oversize stream. In terms of assessing performance, the manufacturers have a curve that relates E_u to the rated capacity of the screen. This is shown in Figure 14.3-19, and the rated capacity is the ratio of the actual tonnage (or area) to the calculated tonnage (or area) from Equation 14.3-2. The respective area figures for the example are 0.35 and 0.57 m^2,

Table 14.3-6 Screen sizing example

Parameter	Value
Feed rate, dry t/h	15.7
Aperture (square), mm	10
Rock density, dry t/m^2	2.7
Application	Dry
C Factor, dry t/h/m^2	40.12
M Factor	0.972
K Factor	0.88
Q Factors	
Q_1	1.00
Q_2	1.00
Q_3	1.00
Q_4	1.39
Q_5	1.00
Q_6	1.00
Adjusted unit capacity, dry m/t/h/m^2	47.64
Area required, m^2	0.34
Size, mm	**Feed Distribution, %**
13.2	5.6
9.5	16.6
8	11.4
6.7	11.1
4.75	24.1
3.35	10.7
2.8	6.5
Pan	14.0
Percent +10 mm	18.7
Percent –5 mm	34.0

Source: Data from Hilden 2007.

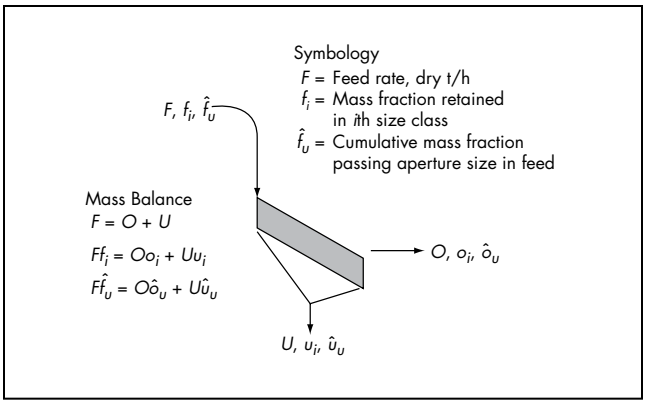


Figure 14.3-18 Screen performance parameters

Table 14.3-7 Sampled data summary for the horizontal screen in Figure 14.3-7

Size, mm	Feed, %	Oversize, %	Undersize, %
13.2	5.6	20.8	0.0
9.5	16.6	61.8	0.0
8	11.4	13.9	10.5
6.7	11.1	3.5	13.9
4.75	24.1	0.0	33.0
3.35	10.7	0.0	14.6
2.8	6.5	0.0	8.9
Pan	14.0	0.0	19.1
Mass fraction	1.00	0.27	0.73
Percent –10 mm	81.3	22.1	100.0
Mass flow, dry t/h	15.7	4.2	11.5

Source: Data from Hilden 2007.

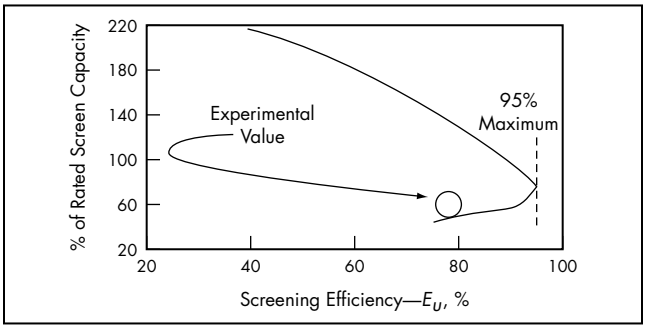


Figure 14.3-19 Percent of rated capacity versus E_u

which give a rated capacity of 59.9%. This point is plotted on the curve for reference.

This comparison indicates that the test screen in Figure 14.3-7 is not performing as well as would be expected at the given operating conditions, and further analysis of the situation would be required to validate the result and improve performance.

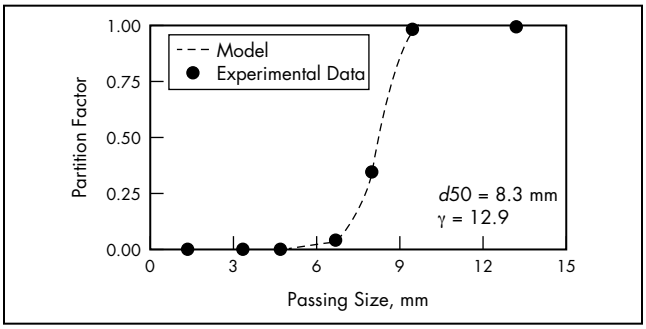
Another very common measure of efficiency is to use the data in Table 14.3-7 to compute an experimental partition curve, which can then be modeled with any number of simple functional forms. The model parameters are efficiency measures. Again, looking to the example in Figure 14.3-7, the experimental partition factors for the various size fractions are computed from Equation 14.3-5 and are defined as the mass fraction of material in a certain size fraction in the feed that reports to the oversize stream:

$$p_i = \frac{Oo_i}{Ff_i} \tag{14.3-5}$$

where

p_i = observed partition factor for size class i
f_i = mass fraction retained in the ith class of the feed
o_i = mass fraction retained in the ith class of the oversize stream

A very common model and one that will be further discussed later is the so-called Rosin–Rammler curve, and



Source: Data from Hilden 2007.

Figure 14.3-20 Experimental and modeled partition function data

the estimated partition factors are computed according to Equation 14.3-6:

$$p_i = 1 - e^{-0.693\left(\frac{d_i}{d50}\right)^{\gamma}} \tag{14.3-6}$$

where

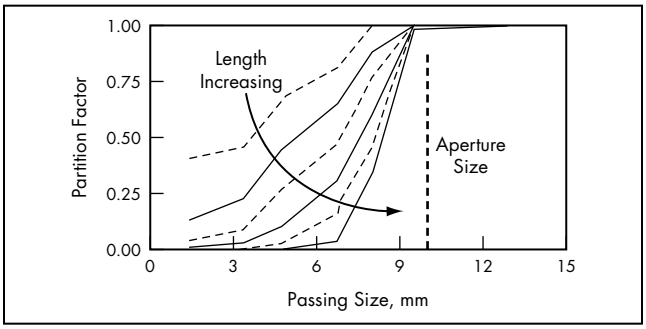
p_i = calculated partition factor for size class i
d_i = the characteristic size for size class i
$d50$ = cut size—particles of this size have an equal chance (50/50) of going to the undersize or oversize
γ = sharpness of separation (larger values mean better separation)

The parameters ($d50$ and γ) are obtained by minimizing a least-squares objective function using nonlinear regression (e.g., Solver in Microsoft Excel). That is, the parameters are selected in such a way that the sum of the squares of the differences in the observed and calculated partition factors is minimized. Figure 14.3-20 presents the results for the screen in Figure 14.3-7.

One would normally expect the cut size to be close to the throughfall aperture size, and it is clear that the value is a little low here (in part because of the way the characteristic size was defined for this study). The sharpness of separation for screens cutting in the aperture size range is usually around 6, so a value of close to 13 suggests that efficiency is high. Taken together, the implication is that the screen is not processing the finer nearsize material very well, which would require further investigation.

Interestingly, the data acquired for the screen in Figure 14.3-7 permit the calculation of the cumulative partition curve as a function of the position along the length of the deck. The experimental results are shown in Figure 14.3-21. Given the explanation related to crowded and separated screening given previously, it is no surprise to see that the fine material moves quickly through the screen—collectively, particles do not have to progress far down the deck before all of the very fine material has passed through the apertures (and the corresponding $p_i = 0$). For the fine, nearsize material, it is clear that the process is a lot slower.

The periodic measurement of screen efficiency is as important to the maintenance of process performance as the periodic measurement of, for instance, bearing condition is to the maintenance of mechanical performance. Unfortunately, it is often the case that the process aspects of screens are



Source: Data from Hilden 2007.

Figure 14.3-21 Partition factors as function of position on deck

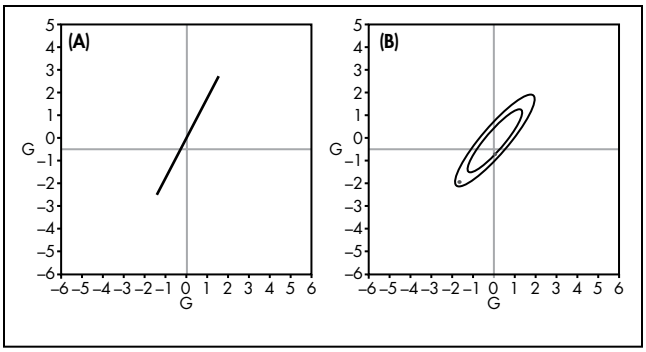


Figure 14.3-22 Feed end orbits in (A) normal and (B) abnormal operation

ignored. However, they are robust devices, and they often have a direct impact on the performance of comminution and separation equipment, for which a systems thinking approach must be applied.

Recent Developments

This subsection provides a brief introduction into some of the developments emerging in the vibrating screen market.

Real-time condition monitoring. Using vibration, thermographic, and occasionally other specialty sensors to automate the more usual manual inspections of predictive maintenance is becoming increasingly commonplace. Where it is done, this is still usually reserved for the typical applications of monitoring bearing health. Manufacturers such as Metso (Screen Security Package) have taken this a step further by using additional accelerometers to compute screen orbits, which can then be compared to expected values from run-in tests and/or to finite element modeling to analyze the mechanical and process health of the screen. Figure 14.3-22 is an example of normal orbits and the kinds of trends that develop when there is a problem, such as with the supporting springs. Loose media on the screen deck and other kinds of serious problems have their own orbit "signature."

In the case of the normal orbits (Figure 14.3-22A), both are linear and superimposed, showing the same stroke and angle. In the case of the abnormal conditions (Figure 14.3-22B), the orbits are now elliptical and the strokes are different.

The frequency spectrum of the accelerometers also provides useful information on the mechanical state of the deck, loading, and other interesting performance parameters, Moreover, when these signals are combined with others (e.g., from load cells and vision systems) in a kind of sensor fusion approach, the analytical detail and reliability can be dramatically increased.

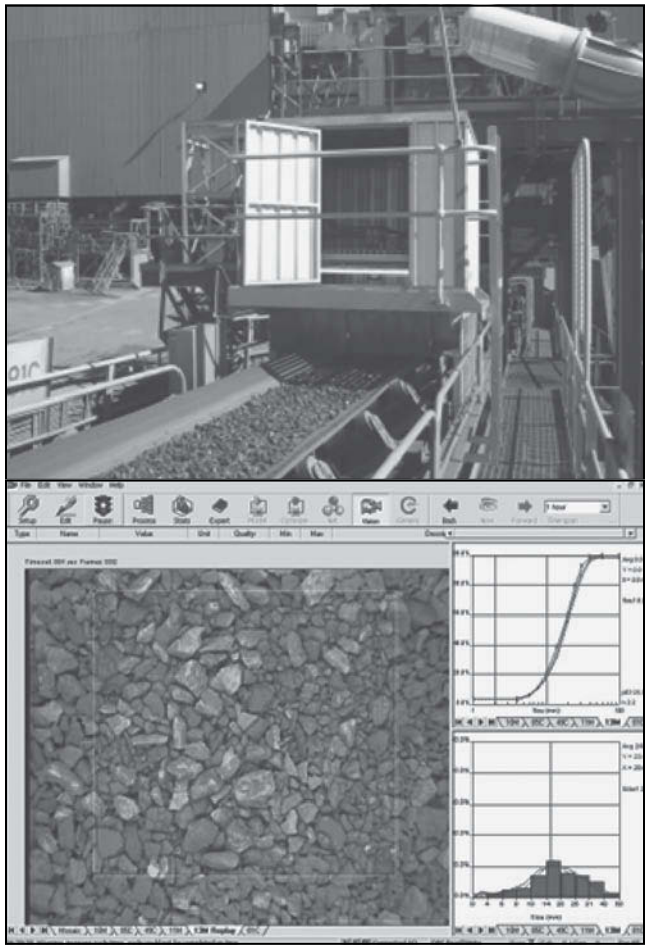


Figure 14.3-23 VisioRock installation and typical output display

Real-time deck motion modulation. As seen in the process matrix of Table 14.3-4, deck motion (stroke, angle, and orbit) all affect performance, and in light of the fact that screen feeds often change (mass flow and particle size distribution), there is a school of thought that says the ability to modulate these parameters on line would support real-time screen performance management. A number of manufacturers are implementing this kind of solution on selected screen models.

Instrumentation. One of the exciting areas of potential for performance monitoring is "machine vision," primarily as applied to measuring feed or product size. Typically, dry screening is very dusty and requires hoods covering the entire screening surface for effective dust control. Therefore, applications on the deck itself (oversize velocity, load, etc.) are challenging. However, in the case where the product streams are discharged onto belts, the standard technology, such as Metso's VisioRock, can be applied. For example, in the production of iron ore, lump size control is very important, and the early warning of a problem is essential if, for example, the screen media is broken, thereby producing off-spec material in the undersize stream. One operator is now using machine vision to detect and react to these types of quality-assurance events, and Figure 14.3-23 shows the VisioRock installation

as well as the output from data processing. In this case, images are analyzed at a rate of approximately 10 per second, which is more than sufficient to monitor everything that passes.

Acoustics also offer interesting potential for analyzing screen performance, but it is very early for research in this area.

Population balance simulation methods. Population balance methods (PBMs) are not new, having been in use for at least two decades, but as the design and operations needs turn increasingly toward simulation to accelerate the activities, higher and higher fidelity is required. These are an element of the Class 2 methods previously mentioned in the "Screen Selection" subsection. Although the industry is moving in this direction, older approaches are still in use in most of the simulation packages. One of these is the model by Karra (1979), which is based on a Class 1 selection method, although in this case it is based on the method of sizing relating to the mass flow rate of undersize in the feed stream.

Using the model in Equation 14.3-6, Karra developed a correlation based on the empirical screen sizing factors used in the screen selection approach. The factors have been modified over time, and their estimation is described elsewhere (e.g., King 2001), but the equation for computing $d50$ is

$$d50 = \frac{G_c h_T}{\left(\frac{\left(T_u/H\right)}{ABCDEF}\right)^{0.148}} \qquad (14.3\text{-}7)$$

where

$d50$ = cut size
G_c = nearsize correction factor
h_T = throughfall aperture (King 2001)
T_u = tons of undersize in the feed (dry t/h)
H = effective screen area (m^2)
A = basic capacity factor
B = oversize factor
C = fine size factor
D = deck location factor
E = wet screen factor
F = bulk density factor

Because a number of the properties for the screen in Figure 14.3-7 have been given, Table 14.3-8 is a shortened summary of the Karra calculation of $d50$. The agreement with the experimental value in Figure 14.3-20 is good.

Karra assumes γ is a constant at 5.9, and in the authors' experience, numbers in the region are pretty typical for coarse screening. This number does not agree very well with the value in Figure 14.3-20, which highlights that Karra's simple two-parameter model approach probably requires additional parameters.

Nevertheless, using the model of Equation 14.3-7 and the feed data in Table 14.3-7, Karra's model can be used to predict the expected size distributions for the undersize and oversize products, and these are shown in Figure 14.3-24 along with the actual experimental results. The predictions are satisfactory, but the importance of γ is clear. The calculated fractional mass splits of 0.26 and 0.74 compare well with the product mass flows reported in Table 14.3-7.

Discrete element simulation methods. In the pursuit of higher-fidelity simulation of screen performance, the ultimate tool is probably discrete element modeling (DEM). Very briefly, this approach is based on fundamental models drawn from physics (e.g., Newton's laws of motion.) Using a mathematical description of the body and motion of the screen; some typical physical properties; and the feed size distribution, flow rate, and particle shapes, the DEM simulation can be run from only first principles (e.g., no correlations and no parameters drawn for experimental data). Although this sounds simple in concept, solving the problem is onerous, as there are millions of particles, each requiring several differential equations to describe translational and rotational motion in time and space. Because of the interactions of the particles among themselves and with the screen itself, the integration intervals are very short—on the order of one millionth of a second. Simulating several seconds of real-time operation can take days, weeks, or months, depending on the problem size, the computational capability, and the simulation code efficiency. It is for this reason that the goal is to combine DEM and PBMs, the former to work out the real mechanisms and the latter to abstract these and provide speedy calculations.

Because of the vast quantities (gigabytes or even terabytes) of numerical data generated in such simulations, it is common practice to use visualization techniques to "see the results." However, the numerical data must be used for any calculations. Figure 14.3-25 presents a snapshot of visualization, and Figure 14.3-26 shows the predicted (lines) and measured (bullets) particle size distributions from a simulation of the banana screen pictured in Figure 14.3-12B. The agreement in Figure 14.3-26 is very good, which provides some insight into the power this tool will bring to screen modeling.

Table 14.3-8 Computing Karra's *d*50 for the screen in Figure 14.3-7

Symbol	Factor	Units	Result
T_u	Feed	dry t/h	15.67
H	Effective screen area	m^2	0.57
h_T	Throughfall aperture	mm	10
G_c	Nearsize correction factor	—	0.82
A	Basic capacity factor	$t/h/m^2$	17.85
B	Oversize factor	$t/h/m^2$	1.38
C	Fine size factor	$t/h/m^2$	1.12
D	Deck location factor	$t/h/m^2$	1.00
E	Wet screen factor	$t/h/m^2$	1.00
F	Bulk density factor	$t/h/m^2$	1.01
$d50$	Cut size	mm	8.34

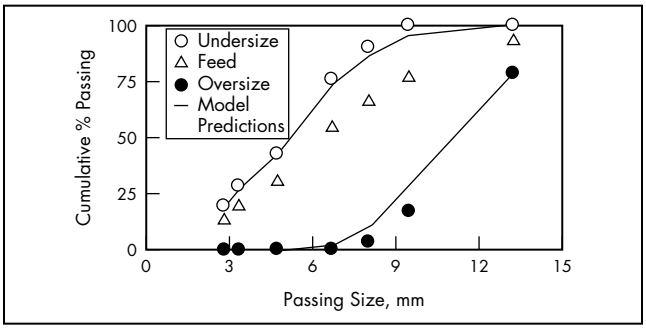


Figure 14.3-24 Karra's model predictions and actual cumulative size distribution curves for the screen in Figure 14.3-7

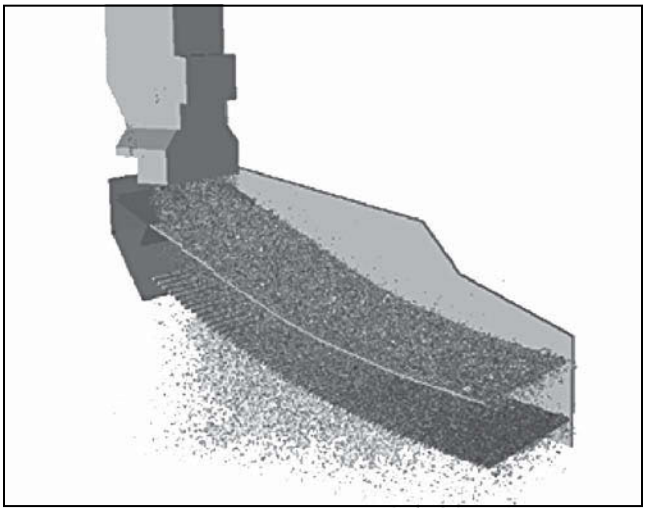


Figure 14.3-25 DEM visualization of screen operation

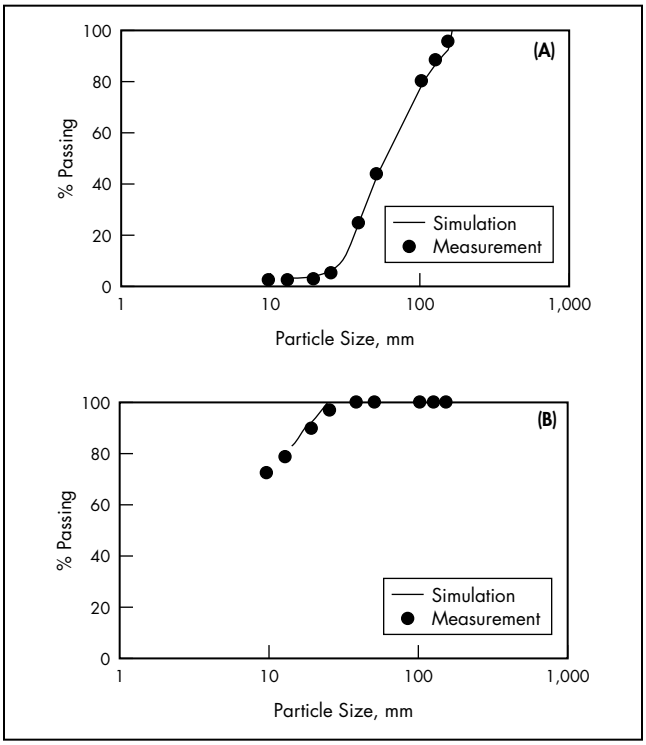


Figure 14.3-26 DEM predictions of (A) oversize and (B) undersize particle size distributions

Screen Media

Because screen media is often the single largest operational expense for a vibrating screen, this chapter would be incomplete without a brief discussion on this topic. The governing objective here is the absolute lowest total cost, which implies that availability, cost, wear life, downtime, and so forth must be considered in making the final media selection.

Three major materials of construction for screen media are steel (punched plate or wire mesh), rubber, and polyurethane. The polymeric compounds of rubber and polyurethane are usually more expensive, but they tend to exhibit superior wear performance in those applications that permit alternatives. Service life can be extended by factors of more than five with polymerics.

Figures 14.3-27 through 14.3-29 show wire mesh, rubber, and polyurethane media, respectively. Media specialists often break down the applications as scalping, standard production (size control and sorting), and washing (sorting).

Wire mesh can have standard weaves, as shown in Figure 14.3-27A, or slotted antiblinding or even custom weaves, as shown in Figure 14.3-27B. The wires are normal high-carbon or stainless steels.

Typical applications for rubber screen decks (Figure 14.3-28) include high-impact scalping type duty or situations where abrasion and/or blinding are a serious concern. The decks are manufactured by punching or molding.

Polyurethane decks (Figure 14.3-29) are usually found in standard production applications and in washing or dewatering applications. They are manufactured by casting or injection molding processes.

The polymeric materials lend themselves to modular installation or paneling. Table 14.3-9 weighs the pros and cons of steel versus the polymerics.

As a cautionary note, although there might be a tendency to choose thicker panels to increase wear life, this will reduce capacity and accuracy and will induce higher levels of pegging and blinding. Media suppliers familiar with best practices can be very helpful in these selections, and it is also recommended that the original equipment manufacturer (OEM) is consulted when changing the design of media.

Standard Screen Sizing and Relative Pricing

Table 14.3-10 provides a glimpse of a typical vibrating screen product line, and while the price data supplied here is somewhat out of date, the relative pricing of the screen equipment is more or less the same.

Fine Screening

In general, wet fine particle classification is achieved by mechanical/hydraulic means (e.g., rake or spiral/screw

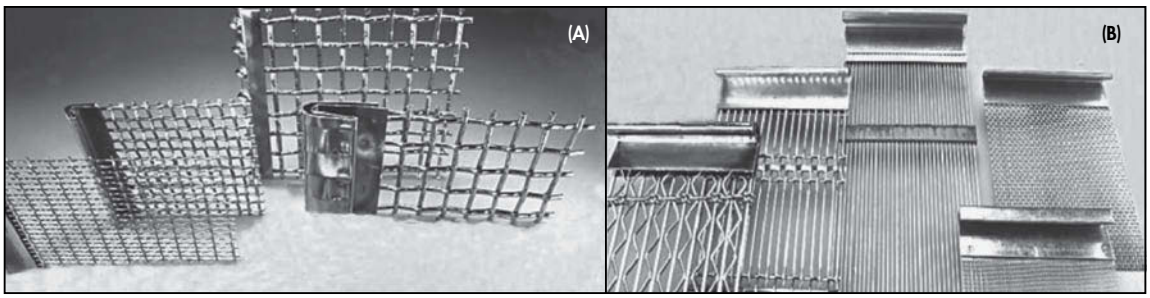


Figure 14.3-27 Wire mesh media

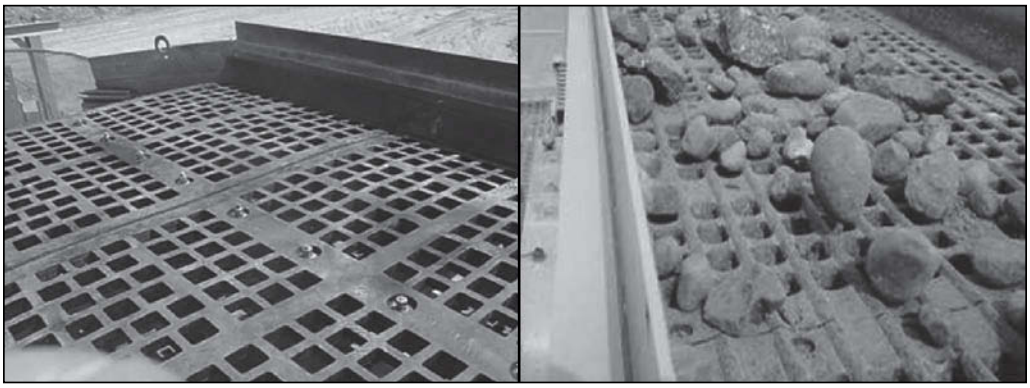


Figure 14.3-28 Rubber screen decks

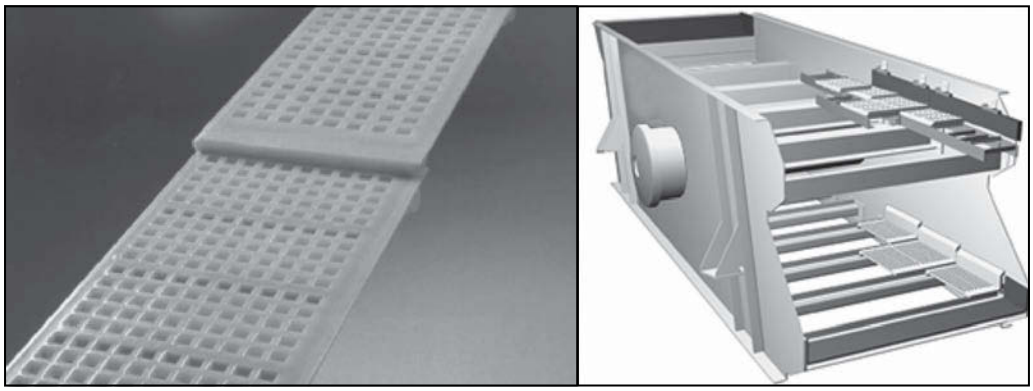


Figure 14.3-29 Polyurethane panels

classifiers), hydrocyclones (discussed later in this chapter), and fine screens. Although the mechanical devices still have their place, fine screens, and especially hydrocyclones, dominate fine and ultrafine classification. Fine screening is the term generally reserved for fine and ultrafine separations, as defined in Table 14.3-1. It is performed dry or wet, although in most mining or quarrying applications, this is a wet process. Consequently, the emphasis here is on wet fine screening.

The principles described previously for vibrating screens apply to most screening processes. One notable difference is that fine screening generally involves much higher rotational speeds (e.g., ~3,600 rpm). Therefore, much smaller strokes are required. Another major difference for fine screening is that the selection of the appropriate screen is as much an art (i.e., past practices in similar applications) as it is a science (i.e., laboratory testing of specific screens on the expected feed material and supplier databases). This complexity is due in large measure to the critically important aspect of fluid mechanics, as drag forces tend to govern wet fine and ultrafine separations. Suffice it to say, the expertise in fine-screen design generally lies with the major supplier organizations, such as Derrick Corporation, Conn-Weld Industries, Kroosh Technologies, Pansep, Multotec Group, and Wedgewire.

The two most common wet fine screening devices are the sieve bend (i.e., a stationary version of the banana screen, Figure 14.3-6) and the more conventional, high-frequency fine screen (i.e., having a more conventional geometry and represented herein by the Derrick product line, including the Stack Sizer [Figure 14.3-5], the multifeed screen, and the repulp screen). Wet fine screening is essentially the only flow-sheet option in a number of applications (see sorting class in Figure 14.3-1), for example, in coal preparation desliming and iron ore coarse silica removal. (A more complete review on this topic is available in Valine and Wennen (2002).

Table 14.3-9 Advantages and disadvantages of steel versus polymeric screen decks

Wire	Rubber/Polyurethane
Advantages • Price • Accuracy of cut • Open area is high (on installation)	Advantages • Longer life • Less downtime • Shock resistance • Accuracy • Reduced pegging • Reduced blinding • Open area (in operation) • Noise reduction
Disadvantages • Shorter life • More downtime • Pegging problems • Blinding problems • Reduces open area (in operation) • Noise levels	Disadvantages • Price • Open area (on installation) (e.g., although these screens tend to have lower open areas out of the manufacturing process than wire mesh, they retain open area much better)

Many other applications provide wet screening as an alternative to hydrocyclones or other mechanical devices, as it offers some significant advantages in classification efficiency (as measured by γ or E_u). In some applications, both sieve bends and high-frequency screens are used in conjunction with hydrocyclones to exploit the physical and classification

Table 14.3-10 Standard screen sizing and relative pricing

Screen Size, W × L, m	Linear Motion, Low-Head, Horizontal, Multiflow Banana Screens*		Inclined Circular Motion Ripl-Flo Screens†	
	Single Deck, $	Double Deck, $	Single Deck, $	Double Deck, $
1.2 × 4.8	70,200	95,160	58,500	93,600
1.8 × 4.8	76,440	138,060	74,880	109,200
1.8 × 6.1	136,500	175,500	117,000	143,520
2.4 × 4.8	113,100	152,100	105,300	127,140
2.4 × 6.1	145,860	269,100	124,800	156,000
2.4 × 7.3	175,500	388,600	179,400	218,400
3.0 × 4.8	117,000	232,440	N/A‡	N/A
3.0 × 6.1	210,600	310,440	155,220	226,200
3.0 × 7.3	237,900	343,200	202,800	249,600
3.0 × 8.5	257,400	374,400	N/A	N/A
3.6 × 6.1	252,720	368,940	N/A	N/A
3.6 × 7.3	280,800	382,200	N/A	N/A
3.6 × 8.5	308,880	444,600	N/A	N/A
4.2 × 6.1	265,200	443,040	N/A	N/A
4.2 × 7.3	304,200	530,400	N/A	N/A

*Low-head and multiflow pricing includes motor and modular polyurethane screen surfaces.
†Ripl-Flo screens include motor or motors and woven wire screen surfaces.
‡N/A = not available.

attributes for each type of separator. In other instances, it may also be possible to feed a wet screen by gravity, rather than the pumping systems more commonly associated with hydrocyclone batteries. Nevertheless, in these kinds of trade-off studies, it generally comes down to the footprint advantages of hydrocyclones versus the process efficiency advantages of wet screening. For example, one study (Vince 2007) in coal indicated that purely from a capacity perspective, two to three Derrick Stack Sizers were the equivalent of one 900-mm (35.4-in.) classifying hydrocyclone. Interestingly, that same study came to the conclusion that the Stack Sizer should be considered for coal-preparation applications with cut points in the 100-to-350-µm range, while sieve bends could be considered for cut sizes in the 250-to-350-µm range. Of even greater interest was the observation that "similar" equipment made by different suppliers can have quite different performance characteristics—*caveat emptor* (buyer beware).

One area of growing interest for fine screening is close fine-grinding (ball mill or even stirred-mill) circuits (e.g., see Wennen et al. 1997). The opportunities to exploit improved efficiency to reduce specific energy and media requirements and to improve downstream metallurgical efficiency (e.g., by minimizing the overgrinding of valuable minerals) are very appealing, both technically and economically. In addition, while hydrocyclones separate based on particle mass (i.e., particle size and density are first-order effects), wet fine screens separate predominantly on size (i.e., density is a higher order effect, mainly related to particle stratification). For example, in iron-ore grinding, the cut size on a screen is the same for the iron minerals and silica, whereas in a hydrocyclone, the cut size for the iron minerals is about one-third of that for silica,

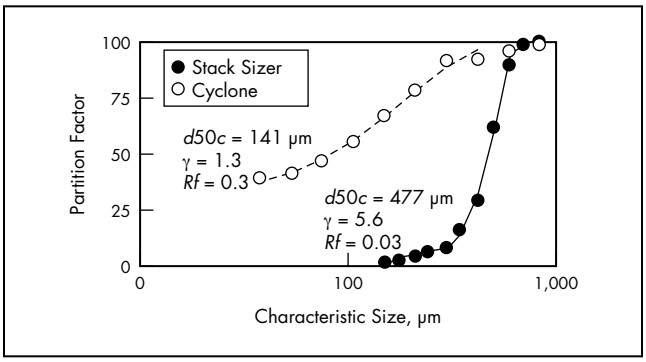


Figure 14.3-30 Partition curves for a 254-mm (10-in.) hydrocyclone and a Derrick Stack Sizer

which is to say that a lot of fine liberated iron mineral can be sent back to the grinding mill by a hydrocyclone.

Consider the partition curves shown in Figure 14.3-30. (The parameter *Rf* is defined later in Figure 14.3-49 and in Equation 14.3-17, but it effectively represents the very fine material hydraulically entrained in the oversize stream—the bypass.) Although there are some "apples to oranges" comparisons in terms of the cut sizes, these example are drawn from the same plant in similar applications (Aquino and Vizcarra 2007); the real interest comes from the comparison of the sharpness of separation (γ) and the efficiency (here inversely related to *Rf*). The benefits of wet screening are obvious, which makes it a technical option to cut sizes ranging down to 100 µm. For the hydrocyclone partition curve, the inflections

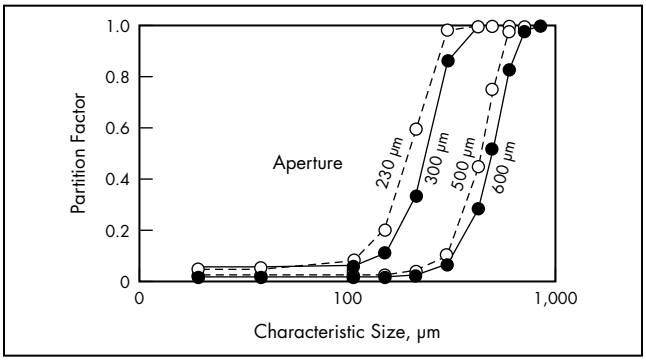


Figure 14.3-31 Partition curves for a Derrick Stack Sizer with different screen apertures

and rather poor sharpness of separation are a partial consequence of the existence of liberated heavy and light minerals in the feed (i.e., the density effect).

Finally, Figure 14.3-31 provides a glimpse of the ability of this fine screen to preserve efficiency over a range of aperture sizes (i.e., the sharpness in separation is good and the bypass is low).

CYCLONES

Cyclones (named for the spiraling motion within the body of the device) are one of a large number of classification devices aimed at separating fine particles in a fluid (water or air). For examples, see Kelly and Spottiswood (1982) or Svarovsky (2000). Cyclone classification is generally reserved for the separation of particles on the basis of size. For this reason, other applications for cyclones, such as thickening and dense-medium separation, are not considered here. Broadly speaking, cyclone classifiers can either treat dry feeds (the air classifier or air separator) or wet/slurry feeds (the hydrocyclone). The emphasis here will be on wet applications.

Unlike the vibrating screen, which employs a physical means (the screen media) to effect a separation, the cyclone classifier uses differential settling properties of particles of different mass. The cyclonic action within the body of the device develops G-forces significantly greater than gravity, which accelerate the separation process. In general, cyclones are robust, compact (small footprint), and relatively efficient size-separation devices.

Figure 14.3-32 shows the most common applications of wet cyclones in mining. Air classifiers are also commonly used in these applications, for example, as a size-control device in dry grinding (most notably in refractory gold, coal-fired power plant pulverizers, or cement clinker applications) and as sorters (usually in de-dusting applications).

Figure 14.3-33 shows the geometry of a standard hydrocyclone, the shape of which clearly resembles the natural phenomenon that gives rise to the name. The separator comprises a feed section, cylindro-conical section, discharge spigot, vortex finder, and overflow. In some cases, usually with larger diameters, each of these is supplied as a separate entity, whereas in others they may be cast into a single unit (i.e., with smaller diameters).

The capacity of a hydrocyclone is a function of its size and, subject to operational constraints, the rate and solids concentration at which the feed (slurry) material is supplied. In some cases, hydrocyclones are gravity fed from surge tanks,

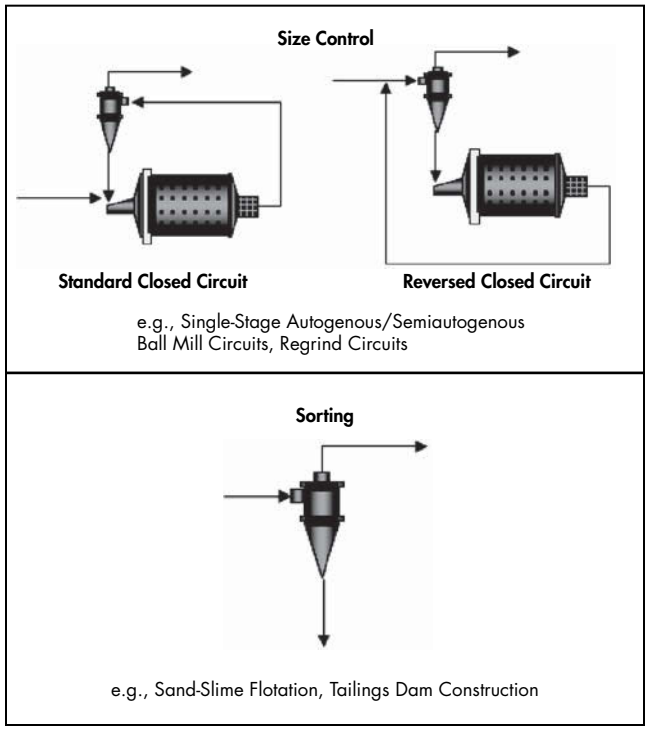


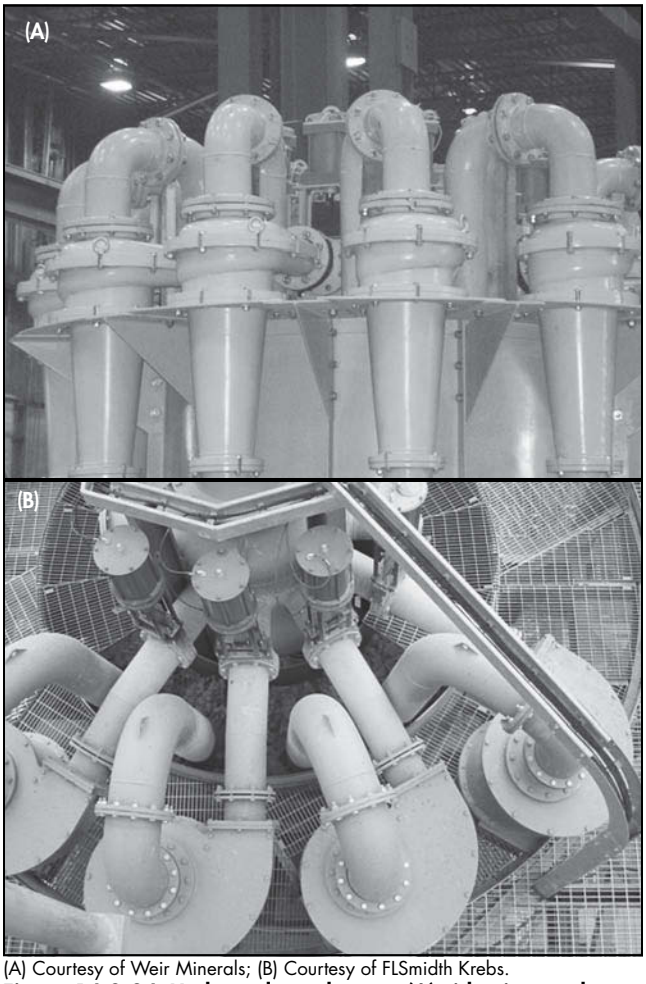
Figure 14.3-32 Examples of the application classes for hydrocyclones in mining



Courtesy of Weir Minerals.

Figure 14.3-33 Hydrocyclone

but the majority of applications use a feed sump and pump arrangement. Consequently, it is common to utilize a group of hydrocyclones operating in parallel to achieve the required process throughput capacity. As Figure 14.3-34 shows, the compact design of the units permits the use of a radial header to distribute feed to a number of hydrocyclones. This is commonly called a hydrocyclone cluster or battery, or a Cyclopac. Moreover, as Figure 14.3-34B highlights, the use of knife-gate valves to open or close a hydrocyclone is very convenient, both from an online process control point of view as well as for maintenance purposes. On the latter point, and given the relative cost of a hydrocyclone, it is typical to remove a



(A) Courtesy of Weir Minerals; (B) Courtesy of FLSmidth Krebs.

Figure 14.3-34 Hydrocyclone clusters: (A) side view and (B) top view

hydrocyclone for maintenance and simply replace it with a refurbished spare.

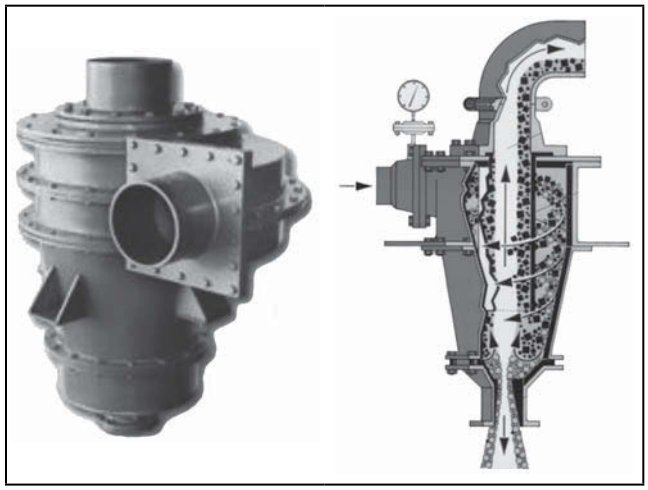
There are a number of variants on the more typical hydrocyclone geometry of Figure 14.3-33. Figure 14.3-35 shows the water-only or automedium cyclone that is frequently used in washing fine (–0.6 mm) coal. (The truncated lower cone section is employed to set up a secondary density separation in a rotating particle bed.) The flat-bottomed hydrocyclone occasionally used in mineral processing applications has a very similar kind of body geometry.

Air separators tend to be designed to treat the whole feed flow, so they are larger units, as shown in Figure 14.3-36. These devices can be either static (such as the hydrocyclone) or dynamic (including moving blades, and so forth, to enhance separation).

Figure 14.3-36 illustrates a typical dry grinding circuit with the air separator used for classification and a cyclone used for solids recovery. The cutaway view on the right illustrates the internals and the drive for this dynamic air separator.

Hydrocyclone Basics

The hydrocyclone evolved into its current role in mineral processing in the period from approximately 1950 to 1970, by which time it had become a key element in the design of most



Courtesy of FLSmidth Krebs.

Figure 14.3-35 Water-only cyclone

large-scale grinding circuits. The hydrocyclone offers high capacities in small footprints and arguably enabled the ongoing development of very large grinding circuits. Although they still have their place, mechanical classifiers were effectively rendered obsolete in plant design by the hydrocyclone. Despite the numerous benefits of hydrocyclones, there have always been issues with efficiency, as introduced earlier in the discussion on fine screening. As today's operators look for ways to address energy efficiency and as competitive technologies emerge, hydrocyclone manufacturers have redoubled their development efforts. As is the case for screens, they are using the very latest simulation tools to help refine existing designs and develop new ones for specific applications, with good success. The following subsections provide a refresher on hydrocyclone basics and introduce some of the latest tools and techniques for the design, analysis, control, and optimization of hydrocyclones.

Principles of Operation

Figure 14.3-37 provides a sketch of a typical hydrocyclone showing the main design components. The principal dimensional variable is the diameter of the upper cylindrical section of the hydrocyclone body, D_c. The inlet (D_i) and discharge (D_o and D_u) dimensions usually lie within some ratio of D_c, (e.g., $0.2D_c \leq D_o \leq 0.45D_c$). Because the inlet is usually more rectangular in nature, to facilitate a reduction in turbulence on entry (involute designs—see Olson and Turner 2002), D_i is defined as the diameter of circle with an area equivalent to the actual inlet area. In the past, smaller hydrocyclones have had lower cone angles (~10°), but because of headroom considerations, larger hydrocyclones tend to have larger cone angles (~20°). Some typical dimensions for a hydrocyclone with D_c = 760 mm (30 in.) are shown in the figure.

The feed enters the hydrocyclone near the top of the cylindrical section, usually through some sort of involute inlet design, which is intended to introduce the slurry into the hydrocyclone with a minimum of turbulence, as this can have a deleterious effect on efficiency and wear. Figure 14.3-34A clearly shows this involute design. The rotational motion set up in the body of the hydrocyclone

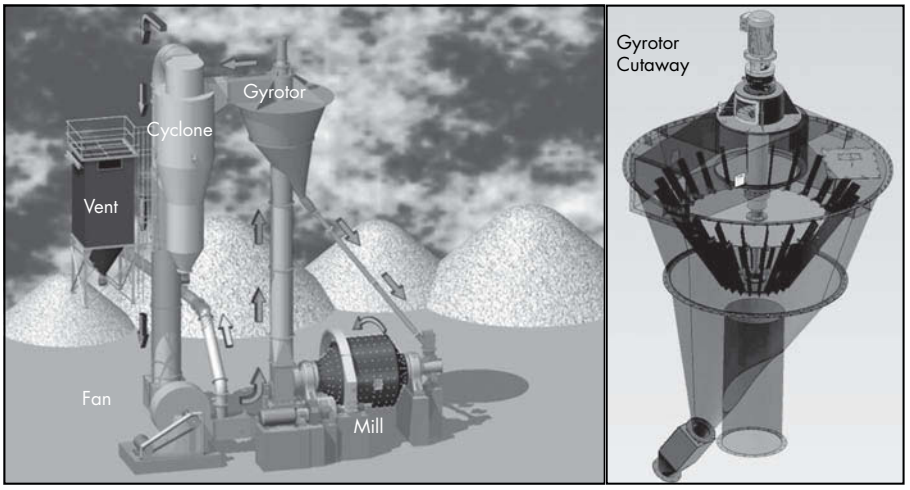


Figure 14.3-36 A Gyrotor air separator in a typical dry grinding circuit

gives rise to the G-forces that accelerate the particle separation process. All of the particles are subjected to the centrifugal forces directing them to the wall of the hydrocyclone, where this material moves downward in a helical fashion to be discharged through the apex, as *underflow*. However, as the particles migrate to the outer wall of the hydrocyclone, they necessarily displace the fluid, which must move inward. The fluid exerts drag forces on the particles and retards their outward motion and, in the case of the finer particles, actually causes them to move inward. This stream moves upward in a helical path to be discharged through the vortex finder, as *overflow*. When the hydrocyclone is operating normally, the centrifugal forces acting on the slurry create low pressure along the central axis, establishing the conditions necessary for development of the characteristic air core.

To look more closely at the separation process, consider Stokes' law for laminar settling, given by Equation 14.3-8. (Whether it is Stokes' law, Newton's law, or something else—as experimental data would seem to indicate—the trends predicted by Stokes' law are seen in practice.)

$$v_t = \frac{(\rho_s - \rho_l) g d^2}{18\mu} \qquad (14.3\text{-}8)$$

where

v_t = terminal settling velocity
ρ_s = solids specific gravity
ρ_l = carrier fluid specific gravity
g = gravitational or centrifugal force acting on the particles
d = particle diameter
μ = carrier fluid viscosity

From the explanation given previously, anything that acts to increase the particle settling velocity increases the chances of particle reporting to the underflow stream. Clearly, the larger the particle, the more likely it is to go to the underflow. Similarly, higher G-forces (meaning higher feed flows or pressure drops) will also act to increase the probability that a particle will find its way to the underflow stream. Particles with higher specific gravities will also have a higher probability of reaching the underflow stream.

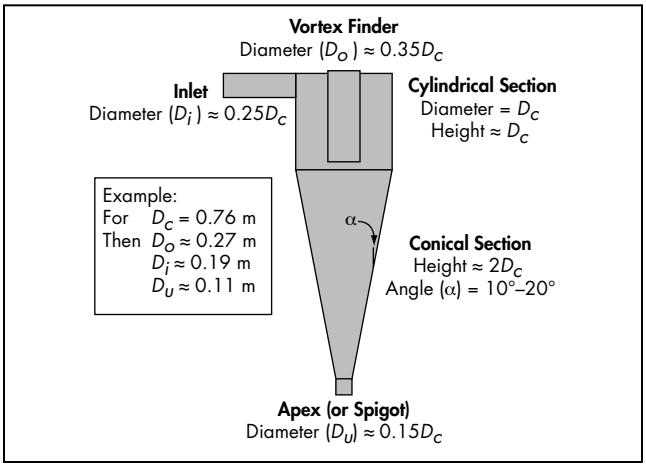


Figure 14.3-37 Principal design elements of a hydrocyclone

However, if the feed material contains a lot of very fine clay material, thereby increasing the effective viscosity, the particles will be less likely to reach the underflow stream. Similarly, with high-density hydrocyclone feeds, there is a significant concentration of fine particles in the carrier fluid, which increases the fluid specific gravity and makes it less likely for particles to reach the underflow stream.

To further complicate things, the size of the vortex finder also has an influence on the separation outcome. Smaller vortex finders will create a finer particle overflow stream, and vice versa. Although to a lesser extent, the apex size will also have an impact, as a smaller apex tends to limit the underflow capacity, diverting some coarser particles toward the upper regions of the hydrocyclone, the finest of which will then be entrained in the overflow stream.

Figure 14.3-38 (absent the air core) was developed by Renner and Cohen (1978) and shows roughly the main regions inside a cyclone. Region A is relatively narrow area in the vicinity of the inlet that is characterized by a size distribution similar to that of the feed. Most of the conical section is Region D, which is characterized by a coarser size distribution similar to the underflow. Region B is the finer material with a size distribution very similar to the overflow product.

Region C is perhaps the most critical region, as it was found to contain significant quantities of nearsize particles, which suggests that they tend to accumulate here until physical capacity constraints force them to move to one product zone or the other. Arguably, this is the region of active classification, and poor operation would be expected if it fails to form properly, possibly because of poor design or excessive feed solids concentrations.

Rather than produce a process matrix for the hydrocyclone, the authors have chosen to illustrate these concepts with the Plitt model (see Flintoff et al. 1987) given in Equation 14.3-9. This is the correlation for the corrected cut size, $d50_c$. For example, increasing the hydrocyclone diameter, D_c, would be expected to lead to a coarser cut size. Similarly, increasing the feed flow, Q, would lead to a finer cut size.

$$d50_c = \frac{CD_c^{0.46} D_i^{0.6} D_o^{1.21} \mu^{0.5} e^{0.063V}}{D_u^{0.71} h^{0.28} Q^{0.48} \left[\frac{\rho_s - \rho_l}{1.65}\right]^k} \qquad (14.3\text{-}9)$$

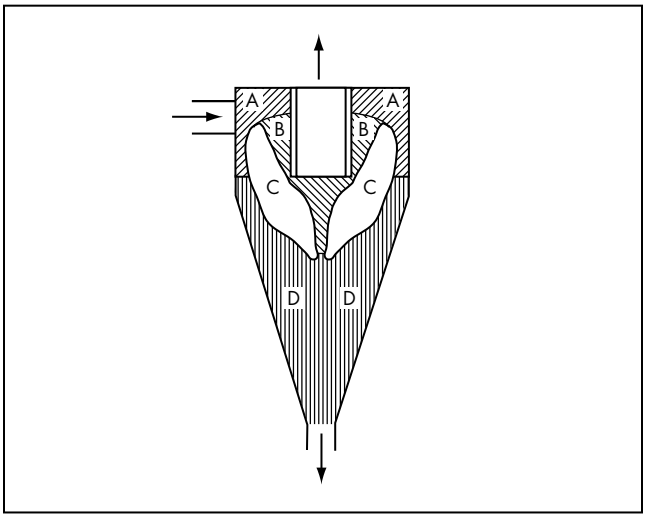
where

$d50_c$ = corrected cut size
C = a calibration constant
D_c = hydrocyclone diameter
D_i = inlet diameter
D_o = overflow apex diameter
V = volume percent concentration of solids in hydrocyclone feed
D_u = underflow apex diameter
h = free vortex height
Q = feed volumetric flow rate
k = settling exponent (e.g., 0.5 in Equation 14.3-8)

A correlation exists for γ as well, but it is much weaker, generally indicating that γ will improve at lower flows (longer residence times) and lower feed solids volumetric concentrations, both of which are intuitively acceptable.

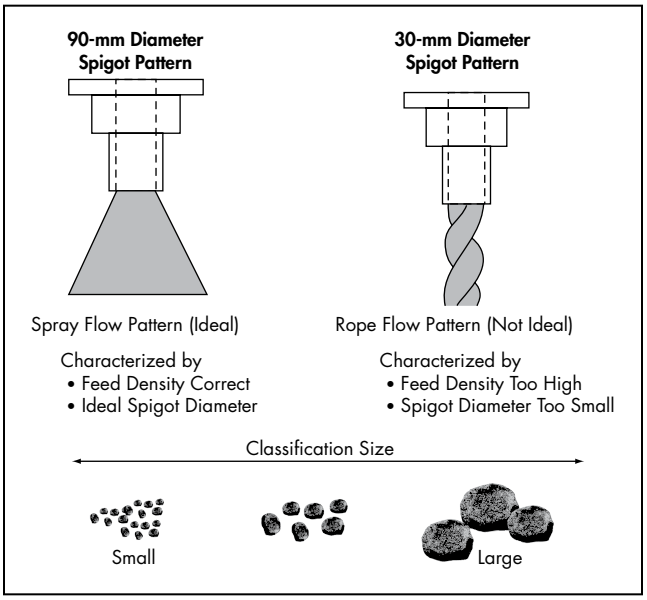
A couple of phenomena are worthy of some amplification. The first is the phenomenon known as roping (a name based on the appearance of the underflow stream in this regime). Roping occurs when the volumetric flow rate of the underflow stream causes the air core to collapse. When this happens, the usual helical spray from the apex alters appearance, as shown in Figure 14.3-39. Consequently, roping will increase the flow of coarser particles to the overflow stream (i.e., a coarser cut size). The impact on efficiency (as measured by γ in a partition curve) appears to depend on the hydrocyclone feed density—higher densities having a more deleterious effect. Roping does tend to maximize the underflow density. Thus, it minimizes the fraction of fine material that reports with the carrier liquid to this stream.

The other phenomenon is particle density. There are many cases (coal, iron ore, massive sulfide ores, and so forth) where liberated particles of widely different densities exist. Equation 14.3-8 indicates that, for a constant settling velocity, the particle size is related to solids specific gravity as $d \propto (\rho_s - \rho_l)^{0.5}$. In practice, and because of other phenomena such a secondary density separation occurring in the rotating bed, the exponent is usually much greater than 0.5, which means that density effects are exacerbated in the hydrocyclone. To illustrate, consider Figure 14.3-40, which shows coal slurry being processed in a hydrocyclone. Sink and float analysis



Source: Renner and Cohen 1978.

Figure 14.3-38 Regions in a cyclone

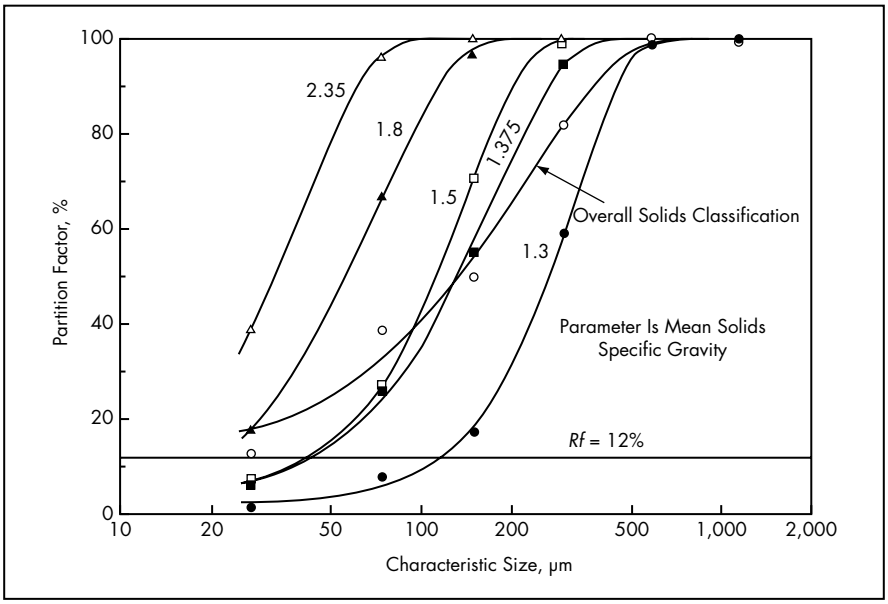


Courtesy of Weir Minerals

Figure 14.3-39 Spray and rope discharge

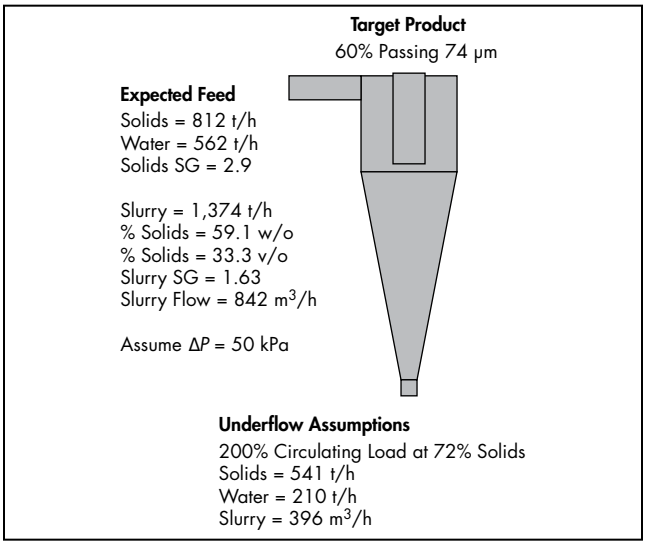
on the products allows the reconstruction of separate partition curves for each specific gravity (SG) class, all of which have a reasonable shape. However, looking only at the overall curve could suggest poor classification.

The last item of interest is hydrocyclone mounting. With the development of larger and larger hydrocyclones, mounting them in a vertical orientation puts a significant head on the underflow stream, which can influence wear and separation. However, tilting these hydrocyclones (usually at ~45°) allows operation at lower pressures while still producing a good underflow product, albeit at a coarser cut size. Maintenance-saving claims for wear in the lower region of the hydrocyclone are impressive (80%, e.g., Schmidt and Turner 1993), but it can be more difficult to perform in-situ maintenance for this kind of installation.



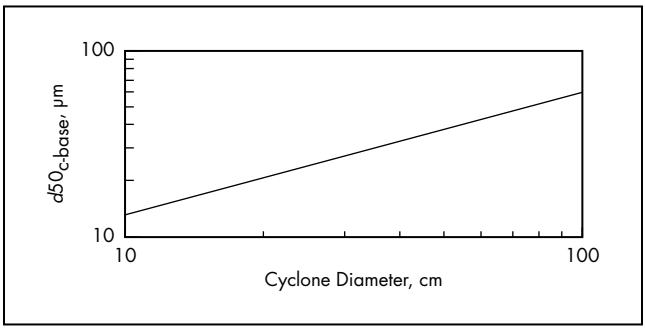
Source: Reprinted with permission of the Canadian Institute of Mining, Metallurgy and Petroleum. Originally published in the *CIM Bulletin*, Vol. 80, No. 905, p. 43.

Figure 14.3-40 Partition curves for coal slurry in a hydrocyclone



Source: Arterburn 1982.

Figure 14.3-41 Hydrocyclone design



Source: Arterburn 1982.

Figure 14.3-42 For standard hydrocyclones, $d50_{c\text{-}base}$ versus hydrocyclone diameter

Hydrocyclone Selection

The selection of hydrocyclones is normally a task best performed with a supplier, as these firms possess the knowledge, expertise, databases, and tools to ensure that the best results are achieved. In most cases, it is possible to avoid pilot testing. However, there are some instances where this is preferable and even inevitable. One of the more challenging design problems is the closed-circuit grinding application, illustrated on the left in Figure 14.3-32. This is challenging because the hydrocyclone feed characteristics depend very much on mill performance, which in turn depends on the hydrocyclone underflow characteristics, which in turn depend on the feed. Experience has given most suppliers a very good understanding of this problem; nevertheless, circuit simulation tools can be useful adjuncts in developing a hydrocyclone design basis. (Public domain hydrocyclone models, e.g., Equation 14.3-9, include most of the basic design parameters, yet these are seemingly not commonly used in the design process, which underscores the experience factor necessary in this process.)

For the purposes of this chapter, the techniques of hydrocyclone design documented by FLSmidth Krebs are briefly reviewed (for more detail, see Arterburn 1982 and Olson and Turner 2002). The required information is summarized in Figure 14.3-41.

A set of standard tests under very specific design and operating conditions was performed to develop a relationship between what is called the base corrected cut size ($d50_{c\text{-}base}$) and the principal dimensional variable, the hydrocyclone diameter, D_c. The outcome of this work is captured in Figure 14.3-42, based on Equation 14.3-10, and is also given in the alternate form as Equation 14.3-11.

$$d50_{c\text{-}base} = 2.84D_c^{0.66} \tag{14.3-10}$$

$$D_c = 0.206d50_{c\text{-}base}^{1.515} \tag{14.3-11}$$

Table 14.3-11 Empirical correlation between the reference size and $d50_{c\text{-}actual}$

Required Overflow Size Distribution (Percent Passing) of Specified Micrometer	Multiplier (To Be Used to Factor the Specified Micrometer Size)
98.8	0.54
95	0.73
90	0.91
80	1.25
70	1.67
60	2.08
50	2.78

Not unlike the screen design methodology introduced earlier, the hydrocyclone design approach illustrated here relies on a number of empirical correction factors to scale the performance of the standard hydrocyclone to the expected industrial operation. In this case, the key scaling parameter is the $d50_c$, and the key design formula is given by

$$d50_{c\text{-}actual} = d50_{c\text{-}base}(CP1)(CP2)(CP3)(\ldots)\,(CD1)(CD2)(CD3)(\ldots) \quad (14.3\text{-}12)$$

where

$d50_{c\text{-}actual}$ = the expected corrected cut size in industrial operation
$d50_{c\text{-}base}$ = the corrected cut size for the standard hydrocyclone
$CP1, CP2, CP3\ldots$ = process related correction factors (e.g., feed volume [~ viscosity] correction factor)
$CD1, CD2, CD3\ldots$ = design-based correction factors, (e.g., vortex finder diameter)

In practice, Equation 14.3-12 is rearranged and used in the form of Equation 14.3-13:

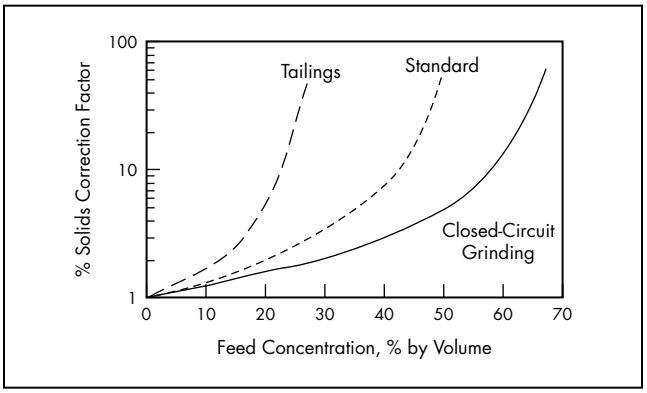
$$d50_{c\text{-}base} = \frac{d50_{c\text{-}actual}}{(CP1)(CP2)(CP3)(\ldots)\,(CD1)(CD2)(CD3)(\ldots)} \quad (14.3\text{-}13)$$

With this short introduction, the basic steps in hydrocyclone design are as follows.

Step 1—Estimation of the $d50_{c\text{-}actual}$. Table 14.3-11 provides the relationship between the overflow size specification and $d50_{c\text{-}actual}$. For the overflow specification in Figure 14.3-41, 60% corresponds to a factor of 2.08, which when multiplied by the reference size of 74 μm gives the estimate for $d50_{c\text{-}actual}$ of 154 μm.

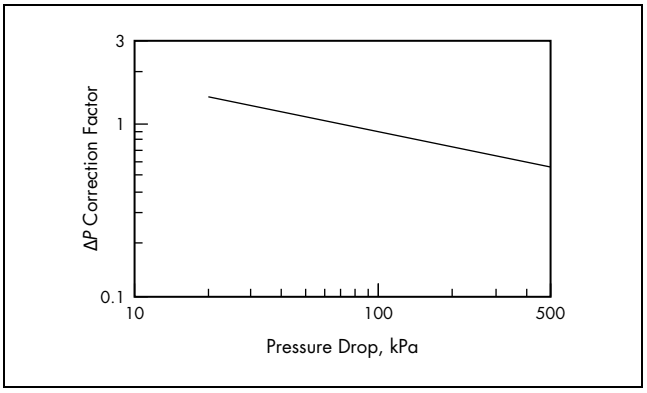
Step 2—Calculation of the correction factors and estimation of $d50_{c\text{-}base}$. Three important published process-related correction factors are the solids feed volume concentration ($CP1$), the operating pressure drop ($CP2$), and the solids specific gravity ($CP3$).

$CP1$ is the correction for the feed volumetric concentration, which is in effect a first-order approximation to viscosity. Of course, viscosity also depends on water temperature, clay concentrations, particle shape, and other factors—clays in fine tailings is what gave rise to the tailings curve in Figure 14.3-43. In this figure, Olson and Turner



Source: Olson and Turner 2002.

Figure 14.3-43 Correction factors for hydrocyclone feed solids volumetric concentration



Source: Arterburn 1982.

Figure 14.3-44 Pressure drop correction factor

(2002) extended Arterburn's method, which was based on the standard curve in Equation 14.3-14.

$$CP1 = \left[\frac{53 - V}{53}\right]^{-1.34} \quad (14.3\text{-}14)$$

In this example the standard curve is assumed, and with V = 33.3%, then $CP1 = 4.09$.

$CP2$ is the correction factor for the operating pressure drop—usually the difference between the measured pressure in the radial header and atmosphere. Larger hydrocyclones tend to run at lower pressures, and vice versa. Figure 14.3-44 and Equation 14.3-15 show this relationship.

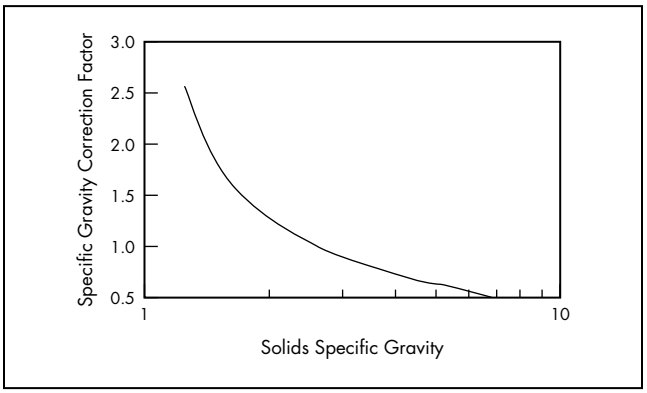
$$CP2 = 3.27\Delta P^{-0.82} \quad (14.3\text{-}15)$$

where ΔP is the pressure drop in kPa.

In this example, it was assumed that ΔP = 50 kPa (7.25 psig), which should give relatively good hydrocyclone performance while minimizing wear. From Equation 14.3-15, $CP2 = 1.09$.

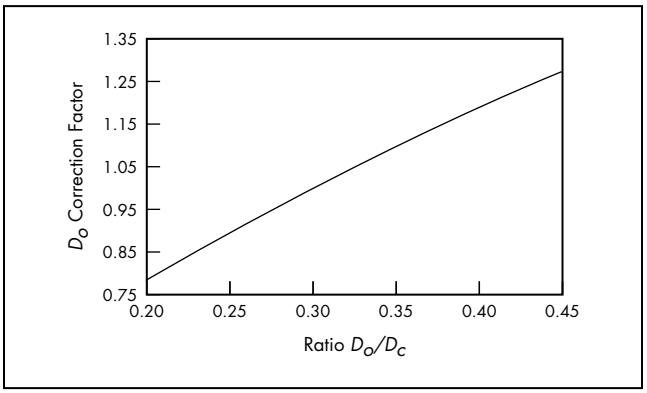
$CP3$ is the correction factor for the solids specific gravity and is shown in Figure 14.3-45 and Equation 14.3-16. For a solids specific gravity of 2.9, the correction factor $CP3 = 0.93$.

$$CP3 = \left[\frac{1.65}{\rho_s - \rho_l}\right]^{0.5} \quad (14.3\text{-}16)$$



Source: Arterburn 1982.

Figure 14.3-45 Specific gravity correction factor



Source: Olson and Turner 2002.

Figure 14.3-46 Vortex finder correction factor

$CD1$ is the correction factor for the vortex finder diameter. In this example the default design of $D_o = 0.3D_c$ is used, but Figure 14.3-46 and Equation 14.3-17 illustrate the nature of the correction.

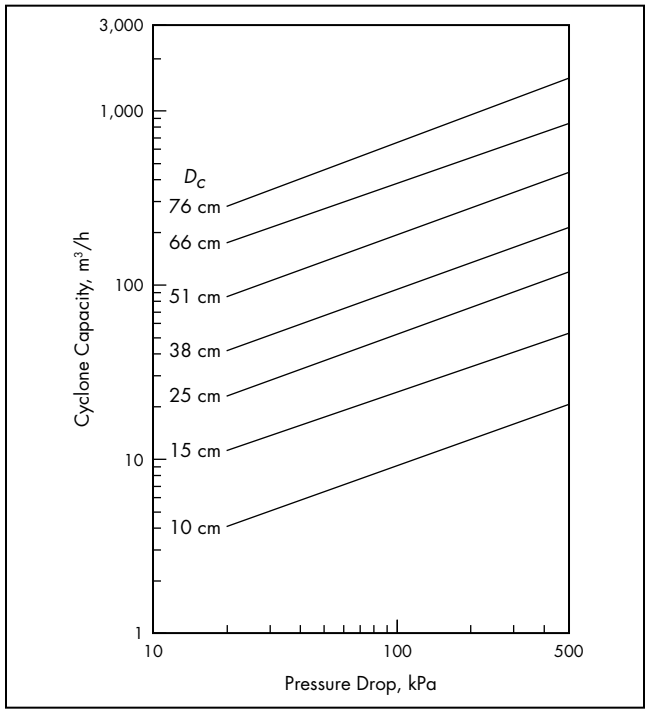
$$CD1 = \left[\frac{D_o}{0.3D_c}\right]^{0.6} \qquad (14.3\text{-}17)$$

Equation 14.3-11 then solves as

$$d50_{c\text{-}base} = \frac{154\ \mu m}{(4.09)(1.09)(0.93)(1)} = 37\ \mu m \qquad (14.3\text{-}18)$$

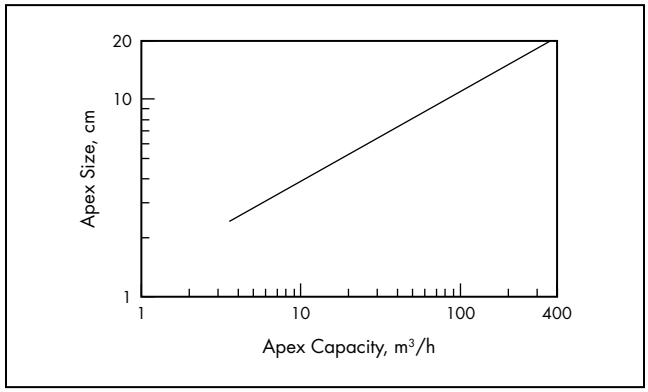
Step 3—Select size and number of hydrocyclones. From Equation 14.3-11 or Figure 14.3-42, the diameter of the hydrocyclone required to give this $d50_{c\text{-}base}$ is 49.1 cm (19.3 in.). Hydrocyclones, like screens, come in certain standard sizes, usually defined by D_c. As seen in Figure 14.3-47, the closest size to the "theoretical" value computed previously is a hydrocyclone with D_c = 51 cm (20.1 in.).

For a pressure drop of 50 kPa (7.25 psig), the capacity of a single 51-cm (20.1-in.) hydrocyclone will be approximately 140 m^3/h (616 gpm). From Figure 14.3-41, the total feed volumetric flow is 842 m^3/h (3,707 gpm), so approximately six hydrocyclones will be required in operation. In practice, two or three additional hydrocyclones would be added to accommodate maintenance as well as peaks in the feed flow rate. It is also not uncommon to provide an additional off-take valve in the radial header to be used for sample-collection purposes.



Source: Arterburn 1982.

Figure 14.3-47 Hydrocyclone capacity correlation



Source: Arterburn 1982.

Figure 14.3-48 Apex size correlation

Step 4—Select apex size. The last step is to determine the apex size, and that comes from Figure 14.3-48. For a total underflow rate of 396 m^3/h (1,744 gpm), the flow through each apex would be ~66 m^3/h (291 gpm), corresponding to an apex size of ~9.3 cm (3.66 in.). The manufacturer's closest size would be chosen, if possible tending to a smaller size in anticipation of wear.

Of course, after the hydrocyclones are installed and in operation, the reality of the application is known, and the optimization process (process variables and design variables such as D_o and D_u) can be modified as required.

Although the suppliers have undoubtedly added many refinements to their proprietary methods for hydrocyclone design, such as screens, the published methods for design have changed little since the 1980s. However, with the advent of some of the new, more efficient hydrocyclone technology

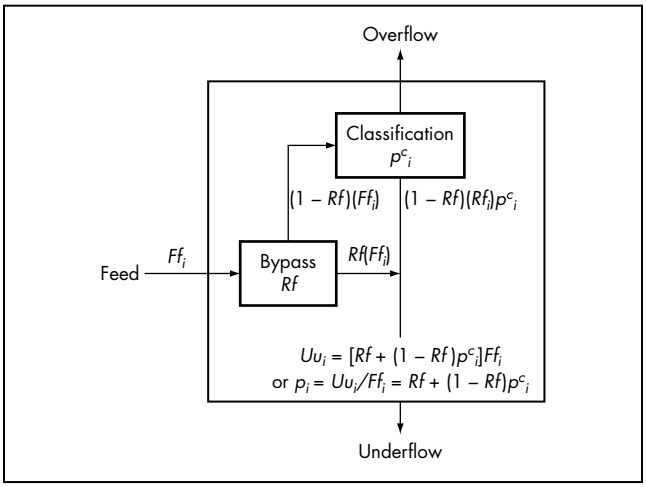


Figure 14.3-49 Conceptual model of hydrocyclone operation

Table 14.3-12 Experimental data for an inefficient hydrocyclone

Size, μm	Cyclone Feed	Cyclone Underflow	Experimental Partition
6,730	0.17	0.24	1.00
4,760	0.41	0.57	1.00
3,360	0.53	0.74	1.00
1,680	6.77	9.49	1.00
850	14.89	20.40	0.98
600	10.21	13.03	0.91
425	11.37	12.97	0.81
300	11.05	10.59	0.68
212	7.96	6.61	0.59
106	12.07	8.86	0.52
74	4.11	2.65	0.46
53	2.80	1.77	0.45
Pan	17.66	12.07	0.49
Total	100.0	100.0	—
Mass	1,237.3	880.9	—
Percent solids	65.1	73.6	
Rf(water)	0.47	—	—

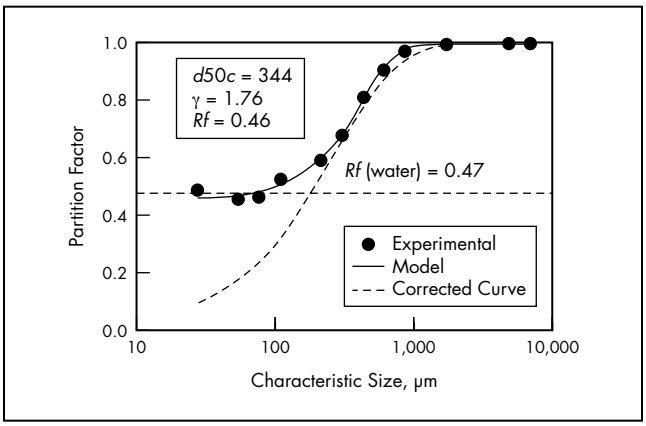


Figure 14.3-50 Partition curve for an inefficient hydrocyclone

to be discussed later, these published methods will require an adaptation.

Performance Assessment

Hydrocyclone efficiencies are usually assessed through the partition curve, and this implies a sampling experiment to acquire the necessary data. However, unlike screens, the fact that hydrocyclones are treating fine particles in slurries introduces another dimension to the simple modeling approach used for screening (Equation 14.3-6). The liquid phase reporting with the underflow stream carries with it fine particles that should have otherwise reported to the overflow stream. A common conceptual model is illustrated in Figure 14.3-49. It is assumed that a fraction of the feed material is bypassed directly to the underflow stream and the rest of the material is then subjected to the classification process (characterized by γ and $d50c$). (In this figure, upper case letters F and U are mass flows in the feed and underflow, while f_i, and u_i are the mass fractions retained in the ith size class. $p^c{}_i$ is the corrected partition factor for the ith size class, usually given by Equation 14.3-6. Modeling convention is to number the size classes from 1 (coarsest size fraction) to N (the so-called pan fraction) from the sieve analysis. For example, in Table 14.3-12, the particles greater than 6,730 μm will be size Class 1, those falling into the interval 6,730 × 4,760 μm, will be size Class 2, and so on.

Thus, the basic partition curve model for hydrocyclones is given by

$$p_i = Rf + (1 - Rf)\left[1 - e^{-0.693\left(\frac{d_i}{d50_c}\right)^{\gamma}}\right] \qquad (14.3\text{-}19)$$

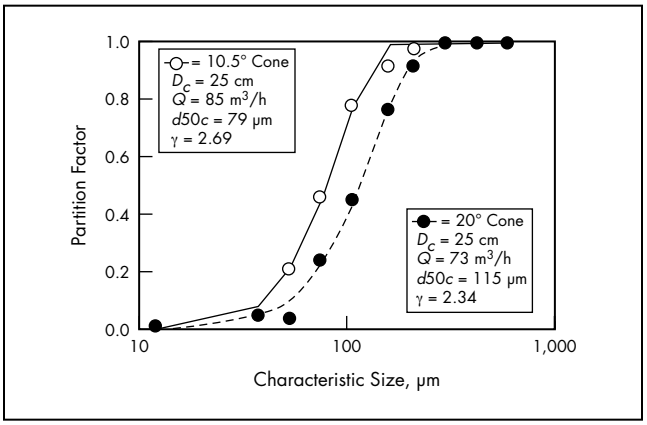
where Rf is the the bypass fraction.

Table 14.3-12 provides some experimental data for an operating hydrocyclone. Figure 14.3-50 compares the experimental data with modeling results with Equation 14.3-19, and the fit is seen to be very good. (The inefficient hydrocyclone was chosen for this illustration as the data are used to explore other measures of efficiency.)

The corrected partition curve (the term in the braces in Equation 14.3-19) is shown as the dashed line in Figure 14.3-50. Removing the effect of Rf in this way permits a more objective assessment of $d50_c$ and γ. The $d50$ for the experimental data in Figure 14.3-50 is about 106 μm, not very close to the $d50c$ value of 344 μm. Of course, this finer cut is because of the large number of fine particles carried to the underflow by the water—the Rf factor.

To amplify on Rf, the very fine particles ($d_i \rightarrow 0$ mm) are assumed to be more or less trapped in the water by virtue of the extraordinary drag forces. Hence, it is expected that the Rf deduced from the partition curve analysis will be equivalent to the water recovered from the feed to the underflow stream. In a well-constructed and executed experiment, the Rf estimated from curve fitting and the Rf(water) calculated from the water split should indicate good agreement, as evidenced in Figure 14.3-50.

For completeness, not all experimental partition curves behave like those for a homogenous ore (i.e., a low-grade ore with a gangue matrix of approximately constant specific gravity). For example, it is not uncommon to see the so called "fish-hook" behavior at the fine end of a partition curve, where the p_i values actually dip significantly below the expected Rf(water) value, before coming back to this level at the



Source: Krebs Engineers 2000.

Figure 14.3-51 Effect of cone angle on hydrocyclone performance

ultrafine sizes. While beyond the scope of this work, there are ways to model this if necessary, but the physical explanations are not so straightforward. Similarly, there can be inflections at the coarser end of the curve, so called "humps" or even "dips," and these are usually related to the separate classification action of liberated heavy and light mineral particles. In this latter case, one must separate these fractions experimentally (or mathematically) and model the curves separately.

From an operational point of view, maximum efficiency is achieved when γ is maximized (values of ~2.5 or larger are considered "good") and *Rf* is minimized (values of 0.3 or less are considered "good"). In closed-circuit grinding, *Rf* is a clear indication of the amount of fine material needlessly sent back to the mill. In practice, there is always a compromise between efficiency and operational objectives. In general both γ and *Rf* will improve with a lowering of the solids volumetric concentration in the feed. *Rf* can be also reduced using flat-bottom or inclined hydrocyclones, both of which maximize the underflow solids content.

A final word of caution on partition curves is that there are numerous simple two-parameter functional forms used to characterize the corrected partition curve. They all have in common the parameters $d50c$ and γ. However, while the $d50c$ values will usually compare well, the same is not true for the γ values. It is important that one be aware of the underlying functional form in drawing any comparisons between plant results and those quoted by others. To very quickly illustrate, another commonly used model is given in Equation 14.3-18.

$$p_i = Rf + (1 - Rf)\frac{\left[\exp\left(\gamma\frac{d_i}{d50_c}\right) - 1\right]}{\left[\exp\left(\gamma\frac{d_i}{d50_c}\right) + \exp(\gamma) - 2\right]} \quad (14.3\text{-}20)$$

When Equation 14.3-18 is fit to the data in Figure 14.3-47, the parameter values are $d50_c$ = 340 µm, Rf = 0.44, and γ = 2.13. For a given data set, γ values from the model in Equation 14.3-20 are always higher than those from the model given by Equation 14.3-19.

A number of other efficiency measures, such as imperfection and probable error, are sometimes used with hydrocyclones. Many of these are based on the partition curve. In addition, some people use the efficiencies defined earlier for screening (Equations 14.3-3 and 14.3-4). To illustrate, the partition curve in Figure 14.3-50 is seen to be rather inefficient, both in terms of γ and especially in terms of *Rf*. (This is a high-density feed with older model hydrocyclones.) The corresponding figures are E_u = 62.2% (fines removal from oversize) and R_u = 41.2% (fines removal from feed to overflow). Not surprisingly, these poor performance figures, especially R_u, have been the main drivers to explore fine-screening solutions and more efficient hydrocyclone designs.

Recent Developments

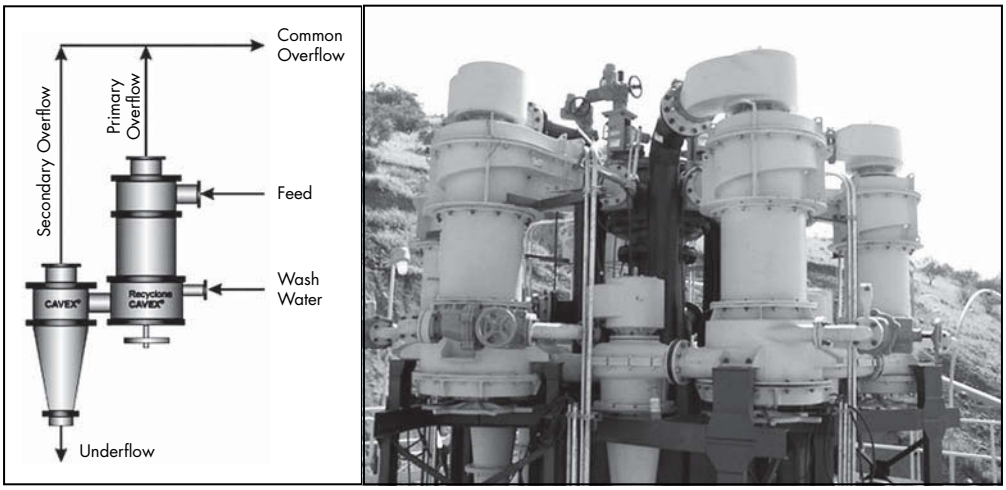
Design enhancement on existing hydrocyclones. Recently, both FLSmidth Krebs (gMAX) and Weir Minerals (Cavex) have introduced new, more efficient hydrocyclone designs. In both cases it appears that design improvements were made in the feed inlet to minimize turbulence when the new feed enters the body of the hydrocyclone, which impacts capacity, wear, and efficiency. Moreover, both OEMs made changes to the hydrocyclone body design as well. FLSmidth Krebs (Krebs Engineers 2000) exploited the potential for efficiency improvements associated with increasing residence time in the critical conical section of the hydrocyclone, as shown in Figure 14.3-51. An extensive redesign campaign yielded optimal combinations of cylinder and cone length, as well as cone design for each standard gMAX hydrocyclone diameter.

Mohanty et al. (2002) reported that a conventional hydrocyclone separating a fine coal feed (~45 µm cut target) yielded an efficiency of R_u = 37%, while the gMAX improved this to 68%. Adding the FLSmidth Krebs Cyclowash attachment (a means of injecting water into the lower cone section of a hydrocyclone to remove fines) gave a further increase to 70%. (Further technical details may be found on the supplier's Web site.)

New hydrocyclone systems. In addition to the work underway to improve traditional hydrocyclone designs, new designs are also under study by equipment manufacturers and researchers alike, although in general these are application-specific products. Two good examples of this are the three-product hydrocyclone (University of Cape Town) and the ReCyclone (Weir Minerals).

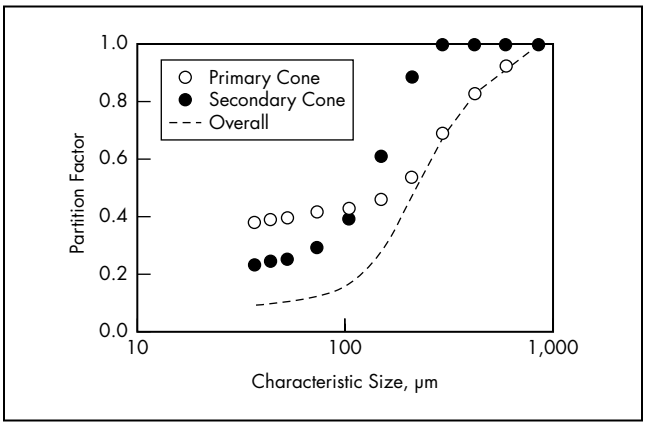
Briefly, the three-product hydroyclone (e.g., Mainza et al. 2004) is intended for use in applications where there are components of distinctly different densities. In conventional hydrocyclones, the heavier minerals concentrate near the hydrocyclone wall, which creates a heavy medium that selectively sends the larger, lighter particles toward the inside of the hydrocyclone, where they can be sent to the overflow. In the case of the platinum ores in South Africa, the lighter silica gangue contains the platinum group metals, while the heavier chromite is barren. Larger silica particles that report to the overflow represent potential recovery losses in the flotation process. The three-product hydrocyclone uses concentric vortex finders to produce two overflow products, and the inner vortex finder collects the fine particles, which are sent directly to flotation. The outer vortex finder collects a coarser middlings overflow, rich in coarse silica, which is then screened to ensure that the coarser material gets to the mill for further grinding, while the finer heavy material joins the flotation feed. Development work continues with this system, which would appear to have other applications in the mineral processing industry.

The ReCyclone (D. Switzer, E. Castro, and J. Lopez, personal communications) uses two-stage classification as a means to reduce *Rf* and increase γ. This concept has been studied and promoted by many academics and a few operators. It



Courtesy of Weir Minerals.

Figure 14.3-52 Schematic and installation of the ReCyclone



Courtesy of D. Switzer, E. Castro, and J. Lopez.

Figure 14.3-53 Partition curves for a ReCyclone

does find application in some coal and iron-ore flow sheets, but generally the extra footprint and the capital and operating expenses associated with such an approach has diminished interest among plant designers. One area where this has been extensively applied is in tailings dam construction, where it is critically important to move as much of the fines as possible from the sand product used to construct the dam. In these cases, two batteries of hydrocyclones (larger primaries followed by smaller secondaries) are common, to ensure that sand percolation rates are sufficiently high to avoid stability problems in the dam. The ReCylone performs these steps in a single unit.

Figure 14.3-52 is an illustration of the ReCylone unit and Figure 14.3-53 presents partition curves for the primary and secondary units and for the overall curve. Note the adjustment system on the primary underflow, which acts conceptually like a variable apex. The effects on *Rf* and, hence, fine particle content in the underflow are clear. All of the case studies reported by D. Switzer, E. Castro, and J. Lopez (personal communications) are on tailings. However, one can easily envisage other applications.

Instrumentation. Although little has been published thus far, the major hydrocyclone suppliers and some third parties are actively looking at technology value additions that would help to improve performance and minimize maintenance costs. For example, hydrocyclones are known to vibrate when they transition from normal underflow spray to a roping regime, and vibration and/or acoustics (e.g., Wedeles 2007) could be applied to diagnose this event or other events such as apex plugging (e.g., tramp grinding balls in the hydrocyclone feed). Another example is the elusive measurement of the relative wear of hydrocyclone liners, an area also under study by pump manufacturers. The interested reader should contact the suppliers directly on this topic.

Computational fluid dynamics (CFD). As was seen earlier in the discussion on screens, most major suppliers are rapidly embracing sophisticated simulation tools as a way to create and screen virtual machines for the purposes of design and optimization. Although this does not preclude the necessary step of field testing prototypes, it does help in the selection of the most appropriate candidate designs, reducing research and development costs and technical risk, and shortening the time to market.

CFD is the simulation technique of choice for fluid systems such as the hydrocyclone. However, at this stage it is impossible to simulate real slurries, and the standard approach is to simulate the fluid and then to use these results with some "tracer particles" (within CFD or in the DEM environment) to gain insight on classification efficiency. This one-way coupling (the fluid influences the particles, but the converse is not true), albeit with dilute suspensions, provides quantitative predictions based on first principles.

Suffice it to say, CFD played a major role in the development of both the Cavex and gMAX hydrocyclone designs described earlier. For the interested reader, the paper by Delgadillo and Rajamani (2007) provides a current vision of the benefits of CFD in advancing the hydrocyclone state of the art.

Media

Briefly, hydrocyclones are usually constructed of urethane or steel. Steel hydrocyclones are lined with wear-resistant materials such as rubber, ceramics, or urethane, and they can sometimes employ special metallic-based compounds. Again, the OEM is in a very good position to provide expert application advice on media choices.

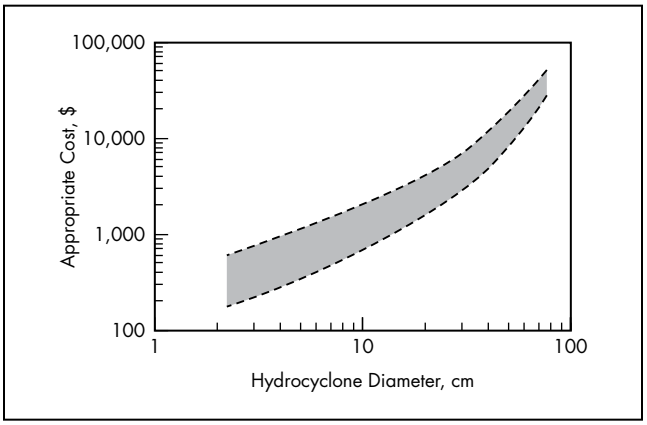


Figure 14.3-54 Typical hydrocyclone pricing

Standard Hydrocyclone Sizing and Relative Pricing

Each hydrocyclone manufacturer has a specific standard product size range, as shown in Figure 14.3-47. Generalizing across several of these suppliers, the usual sizes are approximately 10, 15, 25, 40, 50, 65, and 85 cm (3.9, 5.9, 9.8, 15.7, 19.7, 25.6, and 33.5 in.).

From an investment point of view, the total installation cost depends on the number of hydrocyclones, the materials of construction and lining, the type of radial header, the level of automation (manual versus automatic valves), and so forth. Figure 14.3-54 simply indicates the approximate cost of a single hydrocyclone and demonstrates the usual spread depending on whether lesser or more expensive materials are employed.

ACKNOWLEDGMENTS

All figures and tables not specifically referenced are included courtesy of Metso Mining. The authors extend their most sincere thanks to Derrick Corp., Weir Minerals, FLSmidth Krebs, and our many colleagues at Metso Mining who provided much of the information contained herein, as well as some very helpful feedback.

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CHAPTER 14.4

Gravity Concentration and Heavy Medium Separation

Nimal Subasinghe

Gravity concentration is one of the oldest processes of separating minerals and is aimed at separating heavy minerals from lighter gangue. Gravity concentrators have wide applications in the heavy minerals industry for the recovery of heavy minerals, such as ilmenite, rutile, leucoxene, and zircon, from the lighter gangue. They are used to separate metal sulfides from ores where the sulfides are coarse grained. They have been used to exploit placer gold deposits over many centuries, and as an adjunct in modern mills to recover coarse gold early in the process. When the feed material is comprised of particles with a wider size distribution, gravity concentrators do not produce a clean concentrate and have been used mainly for preconcentration or as a rougher stage. Gravity concentrators are simple devices that are inexpensive to operate and cause comparatively much less environmental pollution than using harmful chemical reagents for the same purpose. Gravity separation may be achieved in either of two ways: relative gravity separation and absolute gravity separation.

In relative gravity separation, all particles settle in the same direction within the device, and large differences in specific gravity (density) between the minerals are needed for effective separation. As settling velocity of particles depends not only on density but also on size, it is important to prepare narrowly sized feeds to these devices. However, as particles of finer sizes settle under gravity at much slower rates, the effectiveness of these devices decreases considerably below about 50 μm. The heavy and light minerals that settle at different rates are made to report to different exit streams by applying a transverse force. There are many devices of this nature, such as jigs, spiral concentrators, Wilfley tables (shaking tables), air tables, Gemini tables, pinched sluices, riffled sluices, Reichert cones, and so on. To improve the efficiency of separation of finer material, centrifugal concentrating devices have been introduced within the past two decades, including Knelson concentrators, Falcon concentrators, Kelsey jigs, and water-only cyclones. These devices use centrifugal acceleration to enhance the movement of finer particles and have been extensively used in the gold processing plants to recover fine gold.

In absolute gravity separation, a fluid or a nonsettling slurry comprised of fine solid particles with a specific gravity (SG) intermediate to those of the two minerals to be separated is used as the medium of separation. Particles of higher and lower specific gravities than the medium move in opposite directions. This technique requires only a small difference in specific gravities of the minerals for effective separation. Heavy or dense medium separation methods are absolute separation methods. Devices of this sort include dense medium vessels and dense medium cyclones.

The amenability to gravity concentration is indicated by the concentration criterion, C, defined (Taggart 1951) as

$$C = \left(\frac{\rho_h - \rho_f}{\rho_\ell - \rho_f} \right) \tag{14.4-1}$$

which has been deduced from Newton's law of settling, where ρ_h, ρ_ℓ, and ρ_f are the densities of heavy mineral, light gangue, and water, respectively. If C is greater than 2.5, the separation may be achieved with relative ease; separation is not commercially feasible for values below 1.25. Table 14.4-1 shows the effectiveness of gravity separation at various feed sizes.

GRAVITY CONCENTRATORS

Gravity concentrators may be classified according to the way particles are made to move relative to each other so that the denser particles will segregate from the lighter ones under the influence of the gravitational force. A gravity concentrator uses essentially one of three mechanisms to provide movement of particles within it: jigging, shaking, and flowing film devices.

Table 14.4-1 Values of concentration criterion required for effective separation

Value of Concentration Criterion, C	Significance for Separation
>2.50	Separation effective down to ~75 μm
2.50–1.75	Separation effective down to ~150 μm
1.75–1.50	Separation possible to 2 mm, but difficult
1.50–1.25	Separation possible to 6 mm, but difficult
<1.25	Relative separations impractical. Absolute separation may be possible down to C of 0.2.

Source: Aplan 1985.

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These gravity separation devices rely on a fluid medium, such as water, to transport the material through the device and also bring about a separation based on settling characteristics of heavy and light particles. The process requires the feed material to be uniformly sized and fed as a slurry. If the feed has a wide size distribution, the smaller heavy mineral particles will report to the same exit as larger light particles because their settling velocities are similar. As the particle size decreases, the differences in gravitational forces acting on heavy and light particles, which bring about differences in their movement, also diminish. Recent advances in the design of gravity separation devices have addressed this problem by introducing "multi-G" devices that impart higher accelerations on smaller particles through centrifugal forces than those due to gravity. These devices are known as centrifugal concentrators. Thus, each gravity concentrator has an optimal feed particle size range for effective separation. The magnitude of the applied acceleration is usually quoted in terms of multiples of the gravitational acceleration, *G*. Figure 14.4-1 shows the classification of gravity separation devices. Table 14.4-2 shows the range of feed sizes suitable for various gravity separators.

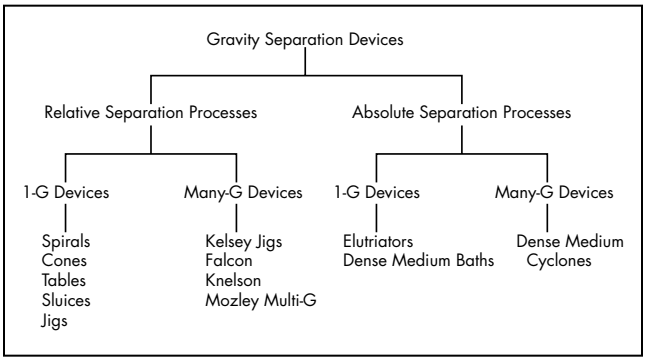


Figure 14.4-1 Gravity separation equipment

Table 14.4-2 Effective feed size ranges for typical gravity concentration devices

Coarse particle concentration [+6.4 mm]	
Jigs	Heavy media hydrocyclone
Heavy media separators	
Intermediate concentration [–6.4 mm + 0.150 μm]	
Jigs	Shaking tables
Heavy media hydrocyclone	Spirals
Water-only cyclone	Pinched sluices
Kelsey jig	Reichert cone
Pneumatic jigs and tables	Bartles–Mozley concentrator
Sluices	
Fines concentration [–0.150 μm]	
Shaking table	Falcon concentrator
Spirals	Knelson concentrator
Pinched sluices	Bartles–Mozley concentrator
Reichert cone	Mozley multigravity separator

Source: Adapted from Aplan 2003.

Jigging Devices

Jigging is one of the oldest methods of gravity concentration. It is commonly used in coal, cassiterite, gold, and iron ore industries. Generally jigs are used for separating coarser material from about 2 to 10 mm in diameter, although efficient separation of particles down to about 75 μm is possible with modified devices. They can handle large tonnages and operate on a continuous basis.

Jigging essentially uses alternating expansion and contraction of a bed of particles. The bed is generally formed over a screen through which the fluid is allowed to flow. Ragging comprised of heavy particles is generally placed over the screen to retain the bed and release the segregated heavies to the hutch during the suction stroke. The bed movement is activated by a pulsating current of fluid, usually water, or by the movement of the screen that holds the bed. The pulsation of the bed may be achieved by means of a plunger or by admitting compressed air at regular short intervals into a limb of a U-shaped tank. Light material is generally discarded through a gate positioned opposite to the feed end.

Gaudin (1939) suggested that three segregation mechanisms may take place within a jig: differential initial acceleration, hindered settling, and consolidation trickling. The extent of prevalence of these mechanisms depends on the properties of the material being processed, such as size and density and the manner in which the force is applied to dilate and contract the bed.

The differential initial acceleration between particles in a pulsating bed is a function of their density and is independent of size. That is, the initial acceleration conveyed to heavy mineral particles by the pulsating current of fluid is larger than that for lighter mineral particles. By applying a higher frequency of pulsation, this effect may be accentuated. This mechanism will segregate heavy minerals beneath the lighter gangue.

Also, during the pulsation stroke, heavy particles within the expanded bed will settle faster than the lighter ones. However, because of the presence of a large number of particles within the bed, hindered settling of particles takes place, the result of particle–particle interactions as opposed to free settling. The differences in hindered settling rates of larger particles are more pronounced than those of smaller particles. Hence, jigs perform better on coarser particles.

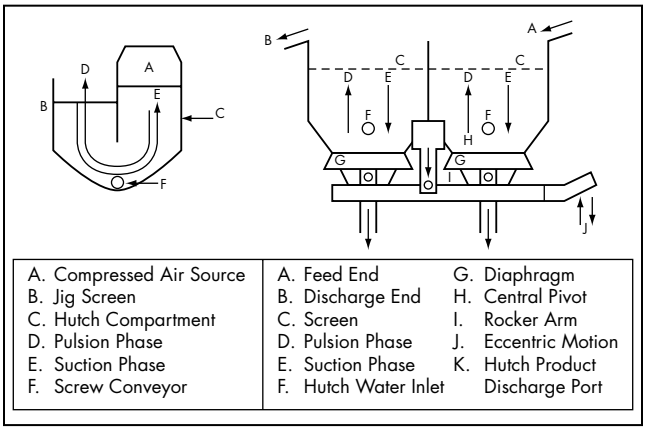
When the dilated bed collapses during the suction stroke, larger particles tend to interlock, leaving voids through which the finer particles may segregate. This process is known as interstitial or consolidation trickling.

Coal Jig

Baum jigs are commonly used for processing coarse coal whereas Batac jigs are used mainly for fine coal. Both these units use compressed air to pulsate the bed. In these units, water is typically pulsed at 60 to 80 pulses per minute and the units handle feed sizes up to 130 mm at capacities of up to 40 t/m^2/h.

Mineral Jig

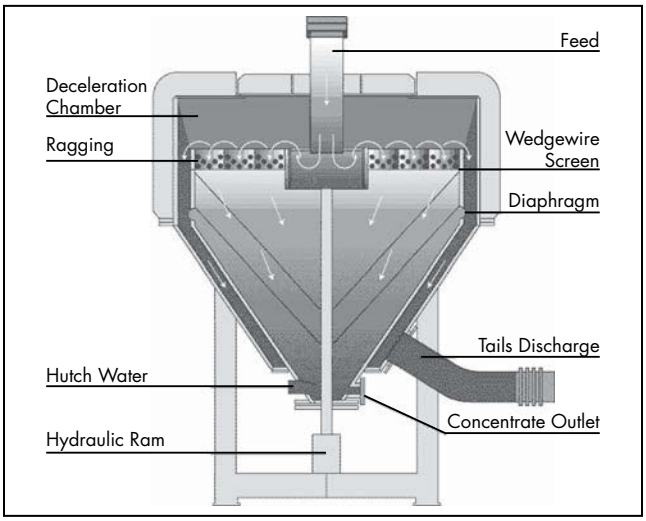
Jigs that treat placer deposits generally have a diaphragm in the hutch area, which is placed underneath the screen that holds the bed. Typical jigs of this type are Cleveland and Pan-American placer jigs, which have a smaller footprint. Harz and Denver jigs have a wall that separates the hutch area and use an eccentrically driven plunger in one section to pulsate the bed. Schematic diagrams of a few jigs are shown in Figure 14.4-2.



A. Compressed Air Source	A. Feed End	G. Diaphragm
B. Jig Screen	B. Discharge End	H. Central Pivot
C. Hutch Compartment	C. Screen	I. Rocker Arm
D. Pulsion Phase	D. Pulsion Phase	J. Eccentric Motion
E. Suction Phase	E. Suction Phase	K. Hutch Product Discharge Port
F. Screw Conveyor	F. Hutch Water Inlet	

Source: Aplan 2003.

Figure 14.4-2 (left) Baum jig, and (right) Pan-American placer jig



Courtesy of Gekko Systems Pty. Ltd., Australia.

Figure 14.4-3 In-line pressure jig

In-Line Pressure Jig

The in-line pressure jig (IPJ) combines a circular bed with a moveable sieve action. The screen is pulsed vertically by a hydraulically driven shaft. The length and speed of the up-and-down stroke can be varied to suit the application. Screen aperture, ragging dimension, and ragging material can also be altered for the application, which varies the recovery of solids to the concentrate stream. It uses less hutch water, compared to traditional jigs. The engineered ceramic ragging may be selected from a range of specific gravities from 1.6 to 5.0. IPJs have been used across many mineral types including gold, sulfides, tin, coal, and diamonds. Figure 14.4-3 shows a schematic diagram of an IPJ.

IPJs are available in a range of sizes handling 2 to 100 t/h of feed material. The solids recovery to concentrate may be varied from about 5% to about 50% by selecting the appropriate ragging density, ragging size, and screen aperture for a given feed size. Lower yields require tighter ragging sizing. The units are supplied with automatic controllers to control the feed rate, pulse frequency, stroke length, and air bleed rate for optimal performance.

Kelsey Centrifugal Jig

The Kelsey centrifugal jig (KCJ) imparts a centrifugal acceleration on the feed and ragging particles while a jigging action is being applied. It enables the effective separation of particles below about 500 μm down to fine sizes, and has found applications in tin, tantalum, gold, nickel, tungsten, chromite, base metals, iron ore, and mineral sands industries.

The centrifugal acceleration is conveyed by a spinning rotor, the speed of which may be varied and controlled. Inside the rotor, a screen of cylindrical shape is spun coaxially with the rotor. The ragging material is spread evenly on the inside of the screen because of the centrifugal acceleration. The feed enters through a fixed central pipe, then accelerates toward and continually rises up the ragging bed. Pressurized water is introduced into a series of hutches behind the screen, which is pulsed through the ragging to fluidize the bed. The pulsing of the bed is achieved via pulse arms connected to pulse pads, which push against flexible diaphragms at the back of each hutch, thereby pushing the water in the hutch through the screen and ragging bed.

KCJs can handle capacities up to 50 t/h solids and typically use about 30 to 50 m^3/h of hutch water. The rotor spins at speeds up to 200 rpm, generating accelerations of 40 *G*. The pulse rate is between 1,800 and 2,200 pulses per minute at a stroke length of 2 to 3 mm. A schematic diagram is shown in Figure 14.4-4.

The size of ragging used depends on the size distribution of the valuable mineral particles in the feed and should be chosen to avoid excessive blinding of the internal screen. Other operating variables include pulsation frequency, hutch water addition rate, ragging type, ragging SG, depth of bed, and so on. These units also have a number of design features, such as automatic screen cleaners and vibration detection. Programmable logic controller–based interlocks and sacrificial shear pins are provided to avoid major damage in the event of a malfunction.

Air Jig

Air jigs have been used in both coal and gold industries. In these devices, a low-pressure, high-volume air source is used to fluidize the particulate bed while a high-pressure, low-volume air source is used to impose the pulsation stroke for stratification. Recently, the air jig has found wider application in the coal industry as this method does not use process water, with the advantage of eliminating the need to dewater fines or manage slurry confinement. Particulate emissions have been minimized by the inclusion of fabric dust collectors. The new units use a punch plate instead of a fine wire mesh cloth to support a deeper particle bed. However, their application has limitations due to particle size, particle shape, moisture, and near-gravity content (Alderman 2002).

Flowing Film Separators

Flowing film separators have been used for centuries to accumulate heavy minerals such as gold and tin. These devices are cheap to make and operate. In flowing film concentrators, the feed slurry is made to flow over an inclined surface, and segregation of the heavy minerals beneath the lighter minerals takes place during the passage. At low flow rates, the thickness of the film is small and the particles generally settle out and roll over the surface. Heavier particles tend to accumulate in the lower regions closer to the surface and travel at lower velocities while the lighter materials travel closer to the surface and

faster. At high flow rates the particles will be suspended in the flow due to turbulent suspension, resulting in a density gradient across the depth of the flow. The particles also experience shearing because of Bagnold forces that give rise to a dispersive pressure on the particles. Commonly used flowing film separators are pinched sluices, Reichert cones, riffled sluices, and spiral concentrators.

Pinched Sluice

A pinched sluice is a device comprised of an inclined plane bottom surface and tapering sides. The feed slurry is fed at the higher end and is made to flow down the inclined surface. The segregated heavy minerals flowing closer to the bottom of the surface are removed through slots placed across the bottom of the channel at the discharge end while the lighter minerals flow over the slot and report to tailings. The depth of flow increases toward the lower end because of the lowering of the area of cross section. These sluices are used by small-scale operators treating alluvial gold deposits and in tin and heavy minerals industries. Figure 14.4-5 shows a schematic diagram of a pinched sluice.

These units are operated at inclinations of between 15° and 20° to the horizontal. Subasinghe and Kelly (1984) demonstrated that at higher inclinations the recovery of coarser particles decreases because of the presence of Bagnold forces acting on the particles. At lower slopes, coarse and dense particles settle out on the deck resulting in poor segregation of heavies in the bed.

Reichert Cone

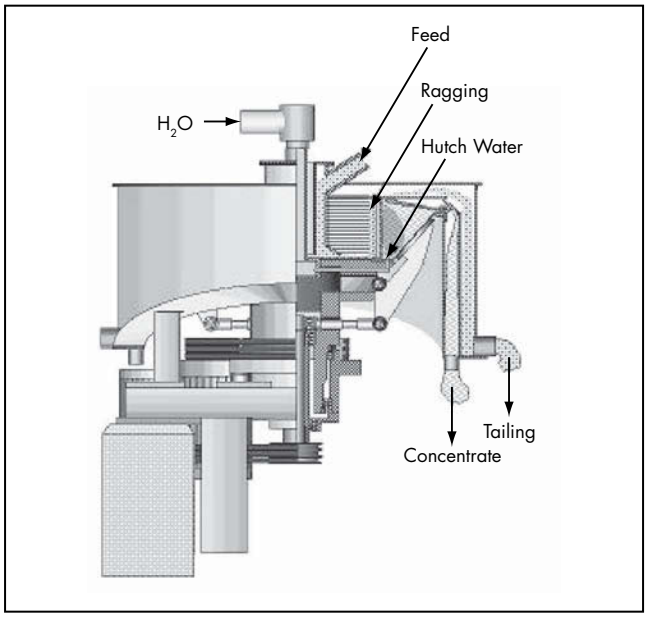
Reichert cones operate in a similar way to that of pinched sluices. The cone is designed as a series of pinched sluices joined in a circular pattern with their sides removed. An inverted cone is placed above the concentrating cone to distribute the feed slurry evenly. Heavy and light materials are removed through annular slots placed near the center column. Several cone pairs may be stacked on top of each other to increase the quality of the separation. Cones are operated continuously. Figure 14.4-6 shows a schematic diagram of a Reichert cone.

Riffled Sluice

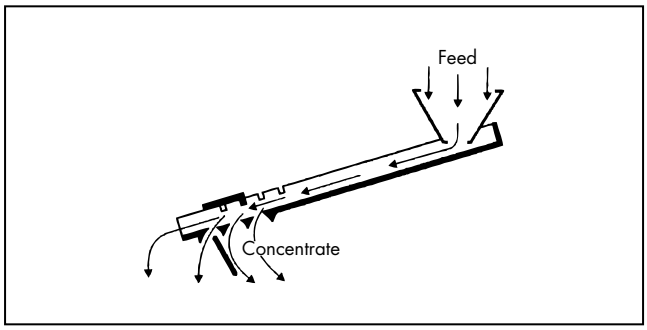
Riffled sluices are commonly used by artisan miners to recover gold. The segregated gold or heavy minerals are recovered behind the riffles placed across the sluice. The feed flow rate, height, and spacing of the riffles determine the size of the recovered particles. If the feed contains fine gold, then a lower flow rate and smaller riffles are used. Sometimes instead of the riffles, corduroy cloth or coir mats are used to trap the fine gold particles. Sluices have been used to process fine cassiterite ores.

Spiral Concentrator

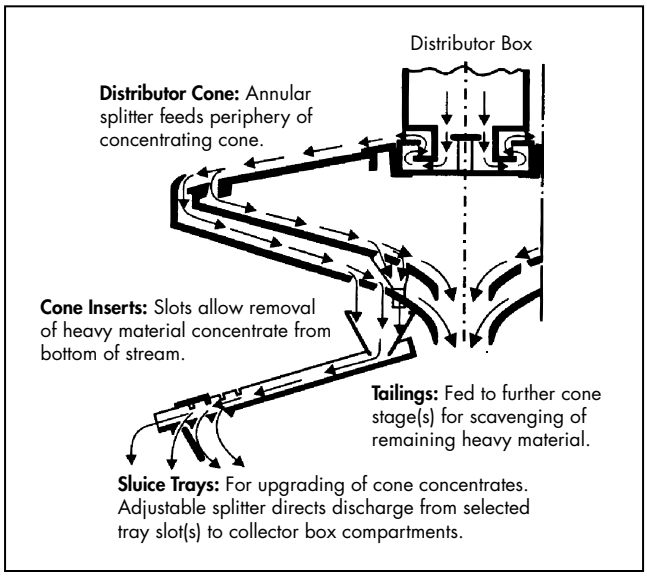
Spiral concentrators have found wide applications in the beach sand industry. In a spiral concentrator, the feed slurry is made to flow down a helical conduit of semicircular cross section. The slurry experiences a centrifugal force and has a tendency to move particles toward the outer rim. Because of the shape of the trough's cross section, these particles then travel inward, closer to the bottom of the trough. The heavy particles settle out of suspension and flow down the spiral closer to the trough's inner edge. There they are subjected to a smaller centrifugal acceleration and take up a trajectory closer to the



Courtesy of DownerEDi–Mineral Technologies.
Figure 14.4-4 Kelsey centrifugal jig



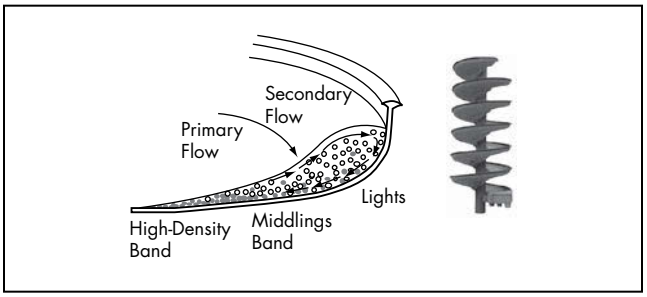
Courtesy of DownerEDi–Mineral Technologies.
Figure 14.4-5 Pinched sluice



Courtesy of DownerEDi–Mineral Technologies.
Figure 14.4-6 Reichert cone

axis of the spiral. The lighter particles move closer to the free surface and are carried toward the outer rim as a result of the centrifugal acceleration. When the slurry reaches the bottom of the spiral, separation of the heavies has occurred, and the heavies may be collected through the inner ports while the lighter material is collected from the outer port. To save floor space, two or more spirals may be incorporated into one unit known as a twin, triple, or quad start, thereby increasing the throughput rate.

The shape of the trough varies with the type of mineral being separated. Coal spirals generally have a flatter profile than mineral spirals. The pulp density may vary between 20% and 40% solids. The feed sizes range from about 2 mm to about 75 μm. The capacity of a single-start spiral may be up to 7.0 t/h, depending on the material being processed. The cut specific gravity depends on the pulp density, feed rate, and size distribution of the feed. In the beach sand industry, spirals are commonly used as a preconcentration stage. Figure 14.4-7 shows a schematic diagram of a concentrating spiral.

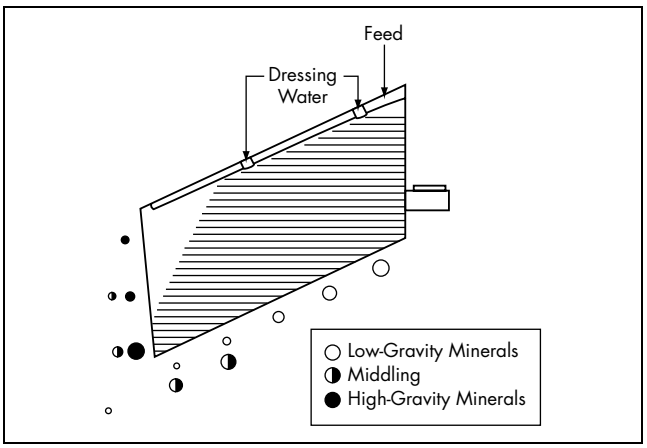


Courtesy of DownerEDi–Mineral Technologies.

Figure 14.4-7 Spiral concentrator

Shaking Devices

The most common shaking device is the concentrating table or the Wilfley table. Concentrating tables are very efficient gravity separators generally used in cleaning concentrates obtained from other devices. Concentrating tables or shaking tables are comprised of a rectangular flat table inclined horizontally by about 15° to 20°. Riffles are placed along the length of the table and the table is made to vibrate in the longitudinal direction in a reciprocating manner, with a slow forward stroke and a rapid return. The feed slurry enters through a port at the top corner and is distributed over the flat surface because of the vibratory motion. Wash water is introduced along the top edge of the table surface, washing the particles down the slope. Heavy mineral particles get trapped between the riffles and travel across the table in a longitudinal direction. The lighter minerals flow over the riffles and are carried down the slope and report to the tailings stream, nearer to the feed end of the table. The heavy mineral concentrate is collected at the opposite end of the table with middling particles collected in between. The feed to tables may vary from about 1 mm down to about 50 μm. The feed pulp density is between 25% and 35% solids. Figure 14.4-8 shows a schematic diagram of a Wilfley table.



Source: Aplan 1985.

Figure 14.4-8 Wilfley table and distribution of table products

Many designs and riffle geometries exist, depending on the type of material being processed. Tables have been in use for processing of minerals such as coal, barites, beach sands, chromite, garnet, iron, manganese, tantalum, tin, tungsten, and zircon. Generally the variables that may be manipulated to increase the efficiency of separation are deck slope, wash water flow rate, and the frequency and amplitude of the vibratory motion. With larger particles, longer strokes and lower shaking frequencies are used. The separation is judged mainly by visual inspection, and modifications to operating variables may be made accordingly. These are low-capacity devices with about 2 and 0.5 t/h/m^2 for 1.5-mm and 100-μm particles, respectively. As they take up much floor space, multiple decks may be stacked up one above the other to save space.

Gemini tables have found wide application in the gold industry as a device for cleaning concentrates of centrifugal devices such as Knelson and Falcon concentrators. For finer feeds, Bartles–Mozley tables are generally used, in which the shaking mechanism is replaced by a smoother oscillator movement to avoid mixing of the bed due to turbulence. Air tables have also been used in the mineral industry to recover fine particles in which the bed is fluidized using air.

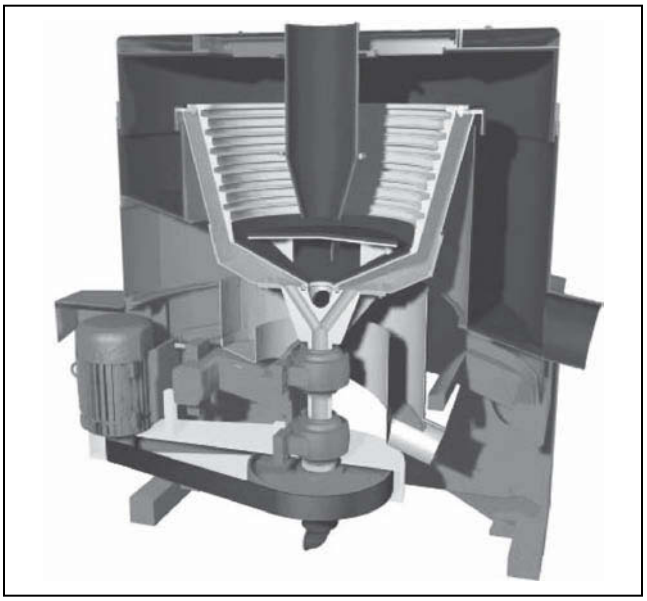
As an aside, air tables were originally developed for use in arid regions but have also been used in the separation of flat, fibrous materials of lower bulk densities such as asbestos and vermiculite. However, these devices demand that the feed be very narrowly sized for their efficient operation. As they also have only limited capacities, their use is limited only to special applications.

Centrifugal or Enhanced Gravity Concentrators

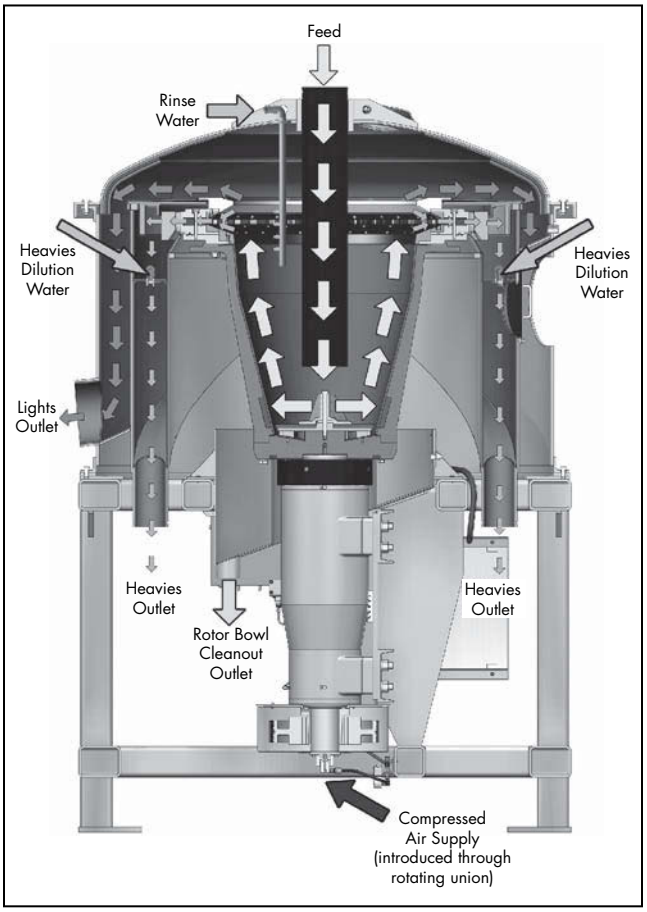
The movement of fine particles that settle under gravitational force within a fluid medium is slow. Even if there is a significant density difference between heavy and light minerals, the separations achieved are inadequate. However, by allowing the particles to traverse a circulatory motion at high speed, a centrifugal acceleration may be transmitted to them that may be several hundred times that of gravity. Devices that use this technique are called centrifugal concentrators. Two devices that have found wide applications in recovering fine gold and other minerals are the Knelson and Falcon concentrators.

Knelson Concentrator

Knelson concentrators, as shown in Figure 14.4-9, are comprised of a slightly conical-shaped rotating bowl with a series of grooves on the outer wall to capture the segregated heavy mineral particles. The feed material enters through a central feed inlet as a slurry. The slurry descends onto the base plate at the bottom of the rotating bowl and is thrown outward because of centrifugal acceleration. As a result,



Courtesy of Knelson Gravity Solutions.
Figure 14.4-9 Knelson concentrator



Courtesy of Falcon Concentrators Inc.
Figure 14.4-10 Falcon continuous concentrator

some of the particles get trapped within the grooves of the bowl forming a bed while the excess get carried upward into the tailings stream by the rising current of water. Injection of water through fluidization ports located in the grooves of the bowl prevents compaction of the concentrate bed. This creates a fluidized bed that acts as a concentrating chamber for heavy mineral particles, under an enhanced separating force. Upon conclusion of the concentration cycle, concentrates are collected manually in batch units or flushed from the bowl into a concentrate launder through a multiport hub in continuous units.

As heavy particles experience a higher centrifugal acceleration, they tend to move through the fluidized bed to displace the light materials and produce a heavy mineral concentrate within the grooves of the bowl. The lighter particles are displaced into the tailings stream by the fluidizing water. Fluidizing water flow rate, bowl rotational speed, and particle characteristics of the feed, such as the size and size distribution, determine the optimal conditions for separation and the cycle time.

Falcon Concentrator

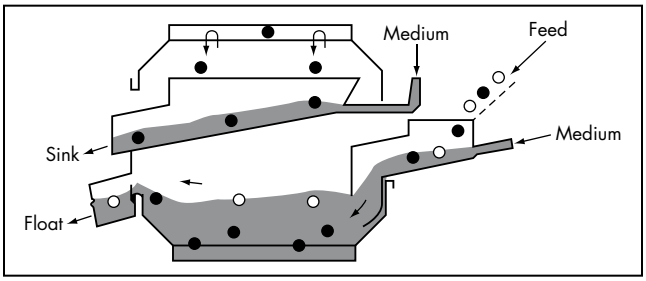
Falcon concentrators rely on a rotating bowl to generate high centrifugal accelerations similar to Knelson concentrators. The feed slurry rises upward over a slightly inclined smooth wall. Stratification of higher-density particles occurs toward the bowl wall, and lower-density particles move toward the center of the bowl. Different models vary in the way they collect and produce the concentrate.

In the semibatch unit, the upgraded product that has been forced against the bowl wall is made to move into a vertical, fluidized collection zone in the upper portion of the bowl. The collection zone consists of a number of rings that have pressurized water injected from the back, effectively elutriating and cleaning the concentrate that sits in these collection rings until a rinse cycle begins. These machines are known as semibatch devices because they continually accept feed during the run cycle but only produce concentrate during periodic rinse cycles. Run times range from 30 minutes to several hours depending on the application, while the rinse times are generally less than a minute. The throughput rates range from 1 to 400 t/h. The centrifugal accelerations range from 50 to 200 *G*. These units are mainly used in precious metal recovery plants.

In continuous devices, at the top of the bowl is a ring of specially designed concentrate flow hoppers, followed by pneumatically controlled variable orifice valves. These valves allow for a constant stream of concentrate to be produced. The continuous concentrator is able to vary mass pull to concentrate by adjusting valve orifice size through changes in air pressure. Because of their high mass pull rates of 5% to 40%, these units are used in the recovery and upgrade of tin, tantalum, tungsten, chrome, cobalt, iron, fine oxidized coal, and uranium ores. The ultrafine units are designed to concentrate material below 50 μm at less than 20% solids and may be typically used in deslime cyclone overflow streams. Figure 14.4-10 shows a schematic diagram of a Falcon continuous concentrator.

HEAVY MEDIUM SEPARATION

Heavy medium separation (HMS) is a very effective gravity separation technique in which a nonsettling heavy medium is used in place of a fluid. The heavy medium is generally



Source: Aplan 1985.
Figure 14.4-11 Heavy medium separator

comprised of a nonsettling suspension of fine heavy particles, the density of which is much higher than that of water. Thus, the concentration criterion described in Equation 14.4-1 increases considerably in this case, which indicates a more efficient separation. Also, since the medium density is between those of the heavy and light minerals to be separated, it gives rise to an absolute separation where the two products are separated into a sinks product and a floats product. This technique requires a much lower density difference between the minerals to be separated, compared to the relative separation techniques described previously.

With HMS it is possible to (a) produce a finished concentrate and a final waste product in one operation, (b) reject a waste product at a coarser size, thereby saving grinding costs, (c) achieve separation at a low operating cost with low maintenance costs, (d) make relatively sharp separations, (e) operate continuously, (f) tolerate feed with wide size distributions, and (g) produce a consistent product for further processing. The main industrial applications of HMS with coarse material feeds are in coal and iron ore industries. HMS has also been used in base metal industries such as copper, lead, and zinc. The most commonly used medium in coal separation is magnetite. Although silica, barite, and galena had been used in the past as the medium solid for ore separations, in recent times ferrosilicon (FeSi) has been used extensively. With finer feed materials, the viscosity of the slurry increases, and the separation efficiency decreases as a result. This situation has been rectified by the use of centrifugal separators such as heavy medium cyclones (HMCs). The water-only cyclones work similarly to an HMC that uses a loess medium instead of an external medium. See Figure 14.4-11 for a schematic diagram of a heavy medium separator.

Heavy Medium Solids

The solids used to make up the heavy medium depend on the type of material to be separated. The heavy medium suspension should be nonsettling, which requires the solid particles to be sufficiently small. For coal, the most commonly used are magnetite and barite, as the medium density should be less than the ash density. For ore separations, more dense solid materials such as ferrosilicon and galena have been used (Aplan 2003). Ferrosilicon contains about 15% Si. Ground ferrosilicon is used for separations at SGs up to 3.2; for higher SGs up to 3.8, atomized FeSi may be used. The main requirement in selecting a medium for a given operation is its ability to be separated from the products. Generally the products are washed and screened to remove the medium. Wet magnetic separators are also used extensively to recover magnetite and ferrosilicon. The medium recovered from the product streams needs to be thickened to the required consistency prior to recycling it back to the separating vessel.

Feed Preparation

It is important to prepare the feed prior to HMS, usually by wet screening. The main purpose is to size the ore into fractions that may be treated in various separating units. More importantly, screening is used to remove the slimes that would otherwise increase the viscosity of the medium, giving rise to inefficiencies.

Medium Recovery

Medium is generally removed from the products in drain and wash screens. Sieve bends are the preferred screen type for medium recovery. Prior to reusing, the medium needs to be cleaned using wet magnetic separators. Magnetic separators are fed at about 30% to 35% solids, and the rate is determined by the type of separator and magnetic susceptibility of the medium.

Heavy Medium Separators

HMS equipment must ensure the feed particles have sufficient time to report to the relevant discharge streams. They generally differ in the way the float and sink products are discharged. The float products will often discharge over a weir with or without the need for paddles or scrapers. The Wemco drum discharges the heavy mineral using lifters while rotating. The Drewboy washer uses paddles to elevate the heavies, while the Norwalt bath uses a bucket elevator. The Dutch State Mines bath uses a drag conveyor with a paddle to deliver the heavies and lights to different discharge ports.

The separations of finer sizes are carried out effectively in units that use centrifugal acceleration to aid in the separation and discharge of particles. HMCs and Dyna Whirlpool separators (DWPs) are the widely used equipment of this type.

Heavy Medium Cyclones

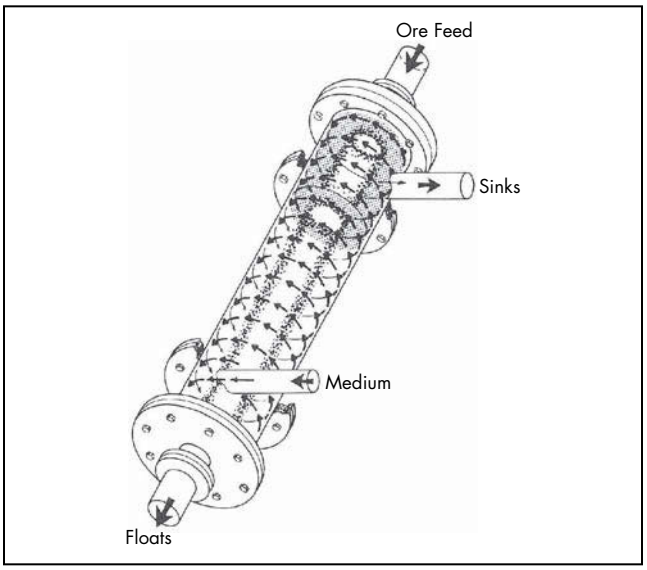
The HMCs are essentially a modified version of hydrocyclones used with a heavy medium as the separating fluid instead of water. In an HMC, the heavy mineral migrates to the outside wall and is discharged through the vortex finder. The shape of the HMC, however, differs from that of a classifying cyclone in that the height of the cyclone is comparatively shorter. HMCs treating coal operate at inlet pressures with a minimum head of about nine times the diameter of the cyclone. The diameters of these HMCs range from about 0.5 m to 1.4 m and have capacities up to 500 t/h. The relative density of the overflow stream should be between 3% and 12% lower than that of the feed stream.

Dyna Whirlpool Separator

The DWP also imparts a centrifugal acceleration to the feed particles. A DWP is essentially a sloping cylinder with cover plates and central openings to which the raw feed is fed centrally at the top end. There are also tangential orifices at either end. The medium enters tangentially at the lower end and migrates toward the top end, leaving the cylinder tangentially along with the heavy mineral. The lights gravitate to the bottom end and leave through a central discharge port. Generally, centrifugal devices such as the HMC and DWP experience higher media losses than static-type devices. A schematic diagram of a Dyna Whirlpool separator is shown in Figure 14.4-12.

PERFORMANCE EVALUATION OF GRAVITY SEPARATORS

Gravity separators aim to separate minerals at a desired separation density. However, because of the presence of middling particles, which have densities between those of heavy and light minerals, a sharp separation is not possible. Also, since these devices rely on differences between settling rates of the two minerals, fine feed material inherently gives rise to inefficiencies. The efficiency of a gravity concentrator is determined by two factors: material characteristics and machine characteristics. The degree of liberation of the feed material that affects the separation efficiency is represented by separability curves, whereas machine characteristics are evaluated in terms of performance curves (Kelly and Spottiswood 1982). The performance curve of an operating machine is represented by a plot of the fractional mass of particles reporting to the concentrate stream against the extent of the property being exploited for the separation. For gravity concentration, traditionally density (or SG) of the particles has been taken as the property being exploited. Such a plot is commonly known as the performance curve, partition curve, or the Tromp curve, which may be construed as a probability plot that indicates the probability of a particle of given SG reporting to the concentrate stream. A typical performance curve is shown in Figure 14.4-13.



Source: Aplan 1985.

Figure 14.4-12 Dyna Whirlpool separator

A flatter curve represents a poor separation, whereas a steeper slope represents a sharp separation. The specific gravity of particles that have equal chances of reporting to either exit is known as the cut specific gravity (SG_{50}) or separation density. For static heavy medium separators the separation density is close to the medium density whereas for centrifugal separators, it is slightly higher.

To compare different operations with different separation SGs, several efficiency measures have been proposed. Measures of efficiency based on performance curves generally used for gravity concentrators are probable error (Ecart probable, E_p), imperfection (I), and the sharpness index (SI). These are defined as follows:

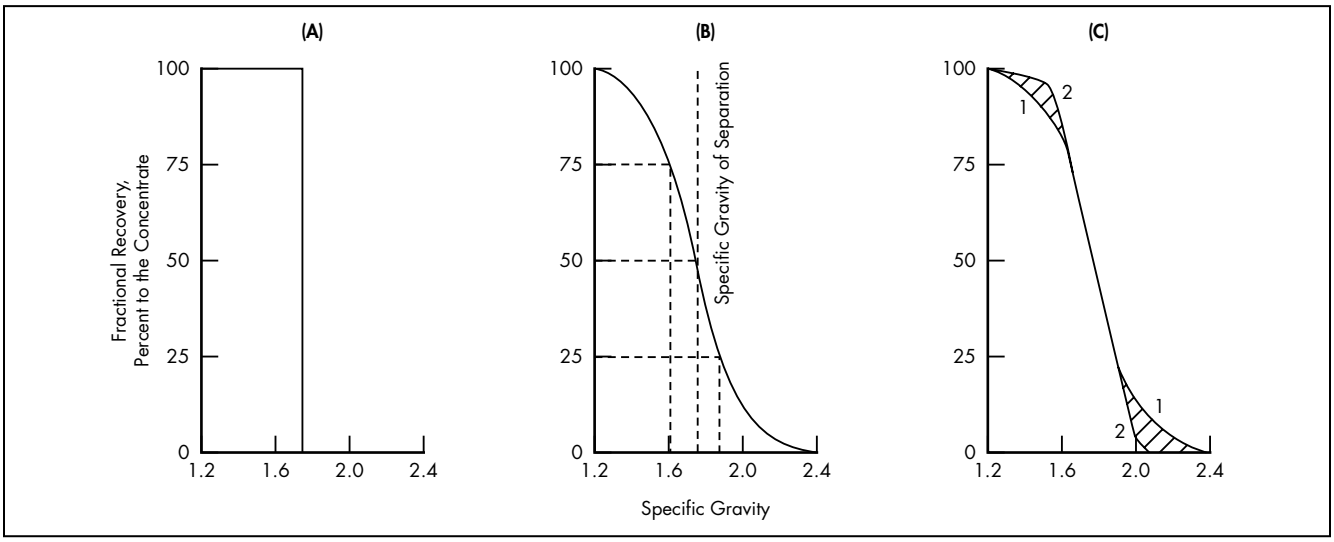
$$E_p = \tfrac{1}{2}\,(SG_{75} - SG_{25})$$

$$I = E_p/(SG_{50} - 1)$$

$$\text{and } SI = \frac{SG_{75}}{SG_{25}} \qquad (14.4\text{-}2)$$

where SG_{25}, SG_{50}, and SG_{75} are the specific gravities at which 25%, 50%, and 75% of the material would report to the concentrate stream, respectively. The percentage of misplaced material measured as an area between curves has also been used as a measure of efficiency.

E_p represents the inverse of the slope in the middle part of the curve but does not take into account the shape of the two ends of the performance curve that represent most of the misplaced material. Generally, the cut specific gravity of



Source: Aplan 2003.

Figure 14.4-13 Performance curves of a gravity separator. Partition curves: (A) perfect separation, (B) actual separation (curve 1), and (C) same E_p as for graph B but with superior recovery of misplaced particles (shaded area between curves 1 and 2).

Table 14.4-3 Approximate E_p values for coal cleaning devices at 1.5 specific gravity

Coal Size and Appropriate Cleaning Device	*Ep* at Stated Size				Broad Size Range of Feed	
	+½ in. (+12.7 mm)	½ × ¼ in. (12.7 × 6.4 mm)	14 × 28 M* (1.7 × 0.6 mm)	20 × 200 M (830 × 74 mm)	Range	*Ep*
Course coal						
Baum jig	0.06	0.16	0.30	—	6 in. × 48 M	0.12
HMS, static bath	0.03	—	—	—	6 in. × ¼ in.	0.03
Intermediate to fine coal						
HMS, cyclone	0.02	0.03	0.05	—	¾ in. × 28 M	0.03
Shaking tables	—	0.07†	0.10	0.20	⅜ in. × 200 M	0.09
Water-only cyclone	—	0.15†	0.20	—	¼ in. × 200 M	0.28

Source: Aplan 2003.
*All mesh sizes (inches) are in Tyler Standard mesh.
†¼ in. × 14 M.

Table 14.4-4 Influence of near-gravity material on the difficulty of separation, process selection, and suggested equipment

Wt % ± 0.1 SG	Degree of Difficulty	Suggested Gravity Process	Suggested Devices
0–7	Simple	Almost any	Jigs, HMS, tables, spirals, sluices
7–14	Moderately difficult	Efficient process	
10–15	Difficult	Efficient process, good operation	
15–20	Very difficult	Very efficient process, expert operation	Heavy media separation
20–25	Exceedingly difficult		
>25	Formidable	Exceptionally efficient process, expert operation	Heavy media separation, close control

Source: Aplan 2003.

separations involving fine particles is higher. Thus, imperfection that takes into account this variation may be used to compare various equipment that process different-sized feeds. Table 14.4-3 shows typical E_p values for coal cleaning devices.

SELECTION OF GRAVITY SEPARATION EQUIPMENT

The selection of gravity separation equipment is based on the response of the ore to sink-and-float tests using heavy liquids. The commonly used heavy liquids are methylene iodide, tetrabromoethane, methylene bromide, and lithium sodium—tungstates whose densities are 3.3, 2.96, 2.48, and 2.85, respectively. For higher-density separations, Clerici solution is used. Generally an organic solvent may be used to dilute the reagents to a required specific gravity. Caution should be exercised in handling these reagents as most of them are toxic.

The data from these tests are presented as washability curves that essentially indicate the grade and recovery of the concentrate that can be achieved at various separation densities. The selection of gravity equipment is based on the amount of material present in the ore that is near the separation density. The amount of ore within ±0.1 SG units of the separation density is known as the tolerance value, which is a reflection of the degree of difficulty of separation. Table 14.4-4 shows the recommended equipment at various tolerance values.

The selection also depends on the feed size of the ore as each equipment type can handle material only within a specified size range.

For the selection of centrifugal separators for fine gold recovery, a gravity recoverable gold test has been proposed (Laplante et al. 2000). This test is a material characterization test that indicates the amenability of the ore to centrifugal gravity concentration. For the purpose of determining optimal operating conditions of a Knelson concentrator, a model based on a mechanistic approach has been proposed (Subasinghe 2007).

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CHAPTER 14.5

Fundamental Principles of Froth Flotation

S. Komar Kawatra

INTRODUCTION

Froth flotation is a highly versatile method for physically separating particles based on differences in the water-repellency characteristics of the surfaces of various rock and mineral species contained in aqueous slurry. The air bubbles generated within a separation vessel collect hydrophobic particles and ascend to the surface with the particles attached to be removed, while the particles that remain completely wetted stay in the liquid phase. Froth flotation can be adapted to a broad range of mineral separations, as it is possible to use chemical treatments to selectively alter mineral surface characteristics so that they have the necessary hydrophobicity for the separation to occur. Many diverse applications, both mineral and nonmineral, are currently in use. Examples of mineral applications include separating sulfide minerals from silica gangue (and selectively from other sulfide minerals), separating potassium chloride (sylvite) from sodium chloride (halite), separating coal from ash-forming minerals, removing silicate minerals from iron ores, and separating phosphate minerals from silicates. An example of a nonmineral application is the de-inking of recycled newsprint. Flotation is particularly useful for processing fine-grained ores that are not amenable to conventional gravity concentration because of their low density differentials that render gravity separation unacceptably ineffective.

The flotation system includes many interrelated components, and changes in one area will produce compensating effects in other areas, as shown in Figure 14.5-1 (Klimpel 1995). Therefore, it is important to take all of these factors into account when developing specific froth flotation applications. Changes in the settings of one factor (such as feed rate) will automatically cause changes in other parts of the system (such as flotation rate, particle size recovery, airflow rate, and pulp density). As a result, it is difficult to study the effects of any single factor in isolation, and compensation effects within the system can sometimes result in an unexpected and undesirable outcome (Klimpel 1995). This makes it difficult to develop predictive models for froth flotation; although work is being done to develop models that utilize readily measurable parameters such as solids recovery and tailings solid content (Rao et al. 1995). Such models incorporate fundamental physical and chemical reactions as well as empirical data that relate to the mineral components, and they are usually validated by various levels of test work (i.e., bench-scale, pilot-plant, and commercial plant trials).

PERFORMANCE CALCULATIONS

A universal method for expressing the effectiveness of a separation does not exist, but there are several useful methods for examining froth flotation processes.

Parameters

The following parameters are used to measure performance.

(a) *Ratio of concentration:* The ratio of concentration is the weight of the feed relative to the weight of the concentrate, which is expressed as follows:

$$\text{ratio of concentration} = F/C$$

where

F = total weight of the feed
C = total weight of the concentrate

One limitation with this calculation is that it uses the weights of the feed and the concentrate. Although these data can be obtained from laboratory experiments, in the plant it is likely that the ore is not weighed and only assays are available. However, it is possible to express the ratio of concentration in terms of ore assays. This can be done starting with the definition of the ratio of concentration (F/C) and using the following mass balance equations:

$$F = C + T$$

$$Ff = Cc + Tt$$

where

F, C, and T = percent weights of the feed, concentrate, and tailings, respectively
f, c, and t = assays of the feed, concentrate, and tailings, respectively

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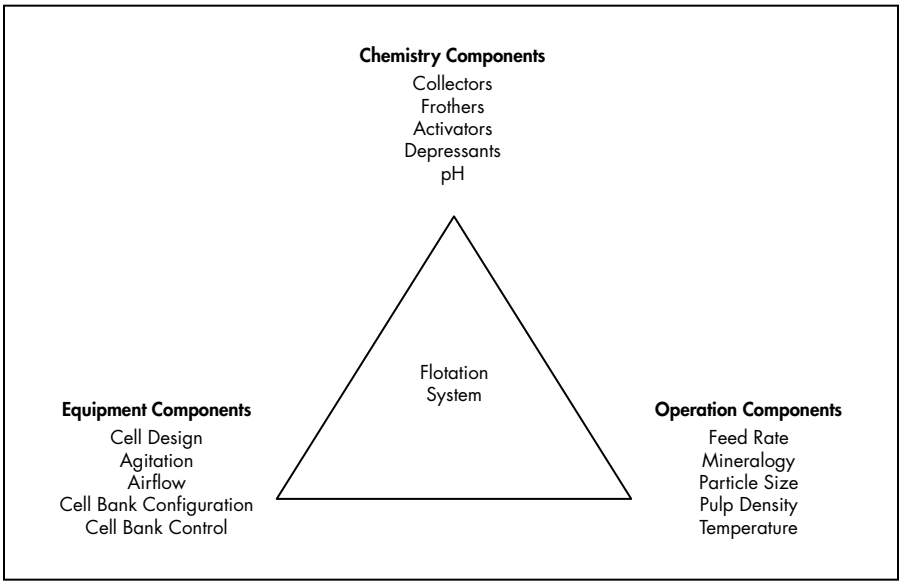


Figure 14.5-1 Interrelated components of a flotation system

So that F/C can be determined, one needs to eliminate T from these equations. Multiplying $F = C + T$ by t gives $Ft = Ct + Tt$, and subtracting this equation from $Ff = Cc + Tt$ eliminates T and gives $F(f - t) = C(c - t)$. Rearranging then gives

$$F/C = (c - t)/(f - t)$$

(b) *Percent metal recovery:* The percent metal recovery is the percentage of the metal in the original feed that is recovered in the concentrate. This can be calculated using weights and assays, for example, $(Cc)/(Ff) \times 100$, or because $C/F = (f - t)/(c - t)$, the percent metal recovery can be calculated from the assays alone using $100(c/f)(f - t)/(c - t)$.

(c) *Percent metal loss:* The percent metal loss is the opposite of the percent metal recovery and represents the material lost to the tailings. It can be calculated simply by subtracting the percent metal recovery from 100%.

(d) *Percent weight recovery:* The percent weight recovery is essentially the inverse of the ratio of concentration and equals $100C/F = 100(f - t)/(c - t)$.

(e) *Enrichment ratio:* The enrichment ratio is calculated directly from the assays as c/f. The weights are not involved in the calculation.

Example Calculations

A copper ore initially contains 2.09% Cu. After carrying out a froth flotation separation, the products are as shown in Figure 14.5-2. Using these data, parameters (a) through (e) are calculated as follows.

(a) *Ratio of concentration:* From Figure 14.5-2, the ratio of concentration can be calculated as $F/C = 100/10 = 10$. If only assays are available, the ratio of concentration equals $(20 - 0.1)/(2.09 - 0.1) = 10$. Therefore, for each 10 t (metric tons) of feed, the plant would produce 1 t of concentrate.

(b) *Percent metal recovery:* Using the example data from Figure 14.5-2, the percent Cu recovery calculated from the weights and assays is as follows:

$$\text{percent Cu recovery} = [(10 \times 20)/(2.09 \times 100)] \times 100 = 95.7\%$$

The calculation using assays alone is as follows:

$$\text{percent Cu recovery} = 100(20/2.09)(2.09 - 0.1)/(20 - 0.1) = 95.7\%$$

This means that 95.7% of the copper present in the ore was recovered in the concentrate, while the rest was lost in the tailings.

(c) *Percent metal loss:* The percent Cu loss can be calculated by subtracting the percent Cu recovery from 100% as follows:

$$\text{percent Cu loss} = 100 - 95.7 = 4.3\%$$

This means that 4.3% of the copper present in the ore was lost in the tailings.

(d) *Percent weight recovery or percent yield:* The percent weight recovery is equal to the percent weight of the concentrate in Figure 14.5-2. It can also be calculated as follows from the assay values given in the figure:

$$\text{percent weight recovery} = 100 \times (2.09 - 0.1)(20 - 0.1) = 10\%$$

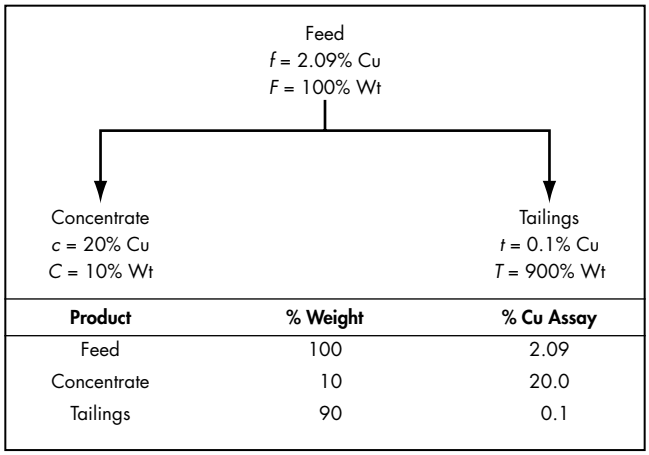
(e) *Enrichment ratio:* The enrichment ratio is calculated by dividing the concentrate assay in Figure 14.5-2 by the feed assay as follows:

$$\text{enrichment ratio} = 20.0/2.09 = 9.57$$

This reveals that the concentrate has 9.57 times the copper concentration of the feed.

Grade–Recovery Curves

Although these calculated values are useful for comparing flotation performance for different conditions, it is most useful to consider both the grade and the recovery simultaneously using a "grade–recovery curve." This is a graph of the recovery of the valuable metal versus the product grade at that recovery, and it is particularly useful for comparing separations where both the grade and the recovery are varying. A set of grade–recovery curves is shown in Figure 14.5-3. If 100% of the



Product	% Weight	% Cu Assay
Feed	100	2.09
Concentrate	10	20.0
Tailings	90	0.1

Figure 14.5-2 Grade–recovery performance of a hypothetical copper ore flotation process

feed is recovered to the product, then the product will obviously have the same composition as the feed, and so the curve starts at the feed composition with 100% recovery. Similarly, if the purest mineral grain that contains the metal of interest is removed, this will be the maximum grade that can be produced by a physical separation, and so the zero percent recovery end of the curve terminates at an assay that is less than or equal to the assay of the purest grains available in the ore. In the curves shown in Figure 14.5-3, the points that are higher and to the right show better performance than the points that are lower and to the left.

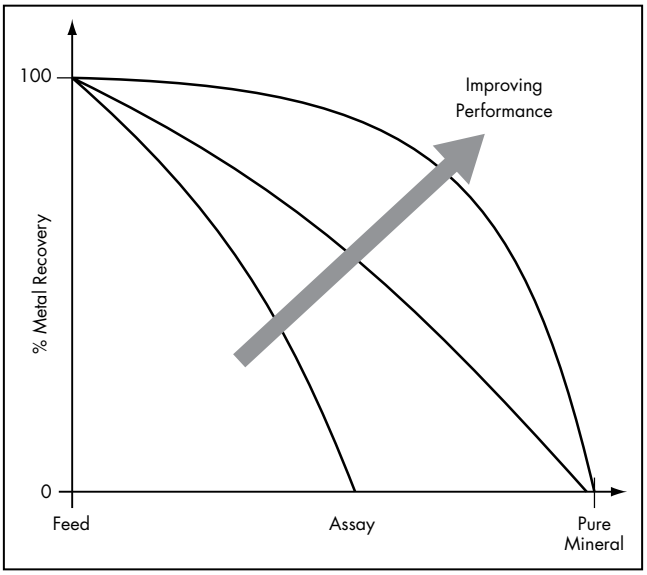


Figure 14.5-3 Typical forms of the grade–recovery curves for froth flotation

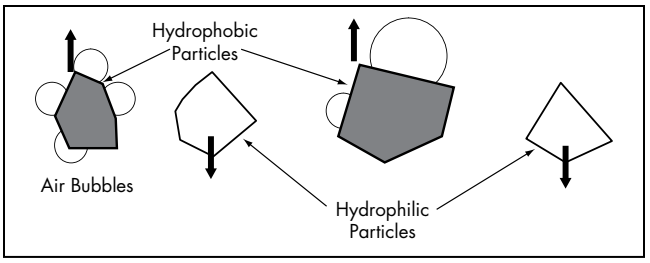


Figure 14.5-4 Selective attachment of air bubbles to hydrophobic particles

HYDROPHOBICITY/HYDROPHILICITY

The basis of froth flotation is the difference in wettabilities of different minerals. Particles range from those that are readily wettable by water (hydrophilic) to those that are water-repellent (hydrophobic). If a mixture of hydrophobic and hydrophilic particles are suspended in water and air is bubbled through the suspension, then the hydrophobic particles will tend to attach to the air bubbles and float to the surface, as shown in Figure 14.5-4. The froth layer that forms on the surface will then be heavily loaded with the hydrophobic mineral, and it can be removed as a separated product. The hydrophilic particles will have limited or no tendency to attach to air bubbles, and so they will remain in suspension and be transported away in the slurry (Whelan and Brown 1956).

Particles can be naturally hydrophobic, or hydrophobicity can be induced by chemical treatments. Naturally hydrophobic materials include hydrocarbons and nonpolar solids such as elemental sulfur. Coal is a good example of a material that is normally naturally hydrophobic, because it is mostly composed of hydrocarbons. Chemical treatments to render a surface hydrophobic are essentially methods for selectively coating a particle surface with a monolayer of nonpolar oil.

The attachment of the bubbles to the surface is determined by the interfacial energies between the solid, liquid, and gas phases. This is determined by Young–Dupré equation as follows:

$$\gamma_{lv}\cos\theta = (\gamma_{sv} - \gamma_{sl})$$

where

γ_{lv} = surface energy of the liquid–vapor interface
γ_{sv} = surface energy of the solid–vapor interface
γ_{sl} = surface energy of the solid–liquid interface
θ = "contact angle"

The contact angle (θ) is the angle formed at the junction between vapor, solid, and liquid phases, as shown in Figure 14.5-5. If the contact angle is very small, then the bubble does not attach to the surface, whereas a very large contact angle results in very strong bubble attachment. A contact angle near 90° is sufficient for effective froth flotation in most cases.

PARTICLE–BUBBLE CONTACT

When particles are rendered hydrophobic, contact with gas bubbles must occur so that attachment to the surface can take place. Contact between particles and bubbles can be accomplished in a flotation cell such as the agitated cell shown in Figure 14.5-6.

In the agitated cell, the rotor draws slurry through the stator and expels it to the sides, creating a suction that draws air down the shaft of the stator. The air is then dispersed as bubbles through the slurry and comes in contact with particles in the slurry that is circulated through the stator.

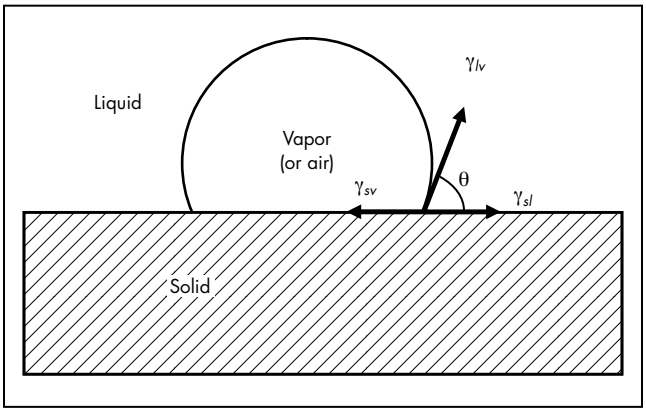


Figure 14.5-5 Contact angle between air bubble and solid surface immersed in liquid

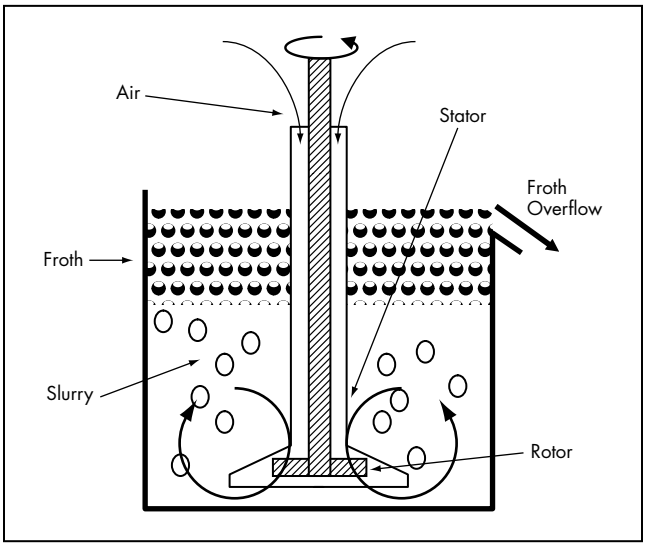


Figure 14.5-6 Simplified flotation cell

Particle–bubble collision is affected by the relative sizes of the particles. If the bubbles are large relative to the particles, then fluid flowing around the bubbles can sweep the particles past without them coming in contact. Therefore, it is best if the bubble diameter is comparable to the particle diameter to ensure good particle–bubble contact.

COLLECTION IN THE FROTH LAYER

When a particle and bubble have come in contact, the bubble must be large enough for its buoyancy to lift the particle to the surface. This is obviously easier if there are low-density particles (as is the case for coal) than if there are high-density particles (such as lead sulfide). The particle and bubble must remain attached while they move up into the froth layer at the top of the cell. The froth layer must persist long enough to either flow over the discharge lip of the cell by gravity or be removed by mechanical froth scrapers. If the froth is insufficiently stable, the bubbles will break and drop the hydrophobic particles back into the slurry prematurely. However, the froth should not be so stable as to become persistent foam, as foam is difficult to convey and pump through the plant.

The surface area of the bubbles in the froth is also important. Because particles are carried into the froth by attachment to bubble surfaces, increasing amounts of bubble surface area allows a more rapid flotation rate of particles. At the same time, increased surface area also carries more water into the froth as the film between the bubbles. Because fine particles that are not attached to air bubbles will be unselectively carried into the froth along with the water (entrainment), excessive amounts of water in the froth can result in significant contamination of the product with gangue minerals.

REAGENTS

The properties of raw mineral mixtures suspended in normal plant circulating water are rarely suitable for froth flotation. Chemicals are needed both to control the relative hydrophobicities of the particles and to maintain the proper froth characteristics. Therefore, many different reagents are involved in the froth flotation process, with the selection of reagents depending on the specific mineral mixtures being treated (Klimpel 1980; Thompson 2002).

Collectors

Collectors are used to selectively adsorb onto the surfaces of particles. They form a monolayer on the particle surface that essentially makes a thin film of nonpolar hydrophobic hydrocarbons. The collectors greatly increase the contact angle so that bubbles will adhere to the surface. Selection of the correct collector is critical for an effective separation by froth flotation. Collectors can be generally classed depending on their ionic charge; they can be nonionic, anionic, or cationic, as shown in Figure 14.5-7 (Glembotskii et al. 1963). Nonionic collectors are simple hydrocarbon oils, whereas the anionic and cationic collectors consist of a polar part that selectively attaches to the mineral surfaces and a nonpolar part that projects out into the solution, rendering the surface hydrophobic. Collectors can either chemically bond to the mineral surface (chemisorption) or they can be held on the surface by physical forces (physical adsorption).

Chemisorption

In chemisorption, ions or molecules from solution undergo a chemical reaction with the surface, becoming irreversibly bonded. This permanently changes the nature of the surface. Chemisorption of collectors is highly selective, as the chemical bonds are specific to particular atoms.

Physisorption

In physisorption, ions or molecules from solution become reversibly associated with the surface, attaching due to electrostatic attraction or van der Waals bonding. The physisorbed substances can be desorbed from the surface if conditions such as the pH or the composition of the solution changes. Physisorption is much less selective than chemisorption, as collectors will adsorb on any surface that has the correct electrical charge or degree of natural hydrophobicity.

Nonionic Collectors

Hydrocarbon oils, and similar compounds, have an affinity for surfaces that are already partially hydrophobic. They selectively adsorb on these surfaces and increase their hydrophobicity. The most commonly floated naturally hydrophobic material is coal. The addition of collectors such as no. 2 fuel oil and kerosene significantly enhances the hydrophobicity of the coal particles without affecting the surfaces of the associated ash-forming minerals. This improves the recovery of

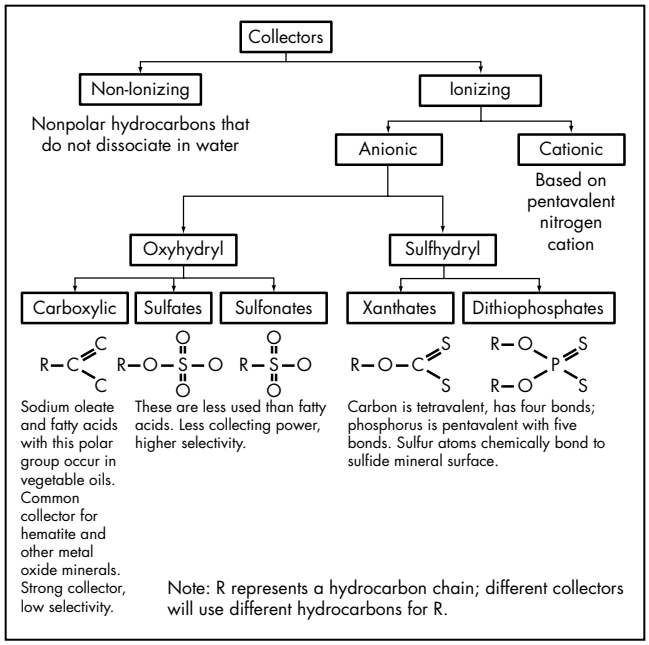


Figure 14.5-7 Basic collector types

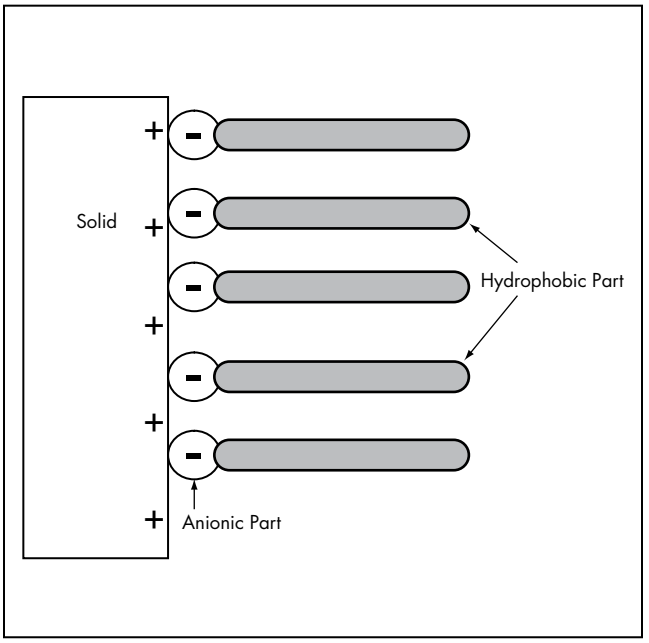


Figure 14.5-8 Adsorption of anionic collector onto a solid surface

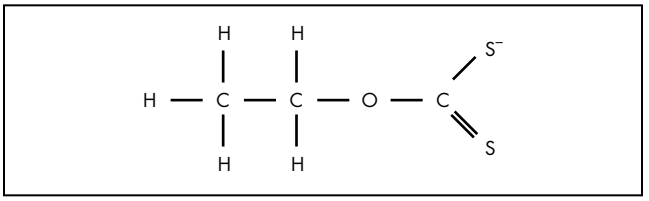


Figure 14.5-9 Structure of a typical xanthate collector (ethyl xanthate)

the coal and increases the selectivity between coal particles and mineral matter. Fuel oil and kerosene have the following advantages over specialized collectors for froth flotation:

- They have a low enough viscosity to disperse in the slurry and spread over the coal particles easily.
- They are very low cost compared to other compounds that can be used as coal collectors.

In addition to coal, it is also possible to float naturally hydrophobic minerals such as molybdenite, elemental sulfur, and talc with nonionic collectors. Nonionic collectors can also be used as "extenders" for other collectors. If another, more expensive collector makes a surface partially hydrophobic, adding a nonpolar oil will often increase the hydrophobicity further at low cost.

Anionic Collectors

Anionic collectors are usually weak acids or acid salts that ionize in water, producing a collector that has a negatively charged end that will attach to the mineral surfaces and a hydrocarbon chain that extends out into the liquid, as shown in Figure 14.5-8. The anionic portion is responsible for the attachment of the collector molecule to the surface, while the hydrophobic part alters the surface hydrophobicity.

Anionic collectors for sulfide minerals. The most common collectors for sulfide minerals are the sulfhydryl collectors, such as the various xanthates and dithiophosphates.

Xanthates are most widely studied and commonly used, and they have structures similar to what is shown in Figure 14.5-9. Xanthates are highly selective collectors for sulfide minerals, as they chemically react with the sulfide surfaces and do not have any affinity for the common nonsulfide gangue minerals (Crozier 1992). The $OCSS^-$ group attaches irreversibly to the sulfide mineral surface. Using xanthates with longer hydrocarbon chains tends to increase the degree of hydrophobicity when they adsorb on the surface.

There are a wide variety of other sulfhydryl collectors (Thompson 2002; Bulatovic 2007), which have somewhat different adsorption behaviors so they can be used for some separations that are difficult using xanthates. These include the following:

- *Dithiophosphates* have greater selectivity against pyrite than do xanthates, and some of these can collect liberated grains of gold. They are also useful in lead sulfide, zinc sulfide, and silver sulfide flotation.
- *Thionocarbamates* selectively float copper sulfides from pyrite at alkaline pH, and they work well for copper sulfides that tend to grind into the slime fraction, such as chalcocite.
- *Dithiophosphinates* are even more selective against pyrite than dithiophosphates, and they are particularly useful for complex polymetallic ores. Although they are selective, they are not particularly strong collectors (produce low levels of hydrophobicity), and so they are often used in combination with other collectors.
- *Trithiocarbonate* has a very high selectivity against pyrite in copper ore flotation.
- *Mercaptobenzothiazole* has improved the recovery of tarnished or semi-oxidized copper and zinc sulfides. It is often used in combination with xanthates.

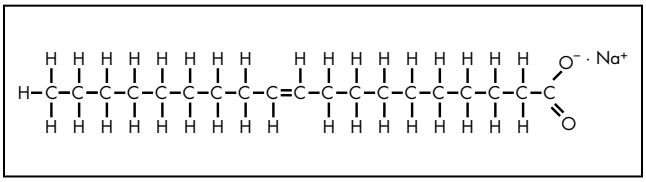


Figure 14.5-10 Structure of oleic acid, a very commonly used anionic collector

- *Mercaptans* can be used in copper flotation, but they are not very popular because of their strong, unpleasant odor.
- *Xanthogen formates* float both copper sulfides and metallic copper, and they have been used to recover both cement copper and precipitated copper sulfide from acid pulps.
- *Monothiophosphates* have strong collecting power for sulfide minerals at pH values as low as 2.0, and they can be used in combination with dithiophosphates at more alkaline pH.

Anionic collectors for oxide minerals. The collectors available for the flotation of oxide minerals (oxyhydryl collectors) are not as selective as the collectors used for sulfide mineral flotation because they attach to the surface by electrostatic attraction rather than by chemically bonding to the surface. As a result, there is some collector adsorption onto the minerals that are not intended to float.

The most widely used oxyhydryl collectors are the carboxylates, with a typical example being the fatty acid sodium oleate, the sodium salt of oleic acid, which has the structure shown in Figure 14.5-10. Numerous other fatty acids can also be used as collectors; these differ from oleic acid mainly in their chain length and their degree of saturation. The anionic group responsible for attaching all of these reagents to the mineral surface is the carboxyl group, which dissociates in water to develop a negative charge. The negatively charged group is then attracted to positively charged mineral surfaces.

Because particles that are immersed in water develop a net charge due to exchanging ions with the liquid, it is often possible to manipulate the chemistry of the solution so that one mineral has a strong positive charge while other minerals have a charge that is either only weakly positive or is negative. In these conditions, the anionic collector will preferentially adsorb on the surface with the strongest positive charge and render them hydrophobic.

Other oxyhydryl collectors include the following (Bulatovic 2007):

- **Alkyl sulfates:** Alkyl sulfates are derived from sulfuric acid and are essentially a sulfate ion with a hydrocarbon group attached to one of the oxygens ($R\text{-}O\text{-}SO_3^-$). They are not much studied as flotation reagents, and because of this their industrial application is limited. Alkyl sulfates are used for flotation of sulfate minerals such as barite ($BaSO_4$), celestite ($SrSO_4$), gypsum ($CaSO_4 \cdot 2H_2O$), and anhydrite ($CaSO_4$).
- **Sulfonates:** These collectors are similar to alkyl sulfates, except that the hydrocarbon group replaces one of the oxygens from the sulfate ion and attaches directly to the sulfur ($R\text{-}SO_3^-$). They are generally manufactured in a poorly controlled way by treating petroleum fractions with sulfuric acid and separating the sulfonates from the acid sludge produced. They are poorly chemically characterized, and they are not much studied as flotation reagents, although they have been available from Cytec Industries.
- **Hydroxamates:** Belonging to the group of "chelating collectors," hydroxamates have structures such as those shown in Figure 14.5-11. These are intended to attach to ions on mineral surfaces at multiple points, with the objective of being more selective than other oxyhydryl collectors. They have been investigated as collectors for malachite, bastnaesite, titanates, and pyrochlore.
- **Sulfosuccinamates:** Derived from carboxylic and succinic acids, the sulfosuccinamates can be considered to be a hybrid of the fatty acid collectors and the sulfonate collectors, with active groups such as shown in Figure 14.5-12. The presence of multiple carboxyl groups and additional sulfonate groups gives them greater collecting power, at some cost to the selectivity. They are used for flotation of cassiterite and monazite.
- **Phosphonic acid:** A typical structure of styrene phosphonic acid is given in Figure 14.5-13, which serves as a basis for a range of collectors that have recently been developed for the flotation of cassiterite, ilmenite, rutile, and pyrochlore.
- **Phosphoric acid esters:** This family of collectors consists of a phosphate core with one or more attached hydrocarbon groups, as shown in Figure 14.5-14. Collectors of this group have been developed for the flotation of apatite, scheelite, ilmenite, rutile, perovskite, zircon, cassiterite, and pyrochlore.

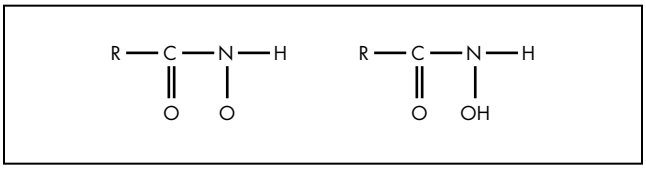


Figure 14.5-11 Typical hydroxamate structures

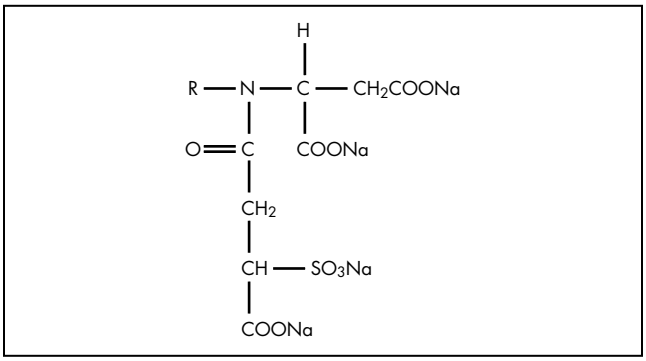


Figure 14.5-12 Active group of sulfosuccinamate

Cationic Collectors

Cationic collectors use a positively charged amine group (Figure 14.5-15) to attach to mineral surfaces. Because the amine group has a positive charge, it can attach to negatively charged mineral surfaces. Cationic collectors therefore have essentially the opposite effect from anionic collectors, which

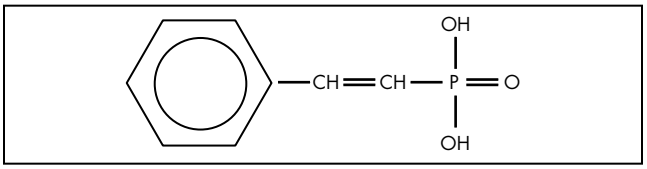


Figure 14.5-13 Styrene phosphonic acid structure

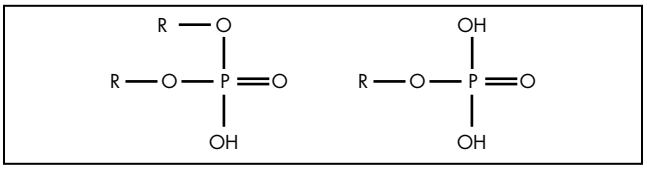


Figure 14.5-14 Structure of dialkylphosphoric acid and alkylphosphoric acid

attach to positively charged surfaces. Cationic collectors are mainly used for the flotation of silicates and certain rare-metal oxides and for the separation of potassium chloride (sylvite) from sodium chloride (halite).

Frothers

Frothers are compounds that act to stabilize air bubbles so that they will remain well dispersed in the slurry and will form a stable froth layer that can be removed before the bubbles burst. The most commonly used frothers are alcohols, particularly methyl isobutyl carbinol (MIBC), or 4-methyl-2-pentanol (a branched-chain aliphatic alcohol), or any of a number of water-soluble polymers based on propylene oxide, such as polypropylene glycols. The polypropylene glycols in particular are very versatile, and they can be tailored to give a wide range of froth properties. Many other frothers are available, such as cresols and pine oils, but most of these are considered obsolete and are not as widely used as they once were. Some work has also been done using saltwater (particularly seawater) as the frothing agent, and the process has been used industrially in Russia (Klassen and Mokrousov 1963; Tyurnikova and Naumov 1981).

Function of Frothers

Klimpel (1995) found that the use of different frothers produced changes in the flotation rate and recovery values in coal flotation and reached the following conclusions:

- When frother dosage was held constant while collector dosage was increased, it was found that the flotation rate increased to a maximum and then decreased. This was observed for all frother types and all particle size fractions. The difference between the frother families studied was that the collector dosage that produced the most rapid flotation rate was different.
- The finest (–88 μm) and coarsest (+500 μm) particles tended to float more slowly than the intermediate-size particles for all of the frother types.
- Changes in flotation rates were due to changes in the coal particle size and to changes in the frother/collector dosage. Although the contribution of particle size was generally more significant, the reagent dosage effect provides a useful means for adjusting the flotation rate in the plant.
- With aliphatic alcohol frothers, the flotation rate maximum was much more pronounced than for the propylene oxide (PO) and combined propylene oxide/alcohol (PO-alcohol adduct) frothers.
- Increasing the frother dosage to increase recovery always leads to less selective flotation, regardless of frother type.
- The PO and PO-alcohol adduct frothers are more powerful recovery agents than alcohol frothers, and they should therefore be used at lower dosages.
- Overdosing with alcohol frothers leads to a slower flotation rate, because excesses of these frothers will lead to eventual destabilization of the froth. This effect does not occur with the PO and PO-alcohol frothers, and therefore overdosing with these frothers leads to high recovery but poor selectivity.
- PO frothers with molecular weights of 300 to 500 are optimal for coal recovery.
- Alcohol frothers tend to be more effective for fine-particle recovery than for coarse-particle recovery. To recover coarse particles, the alcohol frother and the hydrocarbon collector dosages should both be high. The alcohol will still provide reasonable selectivity at these high dosages.
- The high-molecular-weight PO-based frothers are more effective for coarse-particle flotation than the alcohol or low-molecular-weight PO frothers, but they also have a lower selectivity. For both good coarse-particle recovery and good selectivity, the PO frothers should be used at low dosages, with low collector dosage as well. The PO-alcohol adduct frothers are even more effective for coarse-particle recovery, and they need to be used at even lower dosages.
- The optimal frother for high recovery with good selectivity will often be a blend of members of the various frother classes examined. It is reported that such frother blending will give enough benefit to be worth the effort in approximately half of all coal flotation operations.
- Neither the alcohol nor PO-based frothers will change the shape of the grade–recovery curve. Changes in frother type and dosage simply move the flotation results along the curve. Similarly, changes in hydrocarbon collector dosage also mainly move the performance along the grade–recovery curve.
- For medium- and coarse-coal size fractions (–0.5 mm), the total gangue recovered is linearly related to the total coal recovered. It is only for the finest particles (–50 μm) that the gangue recovery increases nonlinearly with increasing coal recovery.
- When floating coals with a broad particle size range, the majority of the gangue reaching the froth is from the finer particle size fractions.
- As the rate of coal flotation increases, the rate of gangue flotation increases proportionately. This is typical of a froth-entrainment process acting on the gangue.

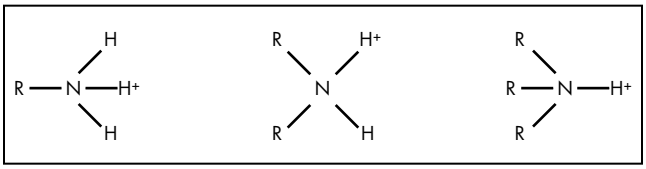


Figure 14.5-15 Primary, secondary, and tertiary amine groups

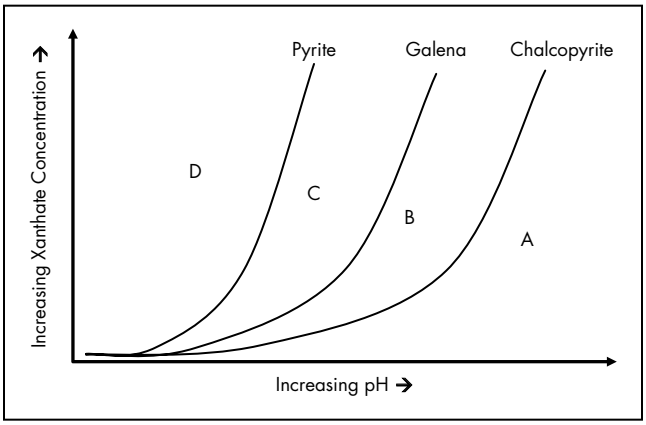


Figure 14.5-16 pH response curves for sulfhydryl collector adsorption on different sulfide minerals

Synthetic and Natural Frothers

The frothers originally used were natural products such as pine oil and cresylic acid. These are rich in surface-active agents that stabilize froth bubbles and are effective frothers. As natural products, they are not pure chemicals, but instead contain components that are effective frother and others that are weak collectors. Although this can have the advantage of reducing the amount of collector that needs to be added separately, it introduces some problems with process control. If the frother is also a collector, then it becomes impossible to alter the frothing characteristics and the collecting characteristics of the flotation operation independently.

Synthetic frothers, such as the alcohol-type and polypropylene glycol-type frothers, have the advantage that their effectiveness as collectors is negligible. It is therefore possible to increase the frother dosage without also changing the quantity of collector in the system. This in turn makes the flotation process much easier to control.

Modifiers

Modifiers are chemicals that influence the way that collectors attach to mineral surfaces. They may either increase the adsorption of collector onto a given mineral (activators), or prevent collector from adsorbing onto a mineral (depressants). Importantly, just because a reagent is a depressant for one mineral/collector combination, it does not necessarily follow that it is a depressant for other combinations. For example, sodium sulfide is a powerful depressant for sulfide minerals being floated with xanthate, but it does not affect flotation when sulfide minerals are floated with the collector hexadecyl trimethyl ammonium bromide.

pH Control

The simplest modifiers are pH-control chemicals. The surface chemistry of most minerals is affected by the pH. For example, minerals in general develop a positive surface charge under acidic conditions and a negative charge under alkaline conditions. Because each mineral changes from negatively charged to positively charged at some particular pH, it is possible to manipulate the attraction of collectors to their surfaces by pH adjustment. Other, more complex effects are due to pH that change the way that particular collectors adsorb on mineral surfaces.

Sulfhydryl collectors such as xanthate ions compete with hydroxide ions (OH^-) to adsorb on mineral surfaces, and so adsorption is a function of pH. This makes it possible for sulfhydryl collectors to be used to progressively separate specific minerals. The pH where the xanthate ion wins the competition with OH^- depends both on the concentration of xanthate in solution and on the specific sulfide mineral present, as shown in Figure 14.5-16.

The curves in Figure 14.5-16 mark the boundaries where the given mineral becomes sufficiently hydrophobic to float. Both xanthates and dithiophosphates exhibit curves of this form, with different pH values and concentrations for each type of collector (Fuerstenau et al. 1985). For example, assume a mixture of pyrite (FeS_2), galena (PbS), and chalcopyrite ($CuFeS_2$). From Figure 14.5-16, if the pH and xanthate concentrations are in Region A, then xanthate will not be adsorbed on any of the minerals, and therefore none of the minerals will be floated. However, if the pH and xanthate concentrations are altered to move into Region B, only chalcopyrite becomes hydrophobic and consequently floats. In Region C, both chalcopyrite and galena will float, and in Region D, all three minerals will float. It is therefore possible to progressively lower the pH to float chalcopyrite first, then galena, and then pyrite, producing concentrates for each mineral and leaving behind any nonfloatable silicate gangue minerals.

Acids. The acids used are generally those that give the greatest control of pH at the lowest cost, with sulfuric acid being the most popular. A key point to bear in mind is that the anion of the acid can potentially have effects of its own that are quite separate from the lowering of the pH. Therefore, acids other than sulfuric acid are useful in some cases.

Alkalis. As with acids, the most popular alkalis are those that are cheapest, with the lowest-cost alkali generally being lime (CaO or $Ca(OH)_2$). However, the calcium ion often interacts with mineral surfaces to change the flotation behavior. In some cases the calcium ions have beneficial effects, while in other cases they change the flotation in undesirable ways. It may therefore be necessary to use sodium-based alkalis, such as sodium hydroxide (NaOH) or sodium carbonate (Na_2CO_3), because the sodium cation generally does not have any significant effect on the particle surface chemistries.

Activators

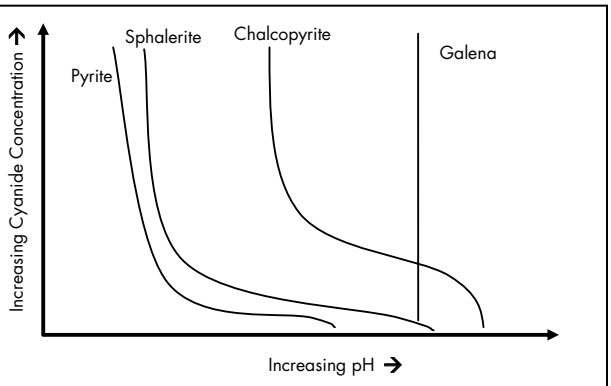
Activators are specific compounds that make it possible for collectors to adsorb onto surfaces that they could not normally attach to. A classic example of an activator is copper sulfate as an activator for sphalerite (ZnS) flotation with xanthate collectors (Fuerstenau et al. 1985). When untreated, xanthate cannot attach to the sphalerite surface because it forms a zinc-xanthate compound that quickly dissolves, as shown in the following reaction:

$$ZnS(s) + \text{xanthate} \rightarrow S(s) + \text{Zn-xanthate (aq)}$$

The surface of the sphalerite can be activated by reacting it with a metal ion that does not form a soluble xanthate, such as soluble copper from dissolved copper sulfate, as shown in the following reaction:

$$ZnS(s) + CuSO_4(aq) \rightarrow CuS(s) + ZnSO_4(aq)$$

This forms a thin film of copper sulfide on the sphalerite surface, which allows for stable attachment of the xanthate, rendering the sphalerite particle hydrophobic and floatable. Other metals such as silver and lead can also be used to activate zinc,



Note: The flotation of galena (PbS) is unaffected by the presence of cyanide, probably due to lead being less readily complexed by cyanide than are iron, zinc, or copper.

Figure 14.5-17 Effect of cyanide depressant on flotation of minerals as a function of pH

but the copper element is cheaper than silver and is less toxic than lead.

It is also possible to adsorb specific ions onto the surface that can promote attachment of the collector. For example, silica (SiO_2) normally has a strongly negative surface charge at approximately neutral pH, and therefore it has little affinity for anionic collectors such as oleic acid. However, calcium ions specifically adsorb onto silica surfaces, and the negative charge of the calcium ions can actually reverse the surface charge, making it positive. It is then possible for the anionic collectors to electrostatically attach to the calcium-activated silica surface.

Depressants

Depressants have the opposite effect of activators by preventing collectors from adsorbing onto particular mineral surfaces. Their typical use is to increase selectivity by preventing one mineral from floating while allowing another mineral to float unimpeded.

Cyanide. Cyanide (CN^-) is a particularly useful depressant in sulfide mineral flotation, as can be seen in Figure 14.5-17. Its activity is believed to be due to its ability to complex with, and in some cases dissolve, a number of metal ions, preventing them from attaching to the xanthate molecules. In particular, it is a strong depressant for pyrite (FeS_2) and can be used to "deactivate" sphalerite that has been activated by copper ions in solution (Fuerstenau et al. 1985).

Lime. Lime is added as either CaO or $Ca(OH)_2$, and when it dissolves it contributes calcium ions that can adsorb onto mineral surfaces. In combination with its strong alkaline nature, this makes it particularly useful in manipulating sulfide flotation. It is less useful in oxide mineral flotation because it can activate the flotation of silica by anionic collectors, causing it to float along with the other oxide minerals.

Organic depressants. A large number of organic compounds are useful as flotation depressants. These tend to be soluble polymers (such as starch polysaccharide) that selectively become adsorbed and coat mineral surfaces, thereby preventing a collector from attaching. An example of this is in the "reverse flotation" of silica from iron ore, where the silica tailings are floated using a cationic collector at a pH of 8.5 to 11, leaving behind the iron oxide minerals. Starch acts as a depressant for iron oxide in this process, preventing it from being floated by the cationic collector.

Recovery of Gangue Particles in Flotation: Entrainment or Hydrophobicity?

Some gangue material is always carried out of the slurry in the froth. For example, in the flotation of coal, a portion of the ash-forming minerals and pyrite will be carried into the froth along with the coal. It is commonly believed that this gangue mineral can often be prevented from floating if the correct depressant can be discovered. In the case of coal, more than 42 chemicals have been reported by investigators as being capable of depressing pyrite in coal flotation, and yet none of them have ever been useful industrially. This is because, in most cases, the pyrite was not truly hydrophobic (Kawatra and Eisele 1992, 2001).

In a froth flotation machine, there are two ways that a particle can reach the froth layer. It can be carried into the froth by attachment to an air bubble (true flotation), or it can be suspended in the water trapped between the bubbles (entrainment). Although true flotation is selective between hydrophobic and hydrophilic particles, entrainment is nonselective, and therefore entrained particles are just as likely to be gangue minerals as they are to be the valuable mineral. If particles are sufficiently coarse, then they settle rapidly enough that they are not carried into the froth by entrainment. As they become finer, particles settle more slowly and so have more time to become entrapped in the froth, and consequently they have less of a tendency to drain away. Clay particles in particular, which are only a few micrometers in size, are very easily entrained. For particles that are less than a few micrometers in size, their rate of recovery into the froth by entrainment is equal to the rate of recovery of water into the froth. For example, if 20% of the water entering a flotation cell is carried into the froth, then up to 20% of the fine particles entering the cell will probably be entrained. The entrainment of coarser particles will be less than 20% due to their greater ability to drain from the froth.

In addition to the entrained particles, gangue is carried into the froth by being physically locked to the floatable particles. In the case of coal, much of the pyrite consists either of submicrometer pyrite grains that are never liberated from the coal, or pyrite particles with surfaces consisting primarily of coal. These will therefore behave as if they were coal particles. The recovery of entrained particles can only be reduced by lowering the fraction of the water recovered into the froth.

The recovery of locked particles can only be changed significantly by either grinding to a finer size to improve liberation, or by rejecting the locked particles along with the least-floatable liberated particles, which sacrifices recovery of the valuable mineral. Depressant chemicals are not useful in either of these cases, as they are only intended for preventing the hydrophobic bubble attachment and true flotation of particles. Before deciding to use a depressant, it is therefore important to first determine whether the particles to be depressed are actually being recovered by true flotation in the first place or whether there are other causes.

EQUIPMENT

In addition to controlling the chemistry, flotation requires a machine for mixing and dispersing air throughout the mineral slurry while removing the froth product (Nelson et al.

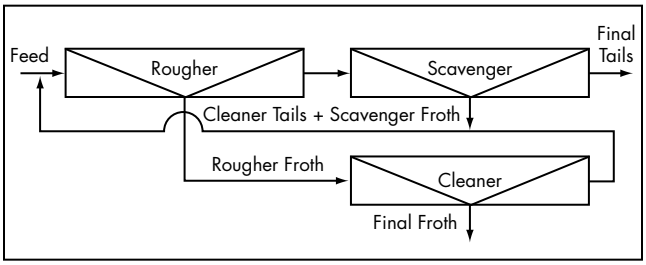


Figure 14.5-18 One configuration for a rougher/cleaner/scavenger flotation circuit

2009). These individual machines are then connected to form a flotation circuit to fully clean the product.

Conventional Cells

Conventional flotation cells consist of a tank with an agitator designed to disperse air into the slurry, as was previously shown schematically in Figure 14.5-6. They differ in the exact location and design of the rotor, the means of air induction, and the size/shape of the tank (Nelson et al. 2009). Considerable design effort has gone into optimizing the hydrodynamics of the system and ensuring that

- The suspended particles do not settle out into dead zones;
- The residence time of slurry throughout the cell is uniform;
- Solid particles are effectively brought in contact with air;
- Air is evenly dispersed throughout the cell;
- The froth layer is collected efficiently without excessive losses of attached particles back into the pulp;
- The pulp level can be accurately controlled;
- Maintenance of the machines is simplified; and
- Power consumption to achieve all of these goals is minimized.

The advances in understanding of flotation cell hydrodynamics has enabled the development of very large machines, with the recent "Supercell" design having volumes as large as 300 to 350 m^3 (FLSmidth Minerals 2009; Nelson et al. 2009).

Flotation cells are relatively simple machines with ample opportunity for particles to be carried into the froth along with the water making up the bubble films (entrainment) or for hydrophobic particles to break free from the froth and be removed along with the hydrophilic particles. Therefore, it is common for conventional flotation cells to be assembled in a multistage circuit, rather than using single flotation cells in isolation. Typically a "rougher" cell bank is used to make an initial, rapid separation of the valuable minerals from the gangue. The valuable mineral concentrate is often then reprocessed in the "cleaner" cells to remove gangue mineral particles that were misplaced into the concentrate. Similarly, "scavenger" cells are used to reprocess the gangue mineral product from the rougher to recover valuable minerals from the tailings. These different cell banks can be arranged in configurations such as the one shown in Figure 14.5-18, although, depending on the nature of the separation, there may be more or fewer stages of cleaners or scavengers added.

The use of a multistage circuit for froth flotation allows a great deal of flexibility, as the recirculation of material between cells can be adjusted to fit the specific needs of the material being processed. One concern is that a large number of flotation cells are typically necessary, which can use a great deal of the available plant floor space.

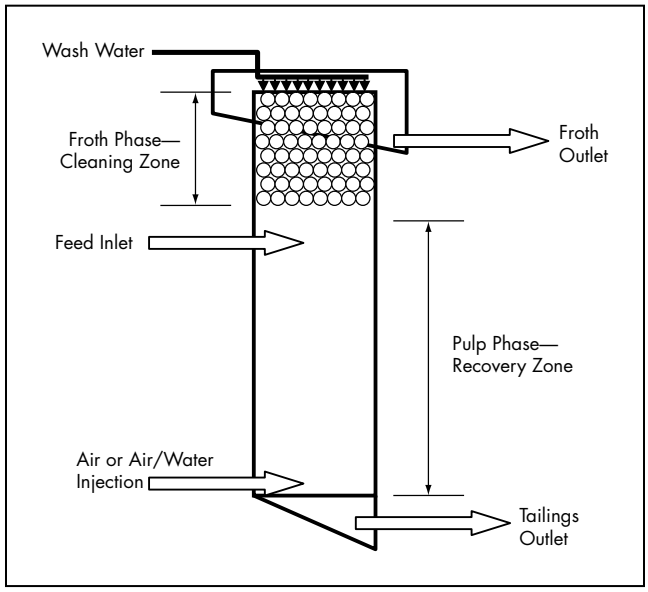


Figure 14.5-19 Flotation column

Flotation Columns

Flotation columns provide a means for improving the effectiveness of froth flotation using a single machine, which can be very useful if floor space is at a premium (Eberts 1986). A column essentially performs as if it were a multistage flotation circuit arranged vertically (Degner and Sabey 1988) with slurry flowing downward while the air bubbles travel upward, producing a countercurrent flow. The first flotation machine design to use a countercurrent flow of slurry and air was developed by Town and Flynn in 1919. It was not until the work of Boutin and Tremblay in the early 1960s that a new generation of countercurrent columns was developed that ultimately became industrially successful (Rubinstein 1995).

A typical flotation column schematic is shown in Figure 14.5-19. The basic principle of column flotation is the use of a countercurrent flow of air bubbles and solid particles. This is achieved by injecting air at the base of the column and feed near the midpoint. The particles then sink through a rising swarm of very small air bubbles.

Countercurrent flow is accentuated in most columns by the addition of wash water at the top of the column, which forces all of the water that entered with the feed downward to report to the tailings outlet. This flow pattern is in direct contrast to that found in conventional cells, where both the air and the solid particles are driven in the same direction. The result is that columns provide improved hydrodynamic conditions for flotation, and thus generally produce a cleaner product while maintaining high recovery and low power consumption. The performance differences between columns and conventional cells may best be described in terms of the following factors: collection zone size, particle–bubble contact efficiency, and fines entrainment (Kawatra and Eisele 1987).

The collection zone is the region where particle–bubble contact occurs, and it differs greatly in size between column and conventional flotation. In conventional cells, contact occurs primarily in the region surrounding the mechanical

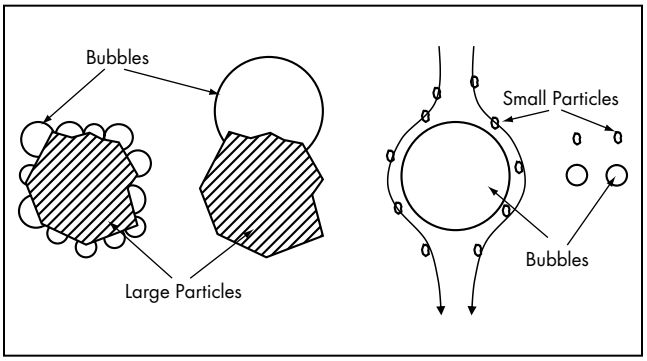


Figure 14.5-20 Effects of relative bubble and particle sizes on froth flotation

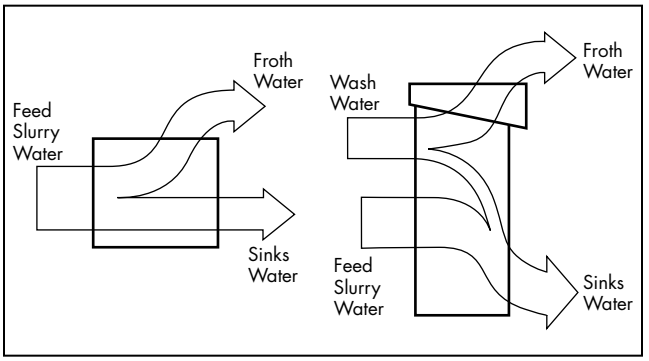


Figure 14.5-21 Effect of wash water on the entrainment of feed into the froth

impeller. The remainder of the cell acts mainly as a storage volume for material that has not yet been through the collection zone. This creates a bottleneck that keeps the flotation rate down. In contrast, flotation columns have a collection zone that fills the entire volume of the machine, so that there are more opportunities for particle–bubble collisions. The reduced level of turbulence needed to achieve a good rate of recovery in columns also reduces the tendency of coarse particles to be torn away from the bubbles that they attach to. Therefore, columns are more effective for floating coarser particles (Kawatra and Eisele 2001).

Columns exhibit higher particle–bubble contact efficiency than conventional machines due to the particles colliding head-on with the bubbles. As a result, the energy intensity needed to promote contact is less and, therefore, power consumption is reduced.

A second beneficial effect in certain types of flotation columns is the reduction of bubble diameter (Yoon and Luttrell 1986). As bubble diameter is reduced, the flotation rate of both the coarser and finer particles is improved. The effects producing the improvement are illustrated in Figure 14.5-20. Coarse particles can attach to more than one bubble if the bubbles are small and the chances of the particle being torn loose and sinking again are therefore reduced. For fine particles, the probability of collision with the bubble is improved if the bubble is small, as then the hydrodynamic forces tending to sweep the particle away from a collision are reduced. The reduction of bubble diameter has the added benefit of increasing the available bubble surface area for the same amount of injected air. Therefore, it is desirable to produce bubbles as fine as possible.

The entrainment of fine waste material in the froth product is a common failing of conventional flotation machines. It results from the phenomena whereby some water is transferred into the froth within the zone surrounding the air bubbles. As a result, fine suspended particles are swept into the froth with this water, even though they are not physically attached to the air bubbles. In most column flotation machines, the entrainment problem is addressed through the use of wash water, as shown in Figure 14.5-21. Where the conventional cell must allow a certain amount of feedwater to enter the froth, the wash water in the column cell displaces this feedwater to the tailings, thus preventing entrained contaminants from reaching the froth. The only drawback to the use of froth washing is that the demand for clean water is increased, which may cause problems in some situations.

In a coal flotation operation, the combined effects of the relatively gentle mixing, the countercurrent flow, and the use of wash water in columns is that, because there is a distance of several meters between the discharge points of the clean coal froth and the gangue tailings, there is a much reduced possibility of coal being misplaced into the tails or of gangue short-circuiting to the froth. The net result is that, depending on the design, a column is typically equivalent to between three and five stages of conventional flotation.

Bubble Generators

The impeller-type mixers that are used in conventional cells are not well suited for use in flotation columns, as they would either need excessively long shafts or rotating seals. In the original flotation column design, bubbles were produced using sintered ceramic air diffusers that produced very fine air bubbles. However, this was found to suffer from plugging problems, particularly in hard water, and for this reason cloth and perforated rubber sheeting alternatives were adopted (Boutin and Wheeler 1967; Dobby et al. 1985). These still tended to require excessive maintenance, and for this reason external bubble generators of various types have been adopted (McKay et al. 1988; Yoon et al. 1990; Davis et al. 1995; Rubinstein 1995). The external bubble generators combine a stream of water with air to produce a mixture of very fine bubbles in water. This mixture is injected into the column. This approach has a number of advantages, including the following:

- The bubble generator is accessible for adjustment and maintenance.
- The column has no porous elements inside that can become clogged or damaged, and so the dispersion of air in the column does not change.
- The bubble generator can be designed to tolerate particulate matter in the water, and so recycle water can be used in the generators.
- External bubble generators can consistently produce very small bubbles in the column.

Axial Mixing Effects

A key issue in flotation column operation is "axial mixing," which is mixing along the vertical axis of the column, as shown in Figure 14.5-22. As the air bubbles rise in the column, they also tend to transfer water to the base of the froth layer. The displaced water then descends again, setting up a strong mixing action. This tendency is greatest if the column is slightly tilted from vertical, as then the bubbles preferentially ascend

on one side while the water descends on the other. This is why it is particularly crucial for flotation columns to be perfectly vertical. The tendency toward axial mixing is also increased if some large bubbles are present, and so performance is optimal if the bubbles are uniformly small in diameter.

Another approach to suppressing axial mixing is the introduction of horizontal baffles made from perforated plates (Kawatra and Eisele 2001). These baffles interrupt the flow of the liquid, as shown in Figure 14.5-22, preventing it from being rapidly carried to the surface or from short-circuiting directly to the tailings. Experiments with horizontal baffles in a flotation column have confirmed that they can greatly improve the operating characteristics of the column without sacrificing capacity, and a sufficiently open baffle design is highly resistant to plugging by coarse particles or debris. Overall, flotation columns generally have superior performance to conventional flotation cells. However, fundamentally, they require automatic control, especially for pulp level control, as they cannot utilize simple tailings overflow weirs, which are commonly used in conventional flotation cells.

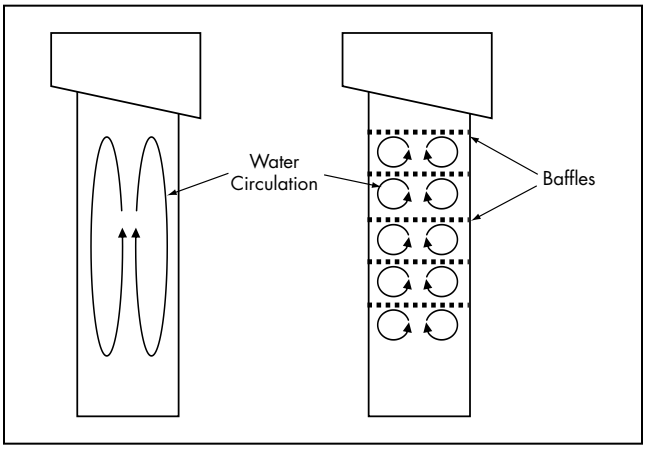


Figure 14.5-22 Comparison of axial mixing in regular columns and horizontally baffled columns

APPLYING LABORATORY RESULTS TO PLANT EQUIPMENT SELECTION

Laboratory Reagent Screening

When the mineralogy of the ore for a new project has been determined, potential reagents for an ore can be based on a review of the relevant literature for similar ores. The sample to be used in laboratory test work should be prepared to minimize the quantity of slimes produced, with a combination of roll crushing and rod milling generally producing a good flotation feed. Because many flotation reagents have a limited shelf life, fresh reagents should be obtained (Thompson 2002).

Flotation experiments to evaluate reagent schemes are generally of two types. Single- and multiple-stage cleaning tests are used to evaluate the relative performance of different reagent types and dosages and show how cleanly the valuable minerals and gangue can be separated. Single-stage "rougher" kinetic flotation tests are used to establish grade–recovery relationships, which are necessary for calculation of residence times and equipment sizes.

Release Analysis

It is often useful to be able to determine the theoretical maximum amount of upgrading that can be achieved by froth flotation, given a specific ore and a particular set of reagents. One approach to this is "release analysis," which is carried out by progressively refloating froth products to collect only the particles that are fully hydrophobic. The procedure is illustrated in Figure 14.5-23.

Release analysis is used for determining the maximum grade of material that can be produced by a froth flotation process. Each stage of flotation removes more of the entrained and poorly floatable particles from the froth, until the final froth product consists only of the most strongly floatable material. It can be considered to be roughly analogous to a heavy-media washability test, except that instead of measuring the separating density that is needed to produce a given grade and recovery, it measures the number of flotation stages needed to achieve a given grade and recovery. It is therefore a valuable method for characterizing the inherent floatability of a particular ore and can be used to produce unbiased comparisons of the relative difficulty of upgrading different ores by froth flotation.

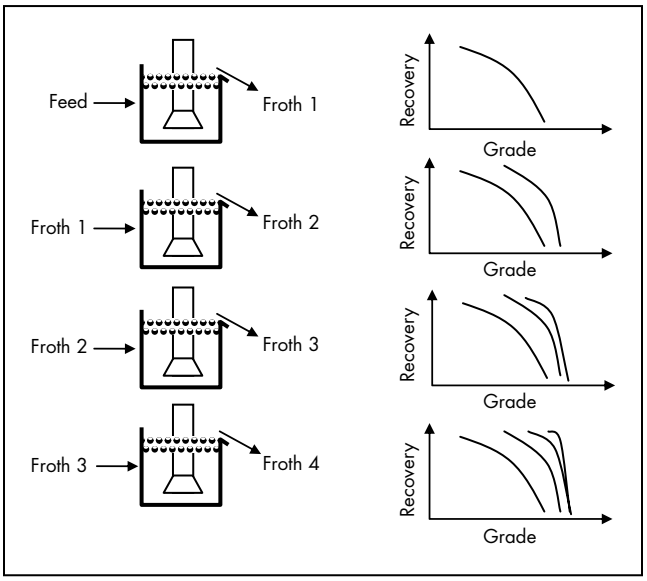


Figure 14.5-23 Release analysis method

Release analysis also provides a means for comparing the performance of conventional flotation with column flotation. The product from release analysis is typically much higher-grade than the product from a single stage of conventional flotation. On the other hand, a correctly operating flotation column will typically provide a product grade that is comparable to the grade of the product from the final stages of release analysis.

Laboratory and Plant Flotation Kinetics

Laboratory kinetic flotation experiments are best carried out in a flotation cell that allows froth to be collected in a reproducible manner over specific time intervals while maintaining a constant level of pulp in the cell. This can be accomplished by use of mechanical froth removal, and a mechanism to continuously provide makeup water to replace the water removed in the froth and prevent the pulp level from dropping and increasing the froth depth in the course of the experiment (Kawatra and Eisele 2001). The time intervals chosen for froth collection will obviously depend on the flotation rate of the mineral, as it is desirable to have at least

five froth increments in order to be able to determine the kinetics properly.

To compare froth flotation experiments in the laboratory with operations in the plant it is necessary to take into account the differences in the way that the two types of cells are operated. This is most easily done by carrying out kinetic experiments to measure the recovery of material to the froth as a function of time.

The following is a simple kinetic equation (Arbiter and Harris 1962):

$$\ln(C_o/C) = kt$$

where

C_o = concentration of the valuable material at the start of the test
C = concentration of valuable mineral in the pulp at time t
k = rate constant

Assuming that all particles of a particular mineral have identical floatabilities, a plot of $\ln(C_o/C)$ versus t will produce a straight line with a slope equal to the rate constant. The higher the value of the rate constant, the higher the collecting power of the reagent used, which provides a means for comparing different reagents in the processing of a particular mineral.

The following equation is a sometimes more useful flotation kinetics model, and it includes a term for both flotation rate and ultimate flotation recovery and takes into account the fact that the particles exhibit a range of hydrophobicities and floatabilities (Klimpel 1995):

$$r = R\{1 - [1 - \exp(-Kt)]/Kt\}$$

where

r = total weight of component recovered at time t
t = time
K = rate constant
R = ultimate theoretical weight recovery at "infinite" time

This model takes into account the fact that the hydrophobic particles vary in size and degree of hydrophobicity, and it is therefore more appropriate than conventional reaction kinetics relationships that are intended to apply to systems of identical molecules. It is particularly useful for correlation of laboratory results with plant results. In conventional laboratory test work, it is common for R to be the most important one in determining the flotation performance, because laboratory tests are often run until all floatable material is recovered. In the plant, it is common for the rate constant, K, to be the most important factor. This is because it is too expensive to provide enough cell volume to recover all material that does not float in a reasonable time. Because of this difference, the results of laboratory studies can be very poor predictors of plant performance. To correct this, it is best to run timed flotation laboratory tests that can produce kinetic data, so that the R and K performances can be determined. Then, based on the residence time of the plant-scale units, it can be determined whether the plant performance is being dominated by kinetics, K, or by the ultimate recovery, R (Klimpel 1995).

Flotation Cell Selection

Flotation cells for use in a plant must be selected based on laboratory and pilot-scale data. Laboratory tests are usually carried out as batch experiments and are generally quite straightforward, although it is necessary to keep the following points in mind:

- The pulp must be agitated sufficiently to keep all particles in suspension.
- It is often necessary to condition the reagents with the minerals for a period of time to ensure good coverage with collector.
- In many cases, adding frother in stages along with makeup water may be necessary to keep the pulp level and froth depth constant.
- The capacity of the cell increases as the percent solids increases, and because of this the best process economics are achieved at the highest percent solids practical.

The following is the most important information obtained from this test work:

- **Optimum grind size of the ore:** This depends not only on the liberation characteristics of the ore but also on the floatability. Excessively coarse particles will be too large to be levitated by attached bubbles, and excessively fine particles will float poorly due to not striking the bubble surfaces, oxidation effects, or other problems. The coarsest material that can be floated is normally around 300 μm, whereas the finest particles are around 5 μm.
- **Reagent addition:** Quantity of reagents needed and the appropriate points in the circuit to add each reagent must be determined.
- **Optimum pulp density:** This is necessary for determining the size and number of flotation cells for a given capacity.
- **Flotation time needed to reach desired recovery:** It is important to be aware that a longer time is required for flotation in the plant than in the laboratory, mainly because of the increased time needed for selected particles to ascend through the large equipment. Typical comparative flotation times for different minerals are shown in Table 14.5-1.
- **Variability of the ore:** Because ore properties vary from one location to another in the mine, it is critical to run experiments with ore from several locations.
- **Corrosive and wear properties of the ore:** This information is needed for selecting appropriate materials of construction for the plant equipment.
- **Circuit design:** The type of circuit, the number cells per bank, the number of flotation stages, and the appropriate locations for recirculation of intermediate products must be determined. The number of cells per bank depends on the flotation characteristics of the material being floated. Typical practice can be as little as three or as many as 17 cells per bank, as shown in Figure 14.5-24.

Example Calculation

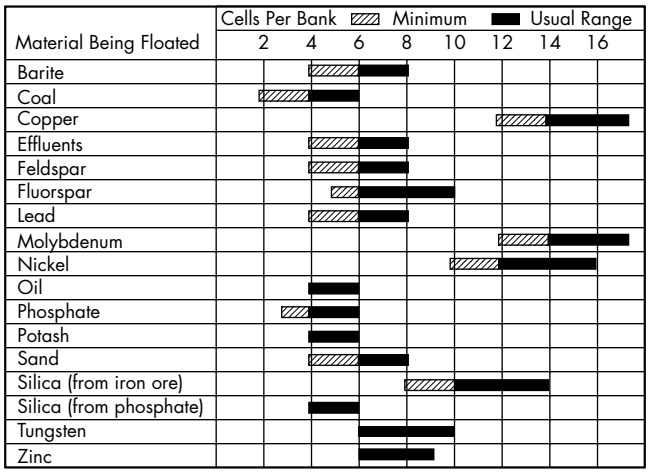
Problem. As an example of sizing flotation cells, consider the following copper ore:

- Specific gravity of dry ore = 2.7
- Optimum percent solids in laboratory flotation machine = 30%
- Optimum laboratory flotation retention time = 6.5 minutes

Table 14.5-1 Comparison of optimum flotation times in the laboratory and in the plant

Material Floating	Flotation Times in Industrial Rougher Flotation Cells, minutes	Usual Laboratory Flotation Times, minutes
Barite	8–10	4–5
Coal	3–5	2–3
Copper	13–16	6–8
Effluents	6–12	4–5
Fluorspar	8–10	4–5
Feldspar	8–10	3–4
Lead	6–8	3–5
Molybdenum	14–20	6–7
Nickel	10–14	6–7
Oil	4–6	2–3
Phosphate	4–6	2–3
Potash	4–6	2–3
Sand (impurities floated)	7–9	3–4
Silica (from iron ore)	8–10	3–5
Silica (from phosphate)	4–6	2–3
Tungsten	8–12	5–6
Zinc	8–12	5–6

Source: Metso Minerals 2006.



Source: Metso Minerals 2006.

Figure 14.5-24 Normal numbers of cells per bank in froth flotation circuits

Assume that the flotation facility will need to process 14,500 t/d of dry ore.

Solution. Using this information, the following can be calculated:

- From Table 14.5-1, the average laboratory flotation time for copper is 7 minutes, and the average plant flotation time is 14.5 minutes.
- The scale-up factor is 14.5/7 = 2.07, and therefore the time needed for flotation in the plant for this particular ore is (6.5)(2.07) = 13.5 minutes.
- A feed rate of 14,500 t/d of dry ore will require a flow of (14,500/1,440 min/d) = 10.07 t/min, and therefore for a 13.5-min residence time, there will need to be (13.5)(10.07) = 136 t of ore retained in the flotation bank.
- The slurry must be 30% (by mass) solids, and therefore, for 136 t of dry ore, there will be (136/0.3) = 453 t of slurry.
- The ore has a specific gravity of 2.7, and therefore the density of the slurry is calculated from the relationship $(100/\rho_{slurry}) = (X/\rho_{solids}) + ((100 - X)/\rho_{water})$, where X is the percent solids, and ρ_{slurry}, ρ_{solids}, and ρ_{water} are the specific gravities of the slurry, solids, and water, respectively.
- Therefore, $(100/\rho_{slurry}) = (30/2.7) + (70/1.0)$, and solving for ρ_{slurry} gives 1.23.
- The total volume of the 453 t of slurry that the cells must hold will be (453/1.23) = 368 m^3. However, when the pulp is aerated, it will consist of approximately 15% air and 85% slurry by volume, so the actual volume of pulp needed will be 368/(0.85) = 433 m^3.
- Cells of 14.15 m^3 (500 ft^3) are a standard size, and therefore the number required if these are used will be 433/14.15 = 30.6 cells (approximately 31 cells).
- Referring to Figure 14.5-24 for copper, between 12 and 17 cells per bank will be required, with 14 to 17 cells being optimum. Two banks of 16 cells would give 32 total cells, which would provide the 31 cells needed with some extra capacity. The cells would then be arranged in the individual banks in accordance with the manufacturer's recommendations.

Design Software

The design of flotation systems is simplified by the use of appropriate design software. Circuit-simulation packages can take data from laboratory test work and can predict the likely performance of a given design of a full-scale circuit. This allows the evaluation of many different arrangements before beginning pilot-scale studies (Collins et al. 2009). This type of software is under active development, so it is necessary to check the current literature to determine what is available at any given time. Such software packages are typically quite expensive and require training to use to their full potential, and they are most effectively used by specialist firms rather than by individual mining companies.

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CHAPTER 14.6

Magnetic and Electrostatic Separation

Partha V. Iyer

The two methods of physical separation—magnetic and electrostatic separation—have developed from very early recognition of the forces that they utilize. Initially only low-field-strength permanent magnets and contact electrification were available, and hence the technologies were of limited value. However, as available magnetic field strengths increased, applications widened. Similarly, the introduction of safe high-voltage supplies opened up new electrostatic separation possibilities. Electrostatic separation is now something of a misnomer, as the most effective use of high-voltage electricity for separation involves small current flows.

Magnetic and electrostatic separation techniques have probably always been bracketed together because historically they were used for dry separation of liberated sand-sized mineral particles. They were mainly applied in the final stages of separation and/or upgrade of beach sand and alluvial tin concentrates (Gaudin 1939; Oberteuffer 1974; Knoll 1997; Svboda 1987; Venkatraman et al. 2003; Wills and Napier-Munn 2006).

Dry magnetic and electrostatic separation are most efficient on closely sized material. In the case of electrostatic separation, brief high-intensity drying can be applied to remove surface moisture and thus enhance differences in surface conductivity. In some cases, the magnetic or electrostatic properties of minerals can be modified by surface treatment at sustained high temperatures.

Wet magnetic separation has found major application in the treatment of fines at iron-ore operations. One of its attractions is its ability to handle a much wider and finer size range than dry systems. It is also applied in the cleaning of heavy media suspensions such as ferrosilicon.

Useful peripheral applications include removal of tramp iron from run-of-mine or coarse-crushed material (magnetic) and removal of dust from roaster and smelter off-gases (electrostatic). One particular attraction of both methods of separation is that they normally create few if any environmental problems and may in fact solve some.

Knowledge of magnetic and electrostatic forces dates back at least to the Greek philosopher Thales of Miletus, who lived about 600 BC. Thales knew some of the magnetic properties of the mineral lodestone, and he was also aware that when amber was rubbed with animal fur, the electrostatic charge produced on the amber (or fur) would attract light, nonconducting particles. The first record of magnetic separation of minerals appears to be a patent issued in 1792 to an English experimenter, William Fullarton, that described the concentration of iron ore. About a century later, in 1886, F.R. Carpenter obtained a U.S. patent for electrostatic concentration of ore.

The application of magnetic separators has extended well beyond either removing tramp iron or concentrating materials that are obviously affected by low-rate (and unsophisticated) magnetic fields. The latest developments in material science and magnet technology have allowed high-intensity and high-gradient industrial magnetic separators with field strengths as high as 6 T (60,000 G [gauss]) to be developed. Development of permanent rare-earth magnets and superconducting magnets has opened new markets for magnetic separators.

The electrostatic separator is still the most reliable and economic unit operation for processing beach-sand deposits rich in minerals such as ilmenite, rutile, leucoxene, zircon, and garnet. Increased environmental awareness has promoted the demand for unit operations that process secondary materials. A classic example is the successful use of electrostatic separators to remove plastics from metals. Triboelectrostatic separators, which can successfully separate two nonconductors, are being used in minerals and plastics separation.

REVIEW OF MAGNETIC THEORY

All materials can be classified as follows, based on their magnetic properties:

- *Paramagnetic* minerals are attracted along the lines of magnetic force to points of greater field intensity.
- *Diamagnetic* minerals are repelled along the lines of magnetic force to a point of lesser field intensity.
- *Ferromagnetic* minerals, a special category of paramagnetic materials, have a very high susceptibility to magnetic forces and retain some magnetism (remnant magnetism) after removal from the magnetic field.

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Magnetic susceptibilities and electrostatic responses of minerals are provided in Table 14.6-1. In this section, the fundamentals of magnetic separation are introduced without detailed explanations of the physics involved.

Magnetic Force or Flux Density

In an electromagnet, an electric charge in motion sets up a magnetic field in the space surrounding it, and a magnetic field exerts a force on an electric charge moving through it. In fact, all magnetic phenomena arise from forces acting between electric charges in motion. Therefore, flux density B at point P, resulting from current element I of length dl, is as specified by Ampere's law:

$$dB = KI\left[\frac{dl(\sin\theta)}{r^2}\right] \tag{14.6-1}$$

where

B = magnetic flux density, weber/square meter (Wb/m^2)
K = constant of proportionality
I = current element, amperes (A)
l = length of current element, m
θ = angle between current element and radius vector to point P
r = distance to P, m

Magnetic flux density B, or force per pole, is defined as the number of magnetic flux lines per unit area normal to the lines. The unit for magnetic flux is the weber (Wb). The unit for term B is webers per square meter, called a tesla (T). The largest values of magnetic induction that can be produced in the laboratory are about 50 to 60 T (500,000 to 600,000 G).

Industrial high-intensity superconducting separators can reach fields of about 5 T (50,000 G). High-intensity induced-roll dry separators reach values as high as 2 T (20,000 G). Low-intensity wet drum separators for concentration of iron ore have drum-surface field strengths on the order of 0.1 T (1,000 G). However, the magnetic force acting on particles depends not only on the magnetic field B, but also on its gradient dB/dz, where z is the direction of the changing field.

Magnetization

A ferromagnetic material can be magnetized simply by being brought close to a permanent magnet or by passing current through a wire winding around the material. The magnetic state of a body can be defined by (1) stating the magnetization of all points within the body, (2) defining the strength of the magnetic poles, or (3) defining the magnitude of equivalent surface currents.

Coulomb's law for magnets is similar to that for electric charges; that is,

$$F = \frac{1}{4\pi\mu_o} * \frac{m_1 m_2}{r^2} \tag{14.6-2}$$

where

F = force, newtons (N)
μ_0 = permeability of a vacuum ($4\pi \times 10^{-7}$ H/m [henry/meter])
m_1 and m_2 = pole strength, A/m
r = distance, m

Magnetization (or, more completely, the intensity of magnetization M) is the total magnetic moment of dipoles per unit volume, in units of amperes per meter or pole strength m per unit area A.

$$M = \frac{m}{A} \tag{14.6-3}$$

Magnetic Field

The strength H of the magnetic field has the same units as does M (amperes/meter) and can be thought of as the cause of magnetization. It is defined as

$$H = \frac{B}{\mu} \tag{14.6-4}$$

where μ is absolute permeability.

In free space, a magnetic field produces a magnetic force given by $B = \mu_0 H$. However, in most applications the space is filled with some magnetic substance that causes an induced magnetization, $\mu_0 M$. Therefore, the total magnetic flux density B is the vector sum of the flux caused by the magnetic field H and the flux resulting from the magnetization M of the material. For ferromagnetic materials, however, the contribution of M usually dominates B:

$$B = \mu_0(H + M) \tag{14.6-5}$$

Permeability and Susceptibility

Magnetic flux density B, strength H of the magnetic field, and magnetization M can be used to compare the magnetic response of various materials. The ratio M/H is a dimensionless quantity called the *volume susceptibility* or simply *magnetic susceptibility*, χ. Similarly, the ratio B/H is called the *absolute permeability*, μ. The *relative permeability* μ_r is defined as

$$\mu_r = \frac{\mu}{\mu_o} \tag{14.6-6}$$

where

μ_r = relatively permeability
μ = absolute permeability
μ_0 = permeability in a vacuum ($4\pi \times 10^{-7}$ H/m)

The relative permeability of diamagnetic materials is slightly less than 1, that of paramagnetic materials is slightly greater than 1, and that of ferromagnetic materials is very high (e.g., for iron with 0.2% impurities, μ_r = ~5,000).

Summary

Paramagnetic minerals have magnetic permeabilities that are higher than the surrounding medium, usually air or water, and they concentrate the lines of force of an external magnetic field. The higher the magnetic susceptibility, the higher the field intensity in the particle and the greater the attraction up the field gradient toward increasing field strength.

Diamagnetic minerals have magnetic permeabilities that are lower than the surrounding medium, usually air or water, and they repel the lines of force of an external magnetic field. These characteristics cause the expulsion of diamagnetic

Table 14.6-1 Minerals and their magnetic and electrostatic responses

Mineral	Composition	Specific Gravity	Magnetic Response			Electrostatic Response		
			Ferromagnetic	Paramagnetic	Nonmagnetic	Conductive		Nonconductive
Actinolite	$Ca_2(Mg,Fe)_5(Si_4O_{11})_2(OH)_2$	3.0–3.2		x				x
Albite	$Na(AlSi_3O_8)$	2.6			x			x
Almandine	$Fe_3Al_2(SiO_4)_3$	4.3		x				x
Amphibole	$(Fe,Mg,Ca)_xSiO_3$	2.9–3.5		x				x
Anatase	TiO_2	3.9			x	x		
Andalusite	Al_2SiO_5	3.2			x			x
Andradite	$3CaO{\cdot}Fe_2O_3{\cdot}SiO_2$	3.8		x		(2)	←	x
Anhydrite	$CaSO_4$	3.0			x			x
Ankerite	$Ca(Mg,Fe)(CO_3)_2$	2.9–3.1		x				x
Apatite	$(F_1Cl_1OH)Ca_5(PO_4)_3$	3.2			x			x
Aragonite	$CaCO_3$	3.0			x			x
Arsenopyrite	FeAsS	5.9–6.2		x	→ (1)	x		
Asbestos	$Mg_3[Si_2O_5](OH)_4$	2.4–2.5			x			x
Augite	$Ca(Mg,Fe,Al)[(Si,Al)_2O_6]$	3.2–3.5		x		x	→	(1)
Azurite	$Cu_3[CO_3]_2(OH)_2$	3.8			x			x
Baddeleyite	ZrO_2	5.6			x			x
Barite	$BaSO_4$	4.5			x			x
Bastnaesite	$(Ce,La,F)CO_3$	5.0		x				x
Bauxite	$Al_2O_3{\cdot}2H_2O$	2.6			x			x
Beryl	$Be_3Al_2[Si_6O_{18}]$	2.7–2.8			x			x
Biotite	$K(Mg,Fe)_3[Si_3AlO_{10}](OH,F)_2$	3.0–3.1		x				(4)
Bismuth	Bi	9.8			x	x		
Borax	$Na_2B_4O_7{\cdot}10H_2O$	1.7			x			x
Bornite	Cu,FeS_4	4.9–5.0		(1)	← x	x		
Brannerite	$(UO,TiO,UO_2)TiO_3$	4.5–5.4		x		x		
Brookite	TiO_2	4.1			x	x		
Calcite	$CaCO_3$	2.7			x			x
Carnotite	$K_2(UO_2)_2V_2O_8{\cdot}2H_2O$	5.0			x	(2)	←	x
Cassiterite	SnO_2	7.0			x	x		
Celestite	$SrSO_4$	4.0			x			x
Cerussite	$PbCO_3$	6.6			x	(2)		x
Chalcocite	Cu_2S	5.5–5.8			x	x		
Chalcopyrite	$CuFeS_2$	4.1–4.3		(1)	← x	x		
Chlorite	$(Mg,Al,Fe)_{12}[(Si,Al)_8O_{20}](OH)_{16}$	2.6–3.2		x				x
Chromite	$(Fe,Mg)(Cr,Al)_2O_4$	4.6		x		x		
Chrysocolla	$CuSiO_3{\cdot}nH_2O$	2.0–2.3			x			x
Cinnabar	HgS	8.1			x			x
Cobalitite	(Co,Fe)AsS	6.0–6.3		x		x		
Colemanite	$Ca_2B_6O_{11}{\cdot}5H_2O$	2.4			x			x
Collophanite	$Ca_3P_2O_8{\cdot}H_2O$	2.6–2.9			x			(3)
Columbite	$(Fe,Mn)(Ta,Nb)_2O_6$	5.2–8.2		x		x		
Copper	Cu	8.9			x	x		
Corumdum	Al_2O_3	3.9–4.1			x			x
Covellite	CuS	4.7			x	x		
Cryolite	Na_3AlF_6	3.0			x	(2)	←	x
Cuprite	Cu_2O	5.8–6.2			x			x
Diamond (natural)	C	3.5			x			x
Diamond (synthetic)	C	3.5		x				x
Diopside	$CaMg[Si_2O_6]$	3.3–3.4		x	→ (1)			x
Dolomite	$CaMg(CO_3)_2$	1.8–2.9			x			x
Epidote	$Ca_2(Al,Fe)_3Si_3O_{12}(OH)$	3.4		x				x

(continues)

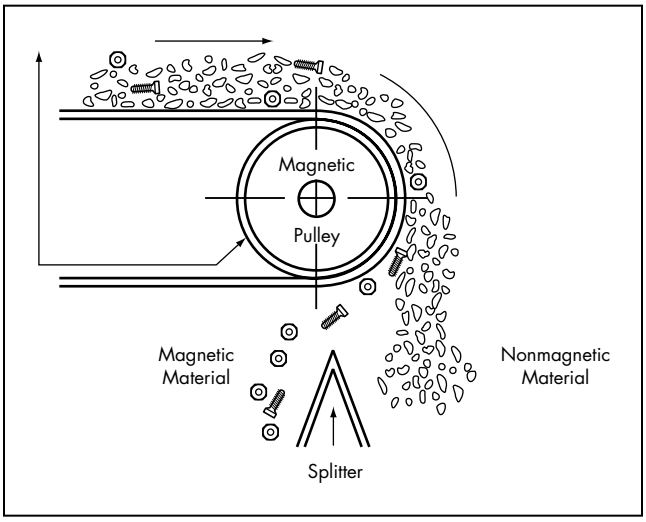
Table 14.6-1 Minerals and their magnetic and electrostatic responses (continued)

Mineral	Composition	Specific Gravity	Magnetic Response					Electrostatic Response		
			Ferromagnetic	Paramagnetic		Nonmagnetic		Conductive		Nonconductive
Euxenite	$(Y,Er,Ce,La,U)(Nb,Ti,Ta)_2(O,OH)_6U_3O_8$	4.7–5.2		x				x		
Feldspar group	$(K,Na,Ca..)_x(Al,Si)_3O_8$	2.6–2.8				x				x
Ferberite	$FeWO_4$	7.5	(1)	← x				x		
Flint	SiO_2	2.6				x				x
Fluorite	CaF_2	3.2				x				x
Franklinite	$(Zn,Mn)Fe_2O_4$	5.1–5.2	x					x		
Gahnite	$ZnAl_2O_4$	4.6				x				x
Galena	PbS	7.5				x		x		
Garnet complex	Ca,Mg,Fe,Mn silicates	3.4–4.3		x	→	(1)		(2)	←	x
Gibbsite	$Al(OH)_3$	2.4				x				x
Geothite	FeO(OH)	4.3		x				(2)	←	x
Gold	Au	15.6–19.3				x		x		
Graphite	C	2.1–2.2				x		x		
Grossularite	$Ca_3Al_2(SiO_4)_3$	3.5				x		(2)	←	x
Gypsum	$CaSO_4{\cdot}2H_2O$	2.3				x				x
Halite	NaCl	2.5				x		(2)	←	x
Hematite	Fe_2O_3	5.2		x				x		
Hornblende	$Ca_2Na(Mg,Fe^{2+})_4(Al,Fe^{3+})[(Si,Al)_4O_{11}](OH)_2$	3.1–3.3		x				(2)	←	x
Huebnerite	$MnWO_4$	6.7–7.5		x	→	(1)		x		
Hypersthene	$(Mg,Fe)SiO_3$	3.4		x						x
Ilmenite	$FeTiO_3$	4.7		x				x		
Ilmenorutile	$(Nb_2O_5,Ta_2O_5)_xTiO_2$	5.1		x				x		
Ilvaite	$CaFe_2(FeOH)(SiO_4)_2$	4.0		x				x	→	(1)
Kaolinite	$Al_2Si_2O_5(OH)_4$	2.6				x				x
Kyanite	$Al_2O[SiO_4]$	3.6–3.7				x				x
Lepidolite	$K(Li,Al)_3(Si,Al)_4O_{10}[OH,F)_2$	2.8–2.9				x				x
Leucoxene	$FeTiO_3$ TiO_2(alteration product)	3.6–4.3		x	→	(1)		x		
Limonite	$HFeO_2{\cdot}nH_2O$	2.2–2.4		x	→	(1)		(2)	←	x
Magnesite	$MgCO_3$	3.0				x				x
Magnetite	Fe_3O_4	5.2	x					x		
Malachite	$Cu_2CO_3(OH)_2$	4.0				x				x
Manganite	MnO(OH)	4.3		x	→	(1)		x		
Marcasite	FeS_2	4.6–4.9				x		x		
Martite (see Hematite)										
Microline	$KAlSi_3O_8$	2.6				x				x
Microlite	$Ca_2Ta_2O_7$ (see Pyrochlore)	5.5				x				x
Millerite	NiS	5.2–5.6		x				x		
Molybdenite	MoS_2	4.7–5.0				x		x		
Monazite	$(Ce,La,Y,Th)PO_4$	4.9–5.5		x						x
Mullite	$Al_6Si_2O_{13}$	3.2				x				x
Muscovite	$KAl_2[AlSi_3O_{10}][F,OH]_2$	2.8–3.0				x				(4)
Nahcolite	$NaHCO_2$	2.2				x				x
Nepheline syenite	$(Na,K)(AlSi)_2O_4$	2.6				x				x
Niccolite	NiAs	7.6–7.8		x				x		
Olivine	$(Mg,Fe)_2[SiO_4]$	3.3–3.5		x						x
Orpiment	As_2S_3	3.4–3.5				x		x		
Orthoclase	$K[Al,Si_3O_8]$	2.5–2.6				x				x
Periclase	MgO	3.6				x				x
Perovskite	$CaTiO_3$	4.0				x				x
Petalite	$LiAl(Si_2O_5)_2$	2.4				x				x

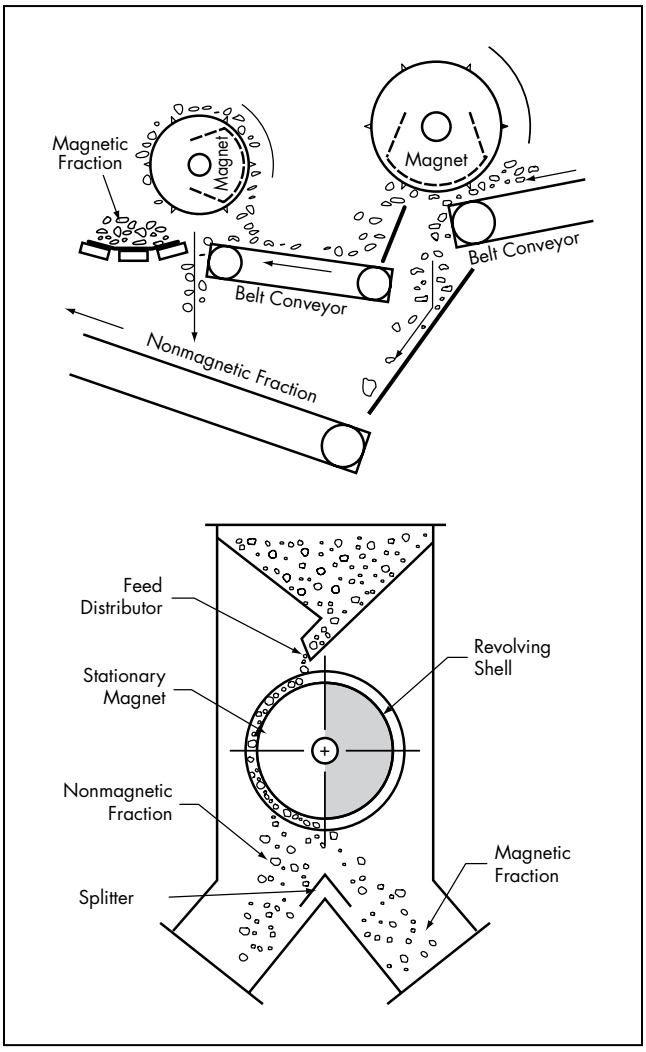
(continues)

Table 14.6-1 Minerals and their magnetic and electrostatic responses (continued)

Mineral	Composition	Specific Gravity	Magnetic Response					Electrostatic Response		
			Ferromagnetic		Paramagnetic		Nonmagnetic	Conductive		Nonconductive
Phosphate (pebble)	(see Collaphanite)									
Platinum	Pt	14.0–21.5			(1)	←	x	x		
Pyrite	FeS_2	5.0			(1)	←	x	x		
Pyrochlore	$(Na,Ca..)_2(Nb,Ta..)_2O_6[F,OH]$	4.2–4.4					x	x		
Pyrolusite	MnO_2	4.7–5.0			(1)	←	x			x
Pyrope	$Mg_3Al_2(SiO_4)_3$	3.5					x	(2)	←	x
Pyroxene	$(Ca,Mg,Fe,Al)_2Si_2O_6$	3.1–3.6			x	→	(1)	(2)	←	x
Pyrrhotite	$Fe_{x-1}S_x$	4.6–4.7	x					x		
Quartz	SiO_2	2.7					x			(3)
Realgar	AsS	3.6					x	x		
Rhodochrosite	$MnCO_3$	3.7					x	(2)	←	x
Rhodonite	$MnSiO_3$	3.6–3.7					x	(2)	←	x
Rutile	TiO_2	4.2–4.3					x	(2)		
Samarskite	$(Y,Er..)_4[(Nb,Ta)_2O_7]_3$	5.6–5.8	(1)	←	x			x		
Scheelite	$CaWO_4$	6.1					x			x
Serpertine	$Mg_6[Si_4O_{10}](OH)_8$	2.5–2.7			x					x
Siderite	$FeCO_3$	3.9			x			(2)	←	x
Sillmanite	$Al_2O[SiO_4]$	3.2					x			x
Silver	Ag	10.1–11.1					x	x		
Smithsonite	$ZnCO_3$	4.1–4.5					x			x
Sodalite	$Na_8[Al_6Si_6O_{24}]Cl_2$	2.1–2.3					x			x
Spessarite	$Mn_3Al_2[SiO_4]_3$	4.2					x			x
Sphalerite	ZnS	3.9–4.0			x	→	(1)	x	→	(1)
Sphene	$CaTi[SiO_4](F_3OH)$	3.3–3.6					x	(2)	←	x
Spinel	$MgAl_2O_4$	3.6			(1)	←	x	x		(1)
Spodumene	$LiAl(SiO_3)_2$	3.1–3.2					x			x
Stannite	Cu_2FeSnS_4	4.3–4.5					x	x		
Staurolite	$Fe^{2+}Al_4[Si_4O_{11}]_2O_2(OH)_2$	3.6–3.8			x			(2)	←	x
Stibnite (Antimonite)	Sb_2S_3	4.6					x	x		
Struverite	$(Ta_2O_5,Nb_2O_5)_xTiO_2$	5.1			x			x		
Sulpher	S	2.1					x			x
Sylvite	KCl	2.0					x			x
Talc	$Mg_3Si_4O_{10}(OH)_2$	2.7–2.8					x			x
Tantalite	$(Fe,Mn)(Ta,Nb)_2O_6$	5.2–8.2			x			x		
Tapiolite	$Fe(Ta,Nb)_2O_6$	7.3–7.8			x			x		
Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$	5.0			x			x		
Thorianite	ThO_2	9.7					x			x
Thorite	$ThSiO_4$	4.5–5.4					x			x
Topaz	$Al_2SiO_4(F,OH)_2$	3.5–3.6					x			x
Tourmaline	$(Na,Ca)(Mg,Fe^{2+},Fe^{3+},Al,Li)_3(Al,Mg,Cr)_6\ B_3Si_6(OH.O,F)_4$	2.9–3.2			x	→	(1)	(1,2)	←	x
Uraninite	UO_2	11.0			x					x
Vermiculite	$Mg_3[Al,Si_3O_{10}](OH)_2{\cdot}nH_2O$	2.4–2.7					x			x
Wolframite	$(Fe,Mn)WO_4$	6.7–7.5			x			x		
Wollastonite	$CaSiO_3$	2.8–2.9					x			x
Wulfenite	$PbMoO_4$	6.7–7.0					x	x		
Xenotime	YPO_4	4.4–5.1			x					x
Zeolite	Hydrous aluminosilicate usually of Ca and Na	2.0–2.5					x			x
Zincite	ZnO	5.7					x	(1)	←	x
Zircon	$ZrSiO_4$	4.7					x			x



Courtesy of Eriez Magnetics.
Figure 14.6-1 Typical magnetic pulley



Courtesy of Eriez Magnetics.
Figure 14.6-2 Magnetic drum separators operating as lifting magnets

minerals down the gradient of the field toward decreasing field strength. This negative diamagnetic effect is usually orders of magnitude smaller than the positive paramagnetic attraction. Thus, a magnetic circuit can be designed to produce higher field intensity or higher field gradient, or both, to achieve effective separation (Falconer 1992).

CONVENTIONAL MAGNETS

Magnets are used in the mineral industry to remove tramp iron that might damage equipment and to separate minerals according to their magnetic susceptibility.

Low-Intensity Magnetic Separators

Low-intensity magnetic separators have flux densities up to 0.2 T (2,000 G). These separators are used mainly to remove ferromagnetic materials (such as iron bolts, bars, cables, and tools), to protect downstream unit operations (such as conveyor belts), or to scalp ferromagnetic materials to improve the performance of permanent or electromagnetic separators used to separate weakly magnetic materials. Low-intensity separators can treat wet slurry or dry solids.

Protective Magnets

The device most widely used to protect downstream operations from tramp iron is a magnetic pulley installed in the head of a conveyor (Figure 14.6-1). These devices remove tramp metals from dry solids. They contain either a permanent magnet or an electromagnet. Many types of magnets can be used—for example, plate magnets, cross-belt magnets, cobbing magnets, grate magnets, magnetic humps, and magnetic filters. An arrangement of magnetic drum separators is shown in Figure 14.6-2.

Wet Magnetic Separators

Low-intensity wet magnetic separators have been the workhorse of the iron-ore industry for several decades. Iron ore that is rich in magnetite has traditionally been enriched by these magnets. The coal industry uses these magnets to recover magnetite or ferrosilicon in a media-recovery circuit.

Several types of separator work on the same principle but have different design features. The common types are counter-rotation drum separators and concurrent-rotation drum separators (Figure 14.6-3).

High-Intensity Magnetic Separators

Separating paramagnetic or weakly magnetic particles requires a higher flux density. This higher density is achieved by means of electromagnetic circuitry that can generate a magnetic force of up to 2 T (20,000 G). For example, in a silica sand processing plant, these separators remove weakly magnetic iron-bearing particles.

Induced-Roll Magnetic Separator

Induced-roll dry magnetic separators are widely used to remove trace impurities of paramagnetic substances from feedstock such as quartz, feldspar, and calcite. The machine contains laminated rolls of alternating magnetic and nonmagnetic discs. A magnetic flux on the order of 2 T (20,000 G) is generated, and very high gradients are obtained where the flux converges on the sharp edges of the magnetic laminations.

A thin stream of granular material is fed to the top of the first roll (Figure 14.6-4). The magnetic particles are attracted to the roll and deflected out of their natural trajectory. Selectivity

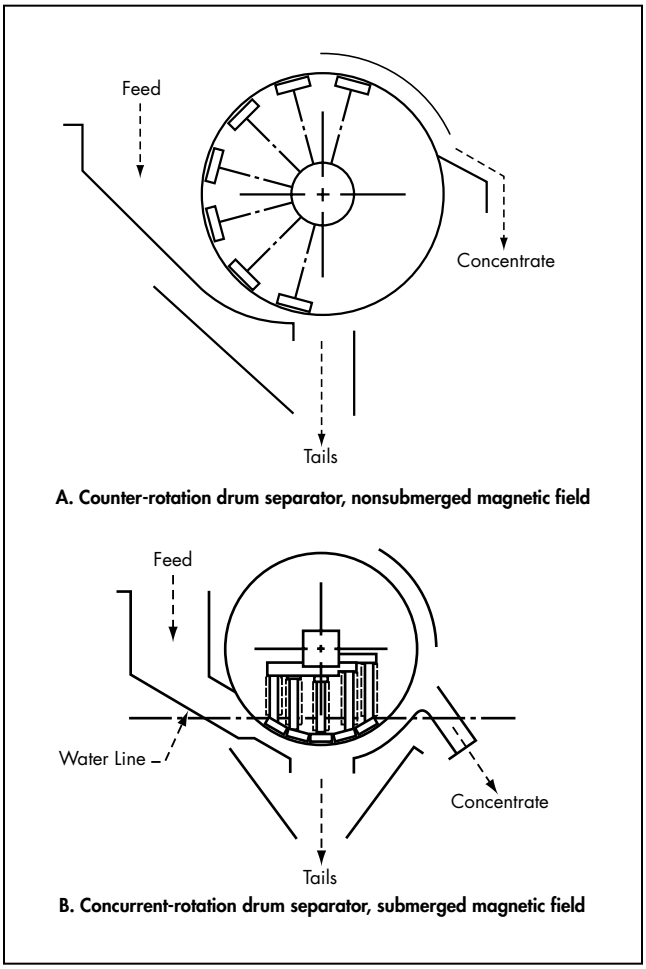


Figure 14.6-3 Wet magnetic separators

is obtained by varying roll speed and magnetic flux. A rather closely sized material must be treated if high selectivity is required. An industrial induced-roll magnetic separator consists of several rolls and can treat up to 10 tph (Figure 14.6-5).

Lift-Type Magnetic Separator

Lift-type magnetic separators are used on granular and powdered material that is dry and free-flowing. This type of separator produces a clean magnetic product because the magnetic particles are lifted out of the stream against the force of gravity, which minimizes entrapped particles (Figure 14.6-6). The selectivity of the lift-type separator is superior to that of induced-roll separators. Their main limitation is lower capacity. The cross-belt separator, a type of lift magnetic separator, has been used to some extent in processing ilmenite, garnet, and monazite in beach sands.

Jones Separator

The Jones separator is a wet high-intensity separator built on a strong mainframe made of structural steel (Figure 14.6-7). Magnet yokes are welded to this frame, and electromagnetic coils are enclosed in air-cooled cases. Actual separation takes place in plate boxes that are on the periphery of the one or two rotors attached to the central shaft.

Feed, which is a thoroughly mixed slurry, flows through the separator by means of fitted pipes and launders, and then

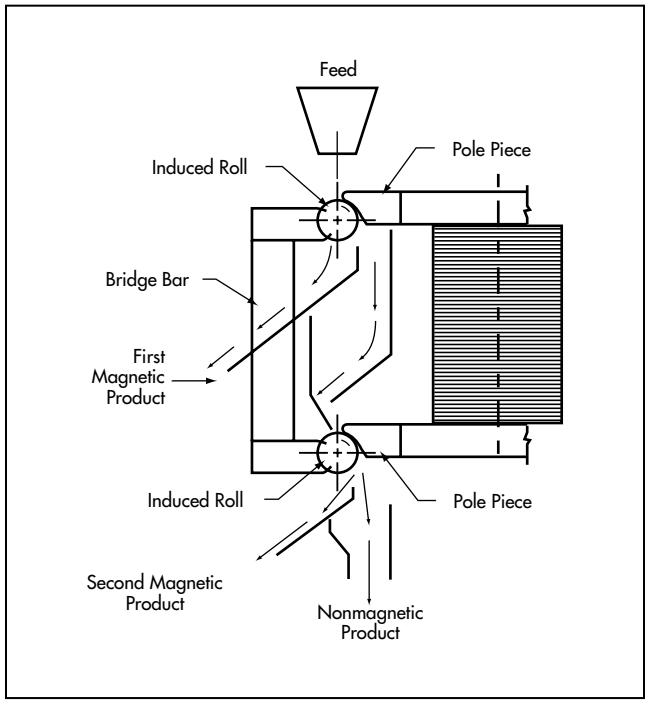
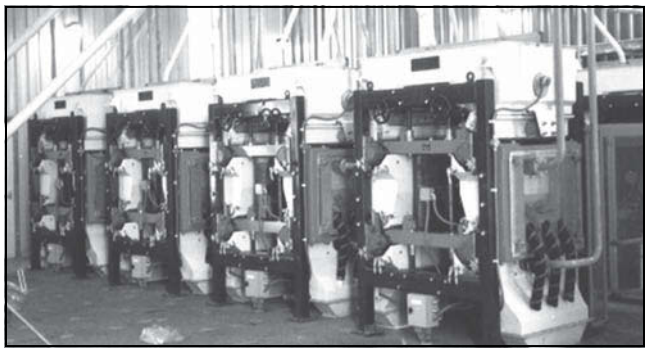


Figure 14.6-4 Induced-roll dry magnetic separator



Courtesy of Outotec USA, Inc.

Figure 14.6-5 Induced-roll dry magnetic separators treating silica sand

into the plate boxes. The plate boxes are grooved to concentrate the magnetic field at the tips of the ridges. Feeding is continuous as a result of the rotation of plate boxes and rotors, with feed points at the leading edges of the magnetic fields. Each rotor has two symmetrically placed feed points. The feebly magnetic particles are held by the plates, while the remaining nonmagnetic slurry passes straight through the plate boxes and is collected in a launder. Before leaving the field, entrained nonmagnetic particles are washed by low-pressure water and collected as a so-called "middlings product." When the plate boxes reach a point midway between the magnetic poles where the magnetic field is essentially zero, the magnetic particles are washed out under high-pressure scour water sprays of up to 5 bars of pressure.

Field intensities of >2 T (20,000 G) can be produced in these machines. They are widely used to recover iron minerals from low-grade hematite ore. Other common applications include removing magnetic impurities from cassiterite

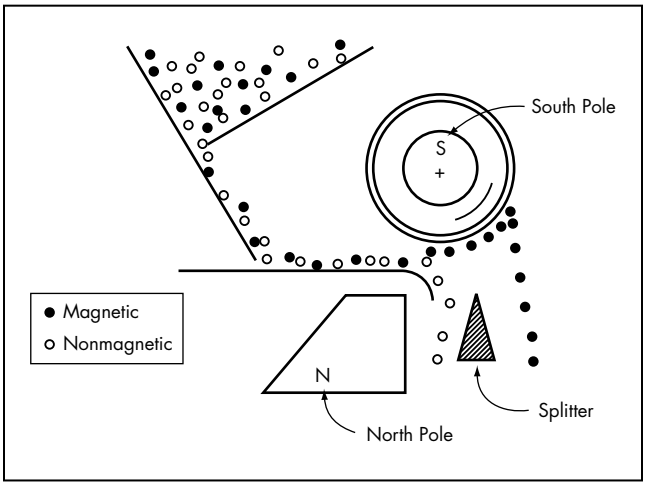


Figure 14.6-6 Lift-type magnetic separator

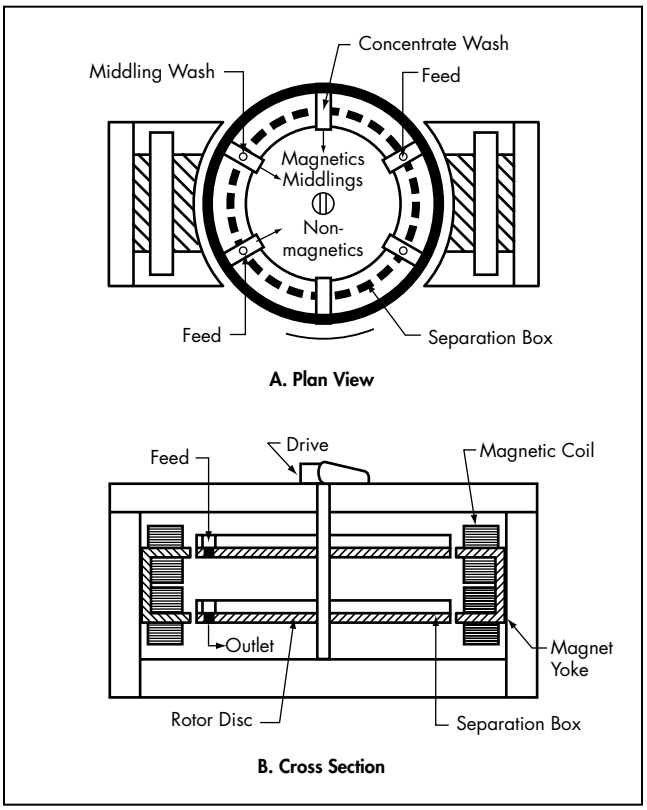


Figure 14.6-7 Jones separator

concentrate, removing fine magnetics from asbestos, and purifying talc.

Frantz Isodynamic Separator

The Frantz Isodynamic separator, introduced in the early 1930s, is the most efficient magnetic separator for separating minerals with field-independent magnetic susceptibilities. The isodynamic field, generated by a bipolar magnet with special pole-tip profiles, provides constancy of the product of the field and the field gradient. However, mineral separation in an isodynamic magnetic field is limited to minerals that have a constant susceptibility at the laboratory scale. Only this category of mineral then experiences a constant force throughout the isodynamic area.

PERMANENT MAGNETS

Most weakly magnetic minerals, such as garnet, ilmenite, and magnetic impurities in silica sand, can be effectively separated with a magnetic separator that has a flux density of >0.6 T (6,000 G). For nearly a century, induced-roll magnetic separators were the only economically viable unit operation in these applications. However, in spite of their considerable success, induced-roll separators have certain limitations in their selectivity and application. The development of permanent-magnet technology during the last three decades has reestablished the importance of magnetic separation and has increased the efficiency of fine-particle separations that were not successful with induced-roll magnets.

Principle and Design

In the last three decades, magnetic separation technology has undergone a revolution. Research in material science and ceramic technology has culminated in the development of new permanent rare-earth magnets and superconducting alloys. Successful adaptation of these magnetic materials, combined with knowledge of magnet geometry, has led to the design and development of a number of new magnetic separators that have opened niche markets that were previously considered beyond the realm of magnetic separation. These new separators are capable of

- Effectively removing magnetic impurities or reducing their concentration (even to parts-per-million levels);
- Producing high-grade mineral separates;
- Operating on virtually no energy, which makes them economical; and
- Generating higher magnetic flux levels, of up 2.1 T (21,000 G).

Dry Permanent Magnetic Separator

Recent improvements in magnet composition and design have led to the development of permanent magnetic separators. These improved rare-earth permanent magnets—for example, neodymium iron boron (NdFeB) and samarium cobalt (SmCo) magnets—have a magnetic attractive force that is an order of magnitude greater than that of conventional permanent magnetic circuits. The two main types of dry permanent magnetic separators that have found wide industrial applications are the rare-earth drum (RED) and the rare-earth roll (RER). They are widely used to separate weakly magnetic materials (such as garnet, ilmenite, and chromite) and to separate magnetic impurities present in low concentrations in silica sand (Dingwu et al. 1997; Dobbins et al. 2009; Wasmuth and Mertins 1997).

Rare-Earth Drum Separator

In a RED separator, NdFeB magnets are uniquely arranged to provide an intense (≤0.9 T or 9,000 G) and "deep" magnetic field perpendicular to the drum surface (Figure 14.6-8). Particles on the drum surface experience uniform flux density that minimizes misplacement of pinned particles to the middlings. Weakly magnetic particles pinned to the drum are carried to the region of no magnetic intensity and released as magnetics. The centrifugal force of the rotating drum throws those particles not influenced by the magnetic field into the nonmagnetic hopper.

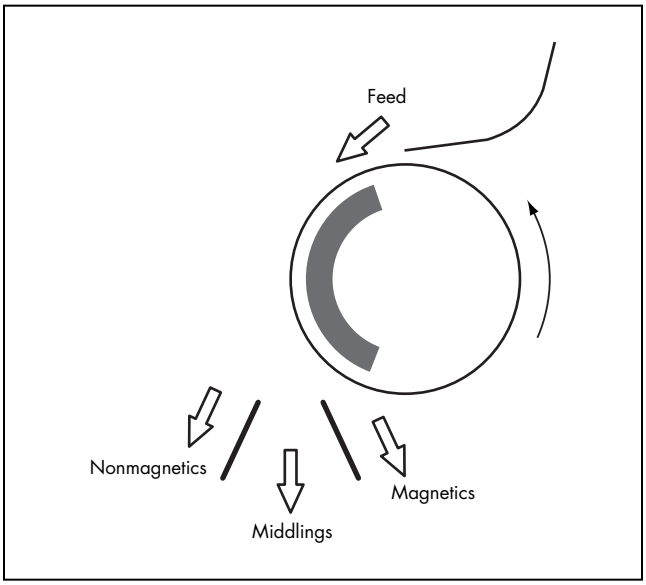


Figure 14.6-8 Operating principle of a rare-earth drum separator

An industrial-scale RED separator usually has three drums (Figure 14.6-9). The top drum is generally a low-intensity (≤0.2 T or 2,000 G) scalper magnet that removes ferromagnetic particles; the nonmagnetic fraction is subsequently treated on the REDs. The main purpose of the scalper is to protect the bottom two REDs as well as to increase their capacity. Some separators have a built-in internal air-cooling system to protect the magnets from overheating when feed is preheated, as in plants that process beach and silica sand.

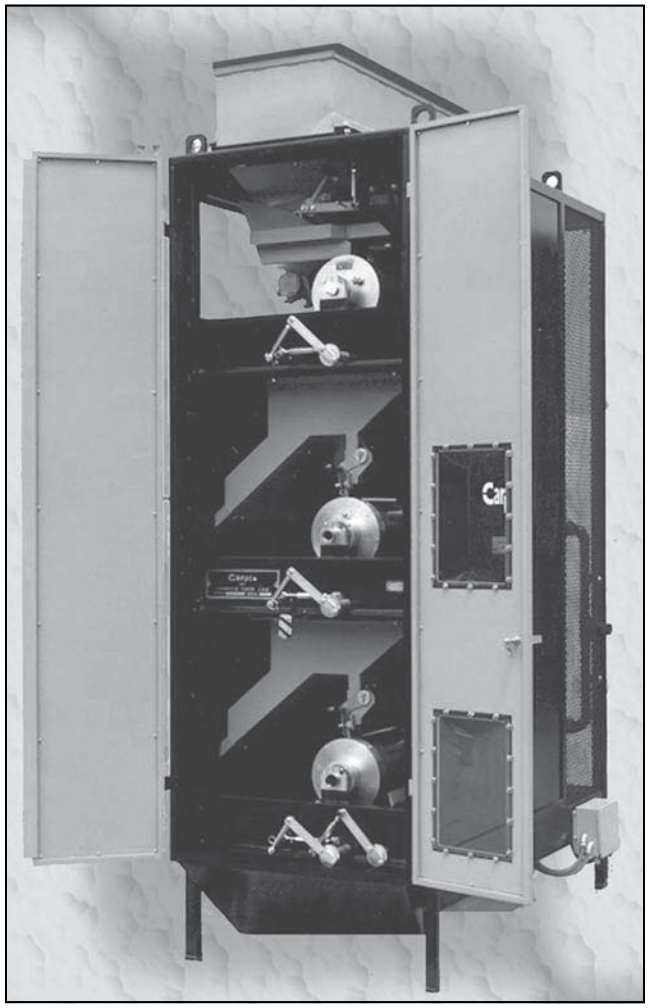
Rare-Earth Roll Separator

An RER separator operates by means of a thin belt (usually 0.076–0.51 mm) that travels at very high velocity. The unique aspect of this separator is the way in which it is configured as a head pulley. Feed passes through the magnetic field, and magnetic (or weakly magnetic) particles are attached to the roll and separated from the nonmagnetic stream (Figure 14.6-10).

Although drum separators can effectively handle coarse particles (12.5 to 0.075 mm), roll separators are very effective in treating fine particles (<1 mm). The capacity of deeper-field drum separators is generally higher than that of roll separators at 7–9 t/h/m (400–500 lb/h/in.) for drum separators but 1.8–5.4 t/h/m (100–300 lb/h/in.) for roll separators. The major advantage of the drum separator is its low maintenance cost, because it does not have a belt that must be replaced frequently. However, both separators have their own niche markets. Drum separators can treat coarser particles (such as garnet, ilmenite, and iron ore) at a higher throughput. Roll separators can produce high-grade, high-purity glass sand products when feed is not preheated.

Considerable research has been conducted to increase the throughput of REDs and RERs without compromising product quality. Most of these improvements have come from improved magnetic-circuit design, increased drum diameter, higher-quality nonmagnetic stainless steel, better manufacturing processes, and more compact layout design.

Case study I: Performance evaluation of RED and induced-roll magnetic separators. A series of tests compared



Courtesy of Outotec USA, Inc.

Figure 14.6-9 Industrial-scale rare-earth drum separator

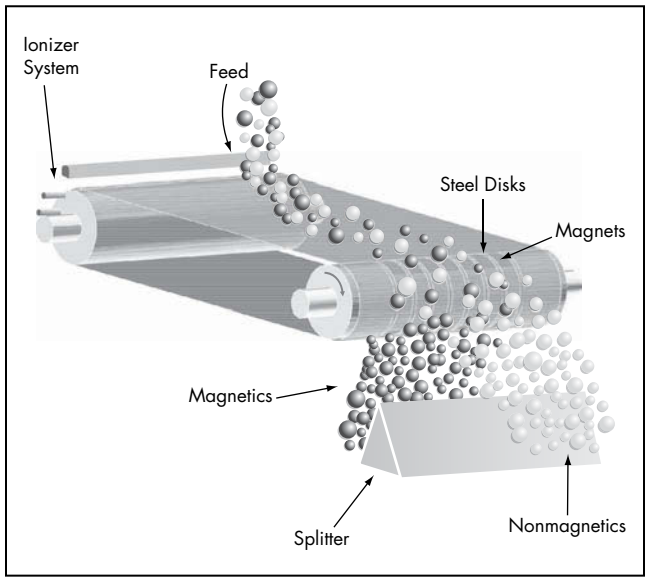


Figure 14.6-10 Operating principle of a rare-earth roll separator

the performance of a RED separator and an induced-roll magnetic separator to process a titanium-rich magnetic feed. The feed contained 76% TiO_2 and 1% Al_2O_3; it was obtained from a beach-sand processing plant. The goal was to produce high-grade nonmagnetic TiO_2 product (Table 14.6-2). Compared to the induced-roll magnetic separator, the RED separator gave product containing nearly twice the TiO_2, 45% rather than 24%, at a comparable product grade (TiO_2 grade of >90% and Al_2O_3 content of ~1%).

Case study II: Performance evaluation of RED and RER separators. A detailed study of the effect of feed rate on RED and RER separators used a garnet-rich heavy mineral sample (Figure 14.6-11). The RED separator was only minimally sensitive to feed rate. Garnet recovery decreased very marginally with increasing feed rate; it was 96.5% at a 2.6-tph feed rate and 95.1% at a 7.8-tph feed rate. In contrast, the RER separator seemed to be very sensitive to feed rate. Garnet recovery was 94.6% at a 2.6-tph feed rate, but fell to 81.3% at a 7.8-tph feed rate.

Eddy Current Separator

In the case of RED and RER magnetic separators, the outer drum or belt rotates; inside the shell, rare-earth magnets are mounted on a stator. In contrast, in an eddy current separator, not only does the nonmetallic outer drum rotate but the inside shell also rotates. The inside shell is a faster-moving rotor containing rare-earth magnets arranged in alternating polarity to induce eddy currents. An induced eddy current sets up a repulsive force in good conductors and thus separates nonferrous electrically conductive metals (such as copper and aluminum) from nonconducting materials.

Wet Permanent Magnetic Separator

A wet permanent magnetic separator (Figure 14.6-12) has permanent-magnet (NdFeB) bars positioned more or less horizontally inside a revolving drum made of stainless steel. This separator provides a field strength of 0.7 T (7,000 G) on the drum surface. The pulp tank is made of stainless steel with a concurrent or semi-countercurrent flow tank design and an adjustable discharge gap at the magnetics discharge end. The separator is equipped with an adjustable valve on the nonmagnetics discharge pipe to help control the flow rate and overflow level. This separator has found industrial applications in processing low-grade (martitic) iron ore.

SUPERCONDUCTING MAGNETS

High magnetic fields (up to 2 T or 20,000 G) are generated by passing current through a resistive coil or by permanent magnets. Development of newer computer models, through the use of finite-element-analysis techniques, has helped to achieve higher magnetic force. However, there is a logical maximum magnetic field for both the resistive coil and permanent magnet. Resistive coils are limited by the intrinsic resistance applied by the windings; the field strength of existing permanent magnets can be increased only marginally by modifying the magnet geometry. In the future, new magnetic materials may help to overcome this limitation.

Principle and Design

Currently, superconducting magnets are the only economically and technically viable way to achieve field strengths as high as 5 T (50,000 G). Most of the industrial superconducting magnetic separators designed and installed in the 1990s and 2000s include expensive cryogenic systems that use liquid helium. This cooling system is essential to help maintain the temperature of the matrix, made of superconducting niobium/titanium alloy, below ~4 K, in order to achieve field strengths as high as 5 T.

Three types of cryogenic system are commonly used:

1. **Closed-cycle liquefier system.** In a closed-cycle liquefier system, the superconductor resides in a bath of liquid helium, and boil-off gas is recirculated through a helium liquefier. Although system installation is quite complex, system performance has been good and reliable, provided that there are no long-term interruptions in the supply of electrical power and cooling water.
2. **Low-loss system.** In a low-loss system, the superconductor windings reside in a reservoir of liquid helium. A very

Table 14.6-2 TiO_2 recovery and composition of nonmagnetic product*

Type of Separator	TiO_2 Recovery, %	Product Grade TiO_2, %	Product Grade Al_2O_3, %
Induced-roll magnet separator	24.2	92	0.96
RED-7000 gauss separator	44.7	92	1.03

*Treated by an induced-roll magnetic separator (feed contained 76% TiO_2 and 1% Al_2O_3; two passes; nonmagnetic retreat) and a RED-7000 gauss drum separator.

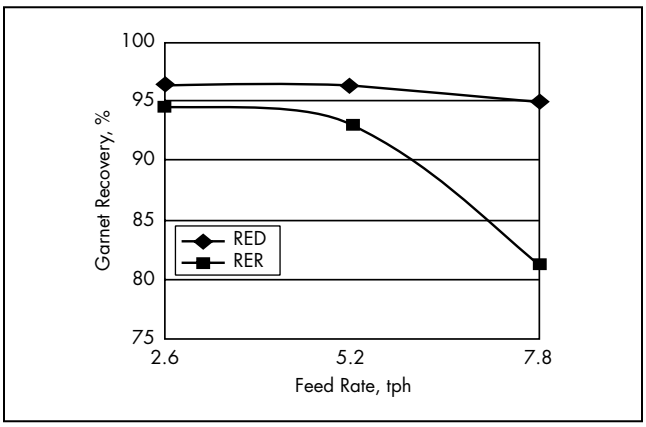


Figure 14.6-11 Performance of RED and RER separators

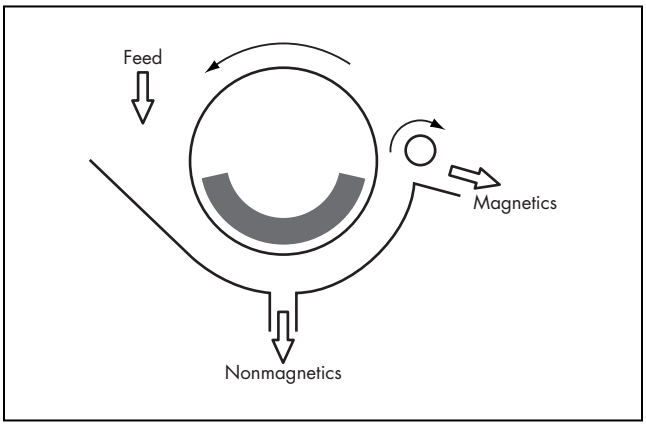
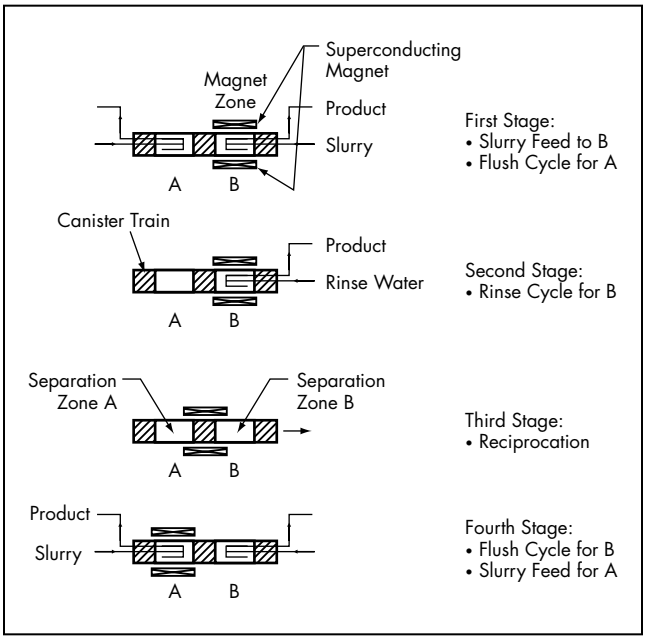


Figure 14.6-12 Operating principle of a wet permanent magnetic separator



Courtesy of Outotec USA, Inc.

Figure 14.6-13 Operating principle of a reciprocating canister superconducting magnetic separator

efficient insulation system enables the magnet to operate for long periods, typically a year or more, between liquid-helium refills. The salient feature of this system is its relative immunity to short-term electrical failures. This feature allows this technology to be used where equipment is operated under difficult conditions.

3. **Indirect-cooling system.** The advent of heat engines based on the Gifford–McMahon cycle, which generate temperatures of ≤4 K, has made it possible to cool superconducting windings without the need for liquid helium. This technique offers great potential for small-scale systems in which the economics of helium supply or the cost of a liquefier cannot be justified. However, a constant power supply is essential for reliable operation.

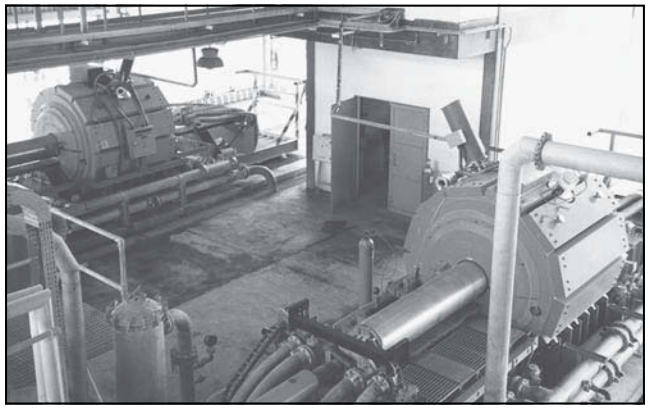
In recent years, however, developments in cryogen-free cooling technology as well as advances in materials science have spurred design of more robust superconducting magnetic separators that do not require expensive helium refills.

In summary, superconducting magnets have two main advantages:

1. Low power consumption resulting from zero resistance of the magnet winding
2. Generation of high magnetic fields

Superconducting Wet High-Gradient Magnetic Separator

In a superconducting wet high-gradient magnetic separator (HGMS), magnetic particles are captured on a stainless-steel–wool matrix contained within the bore of a high-intensity magnet (Carpco SMS Ltd. 1993; D'Assumpção et al. 1995). The high intensity is generated using a superconducting coil. Because these coils have essentially zero resistance, little electrical power is required to energize the magnet. Furthermore, after the magnet is energized, the coil ends can be shorted, leaving the magnet in a fully energized state without any additional power supply. This practice is called operating the magnets in persistent mode.



Courtesy of Outotec USA, Inc.

Figure 14.6-14 Industrial-scale reciprocating canister superconducting magnetic separator

Unloading trapped magnetic particles from the matrix is an essential step that determines the separation efficiency and capacity of the unit operation. Demagnetization is achieved either by deenergizing the magnet (a state commonly referred to as switch-mode) or by moving the matrix canister (referred to as a reciprocating canister HGMS). In reciprocating technology, captured magnetic particles are flushed using a ram to remove the trapping zone from the magnetic field regions. The ram operates on a magnetically balanced canister that houses a multisection separation region with unique and separate trapping zones (Figure 14.6-13).

Units that combine reciprocating canister technology and a low-loss cryogenic system have been used in kaolin processing throughout the world. Figure 14.6-14 shows a typical large-scale reciprocating canister HGMS.

Superconducting Dry Open-Gradient Magnetic Separator

In a conventional dry open-gradient magnetic separator (OGMS), the magnet structure is arranged to provide a region in open space with a highly divergent field. Thus, the magnet geometry supplies both the magnetic field and the field gradient. Paramagnetic material passing through this region experiences a force directly proportional to the field intensity and the magnitude of the field gradient. Compared with a conventional OGMS, a superconducting OGMS offers not only a higher magnetic force but also a deeper magnetic field, which translates to larger separation volumes.

ELECTROSTATIC SEPARATION

Almost all minerals show some degree of conductivity. The electrostatic separation process uses the difference in electrical conductivity or surface charge of the mineral species of interest. It has generally been confined to recovering valuable heavy minerals from beach-sand deposits. However, growing interest in plastic and metal recycling has opened up new applications in secondary material recovery (Knoll and Taylor 1985; Prabhu 1999).

When particles come under the influence of an electrical field, depending on their conductivity, they accumulate a charge that depends directly on the maximum achievable charge density and on the surface area of the particle. The

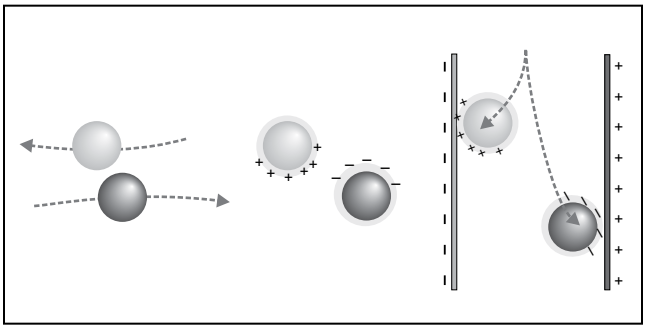


Figure 14.6-15 Work functions after particle–particle charging

charged particles can be separated by differential attraction or repulsion. Therefore, the important first step in electrostatic separation is to impart an electrostatic charge to the particles. The three main types of charging mechanism are contact electrification or triboelectrification, conductive induction, and ion bombardment. After the particles are charged, separation can be achieved by equipment with various electrode configurations.

Triboelectrification

Triboelectrification is a type of electrostatic separation in which two nonconductive mineral species acquire opposite charges by contact with each other (Alfano et al. 1988). The oppositely charged particles can then be separated under the influence of an electric field. This process uses the difference in the electronic surface structure of the particles involved. A good example is the strong negative surface charge that silica acquires when it touches carbonates and phosphates.

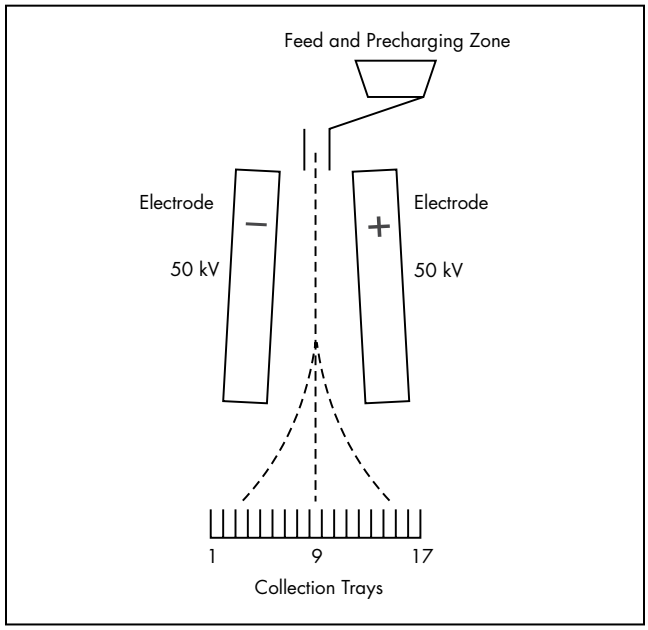
The surface phenomenon that comes into play is the *work function*, defined as the energy required to remove electrons from any surface (Figure 14.6-15). After particle–particle charging, the positively charged particle has a lower work function than does the negatively charged particle.

Over the years, several different types of triboelectrostatic separator have been designed and tested. A few tube-type and plate-type triboelectrostatic separators are now in commercial use processing minerals such as wollastonite, silica sand, and calcium carbonate. Unfortunately, in most applications, these separators suffer from low throughput and poor separation efficiency. The main challenge has been the inability to help particles acquire and retain charge for a desired period of time during the separation process.

Conductive Induction

When uncharged particles, whether conductors or nonconductors, contact a charged surface, the particles assume polarity and the potential of the surface. Electrically conductive minerals do so rapidly. However, in the case of nonconductors, the side that faces away from the charged surface is slower to acquire the same polarity as the surface. Hence, if both conductor and nonconductor particles are just separated from contact with a charged plate (Figure 14.6-16), the conductor particles are repelled by the charged plate, and the nonconductor particles remain unaffected by the charged plate—they are neither attracted nor repelled.

The most common industrial separators working on this principle are plate- and screen-type separators. Feed falls under gravity onto an inclined, grounded plate and into an



Courtesy of Integrated Mineral Technology, Ltd.

Figure 14.6-16 Operating principle of a plate-type electrostatic separator

electrostatic field induced by a high-voltage electrode. These electrodes are generally oval. Conductor particles acquire an induced charge from the grounded plate and move toward the oppositely charged electrode; that is, the particles experience a "lifting effect." Nonconductor particles are generally not affected by this field. Because the lifting effect depends on surface charge as well as particle mass, fine conductors are effectively separated from coarse nonconductors.

Ion Bombardment

When conductor and nonconductor particles placed on a grounded conducting surface are bombarded with ions of atmospheric gases generated by an electrical corona discharge from a high-voltage electrode, both conductor and nonconductor particles acquire a charge. When ion bombardment ceases, conductor particles rapidly lose their acquired charge to the grounded surface. Nonconductor particles react differently; the side that faces away from the grounded conducting plate is coated with ions of charge opposite in electrical polarity to that of the grounded conducting plate. Therefore, nonconductor particles remain pinned to the grounded plate because of electrostatic force (Figure 14.6-17).

An industrial high-tension electrostatic separator that uses the pinning effect to process plastics and metal scrap is shown in Figure 14.6-18. This separator consists of a rotating roll made from mild steel that is grounded through its supporting bearing. The electrode assembly consists mainly of two types of electrodes: a beam or corona electrode and a static-type electrode. The beam electrode, usually connected to a DC supply of ≤50 kV negative polarity, charges all particles and pin nonconductors to the roll. Feed is presented uniformly to the rotating roll surface by a velocity feed system. Both conductor and nonconductor particles are sprayed with ions. Conductor particles rapidly lose their charge to the grounded roll surface and are thrown off by centrifugal force. Nonconductor particles are pinned to and then brushed off the rolled surface.

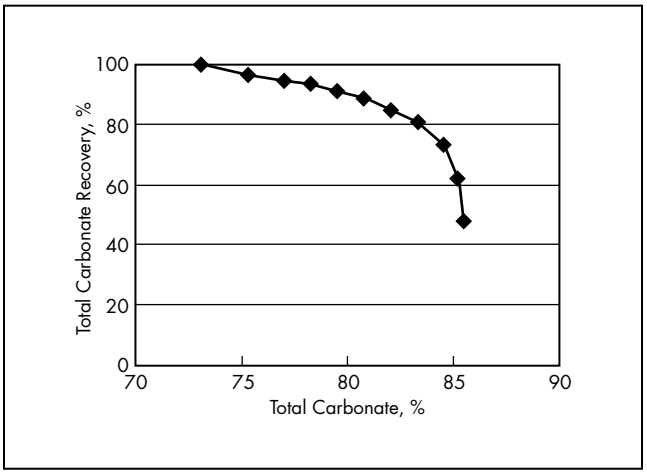


Figure 14.6-17 Operating principle of a roll-type electrostatic separator



Courtesy of Outotec USA, Inc.

Figure 14.6-18 Industrial roll-type electrostatic separator

Both conductor and nonconductor particles are collected in a partitioned product hopper at the bottom of the unit. The operating variables—roll speed, applied voltage, feed rate, splitter position, and electrode combination and position—can be adjusted to achieve effective separation.

After a long hiatus, there has recently been some significant development in designing new and efficient high-tension separators. OreKinetics has taken the lead and developed a new generation of high-tension roll and plate separators that have additional well-insulated static electrodes. This new design has helped improve separation efficiency and reduce the number of reprocessing steps.

ACKNOWLEDGMENTS

This chapter has been revised and updated from Chapter 7 in *Principles of Mineral Processing* (Venkatraman et al. 2003). Thanks go to previous coauthors Frank S. Knoll (retired) and James E. Lawver (deceased) for their contributions to that earlier chapter.

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CHAPTER 14.7

Dewatering Methods

Donald A. Dahlstrom

INTRODUCTION

Liquid–solid separation is a relatively slow-moving field in terms of technological change. Improvements in this process have been evolutionary rather than revolutionary. Water is used to process the majority (between 80% and 90%) of the minerals and coal mined. (A small and decreasing percentage of coal is crushed and screened dry, and industrial minerals, such as diatomaceous earth, bentonite, cement raw materials and clinker, and base metal ores prior to roasting are often crushed and ground to a fine powder using dry processes.) Beneficiation processes usually use water because it allows for greater efficiency, higher recovery, and lower cost per unit of valuable product. In addition, it eliminates air pollution.

Where capital and operating costs are mentioned in this chapter, they should be regarded as order-of-magnitude only and not definitive.

Costs of Liquid–Solid Separation

In general, when considering the need to separate solids from water, as the particles to be separated decrease in size, the cost increases and the capacity per unit area decreases. When the solids are colloids (generally considered to be –10 μm), costs increase even faster. They are difficult to remove by filtration or centrifugation. Usually, a flocculant is added to the mixture to cause the colloids to form large flocculi or agglomerates. Accordingly, liquid–solid separation is a major cost in mineral processing, probably exceeded only by the cost of comminution, flotation, and endothermic reactions.

For example, the capital cost of a coal preparation plant increases by about 30% to 40% if the –28 mesh coal fraction is processed instead of discarded and the process water is recovered (by liquid–solid separation) for recycle and reuse. Operating costs per ton of –28 mesh coal also increase substantially when compared to coarser coals. These costs are due primarily to the use of flotation and the liquid–solid separation steps involved.

At the same time, liquid–solid separation by mechanical means (i.e., sedimentation, filtration, and centrifugation) is much less costly than thermal drying, primarily because those means consume less energy per unit of water removed.

To illustrate some highly efficient liquid–solid separations, consider the following example. A 30-m-diameter conventional gravitational thickener (at a normal design rate for tailings concentration and water reclamation of 0.1 m^2/t of dry solids per day) will process more than 2,400 t (metric tons) of solids per day. With a feed of 15% (by weight) solids, an underflow concentration of 50% (by weight) solids or higher can usually be achieved if the solids contain 50% to 55% particles that are –200 mesh or coarser. This size means that 4.67 kg of water per kilogram of solids has been eliminated and that more than 82% of the water will report to the thickener overflow for reuse. The thickener drive head will be equipped with only a 3.7- or 5.6-kW motor.

In the processing of magnetite concentrates derived from the beneficiation of taconite, disk filters are used to dewater magnetite concentrates before the balling step. For a 1,800 cm^2/g Blaine concentrate (approximately 85% to 90% –325 mesh), a filtration rate of 1,100 kg of dry solids/h/m^2 is used as a design basis (Wolf et al. 1971). The feed concentration would be maintained at 65% (by weight), and the vacuum level should be at 60 cm of mercury by using a vacuum pump capacity of about 1.8 m^3/m^2 of filtration area. Power consumption will be very high because of the high vacuum and flow rate. Considering also the filter drive, compressed air requirements, filtrate pump, and agitator, a total of 26 kW would be required for 10 m^2 of filtration area. This power requirement is equivalent to 2.5 cal/m^2 of filtration area. However, 100 kg of dry magnetite solids per hour are dewatered to the 9% (by weight) moisture required for hauling. At the same time, 4,600 kg of water are extracted. Thus, only 25 kcal/kg are eliminated.

Thermal drying, on the other hand, requires around 4,900 kcal/kg of water evaporated. Obviously, mechanical methods of dewatering have relatively low energy consumption per unit of liquid removed.

Liquid–solid separation is also critical in water discharge, water reclamation and closed water circuits (Wolf et al. 1971). Water that has been used for processing and beneficiating minerals and coals will contain solids that can range in size from a fraction of a micrometer to 6 mm or more. Some streams will

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contain the valuable solids and others the refuse or tailings. In both cases, the solids must be separated out if the water is to be reused. Furthermore, the concentration of suspended solids in recycled water must be low enough so that the water does not contaminate the next product. In the case of iron ore processing, the return water must contain between 100 and 150 mg/L or less to minimize the percentage of silica in the final pellet. Coal requires a suspended solids concentration of less than 1% (by weight).

Water discharge requirements often necessitate even better solids–liquids separation than process requirements. In the United States, state and federal regulations generally require that effluent disposed in lakes, streams, or other public water sources contain no more than 10 to 50 mg/L of suspended solids.

Steps in Liquid–Solid Separation

Separating liquid and solids requires many steps, and these are treated extensively in the literature (e.g., Fuerstenau and Han 2003; Wills and Napier-Munn 2006).

In coal processing, centrifuges and screens are usually used. For minerals beneficiation, ore is usually ground much finer than is practiced for coal processing, often to 80% passing 75 μm or even 80% passing 44 μm (Henderson et al. 1957; Kobler and Dahlstrom 1979). Thus, base metals and iron ore concentrates, which are large-tonnage minerals, will usually be rated according to the percentage of –325 mesh or even –400 mesh solids. Because of their abrasive character, these and other minerals will be dewatered in thickeners and then on filters. Both continuous filters and centrifuges are used with crystalline solids such as potash sulfate. Gravity thickeners are used for both concentrate and tailings (Coe and Clevenger 1916; Roberts 1949; Terchick et al. 1975). Tailings are generally sent to tailings ponds without solids–liquids separation, as their tonnage and volume can be very large. However, more stringent regulation of the construction of tailings ponds ensures that mechanical dewatering methods will be increasingly used in the future (Chironis 1976).

Hydrometallurgical processing always creates abundant colloidal solids during the leach step. Proper liquid–solid separation enables the recovery of the maximum amount of pregnant liquor while minimizing its dilution. Thus, multistage countercurrent sedimentation, countercurrent washing filtration on a single filter, or two- or three-stage filtration with single washes per stage are practiced. Both continuous vacuum filters and pressure filters are involved (Nelson and Dahlstrom 1957; Osborne 1975; Kobler and Dahlstrom 1979).

Current practice at many locations requires "zero discharge," or 100% recycling of water streams. If solids are disposed of in a landfill, the discharged solids must be low enough in moisture (generally 80% solids or more) so that liquids do not leach out by gravity. Thus, in a modern mineral processing plant, efficient liquid–solid separation processes are critical to low-cost processing, and these processes can substantially affect capital and operating costs.

MAJOR INFLUENCES ON LIQUID–SOLID SEPARATION

Many factors external to the liquid–solid separation equipment itself influence its performance and productivity. The most common of these are as follows (Cross 1963; Robins 1964; Scott 1970; Bosley 1974; Silverblatt et al. 1974; Wetzel 1974; Hsia and Reinmiller 1977; Weber 1977; Dahlstrom 1978; Rushton 1978; Sakiadis 1984; Baczek et al. 1997; Lewis 2007), and of which only the first factor will be discussed in detail in this chapter:

- Particle size and shape
- Weight and volume percentage of solids
- Fluid viscosity and temperature
- pH and chemical composition of the feed
- Variation and range in feed quality (i.e., of the first four items)
- Specific gravity of solids and liquid
- Quality requirements of discharge streams from liquid–solid separation steps, particularly as they influence results upstream and downstream

Particle Size and Shape

Size distribution greatly affects liquid–solid separation rates. Stokes' law can be used to illustrate this fact. This law permits the terminal settling velocity (maximum velocity achieved during free fall) to be determined as follows:

$$\nu_t = \frac{(\rho_s - \rho) g D^2}{18\mu}$$

where

ν_t = terminal velocity, m/s
ρ_s = particle density, kg/m^3
ρ = liquid density, kg/m^3
g = gravitational acceleration = 9.806 m/s^2 at sea level at 45° latitude. This value is used for g, the standard gravitational acceleration on this planet, and is usually symbolized by g_c.
D = particle diameter, m
μ = fluid viscosity, kg/m × s

Viscosity is normally measured in centipoise. One poise = 1 g·cm^{-1}·s^{-1}. (All nomenclature is metric [International System of Units], but English units can be used as long as they are used consistently.)

This equation has some limitations. It assumes laminar flow (i.e., a Reynolds number of less than 0.1) and nonhindered settling. In nonhindered settling, a particle is not influenced by the presence of other particles or by the slurry's specific gravity, conditions that require very dilute slurries.

If one assumes a 20-μm particle and an 80-μm particle of the same density and in the same fluid, the terminal velocity is found from the equation to be directly proportional to the square of the diameter. In this case, the 80-μm particle will fall 16 times as fast as the 20-μm particle (see Table 14.7-1).

Unfortunately, mineral slurries contain a variety of particle sizes of different densities and agglomerating behaviors, so the application of the equations is usually difficult. The size distribution of particles in the slurry also has a major effect, so a simple characterization, such as 80% –200 mesh, is not very useful when designing equipment. Percentage of clay-like particles and type of clay are very important variables and can change throughout a mineral deposit. For most heavy metal mineral processing operations, the clay content is usually within manageable limits. For these plants, discharge streams from conventional thickeners are typically 55% to 60% solids, discharge steams from filters are typically 80% to 83% solids (dry enough so it can be stacked with a bulldozer), and discharge streams from high-density thickeners

Table 14.7-1 Drag coefficient and related functions for spherical particles

Reynolds Number, N_{Re}*	Drag Coefficient, C	CN_{Re}^2	C/N_{Re}
0.1	244	2.44	2,440
0.2	124	4.96	620
0.3	83.8	7.54	279
0.5	51.5	12.9	103
0.7	37.6	18.4	53.8
1	27.2	27.2	27.2
2	14.8	59.0	7.38
3	10.5	94.7	3.51
5	7.03	176	1.406
7	5.48	268	0.782
10	4.26	426	0.426
20	2.72	(1.09) (10^3)	0.136
30	2.12	(1.91) (10^3)	0.0707
50	1.57	(3.94) (10^3)	0.0315
70	1.31	(6.42) (10^3)	0.0187
100	1.09	(1.09) (10^4)	0.0109
200	0.776	(3.10) (10^4)	(3.88) (10^{-3})
300	0.653	(5.87) (10^4)	(2.18) (10^{-3})
500	0.555	(1.39) (10^5)	(1.11) (10^{-3})
700	0.508	(2A9) (10^5)	(7.26) (10^{-4})
(1 × 10^3)	0.471	(4.71) (10^5)	(4.71) (10^{-4})
(2 × 10^3)	0.421	(1.68) (10^6)	(2.11) (10^{-4})
(3 × 10^3)	0.400	(3.60) (10^6)	(1.33) (10^{-4})
(5 × 10^3)	0.387	(9.68) (10^6)	(7.75) (10^{-5})
(7 × 10^3)	0.390	(1.91) (10^7)	(5.57) (10^{-5})
(1 × 10^4)	0.405	(4.05) (10^7)	(4.05) (10^{-5})
(2 × 10^4)	0.442	(1.77) (10^8)	(2.21) (10^{-5})
(3 × 10^4)	0.456	(4.10) (10^8)	(1.52) (10^{-5})
(5 × 10^4)	0.474	(1.19) (10^9)	(9.48) (10^{-6})
(7 × 10^4)	0.491	(2.41) (10^9)	(7.02) (10^{-6})
(1 × 10^5)	0.502	(5.02) (10^9)	(5.02) (10^{-6})
(2 × 10^5)	0.498	(1.99) (10^{10})	(2.49) (10^{-6})
(3 × 10^5)	0.481	(4.33) (10^{10})	(1.60) (10^{-6})
(3.5 × 10^5)	0.396	(4.86) (10^{10})	(1.13) (10^{-6})
(3.75 × 10^5)	0.238	(3.34) (10^{10})	(6.34) (10^{-7})
(4 × 10^5)	0.0891	(1.43) (10^{10})	(2.23) (10^{-7})
(4.25 × 10^5)	0.0728	(1.32) (10^{10})	(1.71) (10^{-7})
(4.5 × 10^5)	0.0753	(1.53) (10^{10})	(1.67) (10^{-7})
(5 × 10^5)	0.0799	(2.00) (10^{10})	(1.60) (10^{-7})
(7 × 10^5)	0.0945	(4.63) (10^{10})	(1.35) (10^{-7})
(1 × 10^6)	0.110	(1.10) (10^{10})	(1.10) (10^{-7})
(2 × 10^6)	0.150	(6.00) (10^{11})	(7.50) (10^{-8})
(3 × 10^6)	0.163	(l.47) (10^{12})	(5.44) (10^{-8})
(5 × 10^6)	0.174	(4.35) (10^{12})	(3.48) (10^{-8})
(7 × 10^6)	0.179	(8.75) (10^{12})	(2.55) (10^{-8})
(1 × 10^7)	0.182	(1.82) (10^{13})	(1.82) (10^{-8})

Source: Perry et al. 1984, with permission from The McGraw-Hill Companies, Inc.
*For values of Re less than 0.1, $C = 24/Re$.

are typically 68% to 73% solids (a thick paste but of pumpable consistency).

The designs for thickeners and filtration equipment are usually decided in a program of laboratory testing, and a large safety factor is included when designing equipment. In general, even very finely ground mineral slurry will respond to dewatering if the correct equipment is employed. Although equipment suppliers should be contacted early in the design process, it is important that the client take charge of this process (rather than leaving it up to equipment suppliers), because the results can be interpreted in a variety of ways, which might favor some equipment over others.

LIQUID–SOLID SEPARATION EQUIPMENT

Commonly used equipment for liquid–solid separation in the minerals processing industry can be categorized into the following unit operations: screens, cyclones, thickeners and clarifiers, filters, centrifuges, and tailings ponds.

Screens

Screens are often the first liquid–solid equipment encountered by the ore stream as it enters the processing plant. Screens serve the dual purpose of separating the solids into different size fractions and separating solids from liquids. Typical applications for the latter use are "trash screens," which remove nonmineral particles such as wood chips from the mineral slurry. While this is a somewhat mundane application, it can be critically important to the functioning of downstream equipment, so a trash screen is usually present following the grinding stage in most mineral processing plants. This screen is usually an inclined-deck vibrating screen. In the gold-processing industry, many plants employ coarse carbon particles (1,600 μm, 10 mesh) to adsorb dissolved gold from 75-μm ore slurry. The carbon particles are recovered from the slurry on a variety of screen types, typically inclined-deck vibrating screens (commonly called Dutch State Mines screens or DSM screens), which employ parallel bars/wires at right angles to the flow and a steeply dipping nonvibrated deck (Fuerstenau and Han 2003).

In 2005, vibrating screens with very fine screen decks came into limited use to make a very fine separation, typically at 200 μm (65 mesh), of the product fines fraction from slurry, which is recirculated to the grinding mills. Formerly this job was always done using cyclones or spiral classifiers.

Cyclones

Cyclones are typically configured as a straight-sided cylinder connected to a tapered pipe section. The ore slurry stream is introduced tangentially into the cylindrical section, and it rapidly spins to separate the solids from the liquids. Typically, the dimensions are 25 cm in diameter and 100 cm long, but they can be much larger or smaller. Cyclones are commonly used as size separation devices to separate the fine product slurry from the coarse recycle slurry in the grinding section of mineral processing plants. Size separation is not the subject of this chapter, but the cyclones also serve the dual purpose of making a dense underflow (high solids content, coarse material) from a dilute overflow. This effect is utilized by dewatering cyclones, which are used to separate dense solids slurries to form a nearly clear liquid overflow. In this application they substitute for thickeners.

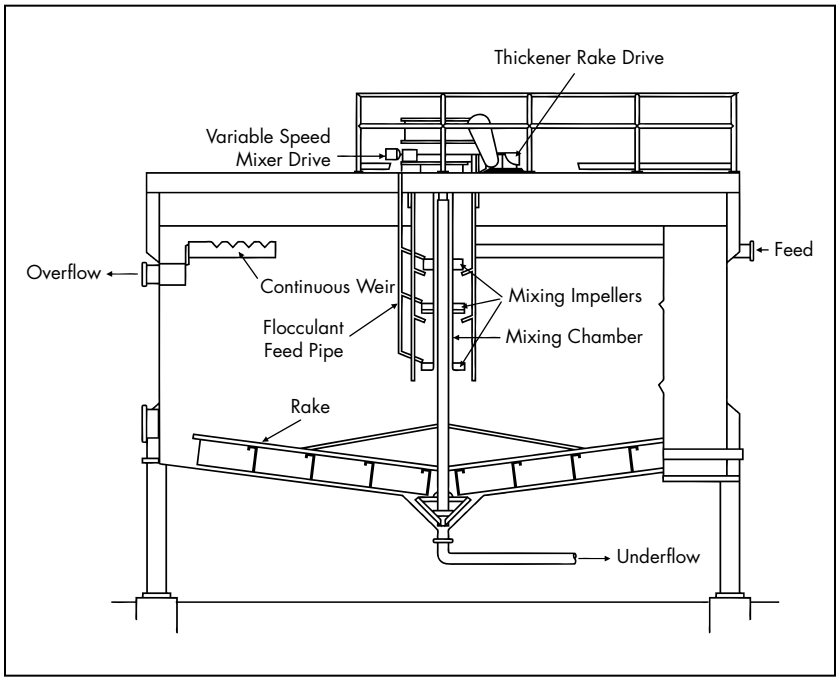


Figure 14.7-1 High-density thickener

Thickeners and Clarifiers

While cyclones employ centrifugal force, which can be many times the force of gravity, thickeners and clarifiers use gravity only to allow settling of mineral particles and separation of a dense slurry, which is pumped from the bottom of the thickener below a clear or nearly clear overflow liquid. Most mineral processing plants employ thickeners at multiple points in the process. Thickeners can range from 3 m to more than 100 m in diameter. Thickeners with diameters in the range of 20 to 50 m are applicable to settle the process stream in medium sized mineral processing plants treating 1,000 to 5,000 t of ore per day.

Where colloidal solids are present (which is usually the case), flocculants will improve operation of thickeners by causing the colloids to form agglomerates or flocculi of much larger size (which still contain probably 95% or more of liquid) that will settle at reasonable rates. In a typical mineral processing plant, flocculants are employed at multiple points, and the total reagent cost is between $1.00 and $5.00/t of mineral processed.

There are three basic types of thickeners: conventional, high-rate, and high-density. They all consist of a large, round, relatively shallow tank with a bottom that slopes toward the center, and they are equipped with a slowly rotating rake mechanism that moves the settled solids toward the central discharge. Overflow liquid is collected in a peripheral launder (trough) and is directed to a pipe outlet. Conventional thickeners can be very large and are sometimes run without the addition of flocculant. Capital cost is relatively high, but operating costs are low. High-rate thickeners can be much smaller in diameter than conventional thickeners for the same duty, and they rely on recycled slurry plus high usage of flocculants. Both conventional and high-rate thickeners discharge solids of the same density, usually 45% to 60% solids.

High-density thickeners have much deeper tanks with steep cone bottoms, and they rely on the auto-compression of the cake to increase density of the discharge solids up to 70% or more (see Figure 14.7-1). At this percent solids, the discharge is a thick paste solids that can be pumped only with special pumps. High-density thickeners are useful where water must be conserved or where a dense paste is needed for other applications such as mine backfill.

Although the overflow of a thickener must usually be of reasonable clarity, the emphasis is primarily on underflow concentration. Clarifiers are thickeners that operate on a dilute solids stream (often the discharge from thickeners) in which the flocculated stream is introduced below a blanket of flocs to capture all the suspended solids into the settled bed at the bottom of the clarifier. Clarifiers may have different internal configurations than the simple circular-rake thickener. For example, clarifiers may be rectangular tanks filled with submerged inclined plates (lamella thickeners), or they may be inclined tubes that reduce the settling distance of the particles to the vertical distance between plates.

The term countercurrent decantation (CCD) thickeners refers to the use of three to ten thickeners operated in a continuous train, in which slurry containing a process solution enters the first thickener and water enters the last thickener. Solids are pumped from the underflow of the first thickener into the next downstream thickener and so forth to the last thickener.

Solution is pumped from the overflow of the final thickener to the next upstream thickener, and it is mixed with the solids as they enter the thickener from the upstream direction. In this manner the solids slurry is continuously mixed and then settled, and the process solution (containing chemicals or dissolved products to be recovered) is gradually displaced by water. A five-stage thickener stream can often recover >95% of the dissolved values from the slurry using a volume of clean

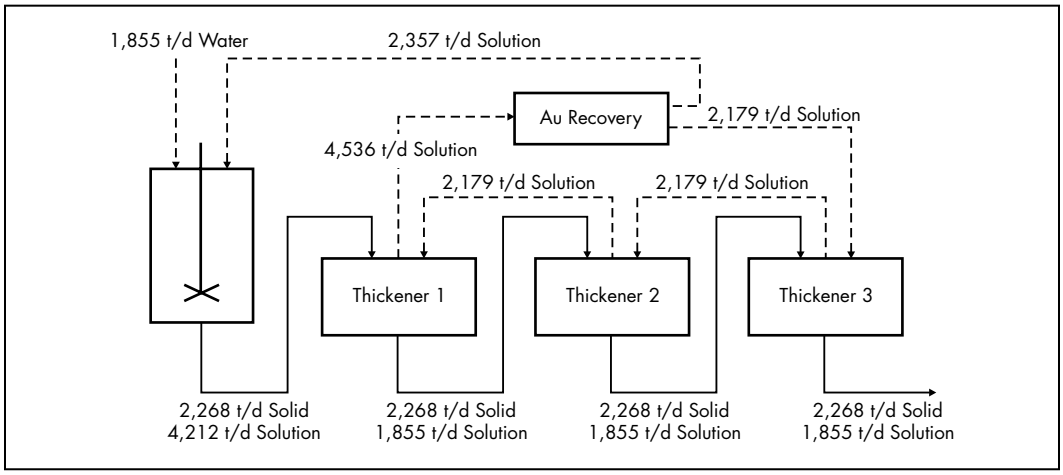
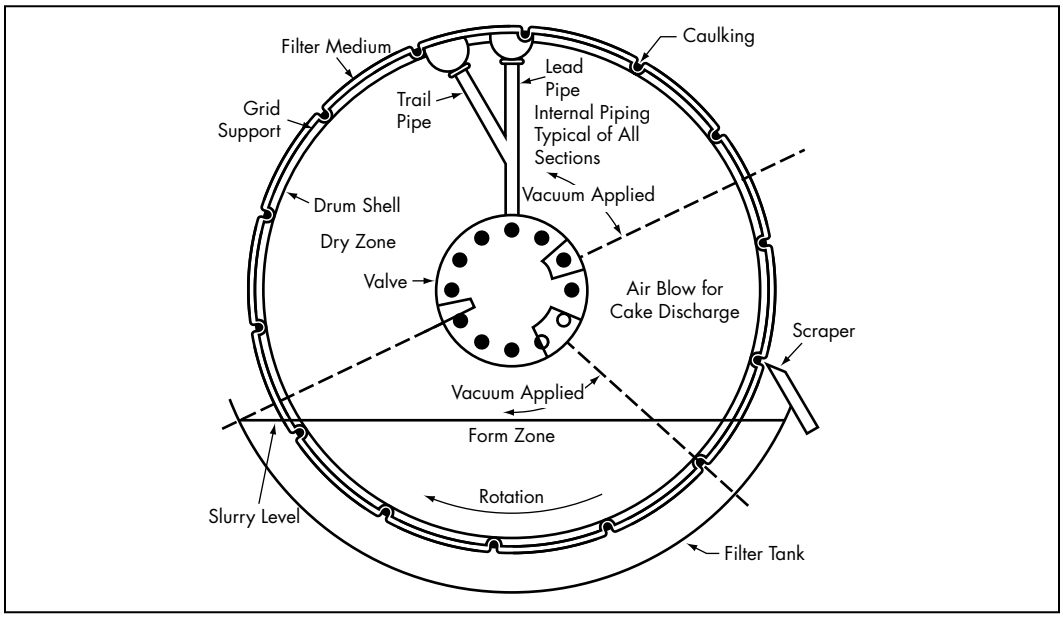


Figure 14.7-2 Three-stage countercurrent decantation circuit for gold ore



Source: Schweitzer 1979, with permission from The McGraw-Hill Companies, Inc.

Figure 14.7-3 Rotary drum vacuum filter

water equal to the volume of process solution entering with the slurry. CCD thickeners are the normal final stage in leach plants that chemically leach metals such as gold/silver, copper, nickel, or aluminum for their ores (see Figure 14.7-2).

Filters

Filtration can be divided into three modes: continuous, batch and semicontinuous, and clarifying. Each of these modes can be further subdivided. Ther are many books devoted to filtration and selection of filtration equipment, and the subject is discussed here in a very limited manner.

Vacuum Filters

Continuous vacuum filters can be rotating discs or drums, or traveling belts (Figure 14.7-3). They are usually used for dewatering mineral concentrates or for dewatering large volume, easy-to-filter materials such as the cleaned coal product of a coal washing plant. Traveling belt filters up to 6 m wide and 30 m long are employed for high-volume or slowly-filtering slurries. Large drum filters (with the filtration surface being the exposed outer surface of the drum) have been employed for the separation of solids slurry from process liquids, although this application has largely been replaced by thickeners.

Low-Tonnage Pressure Filters

Pressure filters are commonly employed in mineral processing plants to remove relatively low-tonnage, dilute solids from process streams. Although there are many types of pressure filters, in mineral processing plants these typically take the form of vertical-leaf filters with multiple parallel vertical leaves in a pressurized tank-like housing, or plate and frame filters with multiple plates sandwiched within a compression framework (Figure 14.7-4).

Typical applications are for achieving high clarifying process streams, where the prefiltration solids density is a few

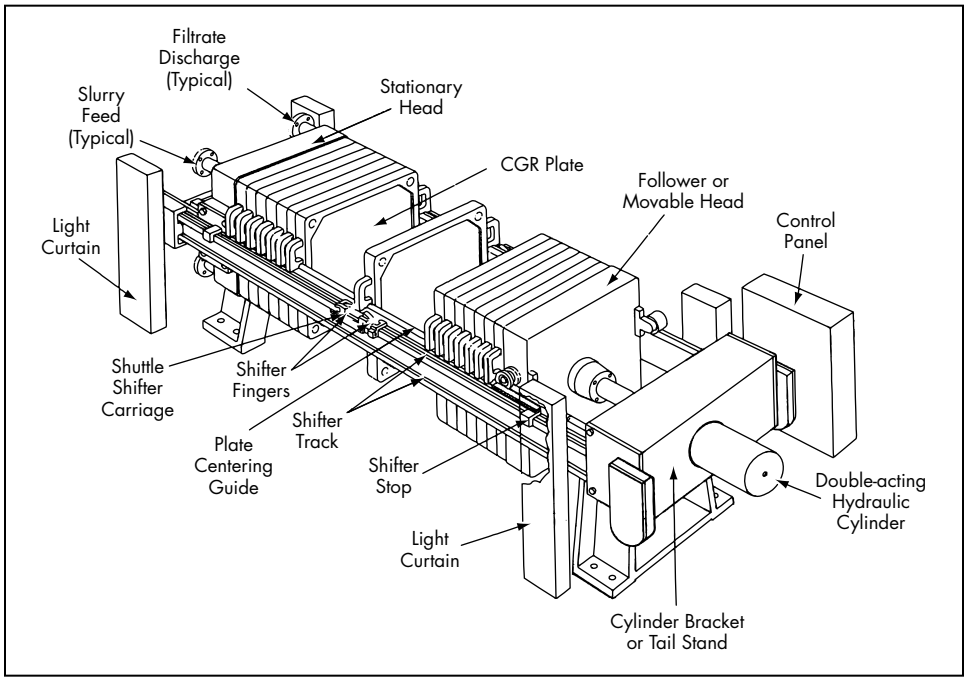


Figure 14.7-4 Plate and frame filter

hundred ppm up to 1%, and for the recovery of precipitated products such as gold and silver after these are precipitated in fine suspended solids form after the addition of zinc dust to a cyanide leach solution.

Clarifying filters typically have multiple parallel leaves consisting of a framework covered with filter cloth, and are usually precoated with diatomite to improve filtration capacity. Plate and frame filters for these low-solids applications are usually opened and closed, and cleaned manually.

High-Tonnage (Continuous) Pressure Filters

Continuous (actually semicontinuous) recessed plate filters are widely used to produce a dry tailings from a dilute slurry in high-tonnage process plants. A typical filter might contain 60 plates, each with a filter surface 1 × 1.5 m on each side of the plate. Such a filter might be able to process 1,000 t/d of slurry in a 12-minute cycle that includes 3 minutes of cake formation and other intervals for washing and blowing. The capital cost of a 1,000 t/d filter is around $2 million, but the final installed cost is around $5 million because the fitler is supported by a complex arrangement of pumps, piping, tanks, air compressors, and discharge conveyors.

Where a conventional tailings pond can be employed (in which the ore slurry is sent to the pond and clear solution is recovered), this is always preferable to tailings filtration, but in many cases topography or environmental considerations make tailings filtration necessary (see Table 14.7-2).

Centrifuges

Centrifuges are used with less-abrasive solids such as fine coal and crystalline solids. They can also beneficiate coal as the concentrate solids are usually appreciably higher in ash content than the feed.

Centrifuges, as their name implies, use a centrifugal force to substantially increase the settling rate of particles. This centrifugal force, created by pumping the feed into a cone-shaped unit, forces the feed to separate into two discharges: a solid-rich fraction and an aqueous fraction. There are no moving parts. However, a choke nozzle on the solids discharge allows adjustment and fine-tuning.

For mineral dewatering, the solid-bowl scroll centrifuge is frequently employed because it continuously discharges solids. The unit consists of a revolving horizontal cone-shaped casing inside which a screw conveyor of similar internal diameter revolves in the same direction but at a slower speed. The feed enters through a central tube of the screw conveyor from where it is subjected to high centrifical forces that cause the solids to settle on the inside of the casing where a screw conveyor "scrapes" and transports them to the discharge port located at the tapered end of the unit. Bowl diameters are usually from 15 to 150 cm. They have a throughput of 0.5 to 50 m^3/h of liquid and 0.25 to 100 t/h of solids with the moisture content of the discharge varying from 5% to 20% (Wills and Napier-Munn 2006).

Tailings Ponds

Tailings ponds are widely employed at mineral processing plants. Mineral slurries are pumped to the ponds, the solids settle and consolidate to form a stable mass, and the clear liquid is recycled to the plant or discharged to the environment. For chemical containment, tailings ponds may be lined with plastic liners and leak detectors, but where the process involves only nonchemical processing (physical methods such as gravity concentration or flotation), the tailings pond may be a simple excavated area with a compacted soil base. The fine solids in the tailings usually serve as an effective seal to prevent most escapes of process water.

Tailings pond management and construction (which is a continuous process as more tailings are created) may be a significant part (5% to 8%) of overall plant operating cost.

Table 14.7-2 Typical equipment factors for continuous vacuum filters (standard designs)

Filter Type	Submergence, %*		Total % Under Vacuum	Maximum % Cake Wash	Maximum % Dewatering Only	% Required for Cake Discharge	Minimum Cake Thickness, mm
	Apparent	Effective					
Filters Forming Cake Against Gravity							
Drum filters							
Scraper	35	30	80	29	50–60	20	6.4
Roll 1	35	30	80	29	50–60	20	0.8
Belt	35	30	75	29	45–55	25	3.2–4.8
Precoat	35	35	100	29	50	0	0
Disk filters	35	28	75	NA†	45–50	25	9.5–12.7
Filters Forming Cake with Gravity							
Horizontal belt as required			100*		As required	0	3.2–4.8
Horizontal table as required			80		As required	20	19.1

*Horizontal belt filter is based only on effective area (area under vacuum).
†NA = not available.

Where zero discharge is practiced, excess solution may be evaporated from the tailings pond using natural solar evaporation or various types of evaporative sprinkler. Evaporators of the "turbo-mist" variety blow a stream of fine droplets high into the air after mixing the solution with compressed air to form the fine droplets. A typical equipment unit of this type might evaporate 2,000 m^3 of water per month while consuming 45 kW. Capital cost is of the order of less than $150,000. Multiple units can be installed. The heat needed for evaporation can come from simple solar heating of the pond surface, but a larger contributor is the latent heat contained in the moving air mass above the tailings pond.

SUMMARY

Liquid–solid separation and the related problem of water balance is one of the major design issues for any mineral processing plant. Regardless of the mineral to be treated or the process employed, mistakes or design uncertainties in this area are one of the most common reasons for slow plant startup or underperformance. This chapter has intended to identify some of the more common design practices, but a very broad body of published literature, supplier brochures, and Internet information exists on the subject of flow-sheet design and liquid–solid separation. After a liquid–solid application is identified, the engineer should invest substantial time in quantifying the design factors and limitations, and in researching the alternatives, before making the selection of specific equipment.

ACKNOWLEDGMENTS

This chapter is based on the "Liquid–Solid Separation" chapter by the same author in *Principles of Mineral Processing*, published by SME.

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PART 15

Health and Safety

CHAPTER 15.1

Mine Safety

David Laurence

The modern mine site is a different place than that which existed in 1989. Unlike the majority of mines that operated throughout the world at that time, many mines are no longer residential operations; instead they are staffed by a "fly-in, fly-out" work force. Workers at most mines are likely to be employed by contractors rather than the mining company itself and may not belong to a labor union. Many workers joined the industry during the boom times from 2004 and lack experience. All of these factors add to the challenges faced by mine managers to provide a safe place of work for all those employed on a mine site.

Even casual visitors to a mine are required to comply with the stringent safety requirements and the legislative responsibilities placed on mine management to ensure the health and safety of all people. The measures include

- Security procedures at the gate or entrance to the mining lease,
- Random or compulsory drugs and alcohol testing (many mines now have a zero-alcohol limit; many have no tolerance to any drugs, including analgesics),
- A safety induction program (for a contractor or employee, this can take a week to complete), and
- Continuous supervision by an employee.

One of the most significant issues for mine managers is achieving a safe working environment at their mine. "Zero harm" is the phrase that sums up the mission of many mining companies, contractors, and their employees. It is likely to be driven by one or all of the following points:

- The community no longer accepts that injuries are a necessary part of the mining process.
- If a fatality occurs at a mine site, there may be community angst, which grows to outrage if multiple fatalities occur.
- Politicians may act contrary to the mine's best interest but in their best interest.
- Regulator enforcement has strengthened, implying that
  - Mines can be shut down or
  - Managers and directors can be imprisoned or fined.
- The loss in human and financial terms can be immense.
- Remuneration and career progression are tied to safety performance.

Safety is one of the dimensions of sustainable mining practices, along with the other important dimensions of economy, environment, community, and efficiency (Figure 15.1-1). Three of these dimensions—economy, environment, and community—represent the pillars of sustainable development. In mining it is also important to consider safety and efficiency. Efficiency relates directly to the resource and encompasses optimization, technical excellence, nurturing the resource, and so forth (Laurence 2008).

Safety in the mining industry must encompass the full cycle of exploration, development, construction, operations, rehabilitation, closure, and final walkaway. Of course, associated activities such as mineral processing, smelting, and transport are all part of the industry, and excellence in safety needs to apply there as well.

In discussions of mine safety, the terms *dangerous* and *accidents* are not specific enough to engender clarity. Rather than using the word *dangerous*, the term *hazardous* is more meaningful when describing the working environment of mine sites. Hazards are present in the mining industry, but these can be identified and the resultant risks managed. The use of risk management techniques has been universally adopted in the

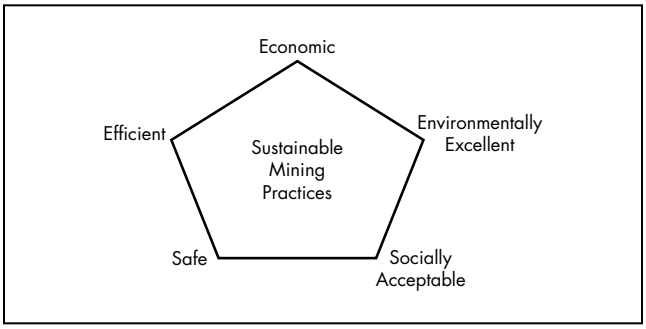


Figure 15.1-1 Sustainable mining practices

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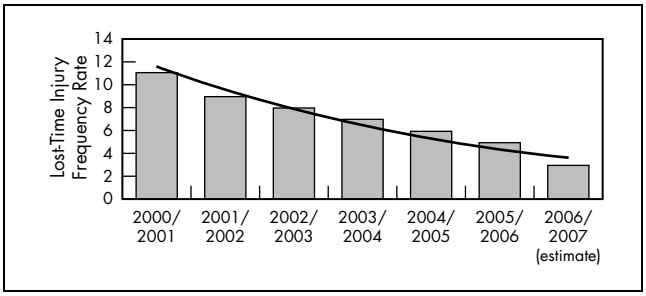


Figure 15.1-2 Australian mining industry lost-time injury frequency rate, 2000–2007

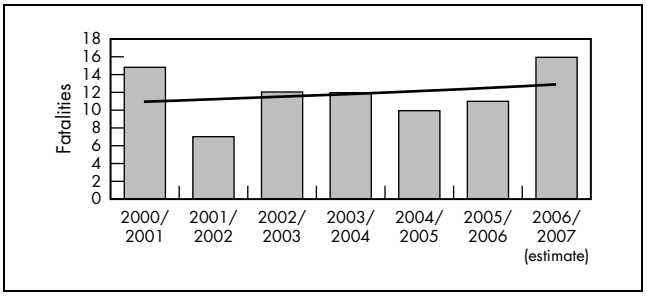


Figure 15.1-3 Mining industry fatalities in Australia, 2000–2007

Australian mining industry, and it has achieved a significant reduction in lost-time injuries and other safety performance measures. In a similar vein, the word *accident* connotes that uncontrollable natural forces may have been involved in an event. The industry prefers to use the term *incident*, with the knowledge that such events are preventable, whereas accidents are not.

Although the content of this chapter focuses on the Australian mining industry, the lessons can be applied anywhere.

DEFINING SAFETY

The Australian Standard 1470:1986, titled *Health and Safety at Work*, defines *safety* as "the provision and control of work environment systems and human behaviour which, together, give relative freedom from those conditions and circumstances which can cause personal damage." The Minerals Council of Australia (MCA), the peak lobby group for the mining industry in that country, defines *safety awareness* as "the state of mind where we are constantly aware of the possibility of injury and act accordingly at all times." On its Web site, MCA describes its vision for safety as "An Australian minerals industry free of fatalities, injuries and diseases." The MCA believes that

- All fatalities, injuries, and diseases are preventable;
- No task is so important that it cannot be done safely;
- All hazards can be identified and their risks managed;
- Everyone has a personal responsibility for the safety and health of themselves and others; and
- Safety and health performance can always be improved (MCA 2009).

Similar values and statements can be found on the Web sites of the major mining companies in Australia and throughout the world.

Exposure to dust, noise, and hazardous substances has long been an occupational hazard for mine workers. Cases of mesothelioma, asbestosis, silicosis, and pneumoconiosis are still appearing in mine workers many years after exposure. Although the health issues are at least as important as the safety issues and, in many cases, are more insidious and can take longer to manifest, the focus of this chapter is on mine safety rather than occupational health issues.

It is also important that workers are not subjected to injury through the use of machines. Mechanization and automation have helped to reduce any adverse interactions between humans and machines. Many mines still use handheld drilling techniques, however, and cases of repetitive strain injury in arms, hands, and fingers continue to occur. Similarly back injuries remain a major component of all workers' compensation claims in the mining industry. Part 15 of this handbook provides detailed information in this problem area.

MEASURING SAFETY IN MINING

Is mining a safe industry? Is *safety* a relative or absolute term? Some would argue that relative to other comparable industries, such as deep sea fishing, logging, and the construction and transport industries, mining is safe. Others would argue that, compared with the nuclear and chemical industries, it is not safe.

Defining how safety is measured is an important first step before attempting to compare mining with other industries. In mining, the two most commonly used measures are lost-time injuries and fatalities. Usually these measures are expressed as frequency rates, e.g., the lost-time injury frequency rate (LTIFR), which indicates the number of lost-time injury events that occur for a given number of hours worked. Figure 15.1-2 shows that on a lost-time injury basis, the Australian mining industry is performing in a continuously improving fashion. The use of this measure alone, however, leaves the industry open to the usual criticisms of reliability of this factor. It is well known that the LTIFR can be manipulated, particularly in this era of flexible work rosters.

One measure that cannot be manipulated is the number of fatalities that occur on mine sites. As shown in Figure 15.1-3, which covers the same period, 2000–2007 (financial years), an increasing number of deaths occurred. Although it can be persuasively argued that the number of employees in the mines increased, the numbers are far from the MCA's vision of "an industry free of fatalities, injuries and diseases." The risk of fatalities is measured by the fatal injury frequency rate (FIFR—the number of fatal injuries per one million hours worked). The Australian national FIFR in 2006–2007 was 0.05 per million hours worked, which was, according to the MCA, higher than the previous year at 0.04. Although an FIFR of 0.05 is below the 10-year average FIFR of 0.07, rates have fluctuated widely from year to year, and a consistent downward trend has not emerged.

According to the MCA, the Australian mining industry is performing relatively well compared with other countries. "The Australian minerals industry average FIFR for the ten-year period 1997–1998 to 2006–2007 was 0.07. Internationally the Australian industry compares favorably to South Africa on this indicator, which recorded an equivalent rate of 0.27 for this period, and the United States of America (USA),

which recorded a rate of around 0.17 for this period" (MCA 2009). Comparisons must be made on equal terms, however, as the indices vary from country to country. For example, in Australia, an LTIFR is defined as the number of lost-time injury events per million hours worked; in North America, it is per 200,000 hours worked.

THE CONCEPT OF RISK

The minerals industry has approximately 10 fatalities per year, based on a 10-year rolling average (MCA 2009). The minerals industry work force is about 120,000 strong, which means that about 1 in 12,000 workers are killed each year.

Is this an acceptable risk? Although no government agency in the world has a prescribed acceptable rate of work-related fatalities, several studies have shown that the community is comfortable with industries that kill less than 1 employee per 100,000 per year, as indicated in Table 15.1-1. This means that fatality risk in the minerals industry is much too high.

It may be useful to begin by examining the mining industry's safety record in comparison with other industries. There is no doubt that many industries have a higher incidence of serious or fatal injuries than mining. It has been estimated, for example, that the construction industry kills one person a week in Australia. The transport sector also results in more deaths than the mining industry in Australia. In a single state, Queensland, 23 truck drivers died in work-related incidents in 2007. Recent safety data is notoriously difficult to obtain, but a 2003 study of fatal crashes involving articulated trucks in Australia indicated that the number of fatalities between 1991 and 2002 (the latest available data), was relatively stable at around 190 per year (Australian Transport Safety Bureau 2003).

In a final comparison and according to recent statistics, in Australia, between 5 and 10 people per year are killed by lightning strikes, approximately the same number as those killed in mines (Bureau of Meteorology 2009).

RISK MANAGEMENT

The mining industry in Australia has embraced the concepts of risk management in virtually every aspect of its business. Risk assessments are usually mandated by legislation and are carried out before a new (major) piece of equipment is introduced or a new mining method, process, or system is implemented. Risk management has been credited with markedly improving safety performance in Australian mines.

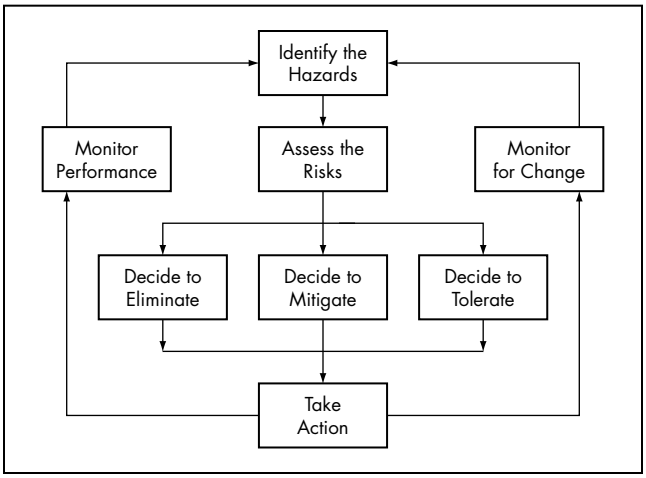
Figure 15.1-4 illustrates the steps usually followed in carrying out a risk assessment. First the hazard is identified. The risk presented by that hazard is then assessed. This can be done in a qualitative or a quantitative manner. Risk is generally assumed to be the product of the likelihood of a loss caused by that hazard occurring and the consequences of it occurring. After the risk is estimated, a decision is made to eliminate, mitigate, or control the risk to an acceptable level. Monitoring is the final step in the process to close the loop.

Hazards are a source of potential harm. In the mining industry, hazards are numerous, and the challenge for the work force, in the first instance, is to be aware of them. One of the reasons that hazards in mining are so great is because of the significant energies involved, be they gravitational, mechanical, chemical, electrical, or other types. The risk analysis allows the individual risks to be plotted on a matrix allowing management to allocate priorities to each.

Table 15.1-1 Estimates of risk

Activity	Risk of an Individual Dying in Any One Year
Smoking 10 cigarettes a day	1 in 200
Deep sea fishing	1 in 260
Influenza	1 in 500
Offshore oil and gas industries	1 in 600
Natural causes, 40 years old	1 in 850
Road accident	1 in 8,000
Road traffic accident	1 in 10,000
Playing soccer	1 in 25,000
Accident at home	1 in 26,000
Accident at work	1 in 43,500
Working in a "safe" industry	1 in 100,000
By fire or explosion at home	1 in 1,000,000
Struck by lightning	1 in 10,000,000

Source: British Medical Association 1987.



Source: Joy 1999a.

Figure 15.1-4 Basic risk-management process

MINE SAFETY LEGISLATION

In May 1970, the British government appointed a committee, chaired by Lord Robens, former chairman of the U.K. National Coal Board, to inquire into the provisions made for the safety and health of people in their workplace. The Robens Report was the major instigator to changing the philosophy and design of safety legislation in the United Kingdom and globally (Robens 1972). Up until that time, health and safety regulation consisted of a number of Acts of Parliament such as the Mines and Quarries Act (U.K. Office of Public Sector Information 1954), which had numerous sets of regulations covering specific risks or activities. Many of these had originated because of past accidents and disasters. Robens felt that this rule-based legislation was too complex, often inadequate, and had built-in obsolescence due to rapid technological changes.

Robens recommended a single broader and more flexible system of legislation based on more effective self-regulation by employers and workers jointly. Workplace health and safety should be effected by a tripartite approach involving workers, employers, and the regulator. The resultant Health

and Safety at Work Act (U.K. Health and Safety Executive 1974) included the following features:

- The duty of the employer to provide safe and healthy working systems, premises, working environments, and equipment, and to ensure that people work safely through training, instruction, and supervision
- The duty on each employee to observe safety and health provisions and to act with due care
- Elimination of prescriptive detail, replacing it with goal setting or performance achievement
- An emphasis on hazard identification, risk management, and risk assessment
- The use of codes of practice and guidelines to assist industry in compliance

In Australia, the New South Wales (NSW) government incorporated the philosophy of Robens in its Occupational Health and Safety (OHS) Act (NSW Government Legislation 1983), which was updated in 2000 and applies to all industries including the mining industry. This is a performance standard or enabling act. That is, it states the standard of performance to be achieved, leaving it to those bound by the act to select for themselves the methods to be used in achieving that standard. It is thus, philosophically, standing apart from the traditional prescriptive legislation, which prescribed for operators how to achieve standards.

The OHS Act acknowledges that the best results will be achieved by influencing attitudes and creating a framework for better health and safety organization and action by industry itself. Key features of the approach are

- Replacement of a mass of legislation by a single statute of general application;
- Provision by legislation of a comprehensive set of duties relating to the basic and overriding responsibilities of employers, employees, manufacturers, and others, which should be simple, easily assimilated precepts of general application (the "general duty" provisions); and
- The use of approved and voluntary standards and codes of practice in order to provide progressively better conditions.

Thus, under this legislation, both employers and employees have duties of care. An employer must ensure a safe and healthy workplace. An employee must take reasonable care of the health and safety of persons (including themselves) while at work and to cooperate with the employer.

Tasmania has a similar act, the Workplace Health and Safety Act (Australasian Legal Information Institute Tasmanian Consolidated Acts 1995). Part 3 of the act describes the general duties and obligations of both employers and employees. Section 9 provides that an employer must ensure, so far as is reasonably practicable, that the employee is safe from injury and risks to health. Section 16 of the act requires an employee to take reasonable care for the employee's own health and safety and for the health and safety of other persons and to comply with any direction given by an employer with respect to any matter relating to health and safety. Under this legislation, all parties, including the employer (and its employees) and the contractor (and its employees), have a duty to contribute to a safe working environment at the mine.

Other countries have different styles of legislation. A feature of the Canadian regulatory framework is the internal responsibility system or IRS. This is a system in an organization where everyone has direct responsibility for health and safety as an essential part of the job (Ontario Ministry of Labour 2000). The position a person holds in the organization does not reduce the obligation; however, the chief executive officer has overriding responsibility for implementing the system. Each person takes the initiative on health and safety issues and works to solve problems and make improvements.

A company's Joint Health and Safety Committee has a key contributory role in making the IRS work well. The keys to a successful IRS involve fixing responsibility at the worker level. For instance, each person must

- Have a sincere wish to prevent accidents and illnesses,
- Accept that the causes of accidents and illnesses can be eliminated or greatly reduced,
- Appreciate that risks can be continually reduced so that the time between accidents and illnesses gets longer and longer,
- Make health and safety an essential part of the job,
- Have a clear understanding of their responsibilities,
- Be able to explain what proactive measures they have taken to ensure health and safety on the job,
- Have a clear understanding of their abilities and limitations,
- Strive to more than meet the minimum standards,
- Strive to cooperate with others to reduce risk,
- Understand and believe in the IRS process,
- Recognize that reprisals and incrimination are not part of the process (Ontario Ministry of Labour 2000).

The IRS is a system that drives responsibility for health and safety down to the operator level. If mine managers can obtain this level of responsibility from their workers, then there is a chance of ensuring the health and safety of employees, which is a general duty under most mining OHS legislation.

INFORMATION SHARING

One significant advance contributing to improved safety performance over the past 20 years has been a greater openness among mining companies and government regulators to share safety information. Although this openness is threatened in some jurisdictions because of the litigation that follows some incidents, there is a great deal of information available, particularly on the Internet. This is mainly in the form of safety alerts, significant incidents reports, and bulletins. Most of these are generated by regulators and designed to inform mine operators about significant incidents that occurred within their jurisdictions. They are also shared between regulators, however, so that an operator of an underground coal mine in Queensland, Australia, for example, is aware of a fatality at a similar mine in West Virginia in the United States. The fatality reports issued by the U.S. Department of Labor's Mine Safety and Health Administration (MSHA) contain detailed information and are designed to ensure that such tragedies do not reoccur. Reports of this nature can be found at the Web sites for MSHA; the Department of Mines and Energy, Queensland, Australia; and the Department of Primary Industries, New South Wales, Australia.

THE HUMAN FACTOR

One body of evidence suggests that despite the best systems and procedures, accidents and incidents do occur and many of these are caused by risk taking or human error. For example, "Recent history in the mining industry in Queensland has shown that incidents and accidents have occurred as a result

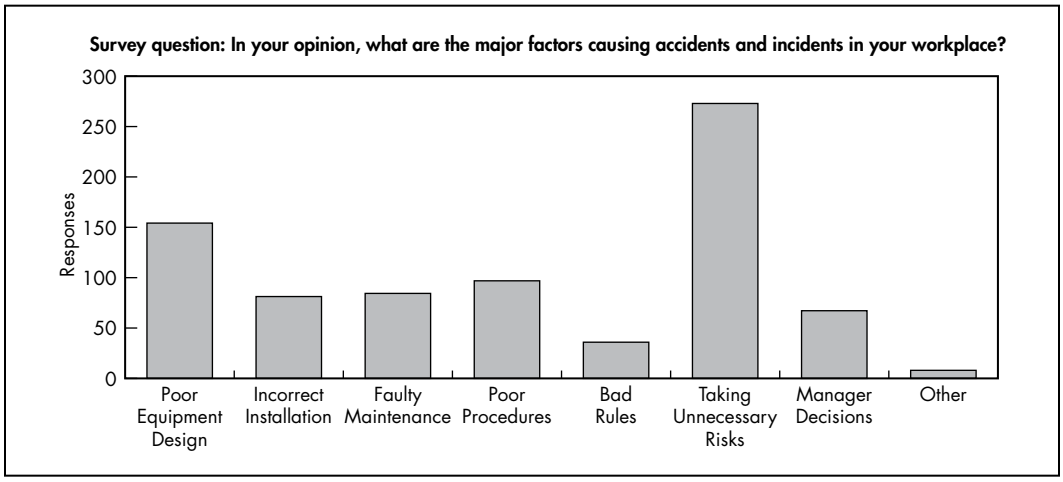


Figure 15.1-5 Miner survey results for causes of accidents and incidents

of unintentional errors or actions that do not recognise the personal risk present. Too often, the serious accidents are a result of conscious risk-taking, such as taking the easy way out, taking a short cut or not thinking about the risks in your workplace before starting work. It is not uncommon to find that the victims knew the potential risk of accidents before the incident" (Queensland Department of Mines and Energy 2002). And "very few people have accidents for which there is no procedure in place and we have a significant degree of accidents as a result of people breaking rules, ignoring rules or simply not knowing about them.…" (Pitzer 2000).

A recent study of risky behaviors of continuous miner operators in underground coal mines found that more than 90% of employees interviewed "expressed misgivings" about safety rules and regulations. There was a clear distinction between operating within the rules and operating normally, with the latter referring to the quickest possible way to complete a task. "The issue is not whether the rules are practical or not, or whether the rules should be simply followed or not. A more serious issue is that possibly the large majority of employees (including deputies/supervisors) operate dangerous machinery every day in underground mines with a basic disposition that safety rules are 'irrelevant, superfluous, non-essential or excessive.' If this is the case, an important resource for limiting the risky behaviour of employees is critically deficient" (Pitzer 2000).

A further study of 39 multiple-fatality coal mine disasters, mainly from the United States, United Kingdom, and Australia, found that serious violations of mining laws were discovered among 33 (Braithwaite and Grabosky 1985). It was found that serious violations either caused the disaster, were components of the causes, or exacerbated the effect of the disaster.

These findings are supported by a 1991 study, commissioned by the NSW Department of Mineral Resources, which found that 83% of fatal accidents from strata movements in underground coal mines in the period 1972–1990 were associated with breaches of rules (Roylett et al. 1991). Human error was simultaneously present with breaches of rules in more than half the fatal accidents analyzed. It was concluded that fatalities would continue to occur unless management found strategies to focus on the development of better support rules, compliance with rules, and improvements in technology to "counter against human error." Subsequent investigations have suggested that, apart from rules breaches, a major contributory cause of these incidents was inadequate mine design, including undersized pillars (J. Galvin, personal communication).

A supplementary study on roof and rib fall incidents commissioned by the NSW Department of Mineral Resources in 1999 updated the findings of the Roylett et al. 1991 study. It found that "to effectively minimise roof fall fatalities the industry must control the behavioural causal factors involved in these fatalities. In particular, miners going out under unsupported roof and not working to the support rules" (Pereira 1999).

Another research project focused on the development of more effective mine safety rules and regulations (Laurence 2002). Responses from almost 500 mine workers were analyzed and some simple guidelines established, in particular regarding rule content. The recommendations were that management and regulators should not continue to produce more and more rules and regulations to cover every aspect of mining. Miners will not read nor comprehend to this level of detail. Detailed prescriptive regulations, detailed safe work procedures, and voluminous safety management plans will not "connect" with a miner. The aim should be to operate within a framework of as few rules as possible, but rules of the highest quality.

Of course, achieving more effective rules and regulations is not the only answer to a safer workplace. Possibly less emphasis should be on content and more about the process, in particular ensuring that a positive safety culture exists and that communication channels are open and working well. This was confirmed in the survey when the expressions "simply bad rules" or "poor rules" were rarely blamed for accidents and incidents, risk taking, or error making. More often the problems of implementation, communication, and learning were the main causal factors.

The mine workers were also asked what they considered to be the major causes of accidents and incidents. Taking unnecessary risks was considered by far to be the most significant response, as shown in Figure 15.1-5.

Human error is defined by Reason (1990) as "the failure of planned actions to achieve their desired ends—without the intervention of some unforeseeable event." Errors can be divided into those where the plan was adequate but actions

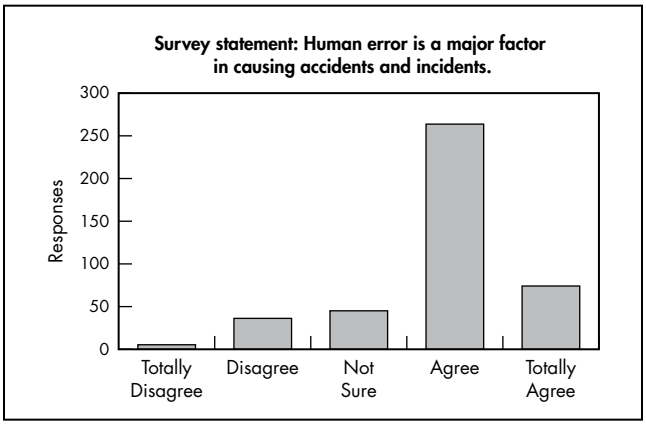


Figure 15.1-6 Miner survey results regarding human error and accidents

failed to go as planned, or those where the actions conformed to plan but the plan was inadequate. The former are known as slips, lapses, and trips, and the latter as mistakes.

There was general agreement among the mining work force that human error is a major cause of accidents and incidents, as illustrated in Figure 15.1-6. The major reason that people make errors is haste, according to the work force, as shown in Figure 15.1-7.

In his analysis of 25 incidents involving fatalities or serious injury, Joy (1999b) found that many were caused by management systems failures. Active human error was also a significant cause. The error was not generally intentional: "most active human error occurs where the person intended to do things correctly." Of the 25 accidents, 46% were violations; 31%, mistakes; 8%, slip/lapses; and 15%, no active human error. In some of these incidents, mine workers

- Were unaware of the rules,
- Were aware of but did not understand the rules,
- Mistakenly applied the rules,
- Ignored the rules,
- Deliberately violated the rules,
- Took a risk,
- Were unable to identify hazardous situations, or
- Were poorly trained or lacked sufficient educational background (Joy 1999b).

Further confirmation of the extent of the contribution of unsafe behaviors to accidents and incidents on mine sites in Australia is shown in Table 15.1-2. The table shows all published incidents over a 4-year period where a human factor can be identified. Such incidents are listed on the Web pages of each government mines' inspectorate in New South Wales, Queensland, and Western Australia. It can be seen that many of these accidents were caused, at least in part, by

- Lack of awareness of procedures or rules,
- Not complying with or ignoring rules,
- Lack of clear instructions,
- Poor communication generally,
- Production taking precedence over safety,
- Overriding or bridging out safety barriers,
- Inadequate training,
- Lack of familiarity with equipment, or
- Fatigue.

EXERCISING DUE DILIGENCE

It is important that mine managers are aware of the need for due diligence should an incident occur at their mine site. In the event of a serious incident, a regulator may wish to see

- A report of the incident;
- Whether it was reported to the authorities as required;
- Whether an accident report was filled out;
- What system of work was being used;
- What equipment was being used;
- The extent of the injured person's qualifications and experience in work of this type;
- Whether the injured person was inducted and trained;
- Who the supervisors and managers were, their qualifications and experience, and what instructions were provided to the injured person;
- Whether a risk assessment was carried out prior to the work;
- Whether a job safety analysis, safe operating procedure, or equivalent was in place; and
- Evidence of a safe system of work.

A SAFE SYSTEM OF WORK

An organization has implemented a "safe system of work" when it has employed and trained competent people, utilized fit-for-purpose equipment, and implemented effective systems and practices, all in a controlled work environment, as illustrated in Figure 15.1-8.

Competent People

To achieve safe production in a mine, competent people must be employed in senior management, in line management, and at the coal face. Evidence of competence includes inductions, training including refresher training, assessments, experience, formal educational qualifications, and so on. In most Australian states, mine managers have to demonstrate in a formal written and oral examination that they are competent to run a mine. They also need a degree or diploma in mining engineering or equivalent. Mine managers in New South Wales and Queensland are now required to hold the G3 certificate, which demonstrates competency in integrating risk management into the mining operation.

Equipment

Clearly the equipment and machinery used on a mine site needs to adequately carry out the task for which it was designed; it needs to be "fit for purpose." In recent times, manufacturers have worked in partnership with mines to continuously improve mining from a safety and ergonomic perspective.

Effective Systems and Practices

In an underground mine, examples of effective systems and practices include a safety management plan, ground control management plan, ventilation surveys, job safety analyses, formal risk assessments, and so on.

A Controlled Work Environment

Because of geological and geotechnical uncertainties, a controlled work environment is difficult to achieve, particularly in an underground mine. This can be overcome to a certain extent by ensuring that geological and geotechnical data is sufficient to enable sound decisions to be made for managing ground support, one of the principal hazards. Ground support measures are usually routine and can thus be considered as

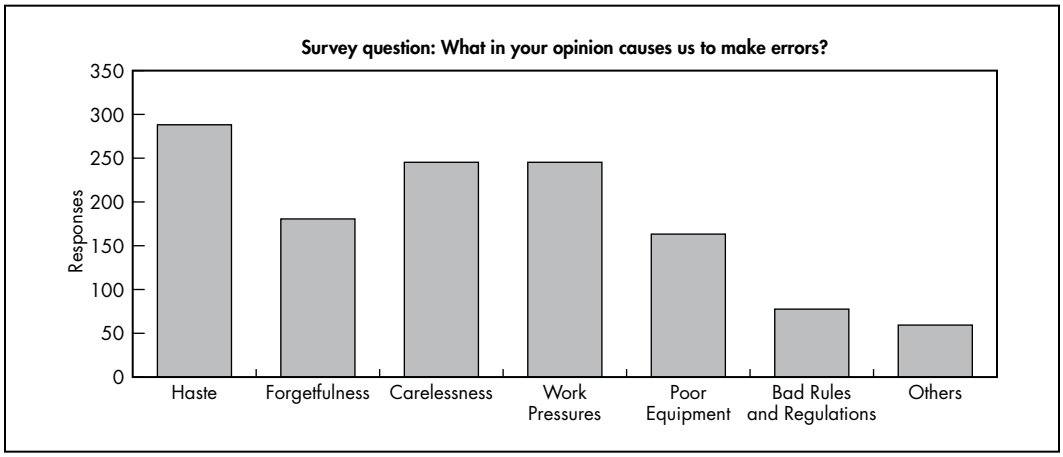


Figure 15.1-7 Miner survey results showing reasons for error

close to a controlled work environment as can be achieved underground.

AVOIDING MINE DISASTERS

In the past, there has been a tendency to focus on low-consequence, high-probability events, also known as "slips, trips, and falls." Doing this may certainly result in a lower injury rate and thus better safety statistics, which of course is one of the common key performance indicators of mine management. Experience shows, however, that if this aspect of safety is the only focus, opportunities may be missed for identifying potentially more damaging events, sometimes resulting in mine disasters.

Increasingly, the community is becoming less tolerant of any injury or human loss caused by mining and associated operations. Society is prepared to accept a certain amount of loss in mining, but community outrage may occur if fatalities are involved.

Reason (1997) argues that all organizational accidents arise from breaching a defense or series of defenses, barriers, or safeguards separating damaging hazards from vulnerable people or assets. Defenses should be present in layers, termed *defenses in depth*, and each defense would protect against the possible breakdown of the one before (Figure 15.1-9). Defenses should

- Provide alarms and warnings when danger is imminent,
- Restore the system to a safe state in an off-normal situation,
- Interpose safety barriers between the hazards and the potential losses,
- Contain and eliminate the hazards should they escape the barrier, and
- Provide the means of escape and rescue should hazard containment fail.

Many organizations harbor what Reason (1997) terms *latent conditions*, which resemble pathogens in the human body—they may be present for many years before they combine with local circumstances and active failures to penetrate the system's many defenses. Examples include poor design, gaps in supervision, manufacturing defects, maintenance failures, inadequate training, and impractical procedures. Latent conditions were present in the *Challenger* space shuttle tragedy, the *Herald of Free Enterprise* sinking, the *Piper Alpha* oil rig fire, and the underground fire at King's Cross Station in London, to name a few.

Using Reason's Swiss cheese example, an accident occurs when there is a connecting set of holes in successive defenses, allowing an accident trajectory to occur, as shown in Figure 15.1-9.

Hopkins (1999) analyzed multiple-fatality events at the Gretley mine in New South Wales and the Moura mine in Queensland, Australia, which resulted from water inrush and a coal dust explosion, respectively. He found the following indications:

- Normalizing of the evidence. In much the same way as the National Aeronautics and Space Administration's management and engineers came to regard O-ring failure in the space shuttle *Challenger* disaster of 1986 as normal due to the frequency of malfunction in earlier launches, Gretley supervisors and managers (and probably the employees as well) normalized the warning signs of water inflow.
- Management reluctance to withdraw workers from danger. Hopkins indicated that at both Moura and Gretley, common sense would dictate that workers not be placed in danger until the hazard had passed or been eliminated.
- A culture of denial. "New evidence appears reliable and informative if it is consistent with one's initial beliefs; contrary evidence tends to be dismissed as unreliable, erroneous or unrepresentative." There were strong production pressures at Moura, which may have resulted in a tendency to dismiss any contrary evidence. Hopkins calls this the culture of denial, "a set of beliefs which enabled management to deny that there was any immediate danger, no matter what the evidence."

Organizational and systemic factors are thus recognized as major contributors to mine disasters. Human factors also played a part in these mining disasters, namely

- Communication failures,
- Failure to recognize warning signs,
- Lack of knowledge about the hazards,
- Lack of training,
- Errors,
- Poor judgment, and
- Risk taking.

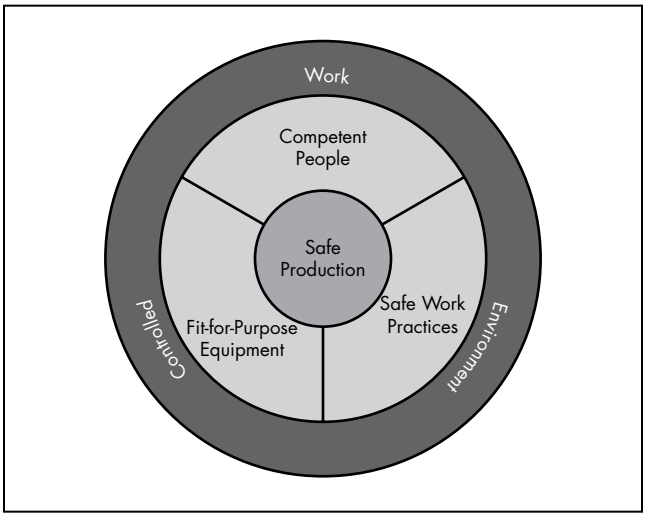
Table 15.1-2 Extracts from significant incident reports 1998 to March 2002, Queensland and New South Wales, Australia

Incident	Behavioral Contribution to Cause of Incident
	Queensland Mines
Highway trucks collide	Local **road rules were not clearly understood and followed.**
Light vehicle drives off high wall	Operators should ensure that they are provided with **clear work instructions**, particularly where they are working in an area of the mine with which they are not familiar.
Catastrophic failure of a dragline boom	Unconfirmed reports indicate that draglines are now being loaded above their rated suspended load, in **a drive for greater productivity**.
Dump truck with unbalanced load becomes unstable	**A standing instruction** that large rocks are not to be loaded but reblasted **was ignored** by the shovel operator.
Driller reports for work in an intoxicated state	The driller's offsider was disciplined and given a mine site reinduction where the **statutory requirement of not reporting for duty in an intoxicated state** was stressed. His workmate was also given the same mine site reinduction.
Methane monitors were bridged out of circuit in two mines while producing coal.	On each occasion the tailgate methane **sensor was bridged out of circuit** during calibration checks and not recommissioned after these routine checks were completed.
Dumping overburden near coal crew operations	A dump truck operator dumped overburden outside of the allotted dump area **without knowledge or consideration of the potential consequences**.
An operator had two fingers pinned and a third severely lacerated when a bolting rig autoretracted.	Operator was attempting to do multiple tasks consecutively and **pinch points were not guarded or identified**.
Collapse of bridged material in pugmill feed bin	The plant operator walked along the edge of the top of the feed bin with **no safety belt/harness attached**.
Rigger struck on head by 500-kg assembly	The crews dismantling the crusher **did not understand** the function of the component parts or **were unaware** of the condition of the component parts.
Dozer runs over light vehicle	The mine site **procedure** for entry into a work area **was not followed**.
Truck falls backward over tipping edge	**Ignorance of safe work practices**; the truck driver was **not familiar with a safe procedure** for tipping over the edge of a stockpile.
Rollover of rear-dump truck	The height and size of the rill was inadequate and **not identified as a risk** by the operators. Lack of handover between shifts led to hazards in the workplace **not being communicated**. The dozer operator on the previous shift **had not worked to the mine's standard** for a dump area by allowing the dump face to become misaligned.
	New South Wales Mines
Two large trucks out of control	Maintenance, testing, and inspection of the braking systems were inadequate. Drivers **had not been given training** in emergency response procedures.
Danger from a misfire	**Communications were not clear** between the day shift and the night shift about the result of the refiring.
Unstable structure crushes workman	For some time before the accident, both the injured person and his supervisor **had worked more than 11 hours** a day.
Quarry worker killed by front-end loader	Management **should have safe work procedures in place** for people who need to approach mobile equipment. Everyone at a mine, including contractors and visitors, **should be trained** in the procedures.
Operator crushed by overturned forklift	**No standard operating procedures** were available. **No formal, refresher, or emergency training** had been given. The vehicle appears to have been **traveling too fast** for the conditions; the driver was **not restrained by a seat belt.**
Electric shock from damaged circuit breaker	The electrical maintenance engineer and the electrical staff generally were **not aware of this hazard**, even though it was reported in previous safety alerts.
Continuous miner drill rig fatally crushes tradesman	Managers should also address **basic hazard recognition and control skills training** as a priority.
Vehicle brake failure while travelling in a decline	If the truck were driven in third gear in the decline, it would have eliminated the need to use the brakes.
The 415-V conductors shorted out, and the power tripped off on short circuit.	The electrical contractor was **not aware** of the site *Electrical Management Handbook*.

Source: Data from Queensland Department of Mines and Energy 1998–2002; NSW Department of Mineral Resources 1998–2002.
Note: The behavioral issue in each incident is emphasized in bold.

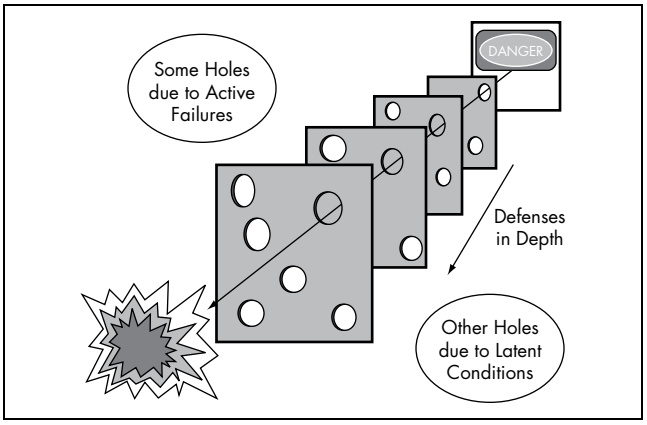
Disasters in the mining industry are usually followed by a political and/or bureaucratic response in the form of inquiries, inquests, changes to legislation or restructuring, and re-resourcing of mines departments. Some ideas for reducing these potential disasters include the following:

- Better communication in both directions at mine sites, both top-down and bottom-up, will reduce the opportunity for a high-potential incident.
- Relying solely on sophisticated safety management systems is not enough; it is necessary to ensure that employees understand and are trained in procedures.
- Failure to comply with existing regulations can contribute to the making of disasters, as can out-of-date regulations. On the other hand, strict adherence to rules also causes disasters, as was shown in the *Challenger* explosion.
- There is a need to avoid normalizing abnormal events and to be aware of incremental changes in processes.



Source: Adapted from Nertney 1987 and Joy 1999b.

Figure 15.1-8 Work process model or Nertney wheel of safe work



Source: Reason 1997.

Figure 15.1-9 Reason's Swiss cheese model

- The development of active foresight, or what Reason (1997) describes as constant vigilance, will also help to reduce disasters.

CONCLUSIONS

Advances in mine safety over the past two decades have been significant. Apart from the obvious advances in technology that have made tasks more efficient and less physically intensive, other initiatives include

- Changes to the regulatory framework in the form of goal or performance orientation rather than prescribing how to achieve safety;
- Stronger enforcement and higher penalties for mine operators if an incident occurs;
- Better training and standards for the work force;
- Better systems and practices;
- More rigorous due diligence;
- More awareness of the importance of the human factor;
- Extra emphasis on reducing the risk of a large event occurring, rather than focusing only on the less damaging slips, trips, and falls;
- More sharing of safety information among mining companies and regulators, particularly through governmental Web sites.

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CHAPTER 15.2

Health and Medical Issues in Global Mining

Frank H. Fox

INTRODUCTION

Mining normally takes place in remote areas, and as resources become increasingly scarce, new deposits are only found in the more inaccessible and inhospitable parts of the world. These are often in countries with poorly developed infrastructure and are characterized by extremes of terrain and climate, all of which contribute to potential medical and health challenges. Climatic extremes (heat, cold, high and low humidity, altitude, and hyperbaric conditions) increase the possibility of relatively minor medical conditions becoming major impediments to health, safety, and productivity and pose challenges to even the fittest individuals. Remote locations also bring psychological stresses that may manifest in the form of alcohol and substance abuse.

With a global shortage of skills in the mining industry it is likely that the work force will be a mixture of local and expatriate employees. Health problems may range from endemic diseases such as malaria to the psychological stress of isolation and continuous shift systems that can lead to wellness issues as well as alcohol and substance abuse.

Prevention of medical and health problems in mining operations (from exploration to production environments) starts with anticipation and recognition of the potential issues and proceeds with a thorough risk-assessment process inclusive of projected frequency and severity of incidents, development of risk-control measures, and allocation of adequate resources to implement the controls to manage the risk.

GENERAL APPROACH

The key to managing all potential health problems is prior risk assessment at the design stage of operations, before travel and before operations begin. During operations, continuous risk assessment is vital as things change. This is based on anticipation, recognition, evaluation, and control. Additionally, awareness of the differences between health and safety needs must be clear; too often, safety concerns are given priority over health issues. This is a short-term focus that must be corrected.

Global travel is so commonplace that travelers are often complacent about potential health risks. Yet the risks may be significant, even for local travel to mine sites within one's home country. And while mining and its associated processes have become safer over the years, and engineering design and equipment have improved, the extraction of minerals remains an inherently hazardous task. Mining requires close attention to detail to avoid these hazards.

Historically, mining has focused primarily on accident rates as the primary driver for performance measurements and safety awards. Improvements in engineering and processes have resulted in fewer accidents, but similar improvement has not been achieved for illnesses. In fact, some engineering improvements may have inadvertently contributed to an increase in health issues such as obesity, risk of cardiovascular disease, and cumulative trauma as more tasks have been engineered to reduce physical effort.

The following factors should be considered when planning for optimum health of mine workers: location of the site, climate, endemic diseases, local infrastructure (health care, transport, and accommodations), local legislation, the make-up of the work force (local, expatriate), the process (ore, products, and effluent), and shift system (continuous operation, fly-in/fly-out, etc.). If the site is an exploration base, travel to and from the site may pose a health risk and should be considered.

There are three ways in which health can be approached with regard to mining: those issues that affect personal health, those that concern occupational health, and those that relate to health in the community. These overlap in a number of areas but can be usefully approached in these subdivisions. A few of the more well-known conditions will be presented together with an approach for management from the mining engineer's perspective. Mining engineers will gain a basic understanding of the health issues that face them and an approach to dealing with them.

PERSONAL HEALTH

Personal health conditions and behaviors start at home, are taken to the mine site, and continue with residence there. A worker's lifestyle (diet, exercise, sleeping habits, stress level, substance abuse, etc.) has as important an affect on health and safety as conditions at the mine itself. Hazards at

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the mine site will include the location and local infrastructure of the mine and the occupational health hazards that may be encountered in the mining environment. In addition, there may be local climatic conditions that pose a risk: excessive humidity or low humidity, high altitude, and exposure to ultraviolet radiation.

Exposure to infectious diseases can occur both on and off the job. The specific diseases vary from country to country and cover the entire spectrum of bacterial, viral, protozoal, fungal, and parasitic infections. Exposure to disease occurs through personal contact, insect vectors, food sources, contact with animals, and water.

Malaria, tuberculosis (TB), and HIV/AIDS (human immunodeficiency virus/acquired immunodeficiency syndrome) are three prevalent diseases that are currently receiving worldwide attention. These diseases are not only important at a personal level but also as social issues in the communities surrounding mines in many developing countries. Ultimately they are an important factor in the productivity of the work force. These will be treated separately, as they illustrate the spectrum of involvement and intervention that may be needed in mining.

Traveling—locally and globally—presents its own unique set of health risks. Before traveling to exotic or unfamiliar locations, it is wise to seek advice from a health practitioner who is knowledgeable about worldwide medical issues, as well as from local medical practitioners who are experienced in dealing with health concerns specific to the area.

Health insurance is an important consideration when traveling and particularly when engaged in exploration work. Local health care facilities may be less than ideal, and suitable emergency evacuation arrangements should be in place before the work is undertaken. Many companies employ paramedical personnel to provide a basic level of primary and emergency care on-site with additional arrangements in place for evacuation in case of emergency. However, the key in all situations is prevention of the emergency in the first place, which means evaluation of fitness before placement, correct prophylaxis for endemic diseases while on-site, and careful attention to public health measures.

Although often taken for granted, some mention must be made of personal hygiene. Work in hot, humid conditions, underground or topside in tropical climates brings with it potential health problems. Fungal infections, insect bites, and bacterial skin infections are a common source of disease and can become serious problems if neglected. These are often prevented through simple measures and attention to personal hygiene.

OCCUPATIONAL HEALTH

Sir Thomas Legge, the United Kingdom's first Medical Inspector of Factories (appointed in 1897), wrote a simple axiom that remains relevant today:

> All workmen should be told something of the danger of the material with which they come into contact and not be left to find it out for themselves—sometimes at the cost of their lives (Robbins and Landrigan 2007).

This principle is enshrined in occupational health and safety law and various International Labour Organization conventions, specifically in Article 9 of Convention 176 (ILO 1995).

A full discussion of occupational health is beyond the scope of this chapter, but there are a few basic points that should be considered. Occupational health is defined as follows:

> Occupational health should aim at the promotion and maintenance of the highest degree of physical, mental and social well-being of workers in all occupations; the prevention among workers of departures from health caused by their working conditions; the protection of workers in their employment from risks resulting from factors adverse to health; the placing and maintenance of the worker in an occupational environment adapted to his physiological and psychological capabilities and; to summarize: the adaptation of work to man and of each man to his job.
>
> The main focus in occupational health is on three different objectives: (i) the maintenance and promotion of workers' health and working capacity; (ii) the improvement of working environment and work to become conducive to safety and health and (iii) development of work organizations and working cultures in a direction which supports health and safety at work and in doing so also promotes a positive social climate and smooth operation and may enhance productivity of the undertakings. The concept of working culture is intended in this context to mean a reflection of the essential value systems adopted by the undertaking concerned. Such a culture is reflected in practice in the managerial systems, personnel policy, principles for participation, training policies and quality management of the undertaking (International Labour Office 1998).

These objectives might imply that occupational health is a medical issue, whereas in reality the second and third parts of the statement can only be achieved through application of engineering and human factors and ergonomic solutions. Occupational health and safety is thus an engineering problem, not a medical one, so it is essential that the mining engineer has an understanding of the health hazards that are inherent in mining.

Management is responsible for operating the mine and its associated processes in such a way that it does not pose a risk to the health and safety of the work force and the surrounding community. The focus should be on control of exposure to health hazards through the application of the hierarchy of controls from the design stage onward (see Table 15.2-1). Utmost attention to monitoring and maintaining those controls must be observed. This control must be supported by a program of environmental monitoring and medical surveillance of exposed people. None of this can be effective without the active involvement of all employees. Everyone should be informed, trained, and involved in the control and prevention measures that are put in place.

HEALTH RISK ASSESSMENT

The health hazards associated with mining are numerous and cover the entire spectrum of physical, chemical, biological, ergonomic, and psychological stressors. The mining process itself exposes people to dusts, fumes, and vapors of various

Table 15.2-1 Hierarchy of controls

Elimination Eliminate an unsafe process or substance (e.g., use ultrasound to clean components instead of chemicals).
Substitution Substitute a process or chemical with one that is safer (e.g., use water instead of chemical solvents).
Engineering controls Use physical controls that eliminate or reduce exposure (e.g., enclose the process, separate the worker, provide adequate ventilation [exhaust, directional, and natural], soundproof areas where there is a high noise level).
Administrative controls Practice safe work procedures and good housekeeping (e.g., cover containers, clean up spills, rotate jobs, reduce shifts, restrict eating or drinking to specific areas, maintain washing facilities, provide training).
Personal protective equipment Provide appropriate personal protective equipment for workers (e.g., gloves, hearing protection, overalls).

compositions and particle sizes; noise; vibration; radiation (ionizing and nonionizing); and ergonomic factors. Associated with this are ancillary engineering processes that present an additional set of hazards. Factoring in the location of the mine gives a complete picture of the occupational health risks of mining.

The first step is developing a program to ensure health and safety is risk assessment, followed by a program to control, monitor, and mitigate the residual risk. Health risk assessment (HRA) can be a complex process performed by a multidisciplinary team, but it can also be a basic process aimed at identifying the need for expert advice, intervention, and a more formal approach.

Typically, an HRA will take place in three stages: (1) a baseline or initial assessment to identify and prioritize potential risk; (2) an issue-based assessment to refine and quantify the risk identified in the initial assessment; and (3) continuous risk assessment. This final stage is integral to the ongoing risk management program and is a process of renewal that monitors the effectiveness of controls and continually assesses risk against process changes over time. Ultimately, an HRA should allow managers and workers in any industrial environment to answer these questions:

- What are the hazards in my operation or area of work?
- What am I doing to control exposure?
- What am I doing to monitor the controls and are they effective?
- Can I, and should I, do more?

HRA requires an understanding of the process, the hazards, and possible exposures inherent in the mining operation and an understanding of the potential health effects of any exposure. The team that undertakes an HRA therefore needs input from a process engineer, the medical staff, occupational hygiene personnel, line management, human resources, and the workers.

THE MEDICAL SIDE OF OCCUPATIONAL HEALTH

The medical component of occupational health is concerned with the contribution to HRA, assessment of fitness for work, medical surveillance, management of occupational diseases and injuries, and wellness of the work force.

Depending on the infrastructure in the area, the mine may also provide medical treatment for employees, their dependants, and, in some cases, the community. This may extend to public health issues such as disease control and management programs and, in remote sites where the mine provides all the infrastructure, sanitation and drinking water.

Fitness for Work

Assessment of fitness for work has a large impact on safety as well as productivity. It also is a means of preventing exposure of individuals with certain preexisting medical conditions to situations in the workplace that cause them undue risk. An example is the exposure of people with asthma to environments with irritant gases or sensitizing agents. In order for this process to be fair, it is essential that fitness criteria be based on the task and risks inherent in the particular position within the mine. This requires a job specification with fitness criteria derived from the HRA.

Fitness for work can also be extended into the wellness arena. Here the focus is on maintaining fitness through management of lifestyle (diet and exercise), stress, and chronic diseases. This may be achieved through specific wellness and employee assistance programs, or the company's in-house medical services. The athletic facilities provided in many mining towns support this approach to wellness.

Medical Surveillance

Medical surveillance is the process of screening for early signs of occupational disease and any signs of excessive exposure to hazardous agents in the workplace. The process involves recording an individual's occupational and medical history, undertaking a physical examination, conducting certain special tests (such as audiometry and spirometry), and, in some cases, biological monitoring (measurement of chemicals, metals, or metabolites in body fluids). In all cases this should be based on knowledge of the hazards and exposures in the workplace, with only necessary tests being done. In many countries, medical surveillance is a legal requirement, but regardless of location, all employees should be informed of the reason for testing and given the results with an explanation.

Occupational diseases are conditions that arise out of and as a consequence to exposure to a hazardous agent in the working environment. Typically there is a dose–response relationship between the exposure and the onset of the disease; that is, the exposure must be high enough and long enough to cause the condition. Examples of this are the dust diseases of the lung (pneumoconiosis) where an accurate diagnosis is based on exposure to a certain amount of dust over an appropriate period of time. In some conditions there may not be a clear dose–response relationship (such as allergies), and the condition is diagnosed on the basis of exclusion. Although the traditional occupational diseases have largely disappeared through better control of workplace exposures, there is still a risk from the many new chemicals and processes that are being discovered, and it is impossible to predict idiosyncratic reactions to chemical exposure. The mining industry needs to remain vigilant through continuous risk assessment with regard to exposure to potential health risks.

In most countries, exposure to hazardous agents in the workplace is regulated through the use of occupational exposure limits. These are typically based on a time-weighted average exposure over 8 h/d, 40 h/week. Management of exposure to health risks is through the application of engineering controls according to the hierarchy of controls.

The specific engineering approach to some of the health and safety issues falls into the disciplines of occupational (industrial) hygiene and ventilation engineering and is beyond the scope of this chapter.

Emergency treatment and management of occupational injuries is also a part of any medical service on a mine, and is too often the only consideration given. The scope will vary according to the site and the extent of the operation, but adequate preparation must be made for the emergency care and subsequent treatment of all possible injuries on the mine. Services may range from essential emergency care on-site with evacuation to a hospital to comprehensive on-site medical facilities that can provide sophisticated care quickly and efficiently.

Occupational Disease

Occupational disease may be acute or chronic, and despite the preventive measures outlined previously, occupational disease remains all too common, particularly in developing countries where legislation may not be as developed or effectively enforced. Acute conditions due to exposure to hazards in the workplace (such as muscular strains and sprains, dermatitis, and others) require timely diagnosis and treatment. Conditions such as silicosis and asbestosis (which occur after prolonged exposure) remain significant causes of both morbidity and mortality around the world, and despite having been almost eradicated in many countries, new cases of silicosis continue to be common in the developing world. Mining is an inherently dusty operation, and prevention of occupational disease requires a sustained long-term commitment to implementing and maintaining the hierarchy of controls. (Unfortunately, there is often a reliance on personal protective equipment as the main protection strategy.) The insidious onset of most occupational diseases contributes to the lack of a sense of urgency around prevention, and the overlap with some community diseases (e.g., asthma, skin disease, and TB) may contribute to the lack of recognition of the problem. The most common exposures throughout industry and mining include noise, heat, manual handling, vibration, and chemicals, leading to noise-induced hearing loss (NIHL), musculoskeletal conditions, and dermatitis as perhaps the most common diseases. Emerging risks include whole-body vibration, diesel particulate, nanoparticles, and psychological stress. Fatigue (often related to shift work) is also an important factor to consider, from both a safety and health point of view.

While this is not intended to be a text on occupational disease, a few conditions are worthy of brief mention, although this should not detract from the importance of any other condition.

Silicosis

Silicosis is one of the oldest recognized lung diseases due to dust exposure. In its typical form it is caused by exposure to crystalline silica dust at moderate levels over a period of 10 to 20 years. It is identified as a diffuse nodular pattern on the chest X-ray. This is due to the silica particles that provoke an inflammatory reaction and cause scarring of the lung tissue. In its milder forms it may not cause any appreciable symptoms, but with higher levels of exposure the scarring can be so severe that it interferes with the function of the lungs and the heart. In addition, there is an increased risk of TB in workers with silicosis, and silica has also been linked to lung cancer. There is no treatment, and the only solution is prevention through reduction of dust levels and avoidance of exposure.

Despite the large drop in the number of cases of silicosis in the developed world, there is no room for complacency. Surveillance in the United States showed a large reduction in the number of deaths attributable to silicosis between 1968 and 2005, but the incidence has leveled off since 1998 (Attfield et al. 2009). Although the incidence of silicosis has dropped significantly in developed countries through strict control, it cannot be assumed that current occupational exposure limits are entirely safe, and therefore it remains a risk (Park et al. 2002). In fact, the incidence of pneumoconiosis appears to be more closely correlated with the number of people employed in mining rather than the reduction in dust levels (Attfield et al. 2009). Silica is ubiquitous in soil and rock, and since it is not only hard-rock mining that poses a risk, it must be considered a hazard wherever mining is undertaken.

Noise-Induced Hearing Loss

Hearing loss from exposure to noise has been known since biblical times, and tinnitus (ringing in the ears) in blacksmiths is described in the book of Ecclesiastes. In more modern times it was described in coppersmiths and millers by Ramazzini (1713) in the 18th century. Despite its long history, NIHL remains one of the most common occupational diseases today, and it is a significant cause of morbidity.

Hearing loss can present in two ways: (1) acute, from exposure to very loud noise such as an explosion, and (2) chronic, from exposure to noise above 85 db for many years. The mechanism of damage is different between the acute and chronic types of hearing loss, with the acute form associated mostly with damage to the conduction system (the eardrum and small bones in the ear) and the chronic form associated with damage to the hair cells in the cochlea. Of the two forms, the chronic form of hearing loss is commonly referred to as NIHL. It typically presents with a slowly progressive loss of hearing, first in the higher frequencies and then in the frequencies used for speech. In many cases this is accompanied by tinnitus, which makes the disability even worse. The slow onset and the late age at which the condition occurs means that it may be confused with hearing loss due to aging. Occupational exposure to noise is not the only cause of NIHL, as any other source of loud noise (music, gunfire, etc.) can produce the same symptoms. In addition, some drugs and infections may cause a similar pattern of hearing loss. Modern hearing aids may provide some assistance with deafness, but in the presence of tinnitus this is doubtful. The best solution is prevention.

Heat Stress

Exposure to heat is a common problem in mining, whether on the surface or underground, and can lead to significant health and safety risks if not correctly managed. Any situation where there are high temperatures, high humidity, poor ventilation, or high radiant heat sources can result in heat stress.

There are four typical presentations of heat stress: heat fatigue, heat cramps, heat exhaustion, and heat stroke. Heat fatigue is a vague condition of impaired general performance and may be associated with mood changes. The behavioral effects of heat exposure should not be underestimated as they can lead to poor decision making that, in the mining and

industrial setting, can have a significant impact on safety. Heat cramps occur in the muscle groups that are most used in work and are painful spasms that last several minutes but are usually self-limiting. Heat exhaustion is the condition that results from excessive fluid loss and presents with dizziness, sweating, headaches, weakness, nausea, and vomiting. Heat stroke is the most serious of the conditions and results from failure of the thermoregulatory center in the brain. It is almost always associated with prolonged physical exertion in a hot environment and presents with extreme hyperthermia (body temperature above 41.1°C or 106°F); confusion; collapse; and, typically, hot, dry skin (although this is not always the case). If not treated effectively, heat stroke can be rapidly fatal. Heat exposure can also cause a skin condition known as prickly heat, which is an acute itchy dermatitis.

Heat exposure is a combination of the ambient environmental conditions, the heat generated internally by physical effort, and the ability of the body to cool itself. There are four basic mechanisms of heat exchange involved: convection, conduction, radiation, and cooling through evaporation.

The management of risk from heat exposure starts with awareness of the problem, environmental controls where possible, increased fluid intake (200 mL/20 min), and control of physical exertion (increased work breaks) in hot environments. Salt tablets are often mentioned, but this may do more harm than good; simply increasing the amount of salt in the diet is all that is required. Acclimatization does occur after a period in a hot environment, and many deep-level mines have acclimatization programs for new workers. This reduces the risk, but there are some individuals who will remain at high risk for heat disorders. Age (over 50), obesity (body mass index >30), certain medical conditions, and a previous history of heat disorders all increase the risk of heat stroke and should be considered as barriers to working in hot environments. For a more complete review of the management of risk due to heat in the mining environment, see the chapter by Johan Kielblock in the *Handbook of Occupational Health Practice in the South African Mining Industry* (Kielblock 2001).

Fatigue

Mine work, with its physical nature, remote locations, and the need for continuous production, creates an environment in which fatigue is an ever-present risk. Fly-in/fly-out work rosters with 12-hour shifts over prolonged periods and sometimes long commutes make it difficult to avoid fatigue. Despite increased mechanization of the work, the rough terrain and long shifts make the work physically tiring.

The term *fatigue* is used in various ways to describe localized tiredness of muscle systems or in the general sense of tiredness. It can be acute or chronic, and in this chapter the term is only used in the general sense.

Fatigue manifests as increasing difficulty in performing physical or mental activities. This in turn results in poor productivity and poor decision making, which can be a safety risk. In its acute form, fatigue may simply present as sleepiness, but chronic fatigue may lead to more general effects such as mood disorders, stress, digestive problems, heart disease, and mental illness. The causes of fatigue are many, but put simply, fatigue in the workplace is the sum of the interaction between the shift system, the physical workload, and personal factors (such as diet, sleep, overall health, drug use, etc.). These can be combined into an equation that provides a useful reference for managing the risk from fatigue in the workplace (Schutte 2008):

$$F_t = F_{ss} + F_{ew} + F_{pf} \quad (15.2\text{-}1)$$

where

F_t = total fatigue
F_{ss} = fatigue due to the shift system or roster
F_{ew} = fatigue due to ergonomic and work factors
F_{pf} = fatigue due to personal factors

The management of the risk from fatigue starts with recognition and assessment of the risk followed by implementation of an integrated plan to deal with the three components of the equation. Shift rosters should be designed to minimize sleep loss and the accumulation of sleep deficit. Shift lengths should be considered in conjunction with travel time as it is time awake that is important rather than just the length of the shifts. Tasks should be designed to reduce repetitive work, and work requiring sustained physical or mental effort should be broken up with short breaks. Personal factors must also be taken into account and are managed mostly through education, but some arrangements to improve quality of sleep through provision of decent accommodations and recreation facilities at the work site can be made. Where there are rotating shifts, education of the family about the social adjustments needed may also help in ensuring good quality rest for the worker. Diet, general health, and use or misuse of drugs and alcohol should also be addressed in the plan. Numerous guidelines are available regarding the prevention and management of the risks associated with fatigue. An Internet search for government documents on fatigue management will generate many reliable sources for up-to-date information on the subject.

Wellness

Employee wellness is a common part of many occupational health programs, particularly in developed countries, and is concerned with maintaining and improving the health and productivity of employees. This is achieved through targeted interventions on lifestyle management, stress management, and assisting employees in managing chronic illnesses. Employee assistance programs providing access to counseling services are a common intervention. The objective is a fit and healthy work force that has a direct impact on improved productivity and reduced health care costs to the company. In the context of a remote mine site, these programs may cross over into other aspects of community life such as recreational facilities and opportunities, provision of family accommodations, drug and alcohol programs, and many other interventions.

COMMUNITY HEALTH

The community surrounding any mine is intimately linked with the process through employment, emissions from the process, and the impact the mine has on the economy of the area. The influx of people seeking employment creates a demand and a strain on existing health care infrastructure and may provide the stimulus for the creation of additional or new infrastructure.

The mine may have some direct effects (both positive and negative) on the health of the community as well as some indirect effects. The health and welfare of the community also has an important influence on the mine's license to operate, and increasingly there is a call for health impact assessments to be carried out, both before and during operation of the mine.

These impact assessments should address the infrastructure needs, the potential health impact of the mine on the community, and the impact of any common diseases in the community on the mine. Malaria, tuberculosis (TB), and HIV/AIDS fall into this last category and there is a significant interaction in both directions.

Malaria, TB, and HIV/AIDS are perhaps the three most important communicable diseases in the world today; combined, they account for some 4 million deaths worldwide (WHO 2008; UNAIDS 2008; Global Malaria Partnership n.d.). The areas where they occur overlap considerably, and there is some interaction between the three diseases: HIV increases the risk of TB, TB hastens the onset of AIDS, and it would appear that malaria may increase the viral load in HIV infection (although this is not yet well understood). Exposure to silica dust in mining increases the risk of TB, and the in-migration generated by mining employment has a direct effect on increasing the prevalence of malaria and HIV. Since mining often occurs in countries where these diseases are prevalent and there is a direct relationship between mining and the risk of these diseases, it is important that the mining engineer has some basic understanding of the possible effects and interventions required.

The business case for managing these three diseases is compelling: companies that have good programs for managing these diseases in the workplace experience a range of bottom-line benefits that include

- Reduced employee turnover,
- Reduced absenteeism,
- Improved productivity,
- Decreased costs of health care,
- Access to international funding for mine development, and
- Enhanced company reputation and community relations.

The approach to managing these three diseases or any other disease that may be of concern to the mine should be based on a structured assessment of the risk leading to the development of a management program based on that assessment (see Table 15.2-2). Again, risk assessment is the starting point for management. It is beyond the scope of this chapter to describe the approach in detail, and only a brief overview of each disease is given. Because the three diseases have much in common, a case can be made for having an integrated approach to management. An extremely useful reference for an approach to these three diseases is the International Council on Metals and Mining's *Good Practice Guidance on HIV/AIDS, Tuberculosis and Malaria* (ICMM 2008).

Table 15.2-2 Management planning process for HIV/AIDS, TB, and malaria

Risk Assessment (Do we need the program?)	
1. Screening	Decide whether (or not) one or more of the diseases needs to be managed.
2. Scoping	Increase understanding of the nature and extent of the disease and the associated risks.
3. Situation analysis	Develop a health baseline.
4. Stakeholder analysis	Participate in a process of information sharing, problem solving, and identification of potential partners.
Risk Management (What do we need to do?)	
5. Priorities, goals, and objectives	Decide whether (or not) one or more of the diseases needs to be managed.
6. Scoping	Increase understanding of the nature and extent of the disease and the associated risks.
7. Situation analysis	Develop a health baseline.
8. Stakeholder analysis	Participate in a process of information sharing, problem solving, and identification of potential partners.

Source: Adapted from ICMM 2008.

Malaria

Malaria is an infectious disease caused by an intracellular parasite that is spread by the bite of the female *Anopheles* mosquito. The distribution of the disease is mainly tropical (see Figure 15.2-1), but it is also found in other parts of the world, carried by travelers returning from areas where the disease was picked up.

Four species of the plasmodium parasite cause malaria in humans: *Plasmodium falciparum, Plasmodium malariae, Plasmodium ovale,* and *Plasmodium vivax.* Of these, *P. falciparum* is the cause of the most severe form of malaria and has a significant mortality in nonimmune people. It is also becoming resistant to many of the more common drugs used to treat malaria.

The symptoms of malaria mimic those of influenza in the early stages and include headache, chills, fever, malaise, and vomiting. These usually start about 10–15 days after infection, and diagnosis and effective treatment is essential to prevent severe illness (which may have fatal complications).

The diagnosis is made through the use of tests similar to a pregnancy test and examination of blood under a microscope (looking for the parasite in red blood cells).

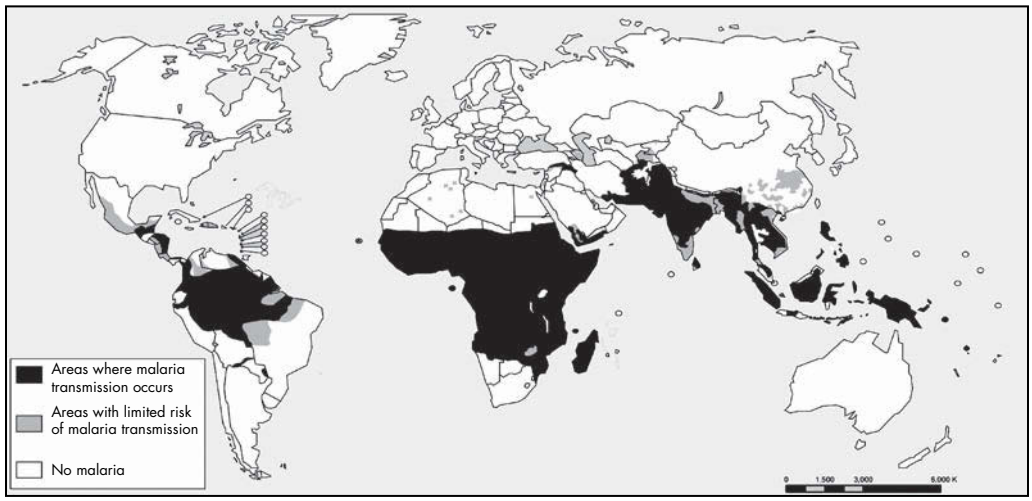
Treatment of uncomplicated cases is with combinations of a drug based on the Chinese plant *Artemisia annua* and another drug, lumefantrine. Complicated cases are usually treated with quinine. Depending on the parasite, further treatment to clear the parasite from the liver may also be required.

People who have not been exposed to malaria all their lives are regarded as nonimmune, whereas those who have been born in and grown up in malarial areas may develop some immunity. This does not mean they do not get malaria; it merely means they are less likely to get the severe forms of the disease.

Malaria Control Programs

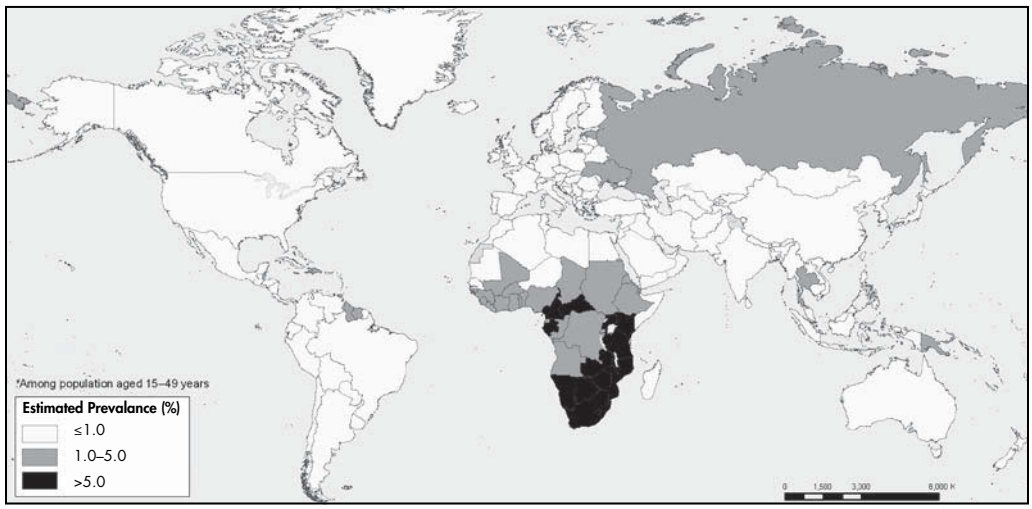
Three components of malaria control are control of the vector, control of exposure to the vector, and chemical prophylaxis. The vector is the female *Anopheles* mosquito, which breeds in stagnant water. It also has certain habits (such as settling on window and door frames) that facilitate transfer to humans. Vector control therefore involves management of sources of stagnant water and spraying campaigns. This is an engineering solution, as drains, sewers, and other sites for stagnant water have to be actively managed. Spraying of buildings with residual insecticides is an important part of many programs, and the specific insecticide used varies depending on the time and place. DDT (dichlorodiphenyltrichloroethane) is still advocated in some areas as it has long residual effects on buildings, but must obviously be used with extreme caution because of its environmental impact.

Control of exposure involves behaviors such as wearing long-sleeved shirts and long trousers, avoidance of being outdoors at dusk when the mosquitoes are most active, the



Source: WHO 2009.

Figure 15.2-1 Global distribution of malaria in 2009



Source: WHO 2007a.

Figure 15.2-2 Estimated HIV/AIDS prevalence in 2007

use of mosquito repellents (both on the person and airborne), and barrier methods such as insecticide-treated bed nets. Chemoprophylaxis involves taking medication to suppress the parasite before it can grow in the bloodstream. These drugs should not be the same as those used to treat the condition, and medical advice should be sought on which is the most appropriate drug for the area and the period of use. It may only be practical for expatriate (nonimmune) employees, but there may be some use for general prophylaxis in the community surrounding the mine, which will reduce the reservoir of the parasite. Expert advice in this regard is essential.

A common myth surrounding malaria is that taking malaria prophylaxis hides the disease and makes the diagnosis more difficult. This is untrue and, in fact, dangerous as the complications are more likely in the nonimmune person who is not taking prophylaxis. It is true that no single prophylactic measure is 100% effective, but these all rely on human behavior. A high degree of suspicion and prompt diagnosis and treatment of malaria is essential.

HIV/AIDS

In 2007, UNAIDS reported that there were an estimated 33.2 million people living with HIV/AIDS, 2.5 million new infections, and 2.1 million deaths worldwide. More than 68% of people living with HIV/AIDS are in sub-Saharan Africa, and 76% of all deaths from AIDS occurred in this region (Figure 15.2-2).

AIDS (acquired immunodeficiency syndrome) is a condition caused by infection with the human immunodeficiency virus (HIV) and has collectively become known as HIV/AIDS. The virus is a lentivirus, one of a group of viruses known as retroviruses that attack the white blood cells of the body (specifically the CD4+ T-lymphocyte). The infection results in a gradual deterioration of the immune system, leaving the individual susceptible to a variety of opportunistic infections and some cancers. Transmission of the virus occurs through contact of a mucous membrane or the bloodstream with infected body fluids such as blood, semen, vaginal secretions, and breast milk. Contact can occur through sexual intercourse,

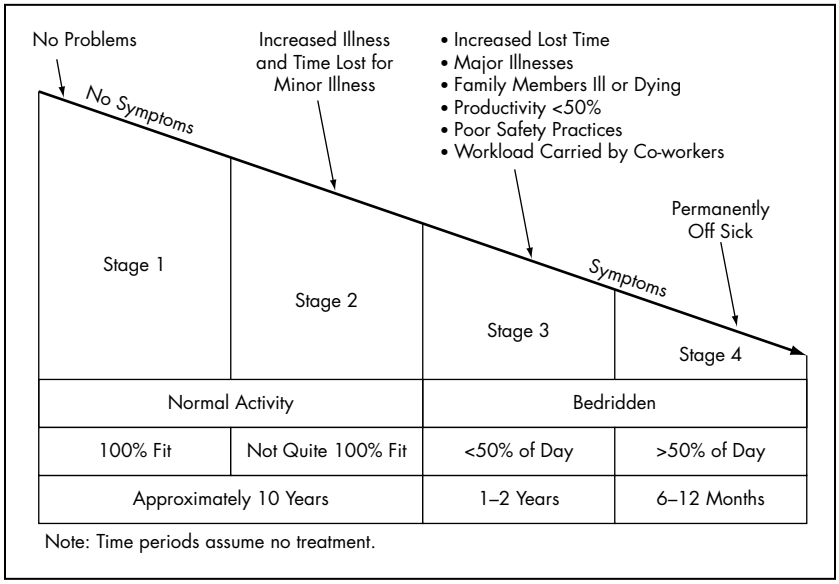


Figure 15.2-3 Stages of HIV infection

blood transfusion, use of contaminated hypodermic needles, and between mother and baby.

HIV infection carries no specific symptoms; in the primary infection it causes a short flu-like illness, but after that, the symptoms are those of the particular opportunistic infection that the person may have. The diagnosis of HIV infection is made by testing a blood (or saliva) sample for the presence of antibodies to the virus. There are various methods for antibody detection and usually the diagnosis is confirmed by a second test using a different test for the antibody. There are also tests for the presence of the virus, and viral counts are done using recombinant DNA techniques, normally used for monitoring the response to treatment.

HIV is no longer the death sentence that it was 15 years ago, and combination therapy with highly active antiretroviral drugs has made this just another chronic disease for many people. However, it is important that treatment is started before the disease has progressed too far. It is also important to prevent the spread of the virus from mother to child during childbirth and breastfeeding, and effective drug regimens exist for this purpose.

Workplace HIV Programs

HIV is preventable, and the primary goal of any workplace program should be to prevent new infections in employees. In addition to this, workplace programs should seek to provide care and support for those who are infected, and build capacity in senior managers to deal with the workplace issues of people living with HIV/AIDS. Any workplace programs developed should be aligned with and support the efforts of the local state health services.

A strong commitment and concerted efforts from management and unions, backed up by company policy, is essential for success. As a minimum, the policy should address the issues of stigma and discrimination, confidentiality, care and support, employment, and workplace voluntary counseling and testing.

The disease typically goes through four stages, and the intervention at each point is different. Figure 15.2-3 shows the stages from an employer's perspective. In the early stages of infection, there are no symptoms and relatively minor illnesses only. This may have little or no impact on productivity; however, this is perhaps the most dangerous stage as the individual is infected, infectious, and very often ignorant of their condition. They are also on a downward slope as far as health is concerned (as measured by the CD4 cell count). Voluntary counseling and testing (VCT) programs not only create an awareness of the disease among those who are HIV negative but also creates opportunities for those who are infected but still well enough to prevent further spread of the disease and maintain their health. Treatment with antiretroviral drugs can be started at the optimal time, and mother-to-child transmission prevented, but only if the infection status is known.

When the disease has progressed to the symptomatic stage, the only option is treatment and prevention of transmission. With early recognition of the infection through VCT and initiation of antiretroviral treatment at an appropriate point, Stages 3 and 4 can be almost indefinitely delayed. Unfortunately, many people leave the diagnosis of HIV and appropriate treatment to a late stage when treatment may not be very effective. However, even at this stage many people are able to return to productive employment and continue to support their families.

HIV/AIDS is seldom an infection that involves only one partner and it is essential that any management program involve the dependents of the employee. Education, access to counseling and testing, and possibly even provision of treatment should be considered. State health programs may provide all this but may need support. In addition, there are numerous donor agencies that can offer help and support in setting up and running HIV/AIDS management programs (ICMM 2008).

Tuberculosis

The global distribution of TB is shown in Figure 15.2-4. Note the similarity with both HIV/AIDS- and malaria-prevalent areas.

TB is a disease caused by a rod-shaped bacterium (bacillus) known as *Mycobacterium tuberculosis.* According to the



Source: WHO 2007b.

Figure 15.2-4 Global distribution of tuberculosis in 2007

World Health Organization (WHO), it infects about one-third of the world's population. Most of this is latent infection rather than active, but WHO estimates that in 2006, 1.5 million people died of TB, with another 200,000 dying of HIV-associated TB (Attfield et al. 2009). In latent infection, the body seals off the bacteria in small numbers and the disease remains dormant. However, this dormant infection can become active if the immune system is compromised by, for example, HIV. The estimated risk of an infected person becoming sick is between about 5% and 10% during their lifetime, but when HIV is a factor, the risk rises to between 10% and 15% per year.

Most commonly, TB causes a respiratory illness (pulmonary TB), but it can infect any organ in the body with devastating consequences. The disease is spread by droplet infection through coughing.

TB is a disease with a slow onset, and the symptoms may be insidious and difficult to diagnose, which means the diagnosis may often be late. Typically it will present with a cough, weight loss, and profuse sweating at night. The cough may be accompanied by bloodstained phlegm, but in the early stages this may not occur. The infected person, unaware, may infect many others before the diagnosis is made.

TB is a preventable and treatable condition. Prevention can occur at two places in the cycle of infection, the first being the interruption of the passage of the bacillus from an infected person to an uninfected person, and the second being the prevention of those who are infected (latent infection) from developing active disease. In the first instance, the focus is on finding and treating infectious cases, and in the second, the focus is on maintaining good health.

Treatment consists of a prolonged course (6 months) of multiple drugs (usually three), and good compliance to the treatment regimen is needed for a cure; this requires rigorous follow-up of cases by health care workers. The treatment program devised by WHO to assist with these requirements is known as "directly observed therapy short-course" (DOTS). It aims to ensure that treatment is taken as instructed and may involve family, friends, co-workers, and health care workers assisting the individual to comply with the treatment course. Despite this, poor compliance with the treatment remains a problem, and in resource-constrained countries, where medical staff numbers are low and drug supplies may not be continuous, drug-resistant strains of the bacteria have emerged. This has been complicated by the HIV epidemic, and multidrug-resistant (MDR) and extensively drug-resistant (XDR) cases are becoming commonplace in some countries. This is an extremely serious situation in some locations as XDR cases mostly have a fatal outcome and MDR cases are increasingly difficult to treat. Education, awareness, and an ability to detect cases is therefore essential for both prevention of the spread of TB and the treatment of those who are ill.

From a mining company perspective there are several things that can be done to manage the risk from TB. First, hostel-type accommodations should be avoided if at all possible. Second, education programs, together with active case-finding, should be undertaken. These can be accomplished through collaboration with local health authorities or through the mine's own health services. Where the risk coexists with HIV, the two programs can be combined.

For a detailed and thorough review of this subject, see the ICMM *Good Practice Guidance* document (ICMM 2008).

CONCLUSION

Although the focus may appear to be on HIV/AIDS, TB, and malaria due to their global importance, it must not be forgotten that there are numerous other infectious and noninfectious diseases that can and do affect the productivity and health of miners. These lend themselves equally to the approach advocated in this chapter, namely, risk assessment followed by a structured management program if the condition is found to be a significant problem to the mining operation. The same approach at an individual level is equally important for the management of personal risk due to the possible health risks that may arise.

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CHAPTER 15.3

Mine Ventilation

Michael A. Tuck

INTRODUCTION

Ventilation in underground mines is an essential element to ensure safe production of minerals. The basic objectives of any ventilation system are clear and simple: to provide airflows of sufficient quantity and quality to provide oxygen, and to dilute contaminants to safe concentrations and remove them from the mine. These basic requirements are enshrined in law in all countries; however, the quantity and quality requirements can vary greatly from country to country depending on a large number of factors as well as the situation being ventilated: coal, metalliferous, industrial minerals, or gassy/nongassy. The overall requirement is that workers must be able to live, work, and travel within an environment that is safe and provides reasonable comfort. Ventilation can also be employed in situations where human intervention is not required, but an adequate oxygen supply is required to operate and/or cool machinery or to provide oxygen to combustion processes.

Mine ventilation can be defined as the application of the principles of fluid mechanics and thermodynamics to the flow of air through mine openings. Air is introduced into the mine from the atmosphere, it flows around the mine, and it is expelled back into the atmosphere by the creation of a differential pressure between the intake and return openings of the mine. To achieve this, the mine ventilation system consists of interconnected airways and working zones within the mine, connections to the surface, fans to produce airflow, and control devices to ensure that the air courses around the mine as required. Controlling a ventilation system requires quantity as well as air quality control, and ventilation planning techniques are employed to achieve this.

This chapter concentrates on the fundamentals of mine ventilation engineering and its application to both coal and hard-rock mining. Other chapters in this handbook deal specifically with individual mine air pollutants.

PURPOSE OF MINE VENTILATION

The primary purpose of ventilation is to maintain an atmosphere that promotes a safe and effective work environment. This includes management of oxygen content and levels of contaminants including toxic gases, combustion products, and dust, plus maintaining workplace temperature conditions at levels where workers can achieve maximum efficiency. Legal limits relating to exact standards for the work environment vary depending on the purpose of ventilation as well as from country to country and state to state within some countries. Other than where prescribed by a definitive legal standard, a useful minimum oxygen concentration to apply to the design of any ventilation system is 19% oxygen by volume.

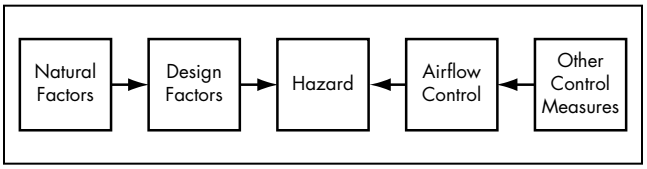
Pollutants enter a ventilation airstream from a variety of sources. The aim of any ventilation system is to dilute any pollutants to a safe level and then remove them. The range of pollutants that can be released into a ventilation system depends on the situation under consideration. Some of the more common types of pollutants encountered include

- Gases,
- Products of combustion,
- Dusts,
- Heat and humidity,
- Radioactive solids and by-products, and
- Diesel particulate matter.

Figure 15.3-1 shows the factors that contribute to hazards encountered in mine ventilation and the methods of control. The factors are divided into those imposed by natural conditions and those imposed by the design and engineering decisions regarding how the mine is to be developed and how ore is to be produced. The natural factors include geology, depth, surface climate, gases contained in the rock mass, ground water and other fluids, the physical and chemical properties of the rock, and the age of the workings. The engineering design of a mine impacts the hazards through the

- Mining method used,
- Method of fragmentation of rock,
- Rate of fragmentation of rock,
- Use of backfilling materials,
- Types of power used underground,
- Types and number of vehicles in the mine,
- Method of ore and waste removal,

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Source: Adapted from McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-1 Factors contributing to hazards and methods of control in mine ventilation

- Types and quantities of materials stored and/or used in the mine, and
- Mine layout and the number and interconnectedness of the airways in the mine.

All mines are different, so the pollutants and their effects vary between mines, locations, and the cycle of operations within a mine (see Appendix B, Coal Mine Gas Chart). As an example, a deep mine in Canada may have issues with cold conditions in the shallower sections of the mine requiring mine heating during the winter months, while at the deeper levels the air has to be cooled by refrigeration. In addition to the ventilation pollutants, other hazards associated within a mine ventilation system are underground fires and explosions.

The main method of controlling the atmospheric conditions in an underground mine is by airflow. Fans generally induce the flow of air in mines, although natural ventilation due to air density changes can provide some airflow in modern mines. Main fans handle the whole mine airflow (singly or in combination) and are commonly sited on the surface, although at a number of mines they are sited underground. Legislation in some areas may require that main mine fans can only be located on the surface and in some cases may have to be reversible. While main fans generally handle all the mine airflow, other fans are used in mines to boost the airflow energy to allow air to be circulated over greater distances. These are called booster fans, and they generally serve specific areas of the mine. In mine development headings, a through-flow of ventilation may not be possible as the drive is single entry. In these cases an intake and return path for the air needs to be developed. In modern mining this is generally undertaken by using auxiliary fans to pass air through ducts to ventilate the blind entries. Alternatives include the use of brattice cloth and in some cases jet fans. Air doors, regulators, stoppings, seals, and air crossings control airflow distribution further.

While airflow is the primary method of achieving the goals of mine ventilation, in some situations the airflow required would become too costly to mine economically. In such cases other methods of control can be employed to reduce the load on the ventilation system so that the ventilation costs can be decreased. These methods include methane drainage, mine heating and cooling systems, dust suppression systems, and various other monitoring and control systems. These basic elements of a mine ventilation system are described in the video *Breathe and Live*, which can be viewed on the Mine Safety and Health Administration Web site (MSHA 1973). This video presentation details the hazards present from a ventilation perspective in underground mining, provides details of ventilation systems and devices, and includes an overview of equipment to provide ventilation and systems of auxiliary control such as heating and cooling systems. It is an old presentation, but it details the fundamentals of ventilation in an easy-to-understand format.

THEORY OF MINE VENTILATION

The background theory of mine ventilation comes from fluid mechanics and thermodynamics, specifically the Bernoulli equation, the steady-flow energy equation, and the Chezy–Darcy law for frictional pressure drop in a pipe. Atkinson (1854) developed the basis for the theory of mine ventilation that is still in use today. Taking fluid mechanics theory as a base and introducing a number of simplifying assumptions, he developed a series of equations that still form the basis of current ventilation theory and modern mine ventilation planning software. As described here, the theory uses an assumption of incompressibility. As modern mines progress to greater and greater depths, this assumption frequently becomes invalid due to changes in air density. Where changes in elevation and temperature produce changes in density greater than 5%, the assumption of incompressibility becomes flawed and must be taken into account.

Mine Airway Resistance (Atkinson's Law)

The Atkinson equation is based on the fundamental work undertaken by the French hydraulic engineers Chezy and Darcy. The Chezy–Darcy equation states the following for a pipe:

$$p = fL\frac{per}{A}\rho\frac{u^2}{2}$$

where

p = pressure drop (Pa)
f = friction coefficient
L = length (m)
per = perimeter (m)
A = cross-sectional area (m^2)
ρ = density ($kg/m^{3)}$
u = velocity (m/s)

For fully developed turbulent flow, Atkinson assumed that the coefficient of friction was a constant. Also assuming that the air density was a constant, he was able to combine the constants into a single factor known as the Atkinson friction factor, k (kg/m^3),

$$k = \frac{f\rho}{2}$$

giving

$$p = kL\frac{per}{A}u^2$$

This is known as Atkinson's equation and can be written in terms of air quantity Q (m^3/s), where $Q = uA$, giving

$$p = kL\frac{per}{A^3}Q^2$$

For any given airway, the length, L, perimeter, per, and cross-sectional area, A, are known. Ignoring the dependency on density, the friction factor varies only with the roughness of the airway lining for fully developed turbulent flow, so all

these factors can be collected together to form a single variable or characteristic number, R, for that airway, where R is known as the airway resistance.

$$R = kL\frac{per}{A^3}$$

This gives the square law of mine ventilation:

$$p = RQ^2$$

where

p = pressure across airway (Pa)
R = resistance or Atkinson resistance ($N \cdot s^2/m^8$)
Q = air quantity (m^3/s)

The equation assumes a constant density; usually this is standard air density of 1.2 kg/m^3. Values of R quoted in design tables usually refer to this standard density. For other air densities, ρ, the following can be used to give the frictional pressure drop and resistance:

$$p = k_s L\frac{per}{A^3}Q^2\frac{\rho}{1.2}$$

$$R = k_s L\frac{per}{A^3}\frac{\rho}{1.2}$$

where k_s is the friction factor at air density 1.2 kg/m^3.

From these calculations it is clear that the degree of roughness of an opening underground (k) has an important influence on its resistance and the cost of passing air through that opening. It also directly influences the heat transfer rate between the rock and the airstream. The friction factor, k, can be determined in a number of ways, details of which can be found in McPherson (1993):

- By analogy with similar airways, from ventilation surveys
- By direct measurement
- From design tables (see Table 15.3-1)
- From geometric data

Airway resistance (R) is an important concept in ventilation engineering. The square law states

$$p = RQ^2$$

This shows the resistance to be a constant of proportionality between the frictional pressure drop, p, and the square of the airflow, Q, at a specified air density. The parabolic form of the plot on a p, Q graph as shown in Figure 15.3-2 is known as the airway resistance curve.

The cost of passing any airflow through an airway varies directly with the resistance of that airway. This can be determined by evaluating the air power consumed in an airway using air power = $p \times Q$ or knowing $p = RQ^2$, resulting in air power = RQ^3.

Using a cost per kilowatt-hour for electricity, the cost to ventilate an airway can be simply determined. The equation also shows that doubling the airflow in a particular airway has the effect of multiplying the air power and hence the ventilation cost by a factor of 8.

Table 15.3-1 Friction factors (referred to an air density of 1.2 kg/m^3) for a variety of mining situations

Type of Airway/Ducting	Friction Factor k, kg/m^3
Rectangular airways	
Smooth concrete lined	0.004
Shotcrete	0.0055
Unlined with minor irregularities only	0.009
Girders on masonry or concrete walls	0.0095
Unlined, typical conditions, no major irregularities	0.012
Unlined, irregular sides	0.014
Unlined, rough or irregular conditions	0.016
Girders on side props	0.019
Drift with rough sides, stepped floor, handrails	0.04
Steel arched airways	
Smooth concrete all round	0.004
Bricked between arches all round	0.006
Concrete slabs or timber lagging between flanges all round	0.0075
Slabs or timber lagging between flanges to spring	0.009
Lagged behind arches	0.012
Arches poorly aligned, rough conditions	0.016
Shafts	
Smooth lined, unobstructed	0.003
Brick lined, unobstructed	0.004
Concrete lined, rope guides, pipe fittings	0.0065
Brick lined, rope guides, pipe fittings	0.0075
Unlined, well-trimmed surface	0.01
Unlined, major irregularities removed	0.012
Unlined, mesh bolted	0.014
Tubbing lined, no fittings	0.007–0.014
Brick lined, two side buntons	0.018
Two side buntons each with a tie girder	0.022
Longwall face with steel conveyor and powered supports	
Good conditions, smooth wall	0.035
Typical conditions, coal on conveyor	0.05
Rough conditions, uneven faceline	0.065
Ventilation ducting	
Collapsible fabric ducting (forcing systems only)	0.0037
Flexible ducting with fully stretched spiral reinforcement	0.011
Fiberglass	0.0024
Spiral-wound galvanized steel	0.0021

Source: McPherson 1993; with permission from Springer Science and Business Media.

Shock Losses

The theory outlined previously enabled the frictional pressure drop to be evaluated for an airway. In addition to the frictional or rubbing resistance, other losses of energy, termed *shock losses*, can be incurred. Whenever airflow is required to change direction, additional turbulent vortices are set up. These consume energy as they propagate and the resistance of the airway can increase significantly. These losses occur at bends, junctions, and changes in cross sections due to obstructions and regulators, and at points of entry/exit from the system.

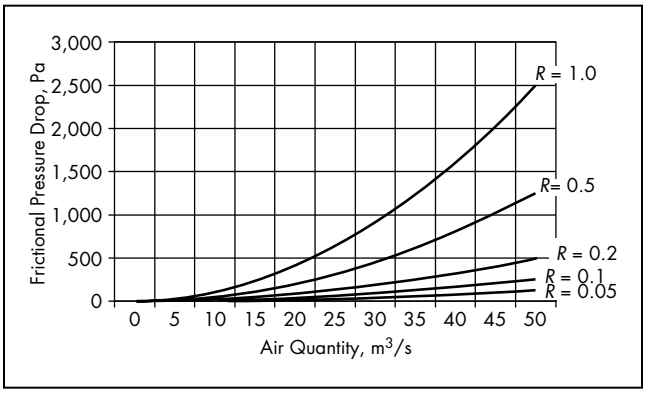
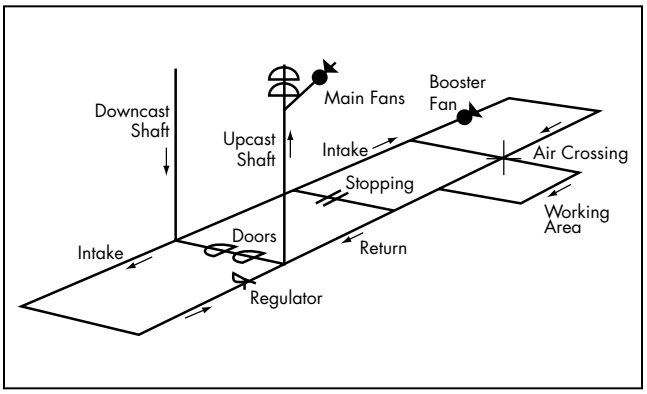


Figure 15.3-2 Mine airway resistance



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-3 Simple mine ventilation system

The effects of shock losses are the most uncertain of all the factors affecting resistance. Two methods are applied to the assessment of the additional resistance caused by shock losses. The head loss or pressure loss caused by a shock loss can be expressed as a shock pressure loss or p_{shock}:

$$p_{shock} = X\rho(u^2/2)$$

where

X = shock loss factor
ρ = air density (kg/m^3)
u = air velocity (m/s)

This can be turned into a shock loss resistance, R_{shock}, using the square law:

$$R_{shock} = R_s = X\rho/2A^2$$

This can also be turned into an equivalent length due to shock loss using the Atkinson equation:

$$R_s = (kL_{eq}\,per/A^3)(\rho/1.2)$$

or

$$L_{eq} = 1.2\,X\,A/(2k\,per)$$

Shock loss factors, X, are listed in McPherson (1993).

VENTILATION SYSTEMS

Every underground mine is unique in terms of its geometry/layout, extent/size, geology, and pollutants. As such, the pattern of airflows and pressure drops through the airways that make up the mine workings are also highly variable. However, certain features of all ventilation systems have some commonality, and classifications of ventilation systems and subsystems can be identified. The aim of this section is to discuss the essential characteristics of underground ventilation systems, describe the main elements of ventilation infrastructure, and introduce some of the technical terms employed by ventilation engineers. Ventilation systems can be broadly classified as follows:

- Mine systems
- District systems
- Auxiliary ventilation systems

Mine Systems

Figure 15.3-3 illustrates a schematic mine ventilation system showing a basic system layout and some of the ventilation control devices that can be encountered. Figure 15.3-3 shows the following:

- Fresh air enters via downcast shafts or other surface connections.
- Air flows along intake airways to working areas where most pollutants are added.
- Contaminated air returns back through the system along return airways.
- The return air passes back up to the surface via upcast shafts or other surface return connections.

However, ventilation is more complicated than this simplistic explanation. Air, like most fluids, will try to follow the path of least resistance from one point to another, and the path to the areas requiring ventilation usually lies along a high-resistance path. In order to get the air to the areas where it is needed, airflow control devices are required. These fall into two main categories: active devices that add energy to air to direct it to where it is needed, or passive devices that add resistance to flow routes to encourage the air to flow along other routes. Examples of active devices include main fans and booster fans. Examples of passive devices include air doors, regulators, air crossings, stoppings, and seals, which by adding resistance increase pressure drop. Both types are discussed in the following sections.

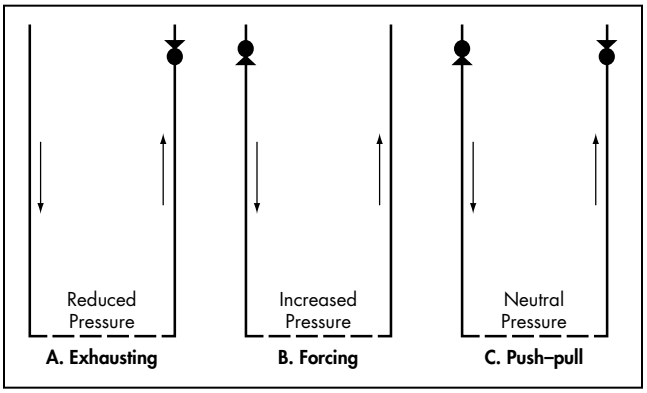
Active Devices

Main fans. Fans are the primary means of producing and controlling the ventilation airflow. Main fans (singly or in combination) handle all the air passed through the mine and are usually located at or near the surface. Main fan systems can be

- Forcing,
- Exhausting, or
- Push–pull (a combination of the two).

Surface fans have the following advantages:

- Simpler installation
- Simpler testing
- Simpler maintenance



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-4 Possible locations for main fans

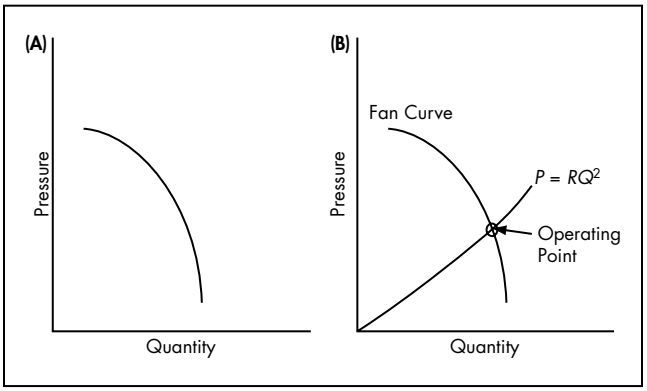


Figure 15.3-5 (A) Fan pressure–quantity characteristic curve, and (B) superimposed mine resistance characteristic and fan operating point

- Easier access
- Better protection in an emergency situation

Main fans can be located underground when fan noise is considered a problem or if shafts need to be completely free of airlocks. Underground fans incur additional problems associated with additional doors, airlocks, and leakage paths.

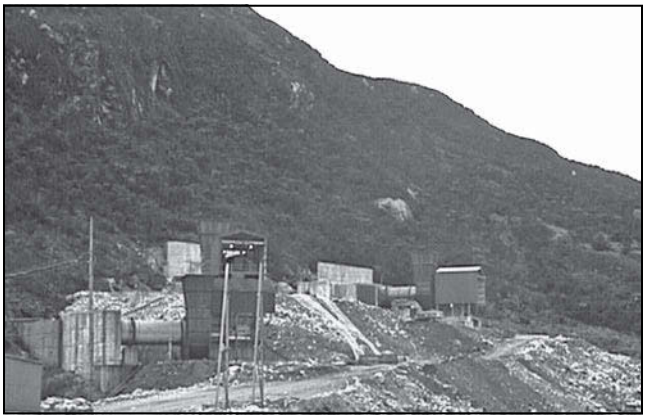
The choice of surface main fan location is illustrated in Figure 15.3-4. In Figure 15.3-4A, exhaust ventilation connects the main fan to the upcast shafts; in Figure 15.3-4B, forcing ventilation connects the main fans to the downcast shaft; and in Figure 15.3-4C, push–pull ventilation connects main fans to both the upcast and downcast shafts.

Most mines ventilate using exhaust ventilation (i.e., connecting the main fan to the exhaust shaft). The choice of whether to force or exhaust depends on four factors:

1. Gas control (control during falls of barometer and fan stoppage favors exhausting)
2. Transportation (selection depends on whether transporting mineral through intake or return airways)
3. Fan maintenance (exhaust fan subjected to air carrying dust, warmer air, and water droplets, thus the combined effects of impaction and corrosion are greater than in forcing case)
4. Fan performance (cooler air implies lower mass flow duty for a forcing fan and hence for a certain airflow, a reduced pressure and running costs). However, this effect is usually not great.

The combination or push–pull system of ventilation is most generally applied in metalliferous mining situations such as where caving techniques are employed and where the zone of fragmentation has extended to the surface. Maintenance of a neutral pressure underground minimizes the leakage of air between the workings and the surface. This is of particular concern where the fragmented rock is liable to spontaneous combustion. Push–pull systems are also employed in multishaft mines where multiple main fans are employed. In this case it offers the potential to achieve a better distribution of airflow, better control over pressure and leakages, greater flexibility, and reduced operating costs. However, multifan systems require skilled adjustment, planning, and balancing.

Fans generate energy and airflow by converting rotational mechanical energy into hydraulic energy. Numerous types of



Courtesy of Calizaya et al. 2000.

Figure 15.3-6 Main ventilation fans at PT Freeport DOZ mine

fans exist; the two most commonly applied fans in mine ventilation are centrifugal fans and axial fans, although in some situations mixed-flow fans are used. Every fan has its own characteristic set of performance or characteristic curves in terms of pressure–quantity, power–quantity, and efficiency–quantity curves. Figure 15.3-5A illustrates a typical pressure–quantity curve for a fan, while Figure 15.3-5B shows a mine system resistance curve superimposed on the fan curve. The point at which the two curves intersect is termed the *operating point* for the fan showing the fan pressure and associated flow. A typical main fan installation is illustrated in Figure 15.3-6.

Booster fans. As air flows through a ventilation network, it loses energy. As mines become deeper and more extensive, the mine resistance increases and the reliance on a single main (i.e., its energy input) results in a reduction in the ventilation volume delivered to the extremities of the mine. Rather than upgrading the main fans to overcome this loss in energy, it is better to boost the energy of the airflow in sections of the mine by adding additional booster fans underground. This has numerous advantages including

- No need to upgrade existing main fan facilities, avoiding capital and operating cost increases; and
- Only a proportion of the whole mine airflow is boosted, saving operational costs compared to having to boost the whole mine airflow.



Courtesy of De Souza et al. 2003.
Figure 15.3-7 Booster fan installation

Figure 15.3-7 shows a typical booster fan facility. De Souza et al. (2003) provide a good example of the design and performance of a booster fan facility and the benefits that can be obtained. These include increased ventilation flows in remote mining sections, improved pressure control, improved energy use, and increased air velocities providing enhanced cooling power in hot mines.

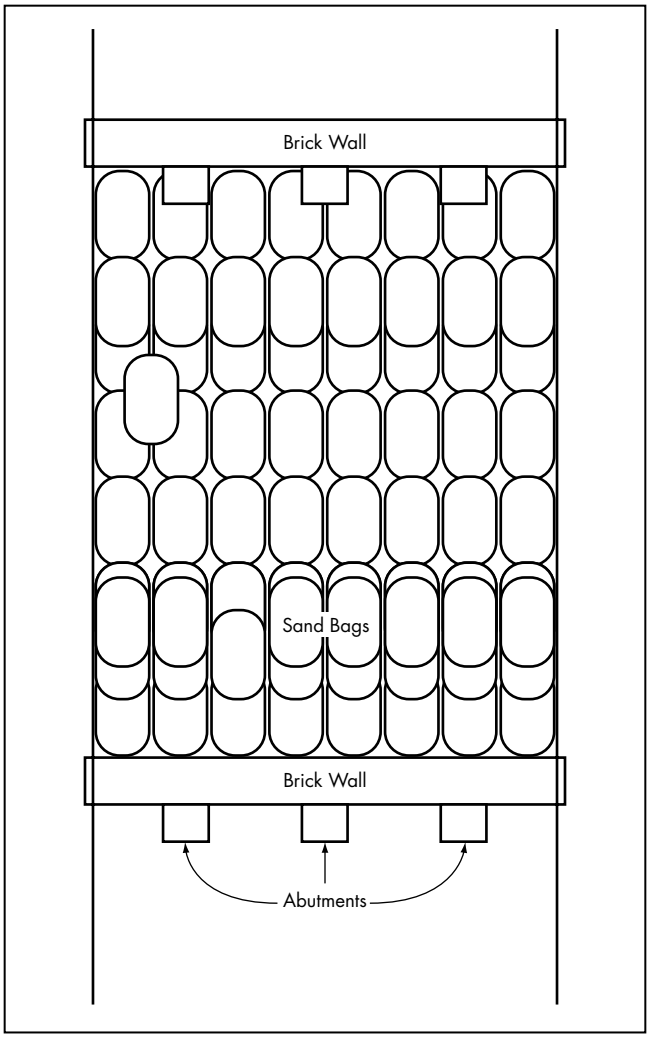
Passive Devices

Air doors and airlocks. When access is still required between intake and return, air doors are employed to minimize short-circuiting of ventilation flow by increasing the resistance of a potential short-circuit route. Ventilation doors between main intake and returns are usually in the form of two or more sets of air doors and termed *airlocks*. Doors should always be hinged to open toward the higher pressure so that they are self-closing. Doors can be constructed with a variety of materials including wood or steel.

Regulators. A passive regulator is simply an air door fitted with an adjustable orifice, the purpose of which is to reduce the airflow in a particular section downstream to a defined airflow by increasing the resistance of the airway. When the airflow in a section of a mine needs to be higher than that normally obtainable, active regulation by the installation of a fan to boost the airflow is required. These are termed *booster fans* and can also be called "active regulation" as the energy of the airflow is increased.

Air crossings. Where intake and return airways need to cross over each other, the leakage between the two must be controlled by means of an air crossing. Air crossings can take the following forms:

- **Natural:** one of the airways is elevated above the other leaving a beam of strata separating the two airways.
- **Intersection:** this is the more usual method. Both airways intersect during construction, and the roof of one or the floor of the other is excavated to expand the zone of intersection. Masonry or concrete blocks or a steel structure with metal or timber shuttering separates the two airways. The high-pressure side is further sealed by the application of shotcrete or another sealant.
- **Prefabricated air crossings**

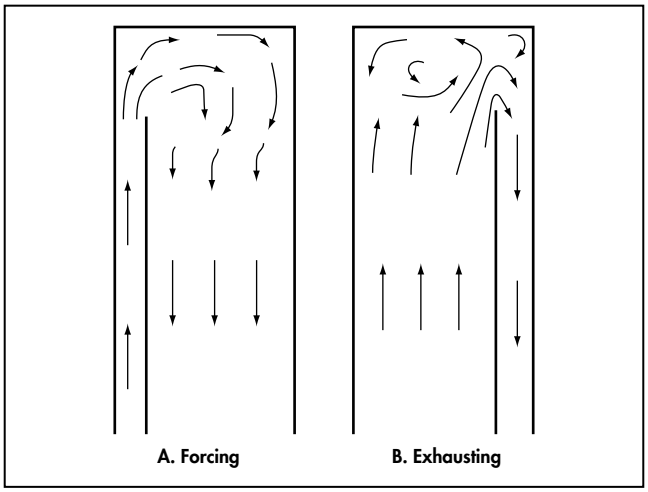


Source: Chalmers 2008.
Figure 15.3-8 Explosion-proof seal

In all cases the material from which the crossing is fabricated should be fireproof and capable of maintaining its integrity in case of a fire. Aluminum and other low-melting-point materials should not be used to construct an air crossing.

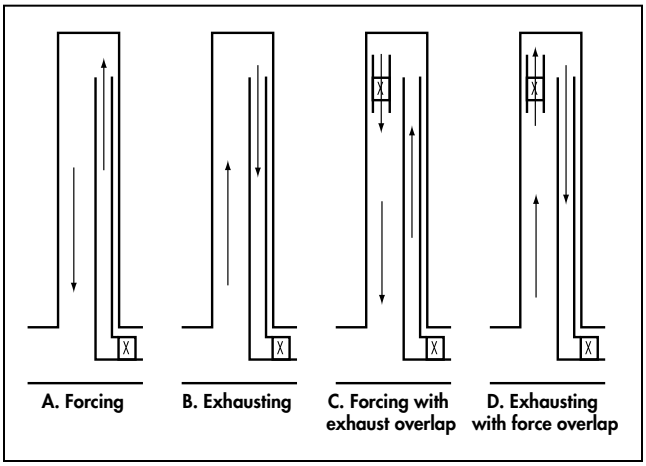
Stoppings and seals. In developing a mine, connections are necessarily made between the intake and return airways for a number of reasons including to provide quick access during mine development and to provide through ventilation. When these are no longer required for access they should be blocked by stoppings. These should be well constructed, keyed into solid rock, and leakage through should be minimized (e.g., by being coated with a sealant). As a short-term method, brattice curtains can be used when pressure differentials are low.

When required to isolate abandoned areas of the mine, seals should be used. Chalmers (2008) details the requirements for sealing abandoned coal mining areas, and many of the details discussed in his paper also apply to other forms of mining. Seals are constructed for a number of reasons, such as preventing unwanted air loss from a ventilation system and to preclude oxygen from entering mined-out areas to prevent spontaneous combustion. Regardless of the reason for the



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-9 Line brattices used in auxiliary ventilation



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-10 Auxiliary fan and duct systems

seal, Chalmers presented a number of characteristics that a good seal should have:

- Highly resistant to airflow
- Strong
- Fire resistant
- Easily and rapidly deployed
- Ability to cope with strata convergence
- Insolubility
- Nonreactive
- Sampling points installed
- Stable, not easily blown over

The exact construction depends on local conditions and materials available; however, a distinction can be made between seals that are designed to purely stop air leakage and a seal that is required to be explosion proof. Figure 15.3-8 shows the typical construction of an explosion-proof seal.

Backfill is becoming more widely used, particularly in hard-rock mining. Backfilling worked-out stopes can also act as a very effective seal.

District Systems

District systems are discussed later in this chapter in the "Coal and Metaliferrous Ventilation" section.

Auxiliary Systems

Systems used to supply air to the working faces of blind headings, sublevels, and so on comprise three basic types: line brattices, fan and duct systems (forcing, exhausting, or overlap systems), and air movers and jet fans.

Line Brattices

Brattice cloth, hung from floor to ceiling, creates an artificial intake or return in a dead-end drive. Commonly employed in room-and-pillar operations, its major disadvantage is the resistance added to the mine ventilation network at the most sensitive point in the system. Also these systems are very leaky. The advantages are that they are cheap, require no power, and produce no noise. Figure 15.3-9 illustrates the use of line brattices.

Fan and Duct Systems

These systems consist of a duct or a multiplicity of ducts connected together to form a ventilation network with a fan or fans to provide the air movement. As such, fan and duct systems can be simple or complex ventilation networks. The main difference between these systems and the ventilation systems discussed previously is that they rely on open atmospheric connections in a factory, mine, or other context to complete the ventilation circuit.

Fan and duct systems basically come in four forms as shown in Figure 15.3-10:

1. Forcing systems
2. Exhausting systems
3. Forcing systems with an exhaust overlap
4. Exhausting systems with a force overlap

Within such systems the usual choice of fan is an in-line axial fan. In mining situations, if the dead-end heading is greater than 30 m in length, such systems are usually the only means of ventilation applicable. Care should be taken to ensure that the pressure and quantity characteristic of the fan is commensurate with the resistance offered by the duct and the airflow required. The latter of these depends on the pollutants to be removed. The resistance of the duct is established in the same way as for other ventilation systems by determining the frictional losses associated with the duct and adding extra losses incurred at bends, changes in the cross section, and entry and exit losses.

The choice of which system to choose depends on the pollutants of greatest concern. The higher velocity emerging from the exit of a forcing duct provides a scouring action, which effectively sweeps the area. This is good for removing gases by providing turbulent mixing. Where heat is the main concern, forcing systems provide high airflows and enhance air-cooling power. The main disadvantage of the forcing system is that pollutants added to the airstream pass back via the open atmosphere along the heading in mines, thus polluting the general atmosphere relatively slowly.

Where dust control is the primary hazard, the exhausting system is preferred as the dust is contained within the ducting and fresh airflows along the heading. The main disadvantage of the exhausting method is the lack of an effective scouring

action as there is no high-velocity jet of air to achieve this. Another disadvantage of exhaust systems is that rigid or reinforced ducting needs to be used. These are more expensive than the flexible ducting types that can be used in forcing systems. The force/exhaust and exhaust/force overlap systems have been developed to overcome the disadvantages of the single forcing or exhausting systems. The selection of which of these methods to use depends on the pollutants under consideration.

Where long lengths of ducting are required (e.g., in tunneling work), the resistance of the duct may be such that multiple fans connected in series may be needed. As ducting leaks, it is not advisable to cluster fans at one end of the ducting system as the pressures developed will enhance leakage. The preferred method is to space the fans out along the duct to avoid excessive gauge pressures. Multiple fans must be interlinked electrically and airflow or pressure transducers used to detect accidental severing or blockage of the duct.

Air Movers and Jet Fans

Air movers are used to enhance and/or control air movement within localized areas. They are freestanding and can be powered by a number of sources including electricity, compressed air, and water. Jet fans, otherwise known as ductless, vortex, or induction fans, are common and come in a number of forms and sizes. A spray of water can also generate airflow.

VENTILATION SURVEYS

A ventilation survey is an organized procedure to acquire data to determine the quantity, pressure, and air quality distribution within a mine. The accuracy and required detail of the survey depend on the purpose of the survey. Surveys are undertaken for a variety of reasons, including

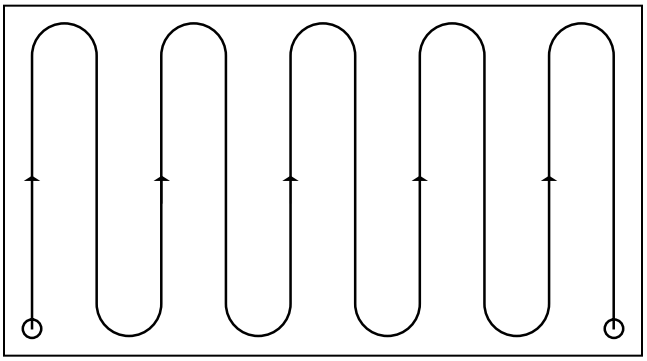
- Mandatory requirements,
- Ensuring that working areas of the mine receive the airflow required in an efficient and effective manner,
- Keeping ventilation plans and network models up to date,
- Verification purposes, and
- Planning purposes.

A major objective of ventilation surveys is to measure and obtain the frictional pressure drop, p, and corresponding airflow, Q, for each branch within the ventilation network. From these data the following may be calculated for both planning and ventilation control purposes:

- Distribution of airflows, pressure drops, and leakage
- Air-power losses and the distribution of ventilation operating costs around the network
- Volumetric efficiency
- Branch resistances
- Natural ventilation effects
- Friction factors

In addition to pressure and airflow measurements, other measurements are taken either as an integral part of the survey or separately. These will include airway area, airway perimeter, wet and dry bulb temperatures, barometric pressures, dust levels, radon levels, and concentrations of gaseous pollutants.

There are two main aspects to a ventilation survey: the determination of air quantity, and pressure drop.



Source: Adapted from McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-11 Path of an anemometer traverse

Air Quantity Surveys

The volume of air passing a given point in an airway or duct, Q, is normally determined as the product of the cross-sectional area of the airway or duct, A, and the mean air velocity, u:

$$Q = u \times A$$

Thus, most common techniques of determining Q are combinations of the methods available for determining area and mean velocity. Modern methods of determining air speed or velocity can be divided into three groups depending on

1. Mechanical effects,
2. Dynamic pressure, and
3. Thermal effects.

Mean velocity at a point can be determined either by a traversing instrument that integrates measurements as the cross section is traversed or mathematically combining fixed-point measurements to arrive at an average air velocity. Details of the methods used to undertake such measurements are described by numerous authors such as McPherson (1993) and are summarized in the paragraphs that follow.

Figure 15.3-11 shows a typical method used to undertake an anemometer traverse measurement. These are undertaken with a vane anemometer or similar type of integrating instrument. The anemometer should be attached to a rod about 1.5 m in length or greater to allow the whole cross section to be traversed. Anemometers are relatively insensitive to yaw; this is misalignment with the airflow, so long as the angle of yaw does not exceed about 5°.

The observer faces the airflow and holds the instrument at least 1.5 m away from his or her body by attaching the instrument to an extendable arm. The instrument is then started while another observer starts a stopwatch. The instrument is then moved as shown in Figure 15.3-11, the aim being to sweep equal areas of the traverse in equal times. The traverse should not take less than 60 seconds. After the whole area is swept, the anemometer is stopped, the total flow through the instrument noted and the time taken, allowing the average velocity to be computed. The process is repeated until three readings yield the same average velocity to within ±5%.

At least two sets of traverses are recommended to be taken at different locations within the airway. The measuring locations should be chosen to avoid potential problems wherever possible. Booking of results should include the following:

- Observers' names
- Location of station, time, and date
- Anemometer readings and corrections
- Air dry and wet bulb temperatures and barometric pressure
- Dimensioned sketch of the cross section
- Calculation of area
- Calculation of air volume flow

An anemometer traverse can also be undertaken at the ends of ducts but not if the duct diameter is less than six times the anemometer diameter. Anemometers can also be used for fixed-point measurements with corrections applied to obtain the real flow. These corrections are determined via traverses undertaken previously at the same location. When the duct/anemometer ratio of 6 precludes application of an anemometer survey, pitot tube techniques are most commonly employed, as described in McPherson (1993).

The measurement of a cross-sectional area can be done in a number of ways:

- Simple taping
- Offset method
- Profilometer
- Photographic methods

Pressure Surveys

The purpose of a pressure survey is to determine the frictional pressure drop corresponding to a certain flow in an airway. Basically there are two methods, the more accurate being the trailing hose or gauge and tube method, which is time-consuming but preferred where foot travel is relatively easy. The second method is the barometer or altimeter method, used where access is difficult such as in multilevel mines and in shafts. Details of these methods can be found in Hemp (1989) and are briefly described in the following paragraphs.

The principle of the trailing hose method is simple: A hose is laid along the length of the airway being surveyed and a manometer or other suitable differential pressure gauge is connected to one end of the hose. In the case of a horizontal airway, this gauge will indicate a differential pressure exactly equal to the pressure loss. When the airway is not horizontal, the differential pressure will differ from the pressure loss; however, methods are available to calculate the pressure loss derived from the airway equations, which are modified versions of the steady-flow energy equation for mining conditions.

In its simplest form, the barometer method involves the simultaneous measurement of barometric pressures and wet and dry bulb temperatures at either end of the airway. Airflow also needs to be measured. If the elevations of the two measuring stations are accurately known, the pressure and/or energy loss can be determined from the airway equations.

MINE VENTILATION NETWORK ANALYSIS

An essential component in the design of new mines is the quantified planning of the distribution of airflows, determining the location and duties of fans and other ventilation control devices to achieve the goal of providing acceptable environmental conditions throughout the ventilation system. In existing mines it is better to plan ahead to ensure that new fans, shafts, and other airways are available in a timely manner for efficient ventilation of the mine and extensions to the mine. Operating mines are dynamic operations with new workings continually being developed and older ones ending their productive life. Ventilation planning should, therefore, be a continuous and routine process.

The laws of fluid mechanics control the behavior of air in airways, ducts, and pipes. As shown previously, with regard to mining this involves using Atkinson's equation, the square law, and in some cases the laws of thermodynamics. Ventilation network analysis is concerned with the interactive behavior of airflows within the branches of a complete and integrated network. Ventilation network analysis problems can be addressed and formulated quite simply as stated by McPherson (1993):

- If the resistances of the branches of a network and how these branches are interconnected are known, how can the distribution of airflows for given locations and duties of fans be predicted?
- If the airflows in specific branches are known, how are the combination of fans and the structure of the network to provide these airflows determined?

Ventilation network analysis is a generic term for a number of techniques, a family, which enable such questions to be addressed. For a ventilation network there are theoretically an infinite number of combinations of fans, regulators, and airway resistances that will produce the desired flow distribution. Practical considerations will limit the number of acceptable alternatives. One essential aspect of any technique to be employed is that it must be easy to use, and be sufficiently rapid and flexible to allow investigation of multiple solutions.

Fundamentals of the Ventilation Network Analysis

The fundamental relationships that govern the behavior of electrical current within a network of conductors were first recognized by Gustav R. Kirchhoff (1824–1887). These same basic relationships are now known as Kirchhoff's laws and are also applicable to fluid networks.

Kirchhoff's first law (hereafter referred to as Kirchhoff 1) states that the mass flow entering a junction equals the mass flow leaving that junction (a consequence of the law of conservation of mass) and can be written as

$$\sum_j M = 0$$

where M is the mass flows, positive and negative, entering junction j. However,

$$M = Q\rho$$

where Q is the volume flow (m^3/s) and ρ is the air density (kg/m^3), thus,

$$\sum_j Q\rho = 0$$

If the density variation around the ventilation network is negligible (i.e., the flow can be regarded as incompressible), then

$$\sum_j Q = 0$$

This provides a means of checking the accuracy of airflow measurements taken around a junction when undertaking a ventilation survey.

Kirchhoff's second law (hereafter referred to as Kirchhoff 2) stated in its simplest form is that the algebraic sum of all pressure drops around a closed path, or mesh, in the

network must be zero, taking into account the effect of fans and/or natural ventilation pressures, a consequence of the law of conservation of energy. The steady-flow energy equation for a single airway can be written as

$$\frac{\Delta u^2}{2} + \Delta Zg + W = \int VdP + F \text{ J/kg}$$

where

u = air velocity (m/s)
Z = height above datum (m)
g = acceleration due to gravity (m/s^2)
W = work input from fan (J/kg)
V = specific volume (m^3/kg)
P = barometric pressure (Pa)
F = work done against friction (J/kg)

Considering a number of branches forming a closed loop or mesh within a network, the algebraic sum of all ΔZ must be zero and the sum of all the changes in kinetic energy, $\Delta u^2/2$, is negligible. Summing the remaining terms around the mesh, m, gives

$$\sum_m \int VdP + \sum_m (F - W) = 0$$

The summation of the $-VdP$ terms is the natural ventilation energy, NVE, which originates from thermal additions to the air, thus

$$\sum (F - W) - \text{NVE} = 0$$

This calculation is in joules per kilogram and can now be converted to pressure units by multiplying by density, ρ.

$$\sum (F - W)\rho - \rho\text{NVE} = 0$$

But if $\rho_F = p$, the frictional pressure drop, and $\rho W = p_f$, the rise in total pressure across a fan, and ρNVE = NVP, the natural ventilation pressure, then Kirchhoff's second law becomes

$$\sum (p - \rho_f) - \text{NVP} = 0$$

Kirchhoff's laws can be applied to fluid networks that have either compressible or incompressible flow. In the case of compressible flow, the analysis is undertaken on the basis of mass flow; in the case of incompressible flow, the analysis is undertaken on the basis of volume flow.

From this point forward, incompressible flow will be assumed. The square law needs to be incorporated within Kirchhoff's laws for incompressible flow.

Kirchhoff 1

$$\sum_j Q = 0$$

Kirchhoff 2

$$\sum_m (RQ\,|Q| - p_f) - \text{NVP} = 0$$

where

R = resistance (N·s^2/m^8)
$|Q|$ = the absolute value of airflow

This ensures that the frictional pressure drop always has the same sign as the airflow and removes the complication of squaring or square-rooting negative numbers.

Deviations from the Square Law

The fundamental relationship of mine ventilation is the square law $p = RQ^2$. This is developed from the Chezy–Darcy relationship and is valid for fully developed turbulent flow. The resistance, R, is a function of the friction factor, k, which is itself related to the Darcy friction factor, f. If the flow regime falls into the transitional zone or the laminar flow zone, f, and, hence, k becomes a function of the Reynolds number, Re, the value of R varies with airflow. In these cases, the square law is inapplicable.

For the transition zone the following relationship applies:

$$p = RQ^n$$

Values of n have been reported to be in the range of 1.8–2.05. In the case of laminar flow, the following equation can be used:

$$p = R_L Q$$

where R_L is laminar resistance.

In the following sections it is assumed that the square law applies.

Methods of Solving Ventilation Networks

Basically there are two methods for solving ventilation networks:

1. Analytical methods formulate governing laws to produce equations that yield exact solutions.
2. Numerical methods iterate until a solution is found to a required degree of accuracy.

Analytical Methods

The following paragraphs describe analytical methods to solve ventilation networks.

Equivalent resistances. This is the most elementary method of network analysis. When two or more airways are connected together in series or parallel, each of these resistances can be combined into a single equivalent resistance in a similar manner to electrical circuits. A mine ventilation network is made up of a number of roadways connected together; these connections between branches can be broken down into roadways connected in series and in parallel.

- Series circuits—The common element is that the air quantity flowing in each branch is the same. The equivalent resistance of n branches connected in series is

$$R_{\text{total}} = R_1 + R_2 + \ldots + R_n \text{ (N·s}^2\text{/m}^8\text{)}$$

- Parallel circuits—The common element is that the pressure drop across every branch is the same. The equivalent resistance if n branches are connected in parallel is

$$1/\sqrt{R} = \sum_1^n 1/\sqrt{R}\,n \text{ (N·s}^2\text{/m}^8\text{)}$$

The square root term in the equation results from the square law of mine ventilation rather than the linear Ohm's law for electrical circuits.

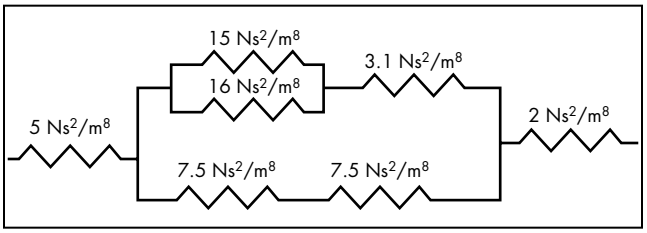


Figure 15.3-12 Ventilation network for equivalent resistance calculation (Example 1)

Example 1. A section of a mine ventilation system is connected together as shown in Figure 15.3-12. If the pressure drop across the system is 1.1 kPa and the resistances are as indicated, determine the flow in each airway.

Solution. First the equivalent resistance of the circuit needs to be determined. Taking the bottom line of the parallel portion of the circuit first, the equivalent resistance is $7.5 + 7 = 14.5$ N·s^2/m^8. The equivalent resistance of the top line now needs to be determined.

Taking the parallel portion first,

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{15}} + \frac{1}{\sqrt{16}} = 0.508$$

Thus $R = 3.875$ N·s^2/m^8, thereby showing the advantage of placing airways in parallel as the resistance of the combination is less than the individual resistances that make up the parallel combination.

This is in series with the 3.1 N·s^2/m^8 resistance, so the resistance of the top line is 6.975 N·s^2/m^8.

The equivalent resistance of the whole parallel section can now be evaluated:

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{14.5}} + \frac{1}{\sqrt{6.975}} = 0.641$$

Thus, $R = 2.433$ N·s^2/m^8.

Combining this in series with the intake resistance of 5 and the return resistance of 2 gives a total system resistance of 9.433 N·s^2/m^8.

The total inlet and outlet flow for the system can now be evaluated from the square law of mine ventilation, $p = RQ^2$.

$$Q = \sqrt{\frac{p}{R}} = \sqrt{\frac{1{,}100}{9.433}} = 10.8 \text{ m}^3/\text{s}$$

Knowing this, the pressure drop for the intake and outlet can be evaluated to determine the pressure drop acting over the parallel element of the circuit:

$$p = 7 \times 10.8^2 = 816.5 \text{ Pa}$$

So the pressure drop over the parallel element of the circuit is 1,100 – 816.5 = 283.5 Pa. This acts across both legs so the flow in the bottom leg is

$$Q = \sqrt{\frac{p}{R}} = \sqrt{\frac{283.5}{14.5}} = 4.42 \text{ m}^3/\text{s}$$

and the flow through the upper leg is 6.38 m^3/s.

To determine the pressure drop over the parallel portion of the upper leg, the pressure drop through the 3.1 N·s^2/m^8 resistance needs to be determined. This is

$$p = 3.1 \times 6.38^2 = 126 \text{ Pa}$$

So the pressure drop over the parallel portion of the upper leg is 283.5 – 126 = 157.5 Pa. The flows can now be determined. Through the 15 N·s^2/m^8 resistance it is

$$Q = \sqrt{\frac{p}{R}} = \sqrt{\frac{157.5}{15}} = 3.24 \text{ m}^3/\text{s}$$

and through the 16 N·s^2/m^8 leg the flow is 3.14 m^3/s.

This example shows the effect of placing resistances in parallel to reduce the overall mine resistance and the dominating effect that the entry and exit from the system has on the overall pressure drop in the system. In other words, the entry and exits from a mine ventilation system are primary consumers of air power in a mine ventilation circuit—not surprising as they carry the full system flow.

Direct application of Kirchhoff's laws. Kirchhoff's laws allow equations for each independent junction in a network and for each independent mesh. Solving these two sets of equations obtains the branch airflows that satisfy both laws. If b branches exist in a network, then b airflows need to be determined, and b independent equations are required.

If a network contains j junctions, Kirchhoff 1 can be written down for each in turn. But as each branch is assumed to be continuous with no intervening junctions, a branch airflow denoted by Q_i entering a junction will automatically imply that Q_i leaves a neighboring junction at the other end of that same branch. Therefore, by the last junction, all airflows will have been symbolized; thus, the number of independent equations from Kirchhoff 1 is $j - 1$.

This leaves $b - (j - 1)$ or $b - j + 1$, with further equations to be established from Kirchhoff 2. It is, therefore, necessary to choose $b - j + 1$ independent closed meshes around which to sum the pressure drop. The direct application of this method to mine circuits may result in hundreds of equations to be solved simultaneously. Computer assistance is required, as manual solution is limited to small networks.

Numerical Methods

Numerous numerical methods exist. The main differentiation between the methods revolves around mesh selection and numerical solving of the equations. The most robust—though not always the most efficient—method was devised by Professor Hardy Cross in 1936 at the University of Illinois (United States) for water distribution systems. This technique is the most widely used and was modified for use in mine ventilation systems by Professors F.B. Hinsley and D.R. Scott at the University of Nottingham (England) in 1951.

Example 2. Figure 15.3-13 shows the system resistance curve for a single representative branch of a ventilation network. If airflow Q is reversed, the frictional pressure drop, p, also becomes negative. When establishing a flow distribution, the true airflow in the branch, Q, is initially unknown. Assume a flow, Q_a, that is less than the true value by an amount ΔQ:

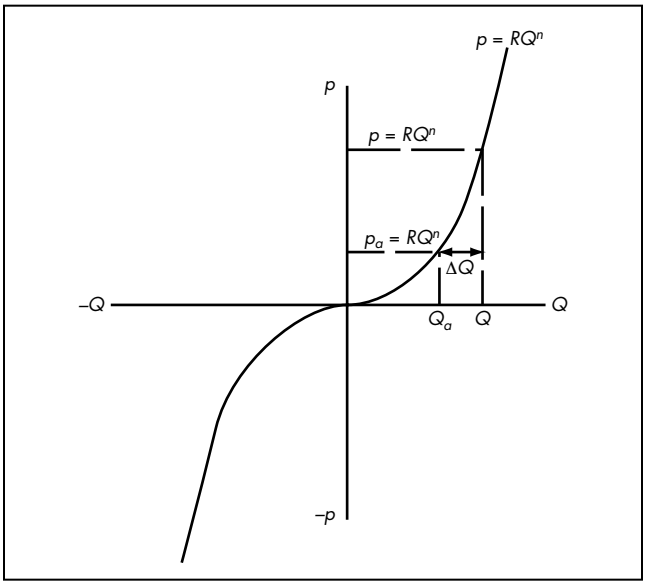
$$Q = Q_a + \Delta Q$$

Now find ΔQ. The square law can be written as

$$p = R(Q_a + \Delta Q)^2$$

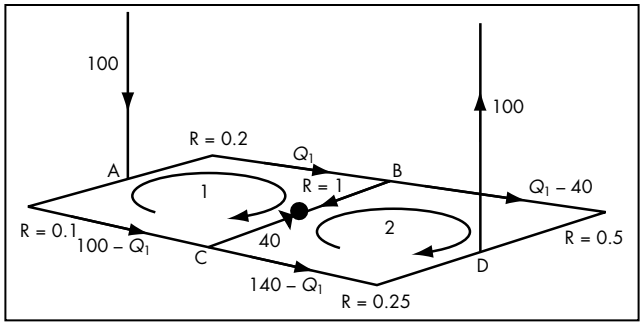
More generally,

$$p = R(Q_a + \Delta Q)^n$$



Source: Adapted from McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-13 Iterative technique to solve ventilation networks



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-14 Network for worked example of direct application of Kirchhoff's laws

Going back to the square law, this expands to

$$p = RQ_a^2 + 2RQ_a\Delta Q + R(\Delta Q)^2$$

The frictional pressure drop corresponding to the assumed airflow is

$$p_a = RQ_a^2$$

where the error in p is

$$\Delta p = p - p_a$$

Substituting from the equations gives

$$\Delta p = 2RQ_a\Delta Q + R(\Delta Q)^2$$

Assuming ΔQ^2 is small compared to $2RQ_a\Delta Q$, the approximation can be written as

$$\Delta p = 2RQ_a\Delta Q$$

Thus, $\Delta Q = \Delta p/2RQ_a$. The technique of dividing a function by its first derivative in order to estimate an incremental correction factor is a standard method of finding roots of functions and is usually called the Newton–Raphson method. The technique can also be applied to meshes as explained in McPherson (1993). For the general case where $p = RQ^n$, the following is true:

$$\Delta Q_m = \frac{-\sum (RQ_a \mid Q_a \mid - p_f - \text{NVP})}{\sum (2R \mid Q_a \mid + S_f + S_{nv})}$$

where

S_f = slope of the pressure/quantity characteristic curve for the fan
S_{nv} = slope of the pressure/quantity characteristic curve for the natural ventilation effects

The Hardy Cross technique can now be summarized as follows:

1. Draw a network schematic and choose at least $(b - j + 1)$ closed meshes such that all branches are represented. Convergence to a balanced solution will be improved if each high-resistance branch is included in only one mesh.
2. Make an initial estimate of airflow, Q_a, for each branch.
3. Traverse one mesh and calculate the mesh correction factor, ΔQ_m, from the equation.
4. Traverse the same mesh in the same direction and adjust each of the airflows by the amount ΔQ_m.
5. Repeat Steps 3 and 4 for each mesh.
6. Repeat Steps 3, 4, and 5 until Kirchhoff 2 is satisfied to an acceptable degree of accuracy.

An example of this method is given in McPherson (1993).

Example 3. Figure 15.3-14 shows a network. The downcast and upcast shafts pass 100 m^3/s and the resistance of each branch is shown. A booster fan in the central branch passes 40 m^3/s. Determine the distribution of airflow and the total pressure developed by the booster fan.

Solution. Inspection of the figure shows that it cannot be solved by equivalent resistances. Therefore, it is a complex network. The given airflows are shown on the figure with flow from A to B denoted by Q_1. The flows in the other branches are expressed in terms of Q_1. The shafts can be eliminated, as their airflows are already known. Thus,

$b - j + 1 = 5 - 4 + 1$ independent meshes are required

where

b = number of branches in the network
j = number of junctions in the network

Meshes along the closed paths ABCA and BDCB are selected. In order to apply Kirchhoff 2 to the first mesh, consider the pressure drops around that mesh branch by branch. Frictional pressure drops are positive in the direction of flow and p_b:

Branch	Frictional Pressure Drop (RQ^2)	Fan
AB	$0.2\ Q_1^2$	–
BC	1.0×40^2	$-p_b$
CA	$-0.1(100 - Q_1)^2$	–

Summing these up according to Kirchhoff 2 gives

$$0.2\ Q_1^2 + 1{,}600 - p_b - 0.1(100 - Q_1)^2 = 0$$

Collection of like terms gives

$$p_b = 0.1Qt_1^2 + 20\ Q_1 + 600$$

For the second mesh,

$$0.5(Q_1 - 40)^2 - 0.25(140 - Q_1)^2 - 1 \times 40^2 + p_b = 0$$

which reduces to, substituting for p_b from the previous equation,

$$0.35Q_1^2 + 50Q_1 - 5{,}100 = 0$$

Solution of this quadratic equation gives Q_1 = 68.83 or –211.69 m^3/s. The only practicable solution is 68.83 m^3/s, thus the flows are

Branch	Air Quantity, m^3/s
AB	68.83
BD	28.83
BC	40
AC	31.17
CD	71.17

The required booster fan pressure is

$$p_b = 0.1(68.83)^2 + 20\ (68.83) + 600 = 2{,}450$$

Ventilation Network Simulation Programs

There are numerous ventilation network simulation packages on the market. These are mathematical models and are designed to simulate real systems. Use of these packages has revolutionized mine ventilation planning (Mine Ventilation Services 2007; Ventsim 2010).

MINE VENTILATION PLANNING

Planning of new mines and extensions to existing mines is a critical part of overall mine planning. Without adequate airflow provided in a timely and efficient manner, likely outcomes include poor environmental conditions for the work force, with associated health and safety issues, and reduced production. Ventilation planning is also essential in the testing of emergency and evacuation procedures.

A generalized approach to mine ventilation planning is shown in Figure 15.3-15. The figure has been simplified for clarity and indicates that a step-like approach should be taken with constant feedback between stages, similar to other mine design problems. All mines are different in terms of the pollutants to be diluted and removed, so generic rules are not applicable to all situations. Ventilation planning should be an integral part of overall mine planning, as well as production planning. With regard to ventilation planning, the following factors should be considered:

- The method should fit with the remainder of the mining operation.
- The method should be flexible.

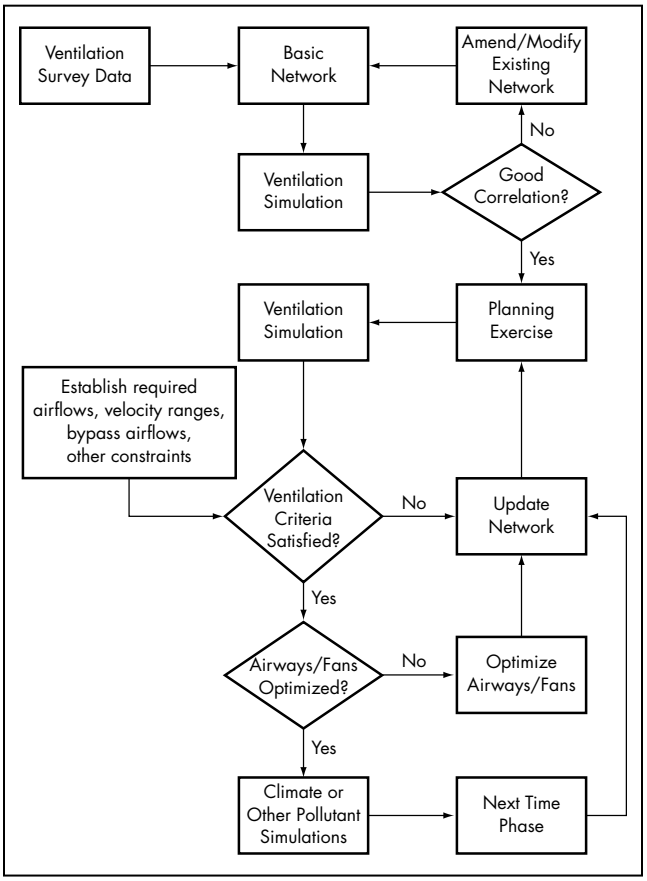


Figure 15.3-15 Generic mine ventilation planning

- The method should be economical.
- The method must be adaptable for extensions and future mining activities.

With reference to Figure 15.3-15, the initial stage is to create a simulation model of the mine. For an existing mine, this should be created using the results of ventilation surveys. For a new mine, typical design data such as friction factors should be combined with geometric data and layout data from other mine planning activities. For extensions to existing mines, a combination of survey data and design data will need to be used. After this has been achieved, the next stage is to test the ventilation network for existing mines to ensure that the results of the simulation correlate well with the real ventilation survey data. If a reasonable correlation (±5%) is not achieved, the ventilation network needs to be modified by examining assumptions and data against reality until a suitable correlation is achieved. Possible reasons why correlation is not achieved include compressibility occurring in the network while an incompressible analysis is being undertaken, airways not conforming to actual design specifications, shock losses not being accounted for, and numerous other reasons. For a new mine, after underground workings exist, ventilation surveys should be undertaken and compared against the simulated network to ensure that the simulations match the mining reality.

After a suitable network simulation has been developed, the next stage is to ensure that the predicted flows are reaching the production and other areas that require ventilation. To do this, the required airflows, air velocity limits, bypass

Table 15.3-2 Recommended maximum air velocities

Area	Velocity, m/s
Working faces	4
Conveyor drifts	5
Main haulage routes	6
Smooth-lined airways	8
Hoisting shafts	10
Ventilation shafts	20

airflow, requirements to eliminate recirculation, and other requirements such as establishing limits for the concentration of pollutants need to be established. In some cases these limits are dictated by law, in other cases not. In some countries and states, best practice is mandated by health and safety legislation.

In the case of gases, the required airflow can be simply determined if the release rate of the gas is known and the required maximum concentration is known as detailed by McPherson (1984). For dust and some other requirements such as bypass airflows, air velocity limits such as those shown in Table 15.3-2 can be applied. Minimum velocity standards can also be fixed; a commonly applied standard is 0.3 m/s (McPherson 1993). For diesel dilution, a common method is to specify airflow per rated kilowatt of diesel power in a location. See Table 15.3-3 for a list of these requirements for several countries (Brake and Nixon 2008). For heat and humidity, heat stress indices need to be applied to define safe work, to reduce work and stop-work conditions, and to undertake climate simulations in addition to ventilation simulations (Tuck 2008).

Other limits can also be applied; for example, in wet exhaust shafts where condensation or water flow results in airborne droplets, the air velocity should not lie in the range from 7 to 12 m/s as water blanketing can occur between this range of velocities (McPherson 1993).

After the required air velocities, quantities, and other criteria have been set, the results of the ventilation simulation need to be carefully checked against these criteria. If the criteria are not satisfied, amendments need to be made to the network such as changing the duty of fans, changing airway lining or other inputs, and the simulation rerun until the criteria are realized.

The third stage involves an economic overview of the system. Mine ventilation systems can be major electricity consumers, so trying to optimize the system to minimize the total capital and operating cost is prudent.

Following this stage, if ventilation alone is not sufficient to dilute and remove the contaminants, the design requirements for auxiliary systems such as methane drainage, mine cooling, and mine heating can be investigated as required.

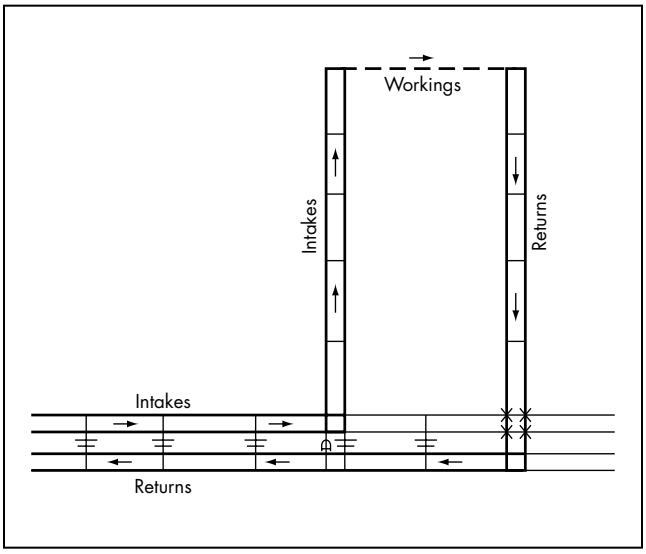
These considerations give a broad, simple overview of the ventilation design process; a fuller treatment can be found in numerous textbooks and conference proceedings (McPherson 1993). However, planning of mine ventilation is not simple, and a lack of care and attention at each stage of the process can result in a system that fails to deliver the outcomes required. A good illustration of this is provided by Brake and Nixon (2008), who examined the requirements to correctly estimate the primary airflow requirements in underground metaliferrous mines. Brake and Nixon found that airflows were underestimated for failure to

Table 15.3-3 Diesel dilution requirements for selected locations

Location	Diesel Airflow Requirement	Comments
Western Australia	0.05 m^3/s per kW	
Queensland, Australia	None	Was 0.04 m^3/s per kW
Ontario, Canada	0.06 m^3/s per kW	
Manitoba, Canada	0.92 m^3/s per kW	Uses 100/75/50 rule
United Kingdom	None	
United States	0.032 to 0.094 m^3/s per kW	Typically design to 0.08 m^3/s per kW
Chile	0.063 m^3/s per kW	
South Africa	None	Exhaust/overlap system required for firing anytime. Typically design to 0.06 m^3/s per kW.
Indonesia	0.067 m^3/s per kW	

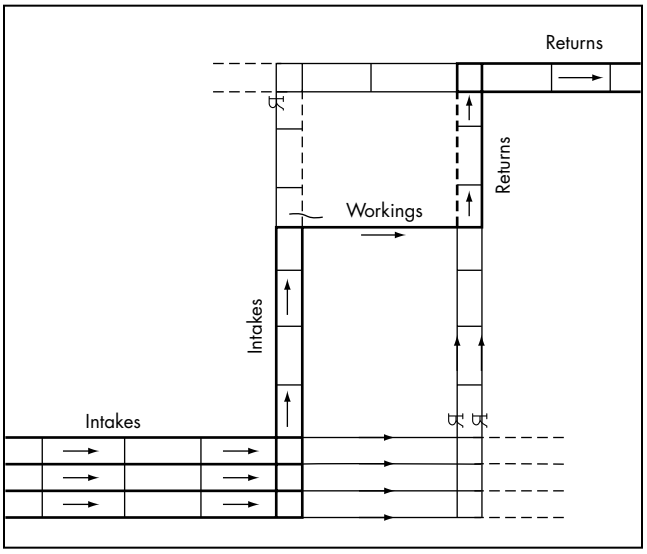
- Provide for leakage in the auxiliary ventilation ducts,
- Provide for leakage in the workings,
- Supply essential anti-recirculation bypass flows,
- Provide for diesel equipment mobility,
- Provide for ramps and other underground fixed plant and infrastructure travelways,
- Understand the relationship between airway dimensions and minimum wind speeds,
- Recognize which is the critical airborne contaminant to be diluted,
- Plan for reasonable capacity increases and other contingencies,
- Provide for likely changes in diesel technology,
- Provide for increased mine resistance,
- Understand the impact of both increased mine resistance and leakage on airflow requirements and fan performance,
- Understand that providing extra capacity in a ventilation system at the start of the project while it is an initial additional capital expenditure is less expensive than fixing the ventilation at a later date, and
- Properly assess ventilation planning and implementation lead times.

An important part of mine ventilation planning is that it should be a continual process and that the effects of time need to be carefully assessed. A good example of this has been reported by Sedlacek (1999). In this paper, four scenarios were investigated to enable the Kidd Creek mine (Canada) to develop to greater depths with the potential to extend to 2,987 m below the surface. The situation in 1998 involved the use of several fans arranged in a series of parallel arrangements to provide the required airflow. In total, 220 intake and return level exhaust fans were in use. In addition, the airflow within the system was seriously constrained by an inadequate return airway system. Sedlacek details four possible solutions to the long-term ventilation needs for Kidd Creek; of these, the preferred option involved the mine being separated into 15 distinct ventilation districts or circuits, each controlled by a single major supply fan. The overall volume of air is controlled by strategically located single exhaust fans placed at or close to the top of the major exhaust raises. This system reduced the number of main fans from more than 200 to 18 and enabled considerable savings in operating costs.



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-16 U-tube ventilation



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-17 Through-flow ventilation

A more modern approach to ventilation design is the potential to apply ventilation on demand to working stopes (Hardcastle et al. 1999). The principle can also be applied to mine cooling systems (Gundersen et al. 2005). Production stopes in mining do not require the same amount of air continuously; currently, ventilation design assumes worst-case conditions for specifying the airflow required—good from a safety perspective, but poor regarding economics. The principle of ventilation on demand is to match the air supply to the requirements of individual stopes depending on the point in the production cycle where they are currently operating. This has the potential to reduce overall mine airflows and to achieve considerable savings in power costs. The principle is still in the early stages of development. One main concern is that mine ventilation systems are highly inertial in nature and the changes induced to provide ventilation on demand for a specific stope could have a poor impact elsewhere in the system, even if only for a short, transient period of time. This could have serious safety implications.

COAL AND METALLIFEROUS VENTILATION

Before describing some of the main points of ventilating coal and metalliferous mines, a general overview of the two main forms of ventilation applied in both cases will be made. There are two broad classifications: U-tube systems (Figure 15.3-16) and through-flow systems (Figure 15.3-17).

Figure 15.3-16 shows the principle of U-tube ventilation. Air flows toward and through the workings, then returns along adjacent airways separated from the intakes by stoppings and doors. Room-and-pillar (R&P) layouts and advancing longwalls tend to be of this type. A problem with this system is the close proximity of the main intake and return airways; there can be a high pressure drop across the control devices separating these, such as doors and stoppings, which can lead to high leakage losses from the main intake to the main return before the working areas.

Figure 15.3-17 illustrates the through-flow system. In this system, intakes and returns are separated geographically, and adjacent airways are all intakes or returns. There are far fewer stoppings and air crossings, but additional regulators are required to control the airflow through the working area. Examples include the parallel flows from downcast to upcast across the multilevels of metal mines and the back-bleeder system of longwall retreating. In these systems, although leakage occurs, it is generally smaller and can be used to enhance flows as, for instance, at the return end of the working zone, which in a hot and humid mine is generally one of the main problem areas; the leakage flow increases the air velocity, enhancing the cooling power of the airflow.

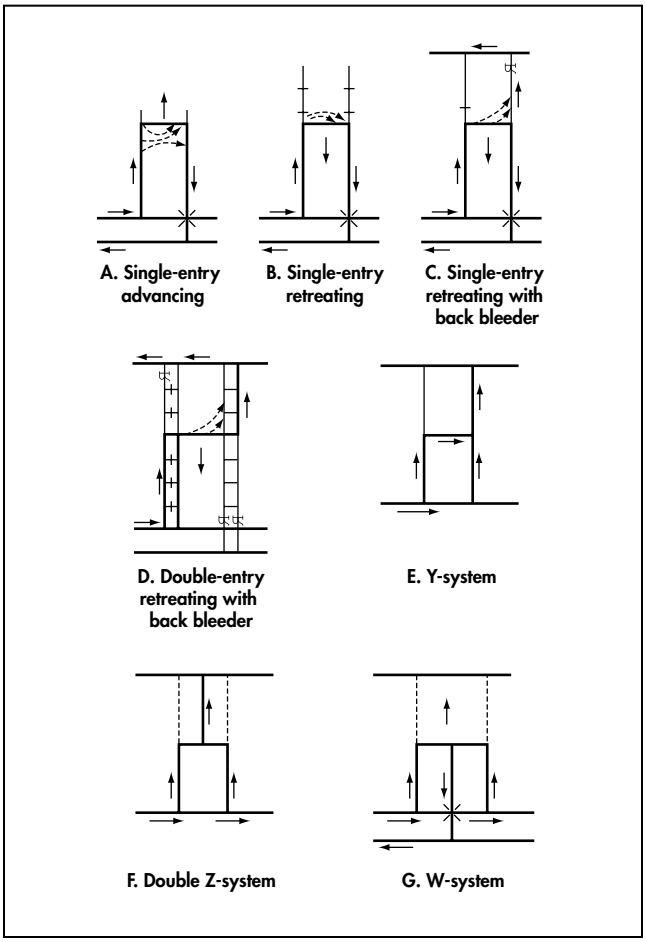
Division can be made between stratified deposits (longwall or R&P systems) such as coal, gypsum, potash, and trona, and metalliferous ore-body deposits (also known as hard-rock deposits). Stratified deposits usually have a simpler two-dimensional system; ore-body deposits tend to be more complicated. This is due to their three-dimensional nature and the need to have more numerous working sections, not all of which are working at a particular time (in other words, air requirements are on a day-to-day basis, and air must be directed to where it is needed).

Stratified Deposits

Underground coal, potash, and other tabular forms of mineral deposits use either the longwall method of mining or the R&P (bord-and-pillar) mining method. Layouts can vary from country to country, however, the following sections highlight the modes of airflow distribution used. Other differences between these types of mines and hard-rock mines are the dominant pollutants that need to be accounted for. In coal mines, methane gas, dust, and in some cases diesel emissions are the dominant pollutants. In hard-rock mines, diesel pollutants, dust, and blast fumes are of concern. Because all mines will have different critical pollutants, assessment should be made on a mine-to-mine basis.

Longwall Systems

Two main features have influenced the design of longwall systems: methane and other gas control from the waste areas, and the high rate of rock breakage. Both have increased the



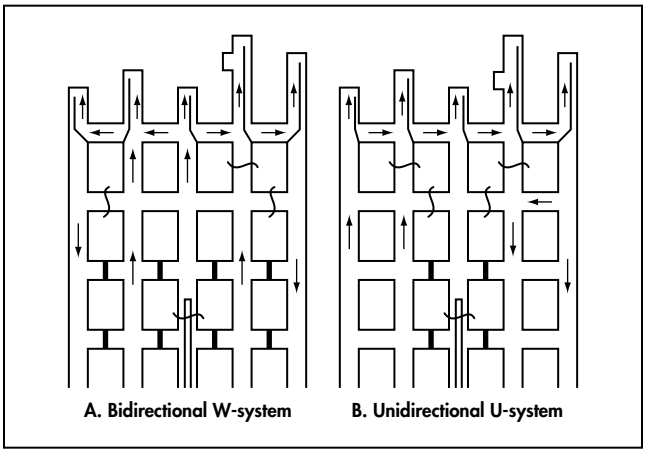
Source: Adapted from McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-18 Longwall district ventilation systems

production of dust, gas, heat, and humidity. A typical layout is shown in Figure 15.3-18.

With reference to Figure 15.3-18, single-entry systems are common in Europe where the deposits are mined at considerable depth and geotechnical issues would require very large pillars to be left if multiple access systems were used. With the advancing system, some leakage occurs through the waste area; control of this is achieved by roadside packing. This leakage air flushes gas out of the waste and can cause problems of high gas concentration at the return end of the face. Leakage of air through the waste can cause gob fires. In the retreating system in Figure 15.3-18B, the amount of air leaking through the waste is generally reduced, but the same problems can occur as in the advancing case.

The back-bleeder system employs a back or bleeder return to constrain the gas fringe back in the waste to prevent flushes of gas onto the face. The method illustrated in Figure 15.3-18C is a combination of U-tube and through-flow ventilation. Multiple opening systems such as those illustrated in Figure 15.3-18D are more common in the United States, South Africa, and Australia, where shallower coal seams are mined. Two or more entries are driven using R&P mining; these serve as the lateral boundaries of the longwall panel. If retreat mining is used, back bleeders can be used to control



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-19 Room-and-pillar development with line brattices (high face resistance)

waste or gob gas. In some states, air from conveyor belt drives cannot be used to actively ventilate production areas; use of multiple entry systems enables this requirement to be achieved.

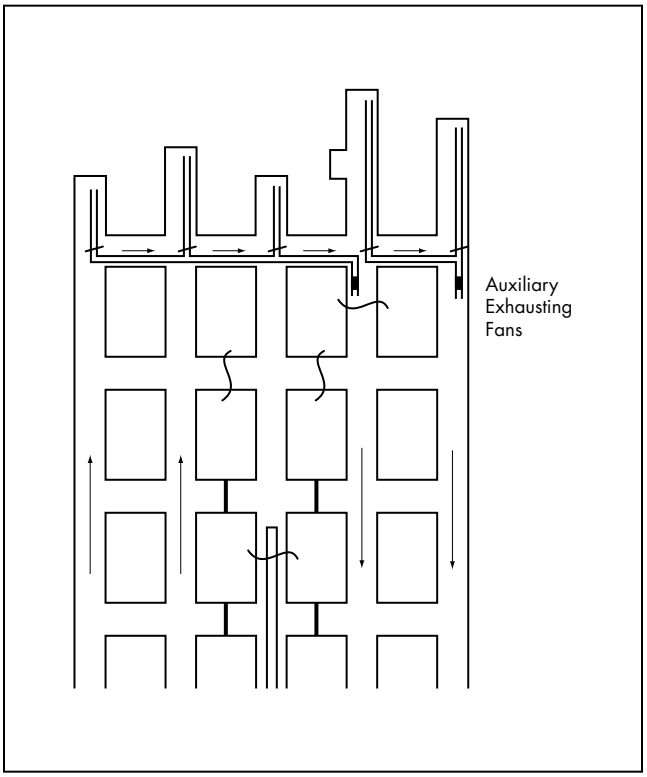
The types shown in Figures 15.3-18E through 15.3-18G are used when the gas is very heavy. An additional fresh air feed helps to maintain gas concentrations to safe levels by feeding fresh air directly to the return end of the face. These systems can also be applied in hot workings.

Room-and-Pillar Systems

In R&P (bord-and-pillar) mining, the potential for leakage is high because of the large number of interconnected entries as shown in Figure 15.3-19. The figure illustrates two methods of ventilating R&P development: a bidirectional or W-system in which intake air passes through one or more central airways with return airways on both sides, and a unidirectional or U-tube system in which intake and return airways are on opposite sides of the panel. In both cases the conveyor is sited in the middle airway and line brattices regulate the air through it.

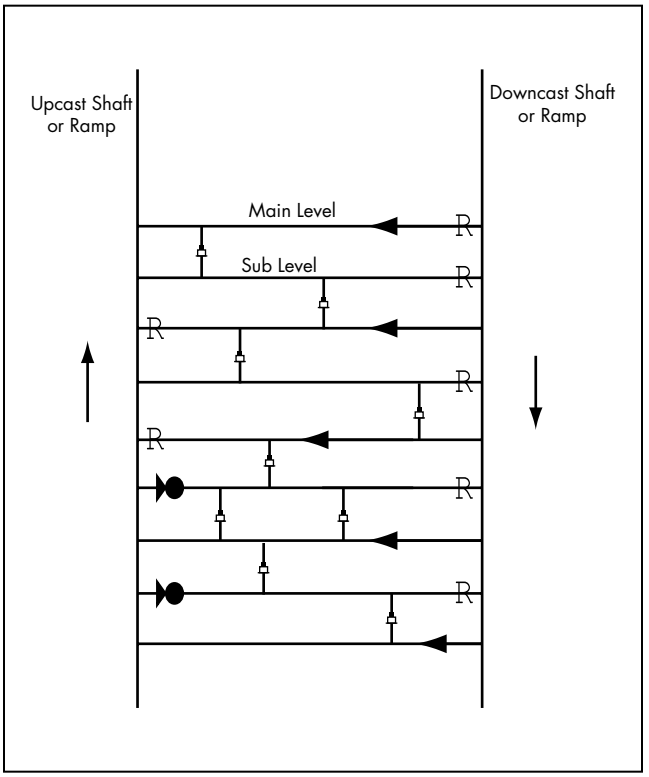
The bidirectional system offers the advantage that the air splits at the end of the panel and each airstream only has to ventilate half the panel. In addition, the rib side gas emission is likely to be heavier in the outer airways. Its disadvantage, however, is that it requires double the number of stoppings, and hence has more leakage paths than the unidirectional system. Thus, the unidirectional system has a higher volumetric efficiency. In both cases the use of brattice cloth to control air distribution at the face end offers a high resistance. This problem can be overcome by employing auxiliary fans and ducts at the face end to force or exhaust air from the headings; the resistance of the face end effectively becomes zero. This is illustrated in Figure 15.3-20.

All the methods shown in the previous figures for longwall mining can also be applied to R&P workings. There are different strategies between R&P and longwall mining. The large number of interconnected airways and high leakage result in R&P layouts offering less resistance to airflow than longwall layouts. Thus, R&P layouts tend to require higher volume flows at lower fan pressures. Volumetric efficiencies of R&P layouts are very low.



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-20 Room-and-pillar development with auxiliary fans (zero face resistance)



Source: McPherson 1993; with permission from Springer Science and Business Media.

Figure 15.3-21 Principle of through-flow ventilation applied to the levels of a metal mine

Ore-Body Deposits

The irregular geometry of metalliferous deposits, coupled with grade variation and fluctuating market prices, results in highly complex mine development in metal mining. Mine development can in a number of cases be related to chasing the grade, dictated by commodity prices at a particular point in time. These factors and the fact that drill-and-blast operations are cyclic in operation also necessitate many more stopes than would be required in a stratified deposit mine, some of which may not be worked on a particular shift. Consequently, metal mines are characterized by a three-dimensional aspect and require much more flexible ventilation arrangements.

Ventilation networks in metal mines are more complex than those for stratified deposits. Figure 15.3-21 illustrates the basic principles of ventilating a metal mine. Air moves in a through-flow manner from a downcast shaft or ramp, across the levels, sublevels, and stopes toward return raises, ramps, or upcast shafts. Airflow control is via fans and/or regulators (represented by R in the figure). Ascentional ventilation takes advantage of natural ventilation effects caused by temperature and humidity variations and avoids uncontrolled recirculation.

Airflow distribution in various stoping systems is highly variable. The guiding principles are as follows:

- Use ascentional ventilation.
- Through-flow ventilation should be used wherever possible; however, use of fan and duct systems is recognized as a need in certain systems.
- Series ventilation should be avoided.
- Leakage through orepasses is a problem; attempt to avoid large pressure drops across orepasses to minimize leakages.
- Wherever practical, every level and sublevel should be provided with its own through-flow of air between shafts, ramps, or raises.

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CHAPTER 15.4

Gas and Dust Control

Pramod Thakur

INTRODUCTION

The provision of an adequate air environment to promote the health, safety, and comfort of mine workers has always been and will continue to be a prime requisite for successful mining operations. Although the definition of an adequate environment varies from country to country, it generally means the provision of sufficient circulating air, often at specified velocities, to maintain at least 19.5% oxygen in the working areas; concentration of solids (dust) and gaseous pollutants such as methane, carbon dioxide, and respirable dusts below specified limits; and temperature and humidity below specified limits.

This chapter will deal only with the techniques used to control solid and gaseous contaminants in the mine air. Liquid contaminants, such as mists, are not an issue in the mining industry. Table 15.4-1 shows the maximum allowable concentrations of contaminants normally encountered in mine air (CFR 2003).

Methane gas is a big concern primarily in coal mines, but some salt and potash mines also have excessive methane emissions. Special techniques have been developed over the past 50 years to keep the mines safe. Respirable dust is a problem common to all mining and processing operations. In the past, diesel particulate matter was a health concern only in metal and nonmetal mines, but it is becoming a health concern in coal mines as the population of diesel equipment in coal mines is increasing. Diesel haulage is likely to replace all trolley wire haulage, leading to improved safety in mines.

Traditionally, all solid and gaseous pollutants in mines are rendered safe by dilution with adequate ventilation air. In deeper mines, methane emission is so high that it is drained prior to mining and postmining. Respirable dusts also need to be controlled before they become airborne. Diesel exhaust is often oxidized to remove carbon monoxide and unburned hydrocarbons, and the diesel particulate matter is collected on specially designed filters to keep its concentration in air below the threshold limit value.

GAS CONTROL STRATEGY

All gas emissions in mine airways must be controlled to keep their concentrations below the statutory limits. The control strategy depends on the type of contaminants, their sources, and rates of emission. It can range from simple dilution with ventilation air to complex procedures for removal of the contaminant prior to mixing with the mine air or suppression/elimination at the source. An overall strategy comprises the following steps (Hartman et al. 1997):

1. Prevention (e.g., proper blasting, internal combustion engine maintenance)
2. Extraction (e.g., methane from coal mines)
3. Chemical absorption—use of catalytic converter in diesel engines
4. Isolation—sealing of areas in coal mines to contain methane emissions
5. Ventilation—including local dilution at the source with auxiliary ventilation and disposal via the mainstream ventilation

Four cases will be discussed in greater detail to illustrate these strategies:

1. Methane control in coal mines
2. Respirable dust control
3. Diesel exhaust control
4. Radon gas control

Methane Control in Coal Mines

Coal seams and coal-bed methane are syngenetic in origin because they originate from the same plant material. All coal seams are gassy, but they vary in their degree of gassiness (gas contained per ton of coal). Typical gas contents of coal seams vary from negligible to 25 m^3/t (see Appendix B, Coal Mine Gas Chart). Methane and sometimes carbon dioxide are the major components of gas in coal comprising 90%–95% by volume. The remaining gas is primarily nitrogen, helium, argon, and hydrogen. Methane is released into coal mines from coal seams as mining proceeds. It is colorless, odorless, and combustible, and it forms an explosive mixture with mine air in the concentration range of 4.5%–15% by volume. The maximum permitted concentration of methane in mine air is restricted by law to between 1% and 2% in all major

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Table 15.4-1 Permissible exposure limits for mine air contaminants

Contaminant	8-Hour Time-Weighted Limit	Ceiling Limit
Methane (fresh air)	—	1%–1.25%
Methane (return air)	—	2%
Carbon dioxide	—	0.5 %
Carbon monoxide	50 ppm	
Nitric oxide	25 ppm	
Nitrogen dioxide	—	5 ppm
Sulfur dioxide	5 ppm	
Hydrogen sulfide	10 ppm	
Radon (uranium mines)	1.0 working level	
Respirable coal dust	2 mg/m^3	
Respirable silica dust	0.1 mg/m^3	
Diesel particulate matter (coal mines)*	120 mg/m^3	
Diesel particulate matter (metal mines)	160 mg/m^3	
Asbestos	5 fibers/cm^3	

*Applicable only to Pennsylvania and West Virginia in the United States; it is the most stringent standard in the world.

Table 15.4-2 Recent methane explosions in U.S. coal mines

Year	Mine	Deaths
2006	Sago mine, Tallmansville, West Virginia	12
2001	Blue Creek No. 5 mine, Brookwood, Alabama	13
1992	No. 3 mine, Norton, Virginia	8
1989	William Station No. 9 mine, Wheatcroft, Kentucky	10
1983	McClure No. 1 mine, McClure, Virginia	7
1982	No. 1 mine, Craynor, Kentucky	7
1981	No. 21 mine, Whitwell, Tennessee	13
1981	No. 11 mine, Kite, Kentucky	8
1981	Dutch Creek No. 1 mine, Redstone, Colorado	15
1980	Ferrel No. 17 mine, Uneeda, West Virginia	5
1976	Scotia mine, Oven Fork, Kentucky	26
1972	Itmann No. 3 mine, Itmann, West Virginia	5
1970	Nos. 15 and 16 mines, Hyden, Kentucky	38

coal-producing countries, but methane–air explosions are quite common even today. Table 15.4-2 shows a list of mine explosions in the United States from 1970 to 2006. In these explosions, lives were lost in spite of methane drainage in some of the mines. When methane drainage was not practiced and ventilation was the only method of methane control, mine explosions were much more disastrous with very high numbers of fatalities. To minimize the risk of methane explosion, mine ventilation must be supplemented by methane drainage from coal seams prior to mining and postmining when the gas content of coal exceeds 3 m^3/t.

The amount of gas contained in a coal seam depends on a number of factors, such as the rank of coal, the depth of burial, immediate roof and floor, geologic anomalies, tectonic stresses, and temperatures during the coalification process. For methane control, all coal seams can be divided into three categories: mildly gassy, moderately gassy, and very gassy, as shown in Table 15.4-3.

For mildly gassy and moderately gassy mines, ventilation is the primary technique for diluting and disposing of methane emissions. A large mine may have several fans circulating

Table 15.4-3 Gassiness of coal seams

Category	Depth, m	Gas Content, m^3/t
Mildly gassy	≤200	<3
Moderately gassy	200–500	3–10
Very gassy	>500	10–25

50,000–100,000 m^3/min of air in the mine airways to keep methane concentration below statutory limits. Gassiness of a mine is also expressed as total methane emissions divided by the total daily coal production, and the ratio is called specific emissions. It is generally economically feasible to handle specific emissions as high as 30 m^3/t with a well-designed ventilation system, but with increasing methane emissions, a stage is reached when it becomes impossible to stay within statutory limits with mine ventilation alone. Methane drainage then becomes necessary. With a well-planned methane drainage scheme and a good ventilation system, even very gassy mines with specific emissions in excess of 100 m^3/t can be safely operated.

In gassy mines it is best if most of the methane is recovered by carrying the gas in pipelines without mixing it with mine air.

Properly designed and carried out, methane drainage has the following benefits:

- Enhanced mine safety
- Improved coal productivity leading to a lower cost of production
- Reduced number of entries in development sections, which leads to faster development and an improved longwall/development coal ratio
- Reduced number of ventilation shafts, size of fans, and power cost
- Controlled respirable dust due to boreholes drilled in coal seams providing a channel for advance exploration and water infusion
- Defrayed costs through marketing of produced gas and boosting energy production from coal mines

The Gas Emission Reservoir and Its Properties

Methane emissions during the mining process take place in two stages: during development and during actual longwall mining. In the development phase, all methane emissions are realized from the coal seam being mined. Methane emissions on the longwall face are also realized only from the coal seam being mined. In order to achieve a high rate of advance in longwall gate roads as well as the mains, the working coal seam must be properly degassed prior to mining. The most important reservoir properties that must be determined for designing an optimum degasification scheme are

- Gas content and gas isotherm of the coal seam;
- Water content of the coal seam and its chemical composition;
- Reservoir pressure;
- Directional permeability of the coal seam;
- Magnitude and directions of principal horizontal stresses; and
- Compressive strengths and elastic moduli of the coal seam, the roof, and the floor.

For most efficient mining, the gas content of very gassy coal seams must be reduced to <3 m^3/t and reservoir pressures should be reduced to <0.5 MPa. The amount of gas retained

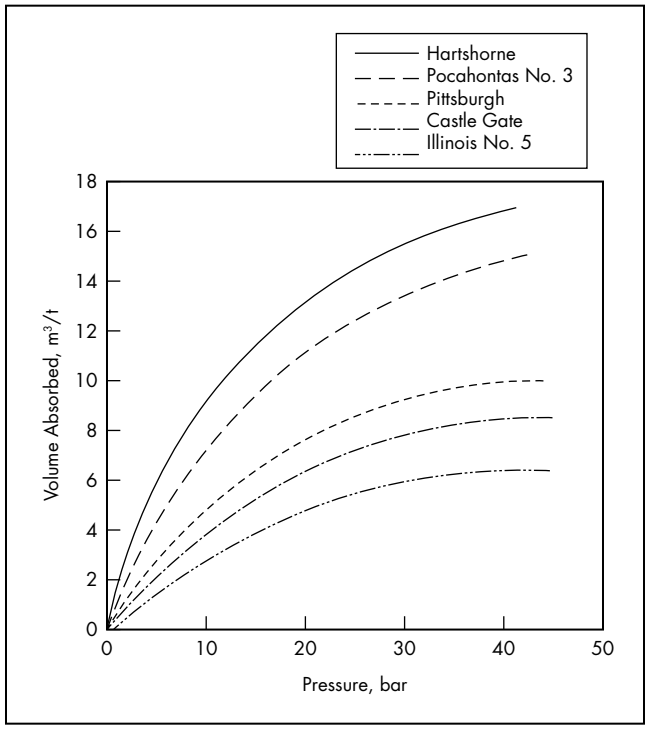


Figure 15.4-1 Gas isotherms for U.S. coal seams

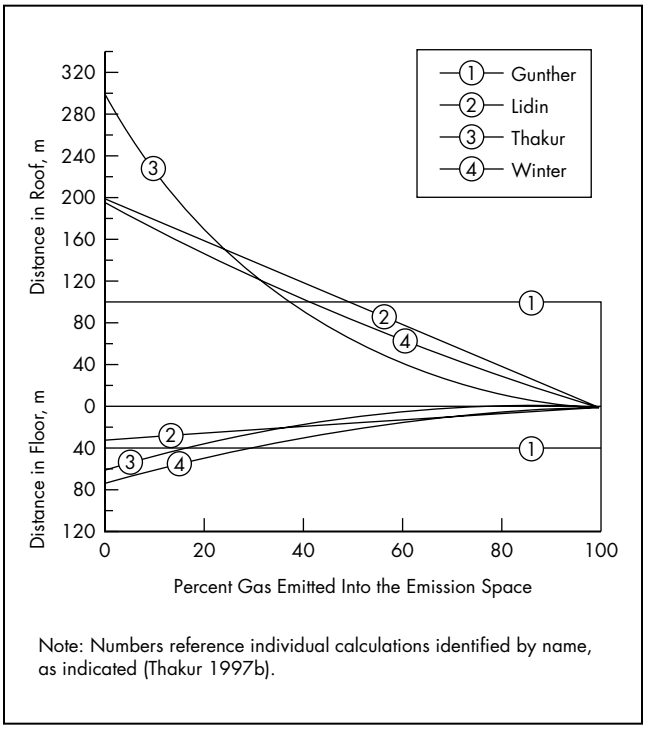


Figure 15.4-2 Vertical limits of the gas emission space

by coal seams at any given pressure can be determined from their gas isotherms. Typical gas isotherms for some U.S. coals are shown in Figure 15.4-1.

In the postmining phase, the gas emission reservoir is considerably expanded. Figure 15.4-2 shows the vertical extent of the gas emission space created by longwall mining and the percentage of gas contents released by various coal seams into the gas emissions space as a function of their distance from the mined coal seam. The vertical dimension of the gas emission space is highly dependent on the width of the longwall face. In general, the wider the longwall panel, the greater the vertical extent of the gas emission space and, consequently, the higher the specific gob gas emissions (volume of gas emitted per unit area of the gob). For optimum methane capture, the specific gob gas emissions for the longwall panel must be calculated from Figure 15.4-2 and subsequently verified by direct measurement of postmining methane emissions.

A very gassy coal seam is usually underlain and overlain by equally gassy, but generally nonmineable, coal seams. Longwall mining fractures them all, and their gas contents are released into the mine. Specific gob gas emissions can be as high as 2 million m^3/ha of the gob area, but it depends on the following conditions:

- Total thickness, gas contents, and proximity of other coal seams to the mined seam in the gas emission space
- Width of the longwall panel
- Presence of geological disturbances (e.g., lineaments, faults)
- Thickness and properties of noncoal strata above and below the mineable coal seam

Coal Seam Degasification

Coal seam degasification can be broadly classified into two phases: premining degasification and postmining degasification. Some of the premining degasification techniques can also be used for commercial coal-bed methane production (see Chapter 11.6). Premining degasification is usually done 3–5 years ahead of mining, but for commercial coal-bed methane production the coal seams should be drilled at least 10 years ahead of mining. This maximizes the return on investments.

Premining degasification. At present there are three techniques commonly used for premining degasification: in-mine horizontal drilling, vertical hydrofractured (hydrofraced) wells, and horizontal boreholes drilled from the surface. Depending on the gas content and depth of the coal seams, one or a combination of these techniques is used to reduce the gas content and reservoir pressure to <3 m^3/t and 0.5 MPa, respectively. Vertical hydrofraced wells and horizontal boreholes drilled from the surface can also be used for commercial coal-bed methane production.

In-mine horizontal drilling. Figure 15.4-3 (plan view of a longwall face) shows a commonly used pattern of drilling in a moderately gassy coal seam (with a gas content >3 m^3/t). The drilling rig follows the continuous miner driving the development section and drills ahead of the mining face to degas the section. Next, a cross-panel borehole is drilled in the longwall panel from the same site to degas it. This pattern is repeated approximately every 300 m. In a permeable coal seam (permeability ≥10 millidarcies), nearly 50% of in-situ gas can be removed prior to mining. A detailed description of this technique is available in published literature (Thakur and Davis 1977; Thakur and Zachwieja 2001). This technique of degasification is equally suited to the room-and-pillar type of mining.

Vertical hydrofraced well. In very gassy, deep coal seams (gas content > 10 m^3/t) it becomes necessary to first drill with vertical hydrofraced wells in order to reduce the reservoir pressure and partially degas the coal seam. Secondary

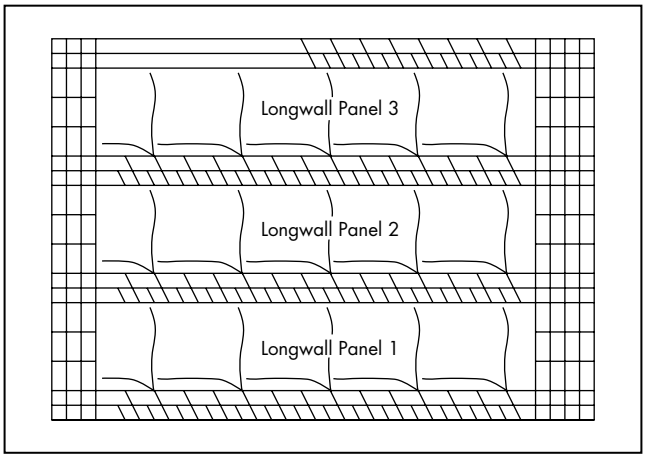


Figure 15.4-3 Coal seam degasification with in-mine horizontal drilling

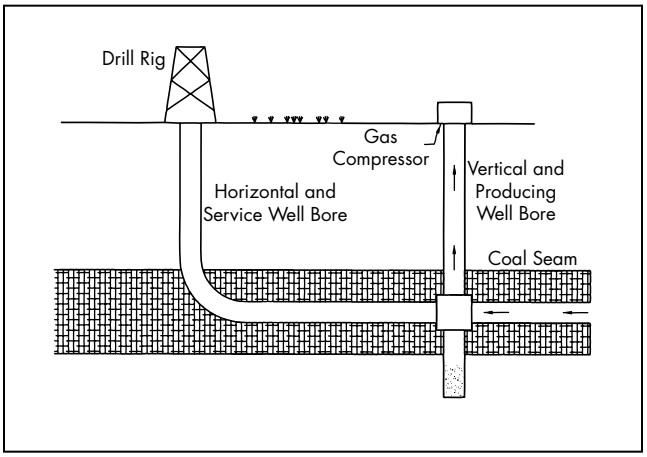
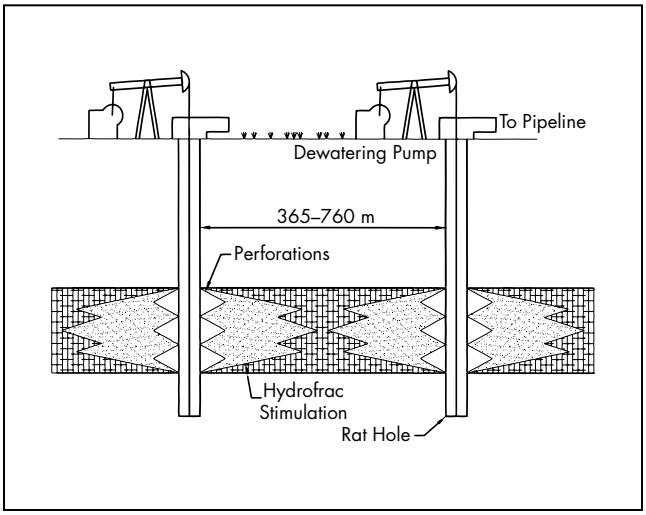


Figure 15.4-5 Horizontal boreholes drilled from the surface



Source: Thakur 2005.

Figure 15.4-4 Vertical hydrofraced wells

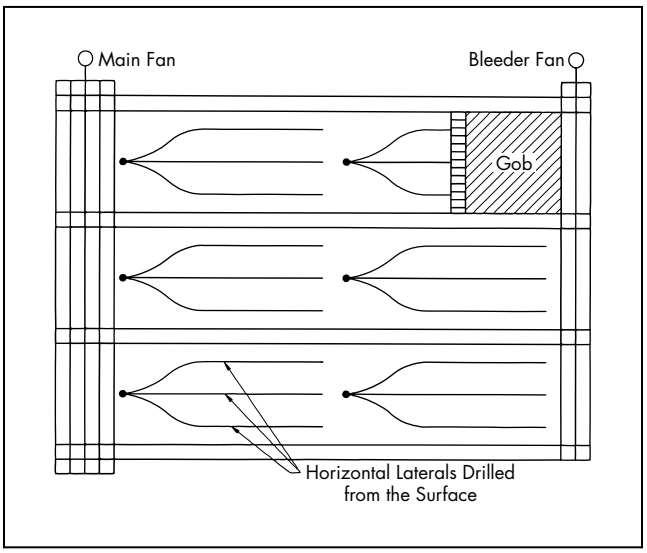


Figure 15.4-6 Coal seam degasification with horizontal boreholes drilled from the surface

degasification is performed by horizontal drilling, either from the surface or underground mine workings. Figure 15.4-4 shows a vertical section of hydrofraced wells. If drilling is done 3–5 years in advance, nearly 50% of in-situ gas can be drained by these wells prior to mining. A 300 × 3,000 m longwall panel in a very gassy mine may need 10–12 vertical hydrofraced wells to achieve this goal. When the reservoir pressure is reduced to <1.5 MPa, the longwall panel is drilled horizontally to remove an additional 25%–30% of in-situ gas content of the coal seam. Well-designed hydrofraced wells can create a vertical fracture in a coal seam that has a length of 300–600 m. Hydrofracing is typically done with nitrogen foam, but plain water, gelled water, and cross-linked gels have also been used depending on the specifics of the coal seam. This technique can be used for both coal seam degasification and commercial gas production.

Horizontal boreholes drilled from the surface. Figures 15.4-5 and 15.4-6 show the common design and application of horizontal boreholes drilled from the surface for coal seam degasification. For successful operation, water produced by the horizontal laterals must be removed. A production well with a sump (drilled below the coal seam to be degassed) is first drilled. Nearly 100 m away, a vertical well is drilled, and when the borehole approaches the target coal seam, it is deviated by 90° to intersect the coal seam and the production well horizontally. The borehole can be laterally extended to a distance between 1,000 and 2,000 m depending on the depth of the coal seam. A water pump is installed in the production well to remove water as well as coal fines, and gas production is maintained. Very expensive instruments are needed to articulate a horizontal borehole to intersect the production well and keep it in the coal seam for most efficient methane drainage. The cost of drilling horizontal boreholes from the surface is, therefore, much higher than the cost of drilling in-mine horizontal boreholes. However, this is the best technique for commercial coal-bed methane production from all coal seams, and its application is increasing (Thakur et al. 2002b).

Postmining degasification. Longwall mining creates a much larger gas emission space than the mined coal seam. Besides the specific gob gas emission of the longwall, the other factor that has the greatest influence on the postmining methane emission rate is the rate of longwall face advance.

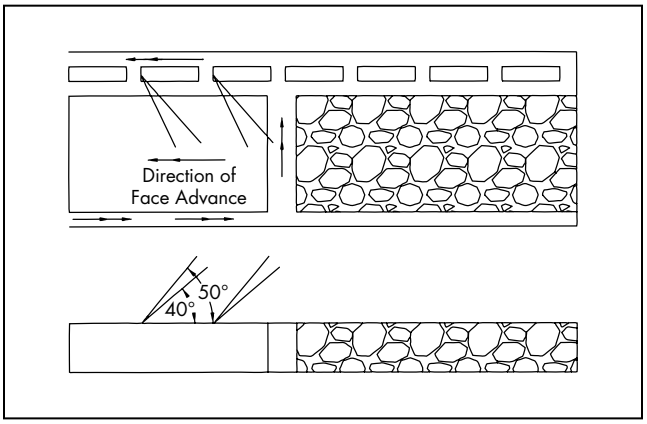


Figure 15.4-7 Methane drainage with cross-measure boreholes

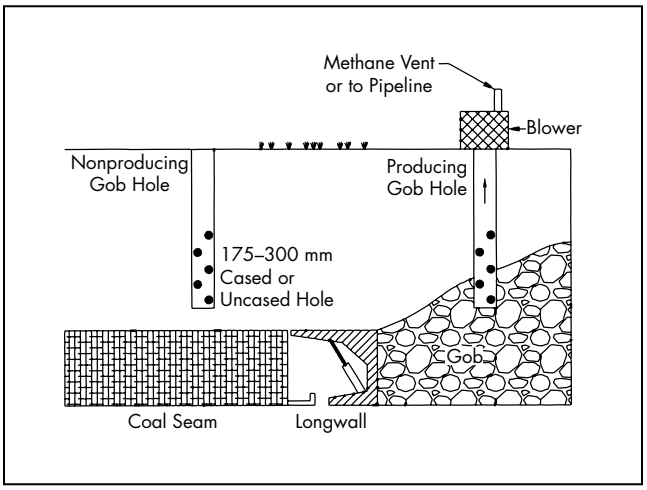


Figure 15.4-8 Methane drainage with vertical gob wells

Methane emission from the gob is linearly proportional to the rate of face advance in hectares per day. Approximate gob emissions in million cubic meters per hectare can be calculated from these two data points. At present there are only two viable techniques for postmining methane drainage: cross-measure boreholes and vertical gob wells.

Cross-measure boreholes. Figure 15.4-7 shows a typical layout of methane drainage from retreating longwall gobs using cross-measure boreholes. Inclined boreholes, 50–100 mm in diameter, are drilled from longwall gate roads into the roof (and sometimes into the floor). These boreholes are inclined at 40°–50° to the horizontal plane and are also slanted toward the gob; they are drilled roughly at 30-m intervals. All boreholes are manifolded to a 100–200-mm diameter pipeline and gas is conducted to the surface. Gas production is often assisted by applying negative pressure of 100–1,000 mm of water gauge. This system of methane drainage is best suited to slow-moving longwall faces in mildly gassy mines. It is popular in European mines but generally unsuitable for gassy and highly productive U.S. and Australian longwall faces.

Vertical gob wells. Figure 15.4-8 shows upward gas movement in the well over the gob. Steel casings of 175–300 mm diameter are anchored in competent rock strata nearly 100–300 m above the working seam, and the bottom 100–300 m is slotted to permit gas entry. Gas production is generally assisted with powerful blowers.

In U.S. coal mines, longwall panels in very gassy coal mines can be 240–400 m wide and can advance 15–30 m/d. The specific gob gas emission in the central and southern Appalachian basin of the United States can be 1–2 million m^3/ha. Hence, the gob gas emission rate can range from 0.5–1.2 million m^3/d. The bleeders with a limited number of entries and ventilation air (8,000–10,000 m^3/min) can capture only 0.1–0.2 million m^3/d of methane. The remaining gob emissions must be captured by vertical gob wells resulting in methane capture ratios of 65%–85%. Completion techniques for these gob wells are described elsewhere (Davis and Krickovic 1973; Mazza and Mlinar 1977). Although the spacing of these gob wells on a longwall panel can be calculated if good data are available, the best method is to go by direct experience in the local basin and the mine. Generally, a spacing of 3–6 ha per well should yield 70%–80% of methane capture if assisted with well-designed blowers.

This method of postmining degasification has also been successfully used in retreat mining of room-and-pillar sections.

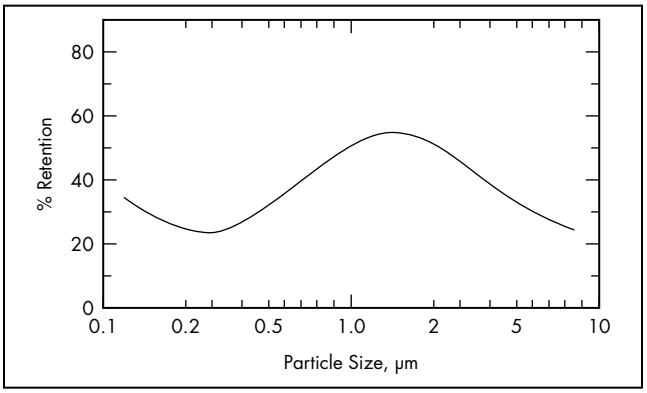
Table 15.4-4 Classification of dusts

Classification	Minerals
Fibrogenic dusts (harmful to respiratory system)	Silica (quartz, chert), silicates (asbestos, talc, mica), metal fumes (nearly all), beryllium ore, tin ore, iron ores (some), carborundum, coal (anthracite, bituminous)
Carcinogenic dusts	Radon daughters, asbestos, arsenic
Toxic dusts (poisonous to body organs, tissue, etc.)	Ores of beryllium, arsenic, lead, uranium, radium, thorium, chromium, vanadium, mercury, cadmium, antimony, selenium, manganese, tungsten, nickel, silver (principally the oxides and carbonates)
Radioactive dusts (injurious because of alpha and beta radiation)	Uranium ore, radium ore, thorium ore
Explosive dusts (combustible when airborne)	Metallic dusts (magnesium, aluminum, zinc, tin, iron), coal (bituminous, lignite), sulfide ores, organic dusts
Nuisance dusts (little adverse effect on humans)	Gypsum, kaolin, limestone
Inert dusts (no harmful effect or cellular response in lung)	None

Respirable Dust Control

When mineral (metal and nonmetal) and sedimentary deposits such as coal are mined, processed, or transported, fine dust is produced by the crushing of the original material. Dust has been defined as solid particles <100 μm in size that can be disseminated and carried by air in suspension. Dust clouds in mine airways constitute two well-known hazards, namely, dust explosions and dust-related lung diseases such as silicosis and coal workers' pneumoconiosis. Table 15.4-4 shows a list of different kinds of dust encountered in mines. They are classified as pathogenic or explosive dusts but in decreasing order of severity (Hartman et al. 1997). Some rate multiple listings.

Dust concentrations in air to sustain an explosion ranges from 20–500 g/m^3. Such high concentrations cannot be created or sustained under normal working conditions. Most dust explosions are preceded by a gas explosion that can kick up the deposited dust, creating a dangerous situation. Dust explosions have been made almost nonexistent by the process of



Source: ILO 1965, © International Labour Office.

Figure 15.4-9 Alveolar retention of particles of unit density

mixing inert dust such as gypsum or limestone dust with combustible dust in very high proportions. For example, mixing limestone dust with deposited coal dust so that the incombustible content is at least 60% will render the mixed dust nonexplosive.

Dust particles finer than 10 μm (unit density spheres) do not generally settle in the turbulent airflow in the mine airway. This size range is called respirable dust. The human respiratory system consists of a series of branching passages decreasing in size but increasing in number. Thus, starting with inhalation either through the nose or mouth, the air passes in succession through the trachea, bronchioles, alveolar ducts, and finally into alveolar sacs where gaseous exchange of oxygen takes place. The lungs act as both a filter and a scavenger for dust particles.

When dust-laden air is inhaled, only a small fraction of it is deposited in the lungs. The mechanisms of deposition are impaction, sedimentation, and diffusion, much the same as in the case of a simple paper filter. Figure 15.4-9 shows a typical deposition and retention curve for the portion of the lungs below the bronchioles, popularly known as the pulmonary deposition curve (ILO 1965). Thus particles >10 μm are deposited by impaction in the nose and the top of the trachea. Particles <10 μm are deposited by sedimentation in the upper respiratory region with progressively increasing efficiency as the size decreases. A maximum is reached around 2 μm, below which there is a decline. A minimum is reached around 0.3 μm, beyond which there is further increase in depositional efficiency due to diffusion forces.

The scavenging action of the lungs is equally important. Practically all dust particles deposited in the upper respiratory tract are trapped by cilia (hair-like structures lining the trachea) and transported to the mouth where they are spat or sneezed out or swallowed. Since the digestive system is sturdier than the lungs, the swallowed particles seldom do any harm. Dust particles deposited in the alveolar ducts and sacs are ingested by migratory cells, the phagocytes, and transported to lymph nodes with varying degrees of efficiency.

With continued exposure to high concentrations of dust, the lungs' parenchyma accumulates a critical mass and body tissues begin to react to the dust particles causing lung diseases. Depending on the type of dust inhaled, the lung disease is commonly called silicosis (for silica), asbestosis (for asbestos), silicatosis (for silicates), siderosis (for iron ore), and coal workers' pneumoconiosis (for coal dust, also known as black lung). Almost all industrialized nations have set up

Table 15.4-5 Respirable dust permissible exposure limits

Dust Type	PEL	Comments
Crystalline silica	0.1 mg/m^3	U.S. standard; other standards are less stringent
Silicates		
Asbestos	5 fibers/cc	>5 μm
Mica	706/million m^3	
Cement	1,056/million m^3	
Talc	706/million m^3	
Coal dust	2 mg/m^3 or 10/% crystalline silica	When silica <5% When silica >5%; U.S. standard is the most stringent
Metallic Dusts		
Mercury	0.05 mg/m^3	
Lead	0.15 mg/m^3	
Antimony, arsenic	0.5 mg/m^3	
Manganese	5 mg/m^3	
Iron, zinc molybdenum	5 mg/m^3	
Uranium (insoluble)	0.2 mg/m^3	
Vanadium (V_2O_5)	0.5 mg/m^3	

Source: Adapted from ACGIH 1972.

a permissible exposure limit (PEL) for the exposure to various dusts. The PELs for various respirable dusts are shown in Table 15.4-5. They are adapted from the general guidelines provided by the American Conference of Governmental Industrial Hygienists (ACGIH 1972).

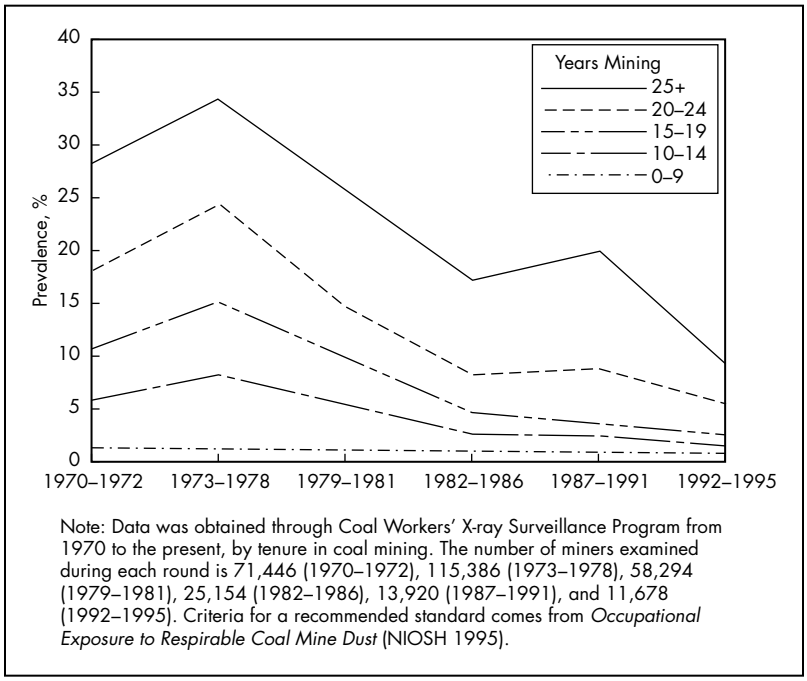
These PELs were developed and implemented during the past 40 years. In general, they are more stringent than the old standards, and the impact on workers' health has been favorable. A fine example is the 70% decline of coal workers' pneumoconiosis (CWP) in U.S. coal mines during the past 40 years as shown in Figure 15.4-10.

In order to minimize, if not totally eliminate, the incidence and prevalence of any dust-related lung diseases, a holistic approach comprising the following requirements are necessary (the last two requirements will not be discussed in this chapter) (Thakur 1997a):

1. Better personal exposure monitoring (daily and cumulative)
2. Better mining and dust control technologies
3. Recognition of disease-causing characteristics of each dust, particularly epidemiological data on dust exposure (dosages) and related occurrence and progression of lung diseases
4. Lifestyle intervention program including cessation of smoking and medical surveillance

Personal Exposure Measurement

Earlier dust sampling instruments such as the impinger and konimeter were nonsize selective and simply measured the number concentration of dust in the air. The concentration of respirable dust was expressed as the number of dust particles per unit volume of air. Other instruments based on the same principle of particle counting were introduced later, namely, the membrane filter and the thermal precipitator. All of them took a long time to determine the dust concentration, in addition to which they were labor and laboratory time-intensive.



Note: Data was obtained through Coal Workers' X-ray Surveillance Program from 1970 to the present, by tenure in coal mining. The number of miners examined during each round is 71,446 (1970–1972), 115,386 (1973–1978), 58,294 (1979–1981), 25,154 (1982–1986), 13,920 (1987–1991), and 11,678 (1992–1995). Criteria for a recommended standard comes from *Occupational Exposure to Respirable Coal Mine Dust* (NIOSH 1995).

Figure 15.4-10 Prevalence of CWP Category 1 or higher

A simple but real-time dust measuring device was needed that could be used for evaluating dust conditions underground. Several dust concentration measuring instruments were developed in the 1970s that were based on light-scattering measurements. The tyndalloscope was the first instrument designed on this principle, but it did not perform very well compared to other instruments. It was very sensitive to the size distribution of dust particles and needed frequent calibration. Subsequently three instruments—the Tyndallometer, Simslin, and GCA Ram—were designed to measure scattered laser light in Germany, the United Kingdom, and the United States, respectively. They were also sensitive to the size distribution of dust particles. But their biggest drawback was their inability to distinguish between dust and water particles. Water sprays used to suppress dust significantly affected the dust concentration readings (Thakur et al. 1981). During the same time period, the dust measurement technology was required to be size selective. The new criteria for sampling and measurement of dust concentration were as follows:

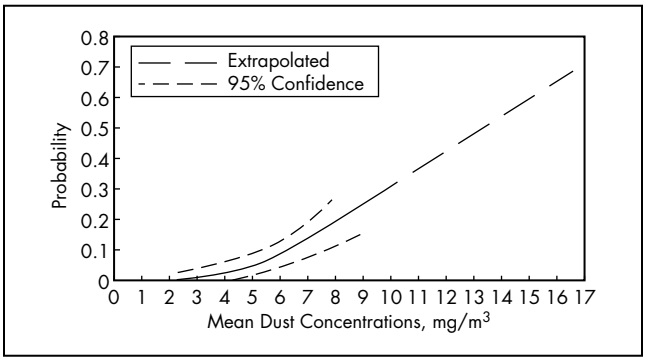
- The sampling device should be size selective and it should mimic the human nose.
- The dust concentration should measure the "mass" of dust per unit volume of air.

Hartman et al. (1997) compared size-selective curves for respirable dust from the British Medical Research Council (BMRC)/Johannesburg and the U.S. Atomic Energy Commission. The BMRC/Johannesburg had 0% collection for 10-µm particles, 50% collection for 3.5-µm particles, and 100% collection for particles <2 µm. This was achieved by classifying the respirable dust with a horizontal elutriator. The U.S. Atomic Energy Commission developed a size-selection device using a cyclone that had 0% collection at 7.1 µm, 50% collection at 3 µm, and 100% collection at 2 µm or smaller particles.

The second criterion required that dust samplers should measure the mass of dust per unit volume in air. This conclusion is based on the findings of the National Coal Board's (United Kingdom) pneumoconiosis field research study (Jacobsen et al. 1971). Dust exposures of a large cohort of coal miners were measured and correlated with changes in their lung radiographs. Figure 15.4-11 shows a quantitative relationship between the coal mine dust exposure and the risk of developing simple CWP. The latter is characterized by the presence of small nodules in the lung and classified as Category 1, 2, or 3 according to the profusion of the nodules. Simple CWP Categories 1 and 2 can be reversed by moving the mine worker to an area with much less or no dust exposure. In Category 3, pulmonary massive fibrosis sets in, and it is generally irreversible.

MRE gravimetric dust sampler. The MRE gravimetric dust sampler was developed by the Mining Research Establishment (MRE) of the National Coal Board. It is the standard dust sampling instrument for most industrialized nations, including the United States, Canada, the United Kingdom, and Australia. It is based on the BMRC/Johannesburg size-selection curve and uses a horizontal elutriator for this purpose. The classified dust particles are collected on a membrane filter. The preweighted filter is weighed again after sample collection, and the differences in weight divided by the volume of air sampled yields the gravimetric dust concentration. The only drawback of this instrument is its size and weight. It measures the mass concentration of respirable dust but cannot be conveniently carried by a mine worker.

The U.S. personal gravimetric sampler. This sampler follows the U.S. Atomic Energy Commission size-selection curve and uses a nylon/metal cyclone to classify dust particles. It consists of two basic components: a cyclone assembly with a preweighed filter cassette, and a rechargeable battery-operated diaphragm pump with calibrated flow rates. The



Source: Jacobsen et al. 1971.

Figure 15.4-11 Probability that a worker with no pneumoconiosis will be classified as Category 2 or higher after 35 years of exposure to various concentrations of coal mine dust

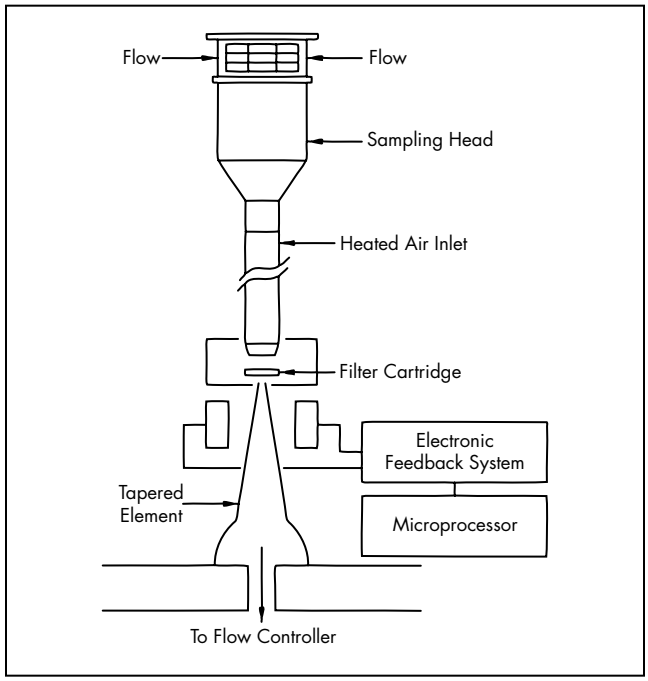
pump is worn on a miner's belt but the cyclone-filter assembly is clipped onto the miner's collar to be as close to his or her nose as possible. Thus, the sampler collects the dust particles that the wearer is likely to breathe and gives a better measure of mine worker's exposure.

The two instruments have excellent correlation. The personal sampler operating at a flow rate of 2 L/min yields a respirable dust concentration that is multiplied by a factor of 1.38 to make it compatible to the MRE sampler measurement. The 2.00 mg/m^3 dust standard for U.S. coal mines is based on the MRE sampler reading. For noncoal mines, the personal sampler draws air at 1.7 L/min and the measured dust concentration is the dust standard; that is, no correction is applied (CFR 2003).

Instantaneous dust monitors. Although the MRE sampler and the personal sampler are required instruments for compliance with dust standards, they are not able to give instantaneous dust readings that can be used for immediate engineering control. Two instantaneous reading instruments are currently in use in the mining industry, the MINIRAM and the TEOMB.

MINIRAM (miniature real-time aerosol monitor). The measuring principle is based on the detection of scattered electromagnetic radiation in the near-infrared region. The MINIRAM senses the radiation scattered by respirable dust particles by means of silicon-photovoltaic hybrid detectors, within a 1-m^3 volume of air. Air passes freely through the open sensing chamber induced by personal movement, ventilation, convection, or a combination of all three. No pump is required. The instrument measures particle diameter in the range of 0.1–10 μm and is independent of air velocity. It is controlled by a single-chip complementary metal-oxide semiconductor microprocessor that provides continuous real-time reading of airborne dust concentration. The instrument is light and small in size and can be easily used anywhere in the mine.

TEOMB (tapered-element oscillating microbalance). As the name implies, the instrument is a real-time mass monitor of respirable dust based on the principle of a tapered-element oscillating microbalance. This instrument consists of a replaceable filter cartridge mounted on the narrow end of a hollow, tapered tube (Figure 15.4-12). The wider end of the tube is fixed. Air is drawn through the filter and dust particles are deposited on it. The tapered tube oscillates but the frequency



Source: Schmitz 1994, courtesy of the IEA Clean Coal Centre.

Figure 15.4-12 TEOMB ambient particulate monitor

of oscillation (initially 252 Hz) depends on the mass of the tapered element, including the filter. The frequency changes as the dust is deposited on the filter, increasing the total mass of the tapered element. The change in frequency translates into dust concentration. A microprocessor collects data on air volume sampled and corresponding dust mass collected, does the calculation, and displays dust concentration in real time in milligrams per cubic meter.

The instrument is sensitive to mechanical vibrations and should not be mounted on mining machines. It is best carried on a person. It is also sensitive to moisture. The intake air stream is maintained at 50°C to minimize this problem.

The TEOMB can operate with flow rates from 0.5 to 5 L/min. The data can be displayed instantaneously, hourly, or at the end of an 8-hour shift. It is highly recommended for engineering control of respirable dust in mines.

Engineering Control of Dust

Because the respirable dust in suspension behaves like a gas, the engineering control of respirable dust, in principle, is similar to that for gases discussed earlier. The sequence of control techniques (listed in descending order of importance and effectiveness) are

1. Prevention of generation of dust and its suspension in air,
2. Suppression of airborne dust on-site,
3. Collection of dust that could not be suppressed, and
4. Dilution with auxiliary and main ventilation.

Use of protective equipment/administrative control in special situations is the last resort, but it is not regarded as a dust-control technique. The following steps illustrate this strategy for dust control in underground coal mining:

1. Coal seams are infused with water and cut with very sharp bits to minimize dust generations.

2. Water mixed with surfactants is sprayed at high pressure to suppress dust on-site.
3. In many cutting and drilling operations, the airborne dust is collected and scrubbed before discharge.
4. Adequate ventilation is provided to dilute the remaining dust and keep the dust concentration below statutory limits.
5. For persons working in return airways, well-designed personal equipment, such as air helmets, is used to protect the workers.

Engineering control of dust is divided into the following types of mining and processing operations:

- Coal mining—continuous mining sections
- Coal mining—longwall sections
- Coal mining—surface
- Metal mining
- Mineral in transport
- Mineral/coal processing

Coal mining—continuous mining sections. All four techniques to control respirable dust have been used in continuous mining sections. Infusing the coal seam with water or water-soluble gels (e.g., sodium silicate and sodium bicarbonate) has been done on a very limited basis because of high cost. Using large and sharp cutting bits also reduces dust generation.

Most common dust control techniques comprise use of water sprays to suppress dust, use of water scrubbers in some cases, and use of ventilation air to dilute the respirable dust concentration below statutory limits.

Water spray systems. Water sprays installed on continuous miners deliver 120–240 L/min water at 0.5–1 MPa pressure at the nozzle. Besides suppressing dust, the sprays also wet the coal, keep the cutting bits cool, and serve as an emergency fire suppression system. Spray nozzles are strategically installed on the continuous miner to achieve these goals. A well-designed spray system can reduce respirable dust concentration by 60%; typical reductions range from 30% to 50%. Ideally, there should be a spray behind each cutting bit delivering 2–4 L/min at 0.3–0.8 MPa. Such machines are known as wet-head continuous miners and they achieve the best dust control.

Water scrubbers. Where seam height permits, the flooded-bed-type scrubber is becoming increasingly popular for use on continuous miners. It is mainly used with blowing-type face ventilation. The dirty air is drawn into a water scrubber by fans and collects the dust. The airflow in the scrubber must be properly matched with the airflow in the heading. A well-designed system with remote operation can cut down respirable dust exposure by 80% (Goodman and Listak 1999). Often, a demister is used to remove water from the scrubber discharge.

Ventilation air. The quantity of ventilation air at the face is perhaps the most important factor in controlling dust exposures. Average air velocity in the face area ranges from 20 to 150 m/min. Air is brought to the face using either line brattice or auxiliary fans and tubing. For best results the end of the line brattice or ventilation tubing should be kept within 4.5 m from the face. When face ventilation is done with auxiliary fans and tubing, it can be either a blowing or exhausting ventilation. Sometimes a combination of both blowing and exhaust ventilation is used for optimum results.

Table 15.4-6 Dust source contribution on longwall faces

Source	Average %	Concentration (median value)
Intake	9	0.33 mg/m^3
Stage loader–crusher	15	0.78 mg/m^3
Shields	23	1.80 mg/m^3
Shearer	53	3.50 mg/m^3

Coal mining—longwall sections. The sources of dust on a longwall face are the intake air, the stage loader–crusher, the shearer, and the shields.

The intake air is usually a minor source of dust unless the belt air is used on the face. Confinement and well-designed water sprays usually at <0.5 MPa pressure can generally keep the contribution of dust from the stage loader–crusher at a minimum. The dust created by shield movement is also contained by mounting water sprays on the shields. The biggest source of dust on longwall faces is the shearer. Table 15.4-6 gives a summary of data obtained from thirteen longwall faces (Kissell 2003).

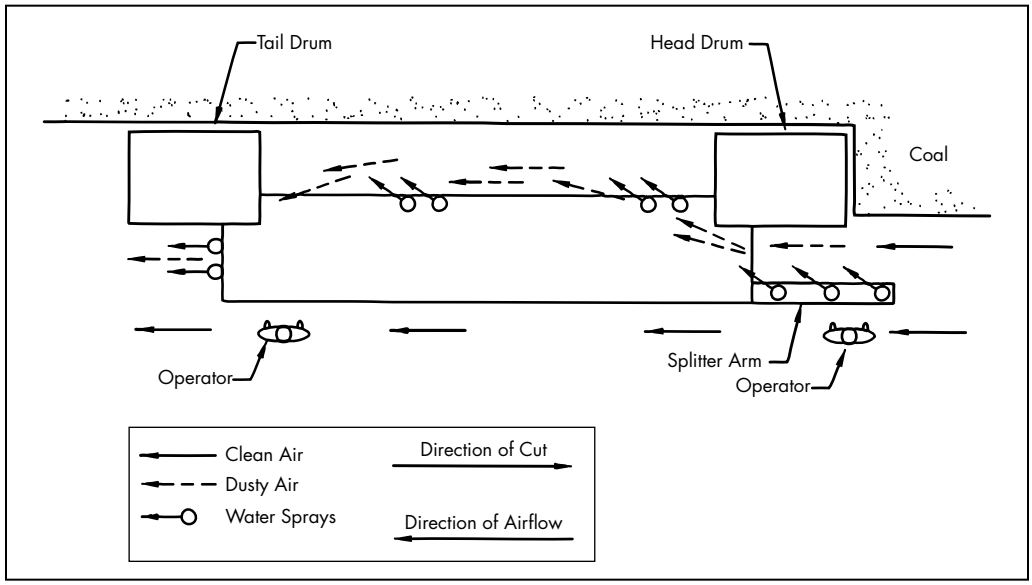
Water infusion. Excellent dust control can be achieved by drilling the longwall for degasification, as discussed earlier, and using the horizontal boreholes for water infusion. Slow infusion at pressures of 0.5–2 MPa has given excellent results, often reducing dust generation by 80% (Nagy et al. 1957). Larger and sharper cutting bits also reduce dust generation. Use of very high pressure water (30–70 MPa) jets to assist coal cutting has also significantly reduced dust generation.

Water sprays. Well-designed water sprays are the next best way to control dust on longwall faces. A minimum of 400 L/min at 0.7 MPa is recommended (Colinet et al. 1997). Ideally, there should be a spray behind each cutting bit but, in general, the larger the number of sprays, the better the dust control. The water sprays, on and around a longwall shearer, can be so arranged that they direct the dust cloud away from the operators and hold it against the face (Figure 15.4-13). This arrangement of sprays is called a shearer–clearer spray system and it reduces shearer-generated dust by 30%–50% (Jayaraman et al. 1985).

Coal mining—surface. Respirable dust in surface mining is created by a number of operations such as drilling, mobile excavation equipment, and haul roads. The situation can become serious under dry and windy conditions.

Drill rigs. Two well-established practices for dust control on drill rigs are dry collection and wet suppression. In a dry collection system, an enclosure is created around the well bore being drilled, which is hooked up to a fan and a filtration system. The negative pressure created by the fan helps contain the dust within the enclosure. The filtration system collects most of the dust and clean air is exhausted to the atmosphere. In the wet collection system, water (often treated with some surfactants) is mixed with the compressed air used to bring drill cuttings out of the hole. Typically less than 4 L/min of water is used for this purpose. The collection efficiency appears to increase from a low rate of 0.8 L/min to a limit of approximately 4 L/min (Zimmer et al. 1987).

Mobile equipment. The operators of mobile equipment such as bulldozers, front-end loaders, and haulage trucks can be best protected by tightly enclosed cabs. In extremely dusty



Source: Adapted from Kissell 1992.

Figure 15.4-13 Air currents when using the shearer–clearer system

situations, filtered air can be pumped into the driver's cab and a slightly positive pressure maintained within the cab.

Haul roads. Wetting the haul roads frequently with water is the most commonly used procedure. To improve the wetting capacity of plain water, hygroscopic salts such as calcium chloride, sodium silicates, or surfactants are often added. The other approach is to create a film of impervious material that can contain the dust. Most commonly used materials are portland cement and bitumen derived from coal or polymers. The best dust control method really depends on the type of road aggregate involved.

Metal mining. Main sources of respirable dust are drilling rigs; blasting; ore- and waste passes; and conveyor belts, transfer points, and roadheaders, if used. Fundamental principles of dust control in metal mines are similar to those for coal mining (i.e., prevention of generation, suppression onsite, collection of airborne dust, and dilution with ventilation).

Drilling rigs. Wet drilling is far superior to dry drilling and can reduce dust concentration by 95% (MSA Research Corp. 1974).

Blasting. Water can be important in controlling blasting dust. The area to be blasted should be thoroughly wetted with water spray beforehand. Good ventilation is also necessary for efficient control of blasting fumes and dust. However, perhaps the best technique to avoid exposure to blasting dust and fumes is to carry out the operation off-shift, when there are no miners in the section.

Ore- and waste passes. Ore- and waste passes create large quantities of respirable dust. Falling ores create a pressure wave discharging dusty air into mine workings. The first step in controlling dust is to design an orepass that is connected with other ore- and waste passes. This relieves the intensity of the pressure wave. The second step in controlling dust is to put the orepasses on exhaust with air velocity at about 60 m/min and discharge the air into a return airway (MAC 1980).

An alternative approach is to confine the dust in the orepass and use sufficient amounts of water to suppress the dust. Water sprays can be installed at the tipping site and the ore can be wetted prior to delivery to the tipping site. However, excessive use of water can create additional problems such as plugging of the chutes, excessive water accumulation in the orepass, and adverse milling conditions.

Conveyor belts, transfer points, and roadheaders. Conventional water sprays are used to suppress dust on conveyor belts, transfer points, and roadheaders. All dust control techniques discussed for continuous mining sections can be used to control dust from roadheaders.

Mineral in transport. Fugitive dust created in mineral transport can be successfully controlled by identifying the points of origin and installing proper control equipment. Major points of origin are loading, transportation, unloading, and stockpiles. Loading and unloading use silos, cranes, front-end loaders, and belt conveyors. For transportation, trucks, trains, and conveyor belts are used. Stockpiles are located at various points in the mineral chain, but primarily at the mines, ports, and the consuming plant. General principles of dust control in transportation are discussed in great detail elsewhere (Schmitz 1994), but they are similar to the guidelines discussed earlier for in-mine dust control.

Prevent dust generation. Every effort should be made to design loading, unloading, and transporting equipment to minimize dust generation. Stockpiles can also be designed to minimize total surface area for a given volumetric content.

Use dust suppressants. Increasing the moisture content of the mineral increases the force of cohesion. Water sprays are the most common devices used to control dust. Sometimes the water is atomized using compressed air or high-frequency sound to improve the efficiency of dust control. Adding surfactants to water also improves dust control. Besides water, many other liquids and chemical additives can also improve dust control. Oil and oil products are highly suited to use as coal dust suppressants because of their surface chemistry. Agglomeration agents such as polymer emulsions are also used for dust control. The main advantage of an agglomeration agent is its longevity of use. Effective dust control may be sustained over several transfer points. Coating agents such as

polymers are mixed with water or are sprayed on bulk materials to form a water-insoluble crust on the material. This prevents the effects of weathering and wind erosion. Foams are used where the use of excessive water can create problems. Only water, compressed air, and foaming agents are needed to achieve this. Foaming agents are high-foaming surfactants, and they may also contain wetting and binding agents.

Contain and capture. Suppressants can successfully prevent dust from becoming airborne but can have high cost and some environmental concerns. They can be supplemented, or at times replaced, by containing and capturing fugitive dust. Capturing techniques require an exhaust fan, hoods, ducts, and fabric collectors/filters. Very high collection efficiency, up to 99%, can be achieved by proper design. The fabric collectors must be cleaned periodically. Wet scrubbers and electrostatic precipitators are also used for dust collection. In an electrostatic precipitator, the dust particles first receive a negative charge from high-voltage DC electrodes, and then they are collected by a positively charged collecting plate. The collecting plates are vibrated to release the dust into a discharge system. Electrostatic precipitators are not recommended for combustible dusts such as coal because of the risks of explosions or fire.

Mineral/coal processing. Sources of dust in a processing plant include belt conveyors, loading/unloading points, transfer chutes, crushers, bucket excavators, chutes, and stockpiles. A very good reference for additional information is the *Dust Control Handbook for Mineral Processing* (Mody and Jakhete 1987).

Personal Protection Equipment

In many countries, approved respiratory protection equipment is required for those working in an area where the respirable dust concentration exceeds the statutory limits. Mainly, they use a filtration system to clean up the air to be inhaled, or they provide filtered intake air with a positive pressure to the workers.

Replaceable filter respirator. A respirator consists of a mask that fits a miner's face and forms a seal against the dusty air. Dirty air is drawn through replaceable filters that can collect 80%–98% of all dust in the air. The filter-holding unit is often plastic but can also be made of metal or hard rubber. Although the respirator does an excellent job of respirable dust control, some personal discomfort may arise owing to breathing resistance and facial irritation caused by the face seal. It also tends to interfere with normal voice communication, as well as eyeglasses and goggles. To eliminate these difficulties, sometimes a much lighter, single-use respirator is used. The whole mask is made of the filter material and covers the entire nose and mouth area of the wearer. However, it does not form a perfect seal against the dusty air nor is it as efficient as the replaceable, heavier filters.

Air helmet. The air helmet is a special hardhat (somewhat bigger than the conventional hardhat) that contains a battery-powered fan, a filtration system for dust, and a face visor. It provides protection for head, eyes, and lungs in a single unit. Although the air helmet is much more expensive than mask filters and weighs about 1.4 kg, it is finding wearer acceptance in many mining operations.

In the air helmet, dust-laden air is drawn through the filtration system by a battery-powered fan and the clean air is directed to the full-face visor over the wearer's face. The air exits the wearer's face at the lower end of the visor. The battery is normally smaller than the miner's cap-lamp battery and is carried on the belt.

If air velocity is not high, the air helmet performs almost as well as the replaceable filter mask with collection efficiencies of up to 84%, but at high air velocity, the collection efficiency appears to go down to about 50% (Cecala et al. 1981). This may be due to excessive turbulence created by the high velocity of air forcing dusty air into the visor.

Personal protective devices are meant for situations when a miner enters and works in an area with dust concentrations above the statutory limits. They should not be required to be used for an extended period of time or (under normal operations) the entire working shift. Although personal protection devices can be effective in reducing dust exposures, they are vulnerable to human errors in their cleaning, maintenance, fitting, and use.

Diesel Exhaust Control

Mineral transport in metal mines has been done by diesel engines for a long time. It has many advantages, but the exhaust must be treated and diluted properly to keep the air environment safe.

Diesel Engines

Diesel engines were introduced in European, Asian, and Australian coal mines nearly 60 years ago, but more recently in U.S. coal mines. Currently, nearly 5,000 units are working in U.S. metal mines and another 3,000 units in U.S. coal mines. The driving force behind this phenomenal growth is mine safety and productivity. Diesel equipment improves safety by eliminating electric shocks, fire, and explosion hazards from trolley wires and cabled electrical equipment. It can also prevent personal injury by eliminating the need to carry small loads (up to 40 kg) by hand in work areas.

The main drawback of using diesel equipment in mines is the contamination of air with diesel exhaust emissions. The major components of diesel exhaust are carbon monoxide (CO), carbon dioxide (CO_2), unburned hydrocarbons (HCs), nitrogen oxides (NO_x), sulfur oxides, and diesel particulate matter (DPM). PELs for all gaseous components of diesel exhaust have been well established. Ventilation can dilute all these gases to below their respective PELs.

DPM consists of

- Solid carbon (includes elemental and organic carbons);
- Liquid and solid hydrocarbons, including polyaromatic hydrocarbons (PAHs);
- Sulfates; and
- Moisture.

The PAHs attached to solid carbon particles are generally believed to be potential carcinogens, but their concentrations are exceedingly low.

DPM Standards

At present, there are no DPM standards in the world that are based on sound epidemiological studies correlating sustained exposure to DPM and the resulting health effects. However, several countries have set arbitrary standards, mainly to minimize DPM concentrations in mine air as much as possible. Table 15.4-7 shows some of these standards.

DPM Control Strategy

The best strategy, by general concensus, to control DPM in all mines and tunneling is to take an integrated approach, as summarized by Thakur and Patts (1999):

- Reduce DPM generation
  - Clean engines
  - Clean (low sulfur) fuel
- Combust and collect generated DPM
  - A catalytic converter on all engines
  - Soot filters where needed
- Dilute DPM
  - Minimum ventilation air
- Monitor DPM emissions and maintain equipment
  - Routine engine performance check
  - Proper maintenance
  - Adequate training to operators and mechanics

Reduce DPM generation. The amount of DPM emitted in grams per brake horsepower is known as the specific DPM emission for the engine and is a measure for "clean" engines. Table 15.4-8 shows the specific DPM emission for some old and new diesel engines. The new engines with electronic ignitions and turbocharging are remarkably clean. Whereas the older engines had a specific DPM emission of about 0.3 g/bhp-h, the new engines have a specific emission of only 0.02 to 0.1 g/bhp-h.

The improvement is brought out by electronic ignition and turbocharging. Another technique to reduce the DPM emission is to derate the engine by restricting the maximum fuel rating. Even a 10% reduction in horsepower rating can cut down DPM emissions by 50% or so (Schnakenberg and Bugarski 2002).

Concurrent to the development of clean engines, much effort has been made to improve diesel fuel quality. Extensive research in the field (Cowley et al. 1993) has shown that fuel properties that most influence DPM and gaseous emission comprise sulfur content, fuel density, cetane number, aromatic contents, and oxygen content. Besides sulfur content, the other two most influential properties are cetane number and fuel density.

Sulfur in fuel creates both sulfur dioxide (gas) and sulfates (solids, counted as a part of DPM). When the current American Society for Testing and Materials (ASTM) D2 fuel (sulfur at 340–500 ppm) is used, nearly 50% of DPM is sulfate. Recently, the U.S. Environmental Protection Agency required all diesel fuels to have no more than 15 ppm sulfur, creating an ultra-low-sulfur fuel. The low sulfur level not only reduces the mass of DPM emitted by the engine but also considerably reduces the number of nanoparticles (<40 nm). In some countries the health risks of inhaling diesel exhaust is ascribed to these nanoparticles. Thus, the use of ultra-low-sulfur fuel is quite helpful in reducing the relative health risks.

Alternate fuels such as Fisher–Tropsch (F-T) diesel and biodiesel have received attention as additives to the petroleum diesel fuel. F-T diesel is produced from coal, biomass, or natural gas. It is a very high-quality fuel with a potential to further reduce exhaust emissions. It has a high cetane number (up to 70), a low sulfur content (<10 ppm), and very low aromatic content (<3%). The benefits are more pronounced in DPM emissions (Norton et al. 1999), owing to very small sulfur content. The only drawback of this fuel is low lubricity, and the addition of some lubricants to the fuel is usually necessary.

Table 15.4-7 World DPM standards

Country	PEL	Measurement Technique
United States (metal mines)	0.16 mg/m^3	NIOSH 5040 method for total carbon
United States* (Pennsylvania and West Virginia)	0.12 mg/m^3	Total DPM divided by nameplate air as specified by the Mine Safety and Health Administration
Canada	1.5 mg/m^3	Respirable combustible dust
Germany (metal and nonmetal)	0.3 mg/m^3	Elemental carbon (EC)
Proposed limits		
ACGIH	0.15 mg/m^3	All particles <1 μm
ACGIH	0.02 mg/m^3	EC particles <1 μm

Source: Thakur et al. 2002a; Schnakenberg and Bugarski 2002.

*U.S. federal regulations do not put a separate limit on DPM but combine it with the respirable dust standard of 2 mg/m^3. They do, however, put a limit on DPM emission rates (MSHA 2001):

- Light-duty outby engines: ≤5 g/h
- Heavy-duty outby engines: <2.5 g/h
- Heavy-duty (permissible) inby engines: ≤2.5 g/h

Table 15.4-8 DPM emissions from old and new diesel engines

Engine Type	Horse-power	DPM Emission, g/h	Specific Emission, g/bhp-h
Old engines			
Cat 3306	150	45.9	0.306
Cat 3304	100	29.7	0.297
Perkins 1006-60 T	152	20.4	0.134
New engines			
DD Series 40	250	5.10	0.020
Deutz F3L912W	40	4.3	0.108
Deutz F3L1011	42	2.6	0.062

Biodiesel is defined as mono alkyl esters of long-chain fatty acids derived from renewable biomass (e.g., corn, soya beans, sugarcane, and switch grass). Pure biodiesels have relatively low sulfur content (<50 ppm) and no aromatics. The cetane number is comparable to D2 diesel fuel (42–48). Since biodiesel is oxygenated (esters contain oxygen molecules), its combustion in diesel engines is more complete than that of petroleum fuels. As a result, there is a substantial reduction in ambient HC, CO, and DPM. A slight increase in the NO_x emissions (caused by a significant increase in nitrogen dioxide, or NO_2) have been observed in neat biodiesels or biodiesel blends (Sharp 1998).

The synthetic diesel and biodiesel fuels can be used in existing engines and fuel injection systems without negatively impacting engine performance.

Combust and collect generated DPM. The second step in the control of diesel exhaust is to combust (oxidize) the gaseous and liquid portions of the exhaust and collect solids on a filtration medium. The process is also known as exhaust aftertreatment. It can broadly be divided into two processes: diesel oxidation catalysts (DOCs) and DPM filters.

The main function of DOCs is to oxidize CO, HC, and some liquids (PAHs) attached to solid particles in the DPM. The by-product is CO_2 and water. It has no impact on solids.

Although the designs of most DOCs are proprietary, they are usually made of either metal or ceramic monoliths

with very high surface area/volume ratios. They are normally coated with proprietary catalysts made of noble metals or base metals that lower the temperature required to oxidize the volatiles in the exhaust. They can reduce the CO and HC concentration in the exhaust by approximately 90%. Even the diesel particulate mass is reduced by about 25% by driving out and oxidizing all liquids attached to solids.

The DOCs usually last the entire life span of the engine. They are relatively cheap, costing less than a few thousand U.S. dollars for the small engines used in mines.

The main reason for the failure of a DOC is deactivation of catalysts by very high temperature (>650°C) and poisoning by lubricating oil additives (phosphorus, zinc, and heavy metals) and sulfur in fuel. Overcatalyzation of a DOC can result in the excessive production of NO_2 and sulfates if the fuel has a high sulfur content. Most DOCs are designed to match a particular diesel engine and, by and large, perform well. They are universally recommended for all diesel engines working in underground/surface mines, especially in conjunction with ultra-low-sulfur fuels. They should be installed close to the engine for best results.

DPM filters are used downstream from DOCs and usually work in tandem. They can be broadly classified in two categories: a ceramic monolith or a fiber filter media.

The ceramic monolith is made of cordierites or silicon carbide. They are often catalyzed using noble or base metals. As the exhaust flows through the monolith, DPM is collected on catalyzed surfaces and eventually burnt to gases by the "auto regeneration" process. Cordierite monoliths are liable to thermal fractures, but silicon carbide monoliths perform better. They have lower thermal shock resistance but their higher melting point makes them more durable. They also create less backpressure because they have a high permeability for gases. The collection efficiency for DPM ranges from 80% to 95% depending on the manufacturer. The life of a ceramic/silicon carbide filter is normally in excess of 5,000 hours. Routine maintenance and periodic "complete regeneration" in an off-board electric kiln is needed. In some less-demanding operations such as diesel equipment working on flat rail tracks, off-board regeneration is becoming the standard and more reliable procedure. The ceramic filters also act as flame and spark arrestors.

When a fiber filter media is used on a diesel engine, it is necessary that the diesel exhaust be cooled in a water bath. The cooled gas goes through a replaceable paper or synthetic fiber filter where solid particles are collected. The collection efficiency ranges from 90% to 97%. Depending on the duty cycle, a paper filter can last one to five shifts in operation, after which it must be replaced. Intake air resistance (usually limited to 7–8 kPa) often warns when a replacement is due. Synthetic filters can be washed, cleaned, and reused. Since the temperature of all significant parts of a diesel engine is regulated to be below 150°C, the exhaust must be cooled to this temperature before being discharged into mine air. Water scrubbers not only cool the exhaust but also act as flame and spark arrestors. Their main drawback is the high maintenance. Recently dry scrubbers have been developed that do not require the exhaust to contact the coolant. The gases are conducted through tubes that are themselves cooled by the bath. In another design, the coolant is conducted in tubes to cool the exhaust. Both designs are commonly used in underground mines. The fiber filter system is usually large in size and quite expensive. They are, therefore, more suited to bigger equipment on the surface or on locomotives in the mines. They would also be essential on permissible diesel equipment such as shuttle cars.

Dilute DPM. The earliest ventilation standard for diesel engines was developed by the United Kingdom's National Coal Board. It required 4 m^3/min of air for each brake horsepower of the engine. The latest standard for ventilation is the Pennsylvania and West Virginia standard. It requires the ventilation air to be the "nameplate ventilation" as specified by the Mine Safety and Health Administration (MSHA). This air is enough to dilute all gaseous components of diesel exhaust to or below their respective threshold limit values. The exhaust treatment system for the engine must reduce the DPM emissions to a level that, when diluted by the nameplate air, the resulting DPM concentration will not exceed 0.12 mg/m^3. With the advent of clean engines, clean fuels, and new DPM filters, it is not difficult to achieve this high standard anymore.

For multiple engines working in one air split, the old MSHA requirement was the 100-75-50 rule. This means that the ventilation air requirement for the first (usually the biggest) unit is the MSHA nameplate air, 75% of the nameplate air is needed for the second unit, and 50% of the nameplate air is needed for the third and each additional unit. The latest requirement is the total sum of all nameplate airs for the diesel units working in each air split (West Virginia Diesel Commission 1998). Ventilation air requirements for diesel engines working in mines in different situations are extensively described in Thakur (1974).

Monitor DPM emissions and maintain equipment. DPM emissions are actually based on the original eight-mode test of the engines in the laboratory. Two factors that may make the original emissions worse are poor maintenance and the age of the engine.

It is obvious to anyone working in a dieselized mine that a poorly maintained engine can aggravate the DPM levels in mine air. This can be avoided by preventive, routine maintenance of the entire diesel fleet. Such maintenance must be carried out by trained personnel on a periodic basis. Daily checking of gaseous emissions should be done to confirm the condition of diesel engines. Ambient air quality examination can also provide a warning that one or more engines are not working properly. All mine personnel should be properly trained in operating the equipment and monitoring its performance. Maintenance of a diesel engine is the key to good performance and cannot be overemphasized.

Unmaintained or poorly maintained diesel engines do show an increase in DPM emissions with age. However, a properly maintained diesel engine (including the change of injectors, if needed) should not suffer any deterioration in DPM emissions during its normal tenure in the mine. Routine maintenance as recommended by manufacturers should be carried out only by trained personnel.

Radon Gas Control

Radon is a chemically inert, gaseous, radioactive product of the disintegration of radium. It is primarily found in uranium mines, although it is present in trace amounts in other mines including coal mines. Both radium and radon are intermediate products in the disintegration of uranium-238 to become lead-206. Table 15.4-9 shows some important steps in the disintegration of uranium to lead with half-lives of the isotopes.

After radon is released into mine air, the decay process continues with formation of radium A, B, C, C′, D, E, F, and G, which is lead. These products are commonly referred to

Table 15.4-9 Uranium disintegration process

Name	Isotope	Type of Radiation	Half-Life
Uranium	$^{238}_{92}U$	Alpha	4.49×10^9 years
Radium	$^{226}_{88}Ra$	Alpha	1,622 years
Radon	$^{222}_{86}Rn$	Alpha	3.825 days
Lead	$^{206}_{82}Pb$	—	Stable

as radon daughters. They are atoms of solid matter having relatively short half-lives. They emit alpha or beta particles and rarely gamma rays, and they are the main cause of health concern.

Radon gas is inhaled and exhaled before large amounts of alpha particles are emitted. The daughter products, however, attach themselves to respirable dust particles and get deposited in the alveolar sacs. Exposure to excessively high concentrations of radon and radon daughters have been linked to lung cancer.

The maximum exposure limit for radon daughters has been set at 1.00 WL (working level), with a yearly cumulative exposure of 4 WLMs (WL months). A working level is defined as that concentration of radon daughter products in 1 L of air that will yield 1.3×10^5 million electron volts (MeV) of alpha energy in decaying to radium C′ (U.S. Department of Health, Education and Welfare 1957).

The average WL for a given 8-hour shift is calculated by dividing the sum of exposure level multiplied by the number of hours by 8. Mathematically,

$$\text{average WL} = \frac{1}{8}\Sigma \text{WL}_i \times h_i$$

where

WL = exposure
h_i = hours

The WLM = {WL × 8} / 173. This is based on the assumption that there are 173 working hours in a working month.

Except for uranium mines, radon gas has not created a health hazard in coal and metal mines. However, it is desirable to check for its presence in all mines and to take action if necessary.

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CHAPTER 15.5

Heat, Humidity, and Air Conditioning

Felipe Calizaya and John Marks

INTRODUCTION

In underground mines, heat and humidity can create adverse environmental conditions that can affect worker performance negatively. Such conditions can even limit the depth to which mining can be extended. Although the human body has a self-regulating mechanism against variations of external heat, under extreme conditions this mechanism can break down, resulting in occupational illnesses such as heat exhaustion and heat stroke.

In the United States, with a few exceptions, heat becomes an environmental problem in mines with workings at depths greater than 1,000 m (3,280 ft). In these mines, ventilation is not sufficient to control the heat loads and must be supplemented by some form of air conditioning. The Anaconda mine in Butte, Montana, and the Magma mine in Superior, Arizona, were the first to utilize air conditioning systems. Both mines are at least 1.5 km (0.9 mi) deep. Other U.S. mines that require air conditioning include the San Manuel mine in San Manuel, Arizona, which is 1,000 m (3,280 ft) deep; the Homestake mine in Lead, South Dakota, which is 2.4 km (1.5 mi) deep; the Sunshine mine in Kellogg, Idaho, which is 1.6 km (1 mi) deep, and more recently, the Meikle mine near Elko, Nevada, where heat became a environmental problem at depths ranging from 300 to 600 m (1,000 to 2,000 ft).

In Canada, heat is also an environmental concern in most deep underground mines. These include Inco's Creighton mine, which is 2,400 m (7,870 ft) below the surface; Falconbridge's Kidd Creek mine, which is 3,000 m (9,845 ft) deep; and Agnico-Eagle's Laronde mine, which is 2,000 m (6,560 ft) deep. Autocompression is the main source of heat in these mines. The ventilation systems in these mines are now being upgraded to include refrigeration plants.

This chapter presents summaries of the following topics:

- Psychrometric properties of the air
- Human heat stress indices
- Instrumentation for determining how thermal loads are estimated
- Heat sources
- Mine cooling
- The principles used to estimate airflow and refrigeration requirements

Sample problems are used to illustrate the various steps involved in determining ventilation requirements and the critical depths. The solution to each problem is found using CLIMSIM, a commercial mine climate simulation program from MVS Engineering, Clovis, California. This software is used to predict the properties of the air at various simulated workings, to evaluate the resulting parameters against the target values, and to determine a feasible solution to the problem.

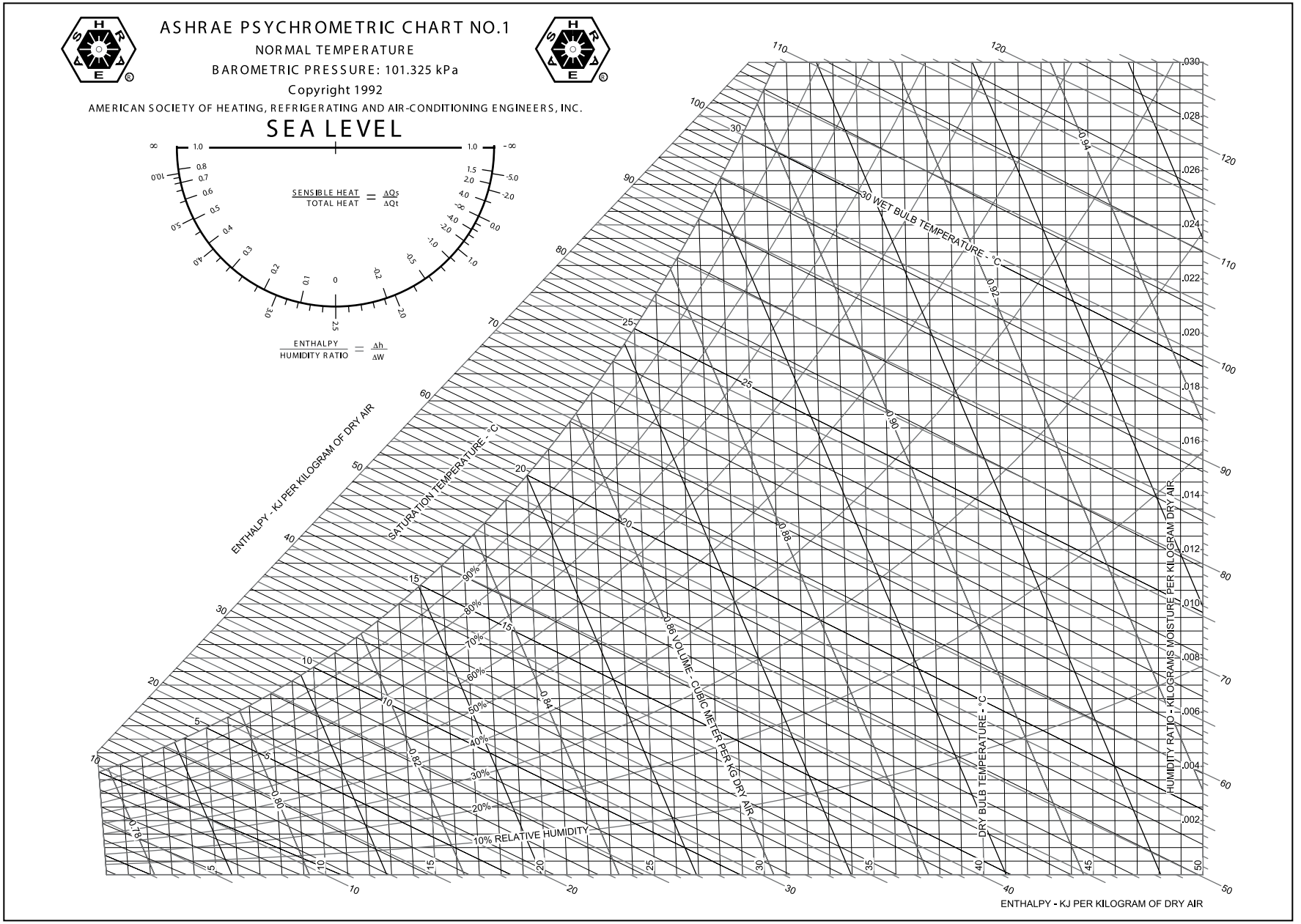
Several books and articles have been written on ventilation and air conditioning. The references at the end of this chapter provide a selected list of publications on this topic. The ventilation engineer is strongly encouraged to study these references.

Definitions

Definitions specific to the control of heat and humidity in underground mines are as follows.

- **Autocompression:** This is the term given to a process by which a column of air moving downward is compressed and experiences a temperature increase. Autocompression is due to the additional pressure of air "stacked" on itself. In a dry shaft, autocompression raises the air temperature by about 1°C (1.8°F) for every 100-m (328-ft) increase in depth.
- **Heat:** Heat is a form of energy and, as such, it can never be destroyed. In mines, heat is produced by chemical reactions such as the oxidation of ores (slow) or the burning of fuels (fast), by friction (pulleys under a stationary belt), and in many other ways.
- **Heat stress:** Heat stress is a qualitative assessment of the work environment based on temperature, relative humidity, air velocity, personal clothing, and radiant energy (environmental factor). Several heat stress indices have been proposed. The most common ones are effective temperature, cooling power, and wet bulb temperature.

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Source: Copyright 2009 ASHRAE Pschrometric Chart Sea Level SI, www.ashrae.org.

Figure 15.5-1 ASHRAE psychrometric chart

- **Heat strain:** The physiological response to heat stress (people factor) is called heat strain. Effects include dehydration, increased heart rate, fatigue, cramps, and heat stroke. Humans have different tolerance levels for heat. They can cope easier with heat than cold.
- **Sensible heat:** This is the heat that when added to or removed from a substance changes its temperature without changing its state.
- **Latent (hidden) heat:** This is the heat added to or removed from a substance that results in a change in state but does not increase or decrease its temperature.
- **Enthalpy:** Enthalpy is the total heat content of the air–vapor mixture per unit weight (sum of latent and sensible heat).
- **Temperature:** Temperature is the degree of hotness or coolness of a substance (measurement of a state).
- **Critical depth:** This is the depth in a mine at which the wet bulb temperature of the intake air exceeds the allowable wet bulb temperature, mainly due to autocompression. Beyond this depth, the mine will require some form of air conditioning.

PSYCHOMETRY AND AIR CONDITIONING

Psychrometry is the science of evaluating the thermal state of moist air. The psychrometric properties of interest to a ventilation engineer are dry bulb temperature (t_d), wet bulb temperature (t_w), dew-point temperature (t_{dp}), barometric pressure (P_b), specific volume (V), humidity ratio (W), and enthalpy (h). Because air is considered a mixture of dry air and water vapor, any three of these are necessary to be known, and the remaining properties can be calculated (McPherson 1993; Hartman et al. 1997). The easiest three to measure are t_d, t_w, and P_b. The determination of all of these properties is a prerequisite to solving air conditioning problems.

Three tools are available for estimating these properties: psychrometric tables, charts, and the fundamental equations. There are several computer programs available that make use of these tools and more advanced numerical solution methods. When tables are not available, the basic properties can be calculated using Equations 15.5-1 through 15.5-8 (McQuiston and Parker 1994).

Specific Volume, V:

$$V = \frac{RT_d}{P_b} \text{ m}^3/\text{kg} \qquad (15.5\text{-}1)$$

where

R = a gas constant, which is 287 J/(kg K)
T_d = the absolute temperature, which is equal to 273 + t_d, K
P_b = barometric pressure, Pa

Saturated Vapor Pressure, P_s:

$$P_s = 0.6105 \exp\left[\frac{17.27 t_d}{t_d + 237.3}\right] \text{ kPa} \qquad (15.5\text{-}2)$$

where t_d is the dry bulb temperature, °C.

It is evident in Equation 15.5-2 that the saturation vapor pressure is a function of temperature only.

Actual Vapor Pressure, P_v:

$$P_v = P_s - 0.000644 P_b (t_d - t_w) \text{ kPa} \qquad (15.5\text{-}3)$$

where

P_s = saturated vapor pressure, kPa
t_d = dry bulb temperature, °C
t_w = wet bulb temperature, °C

Relative Humidity, RH:

$$RH = \frac{P_v}{P_s} 100\% \qquad (15.5\text{-}4)$$

where

P_v = water vapor pressure, kPa
P_s = saturated vapor pressure, kPa

Absolute Humidity (or the "Humidity Ratio"), W:

$$W = 0.622 \frac{P_v}{(P_s - P_v)} \text{ kg/kg} \qquad (15.5\text{-}5)$$

where

P_v = water vapor pressure, kPa
P_b = barometric pressure, kPa

Air Density, d:

This equation gives kilograms of moisture per kilogram of dry air.

$$d = \frac{1}{V}(1 + W) \text{ kg/m}^3 \qquad (15.5\text{-}6)$$

where

V = specific volume, m^3/kg
W = absolute humidity, kg moisture/kg dry air

Enthalpy, h:

$$h = 1.005 t_d + W(2501.3 + 1.86 t_d) \text{ kJ/kg} \qquad (15.5\text{-}7)$$

where

t_d = dry bulb temperature, °C
W = absolute humidity, kg moisture/kg dry air

The *2007 ASHRAE Handbook* gives the following values for standard air: sea level temperature = 15°C and barometric pressure = 101.325 kPa. The barometric pressure, P_b, depends mainly on elevation and can be approximated as follows:

$$P_b = 101.325 (1 - 0.0000225577 Z)^{5.2559} \text{ Pa} \qquad (15.5\text{-}8)$$

where Z is the elevation in meters.

Example 1. Compute the humidity ratio and enthalpy of saturated air at 20°C and at a barometric pressure of 101.325 kPa.

Solution. Perform the following calculations:

- From Equation 15.5-2, for $P_v = P_s$ at 20°C, $P_v = 2.338$ kPa.
- Using Equation 15.5-5, $W = 0.0147$ kg/kg.
- Using Equation 15.5-7, for $W = 0.0147$ kg/kg and t = 20°C, $h = 57.56$ kJ/kg.

Quick estimates of all the psychrometric properties of air can be determined from a psychrometric chart. Each chart applies to air at a given barometric pressure. Figure 15.5-1 shows a psychrometric chart for $P_b = 101.325$ kPa. This is one of a series of charts produced by the American Society of Heating, Refrigerating, and Air-Conditioning Engineers (ASHRAE) for a range of pressures. Although all psychrometric properties can now be calculated using programs such as HotWork from

MVA (Mine Ventilation Australia, Sandgate, Queensland, Australia) (Brake 2008), charts provide a visual representation of process lines that represent changes in psychrometric conditions of the airstream. Figure 15.5-2 shows how the various processes can be followed from an initial state point (point A). Horizontal lines emerging from this point indicate changes in sensible heat, that is, cooling if the process proceeds from right to left or heating if the process proceeds from left to right. A vertical line indicates changes in latent heat—humidification if the process proceeds upward and dehumidification if the process proceeds downward. Combination processes run diagonally.

Example 2. A flow rate of 120 m^3/s of air at $t_d = 30°C$, $t_w = 20°C$, and $P_b = 101.325$ kPa is to be chilled to $t_d = t_w = 10°C$ at the same pressure. First, calculate the following:

- The mass flow rate of dry air through the process
- The kW(R) required to chill the air
- The liquid water condensed in the air
- The relative humidity of the air entering the process

Solution. For $t_d = 30°C$, $t_w = 20°C$, and the specific volume of air is 0.873 m^3/kg, calculate as follows:

- Mass flow rate: $m = Q/v = 137.3$ kg/s, where Q is the airflow rate in cubic meters per second
- Removed heat: $q = m(\Delta h) = 137.3 \times (57.2 - 29.4) = 3{,}817$ kW(R)
- Condensed water: $q_l = m(\Delta W) = 137.3\ (10.6 - 7.7) = 398.20$ g/s (0.40 kg/s)
- Relative humidity: $RH = 40\%$ (from ASHRAE chart)

HEAT STRESS INDICES

These indices refer to the assessment of thermal conditions to which workers can be exposed repeatedly without developing any adverse health effect. They are expressed in terms of quantitative figures (maximum allowable limits) used by ventilation specialists for designing heat control systems. Since the introduction of the first heat stress index in 1923, more than 90 indices have been developed. Based on their application, these can be divided into two groups: empirical and rational. Empirical indices are expressed in terms of an external or environmental factor, such as the wet bulb temperature and the effective temperature. Rational indices are based on the responses of the workers to thermal stress, such as sweat rates and core temperature increases (internal factors). Only the empirical indices are applicable to underground mining conditions.

In most mining countries, these indices are regulated by code (Hartman et al. 1997; Hardcastle 2006). In U.S. mines, heat stress is not regulated. Management must set and enforce the applicable limits. The following are the most commonly used indices (Van der Walt et al. 1996; Del Castillo and Dawborn 2002; Mutama 2005; ACGIH 2008):

- Wet bulb temperature: $t_w \leq 27°C$ (81.6°F)
- Wet bulb globe temperature: $WBGT \leq 27°C$ for continuous work at moderate rate
- Effective temperature range: ET = 28°C (81.6°F) for reduced work load and a maximum of 32°C (89.6°F) when work must be terminated
- Air-cooling power (ACP) for hard workload: 175 $W/m^2 \leq ACP \leq 275$ W/m^2
- Air velocity: $v \geq 0.50$ m/s

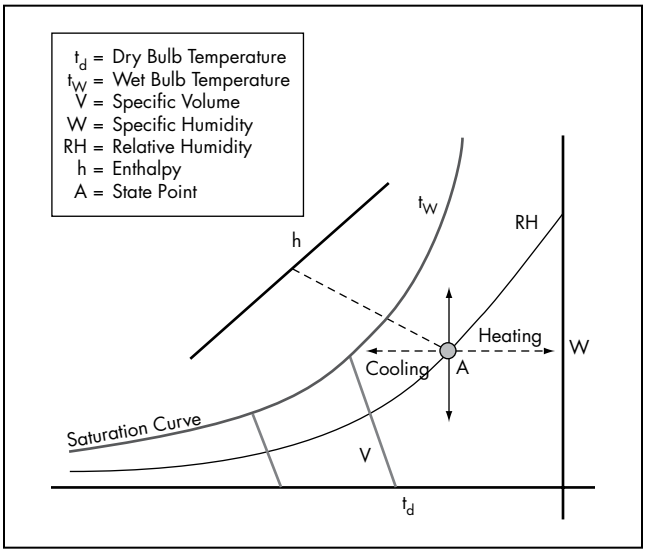


Figure 15.5-2 Psychrometric processes and state point

The wet bulb temperature is the simplest index that can be used by ventilation engineers to assess the work environment. It measures the evaporation of water from a wet bulb thermometer. It can be determined by a sling psychrometer after being whirled or vented at the rate of at least 3 m/s (10 fps).

The wet bulb globe temperature for an indoor environment is measured by a black globe thermometer or calculated using Equation 15.5-9 (ACGIH 2008):

$$WBGT = 0.7NWB + 0.3GT \tag{15.5-9}$$

where

$WBGT$ = wet bulb globe temperature, °C
NWB = natural (or unvented) wet bulb temperature, °C
GT = globe temperature, °C

Effective temperature is an index of relative comfort determined from charts that require the knowledge of three parameters: dry bulb temperature, wet bulb temperature, and air velocity. It is still used in U.S. mines with negligible radiant heat.

Air cooling power is a rational heat stress index based on heat balance between workers and environment. For hard work rates, this index ranges from 175 to 275 W/m^2, depending on air velocity, wet bulb temperature, and type of clothing. For safety, vent systems should be designed for cooling power rates higher than the expected work rate.

Air velocity, as measured by an anemometer, is not an index, but it can be used in combination with the wet bulb temperature. A common lower limit for airways where personnel work is 0.50 m/s (1.6 fps).

The following are sample heat stress indices used in some hot mines:

- Homestake gold mine, South Dakota
  - Wet bulb temperature for acclimatized workers is $26.7°C \leq t_w \leq 29.4°C$ (the lower limit indicates 100% worker efficiency, and the upper limit indicates that a corrective action is needed)
  - The air velocity range is $0.76 \leq v \leq 3$ m/s
  - The wet bulb globe temperature, $WBGT$, is ≤32.2°C



Figure 15.5-3 Digital barometer

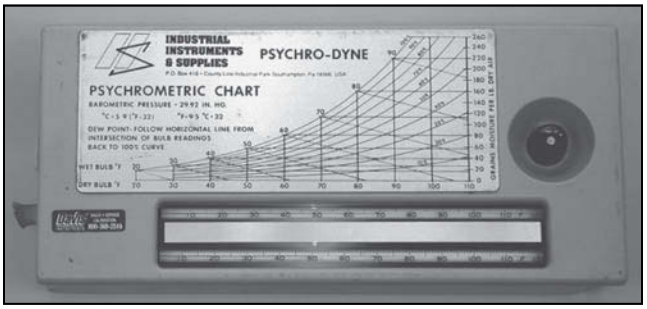


Figure 15.5-4 Portable psychrometer

- Meikle gold mine, Nevada
  - The wet bulb temperature, t_w, for acclimatized workers is ≤27.8°C
  - The effective temperature, *ET*, is ≤26.6°C
- Mindola copper mine, Zambia
  - The wet bulb temperature is 28°C ≤ t_w ≤ 30°C (depending on the air velocity, t_w is 28°C for v = 0.77 m/s and 30°C for 1.99 m/s)
  - Air-cooling power is $ACP \geq 300$ W/m^2

If two heat stress indices of a work environment exceed the design limits, the mine can take one or more of the following actions:

- Install ventilation and/or air conditioning systems (active engineering controls)
- Isolate the heat sources via insulation or blocking off heat sources (passive engineering controls)
- Enact administrative measures

Ventilation and/or air conditioning are generally the preferred control methods for long-term work areas, but they can be very expensive. Isolating heat sources can be done in certain cases, but if active or passive controls are uneconomical, the mine can consider administrative controls. These include

- Increasing workers' heat tolerance by heat acclimatization,
- Assigning workers to hot working areas selectively (on the average about 5% of relatively young workers are heat intolerant),
- Adjusting the work–rest regime by permitting workers to take short breaks to recover sweat rate and body core temperature, and
- Rotating personnel on hot jobs and providing workers with readily accessible cold areas, cold drinking water, and salt tablets. A half-teaspoon of salt per 2 L of water is sufficient.

In every case, the objective is to prevent the body core temperature from rising above 38°C (100°F).

INSTRUMENTATION

The following instruments are used to determine the heat load in underground mines: a barometer, a psychrometer, and thermocouples. Although most of these units are now digital, analog devices are still being used under harsh conditions and as primary standards for calibration.

Barometric Pressure

Barometers are used to measure the absolute pressure of mine air. This pressure is mainly a function of elevation of the working areas. Under standard conditions, the barometric pressure varies by about 500 Pa (0.07 psi) over a 24-hour period. Under special conditions, such as during major storms or cyclones, the pressure difference can be as high as 2,000 Pa (0.3 psi). Aneroid barometers, such as those manufactured by American Paulin System, Cottonwood, Arizona, are still being used for pressure measurement in underground mines. Because of their ruggedness and ease of operation, they are preferred by ventilation engineers. With the advent of electronic technology, digital barometers are gaining a wider acceptance in ventilation surveys, especially in hard-rock mining. They are based on pressure transducers that are highly sensitive to variations of pressure load. Figure 15.5-3 shows a capacitor-type barometer in a rugged enclosure. The instrument basically consists of two closely spaced metal surfaces, one of which is a diaphragm that flexes slightly under applied pressure and alters the gap between them (variable capacitor). This variation in distance is picked up electronically by the sensor. The instrument is highly accurate in the range of 60 to 110 kPa (8.7 to 16.0 psi).

Air Temperature Measurement

The dry and wet bulb temperatures of the air are measured by a psychrometer. This instrument consists of two identical thermometers: one to measure the true air temperature (dry bulb) and the other, covered by a water-saturated wick, to measure the evaporative cooling effect (wet bulb). The drier the surrounding air, the more rapidly the water will evaporate. This causes the wet bulb to depress in temperature until a balance takes places between the cooling effect and the heat gain from the ambient air. A requirement to measure the wet bulb temperature is the air velocity across the bulb. This velocity must be at least 3 m/s (10 fps). Without this velocity, one will always tend to measure a higher humidity level than actual.

Figure 15.5-4 shows a portable psychrometer. This instrument consists of two thermometers (dry and wet) and a small fan. Before any measurement, the wick on the wet bulb must be saturated with distilled water. While the fan is operating, the wet bulb temperature will begin to depress until it stabilizes at its lowest point. The procedure may take from 1 to 2 minutes depending on the dryness of the air. For better accuracy, the wet bulb temperature should be read first. The instrument can be used for temperature measurements in the –7° to 43°C (19° to 109°F) range. A humidity indicator (Figure 15.5-5) is another device used in underground mines. The instrument is based on a thin polymer film sensor that changes its capacitance as it absorbs water molecules. When equipped with a Vaisala HMP41 probe, it can be used to measure relative humidity, absolute humidity, dry and wet bulb temperatures, and dew-point temperature. The operating range of the instrument is from –20° to 60°C (4° to 140°F).

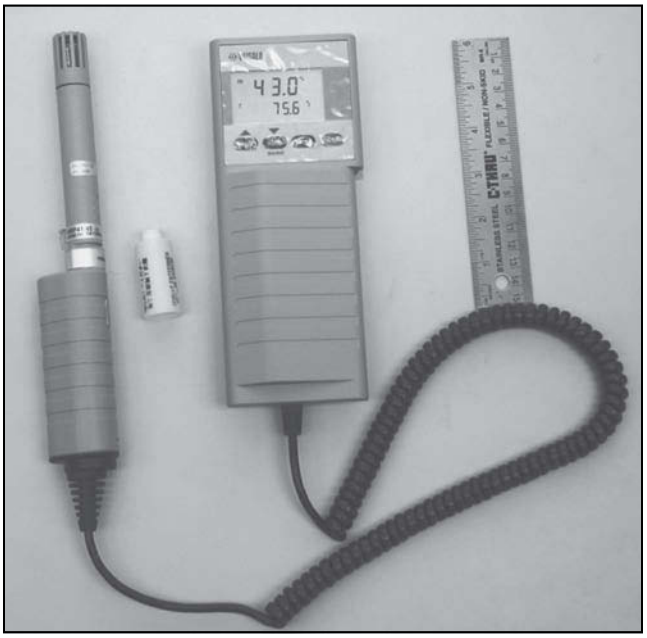


Figure 15.5-5 Digital humidity indicator

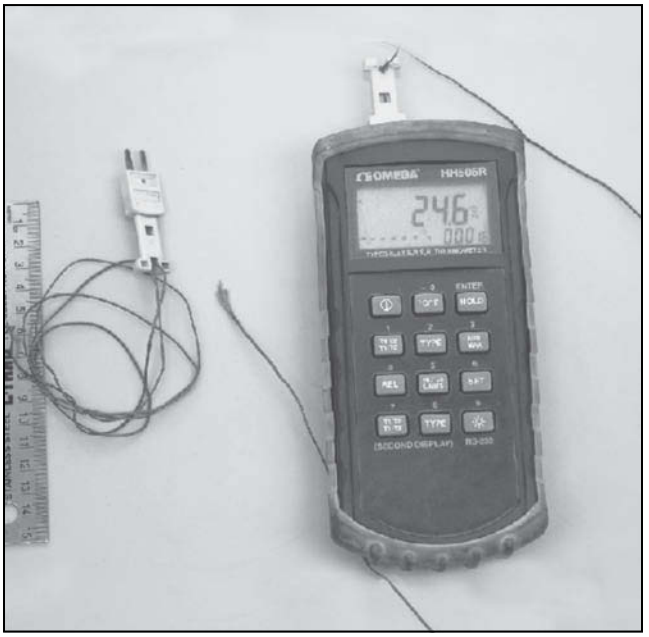


Figure 15.5-6 Thermocouple and digital thermometer

Rock Temperature Measurement

The rock temperature in a mine is usually measured by means of resistance thermometers and thermocouples. A resistance thermometer is constructed of a precious-metal conductor, such as platinum or silver, which exhibits increased resistance when heated. A thermocouple consists of insulated wire pairs with a junction point (bead) at one end and a male connector at the other. For underground applications, J-, K-, and T-type thermocouples are used. When a thermocouple is connected to a temperature indicator (reader), an analog or digital signal is obtained, making it suitable for temperature logging. Rock temperatures can be measured from boreholes that are drilled from underground openings or from the surface. In either case, it is good practice to allow the rock temperature to stabilize and eliminate the local heating and cooling effects. This is usually achieved 8 hours after the drilling is completed (Duckworth 1999). A plot of temperature verses depth is often used to determine the temperature distribution in the borehole and the virgin rock temperature (VRT). Figure 15.5-6 shows thermocouple wires and a digital thermometer. The fine wire thermocouples enable accurate temperature measurements without disturbing the base temperature of the body in which the installation is made. They are available in wire sizes ranging from 0.025 to 0.81 mm in diameter. The K-type thermocouple has an operating range of from –200° to 1,372°C (–328° to 2,502°F). The HH506R data logger (Omega Engineering) is a two-channel temperature-measuring device used with seven types of thermocouples. When recording mode is used, the instrument allows the user to store the maximum, minimum, and average temperatures during a 24-hour period.

HEAT SOURCES

Heat is emitted from various sources in underground mines, including autocompression, heat from wall rock, mining equipment, groundwater, oxidation of minerals, and metabolic heat. A study conducted on heat flow in seven underground mines in the United States has shown that wall rock, autocompression, and mining equipment account for about 80% of the total heat released. In shallow mines with depths of less than 1,000 m (3,280 ft), ventilation alone may be sufficient to remove the heat from the working sections. However, in deep mines with depths greater than 1,000 m (3,280 ft), the autocompression effect causes the wet bulb temperature to exceed the critical temperature of 28°C (82°F), and refrigeration becomes a necessity. The design of a refrigeration system requires a thorough evaluation of all sources of heat.

Heat from Rock

The heat flow from the rock depends on a number of variables, including the virgin rock temperature, rock type, size, shape, and depth of the openings and the presence of water on the rock surface. It also varies with the age of the opening. New and well-ventilated airways rapidly give off heat. After the airway has aged, the rock wall temperature will typically stabilize to within 2°C (3.6°F) of the dry bulb temperature. For rough estimates, Equation 15.5-10 can be used to approximate the radial heat flow, q, from the rock.

$$q = 2\pi\kappa L \frac{t_1 - t_2}{\ln(r_1/r_2)} \text{ W} \qquad (15.5\text{-}10)$$

where

κ = thermal conductivity, W/m °C
L = airway length, m
t_1 = rock wall temperature, °C
t_2 = virgin rock temperature, °C
r_1 = airway radius, m
r_2 = distance into the rock, m

Thermal conductivity is a measure of how well (or poorly) heat is conducted (or resisted) in a certain substance. It represents the temperature profile along each meter of drift length for a given heat flow. However, it does not measure how quickly the heat spreads along the drift. Thermal diffusivity is a measure of how quickly the heat spreads through the substance. Table 15.5-1 shows the thermal properties for

Table 15.5-1 Thermal properties of common substance

Substance	Conductivity, W/m °C	Specific Heat, kJ/kg °C	Diffusivity, $m^2/s\ 10^{-6}$
Water	0.60	4.19	0.14
Dry air	0.026	1.005	21.5
Coal	2.20	1.00	1.29
Copper	380	0.38	112
Gabbro	2.37	—	2.37
Granite	1.92	0.82	3.33
Pyritic shale	3.65	—	2.01
Quartz	5.50	0.88	2.32
Sandstone	1.97	0.71	1.68
Shale	2.39	—	0.90
Rhyolite	3.46	—	1.11
Gneiss	3.18	—	1.63
Concrete	1.20	0.88	0.59

Source: Adapted from ASHRAE 2007 and Brake 2008.

a number of substances determined from laboratory or field tests (ASHRAE 2007; Brake 2008). Temperatures t_1 and t_2 are rock temperatures measured at radii r_1 and r_2, respectively. For rectangular openings, r_1 is approximated by the hydraulic radius (= 2 × area/perimeter). Radius r_2 is the distance into the rock from the center of the opening. The virgin rock temperature t_2 is determined from a borehole that is at least 4 m (13 ft) long and drilled perpendicular to the longitudinal axis of the opening.

A pair of t_i and r_i measurements, together with the length of the opening and the thermal conductivity of the rock, is sufficient to determine the heat flow into an opening. However, to improve the accuracy of the estimates, it is recommended to take a series of measurements from various boreholes drilled radially from the airway.

Example 3. A 400-m-long drift that is 4 m in diameter (r_1 = 2 m) is driven in quartzite rock (Figure 15.5-7). The VRT at r_2 = 10 m is determined to be 50°C. If the rock wall temperature is 30°C, how much heat will flow into the section?

Solution. For κ = 5.5 W/m °C, $(t_1 - t_2)$ = 0°C and $\ln(r_1/r_2) = -1.61$, Equation 15.5-10 yields q = 172 kW. For a given q (measured directly), this approach can be used to determine the thermal conductivity of the rock.

Autocompression

Autocompression is the process in which a column of air is compressed and experiences a temperature increase as it enters a downcast shaft. In dry shafts, the compression occurs adiabatically. On average, the dry bulb temperature increases by 1°C (1.8°F) per 100 m (328 ft). This occurs even when there is no external heat added into the shaft or external work done on the air. The wet bulb temperature also increases with depth, but at a lower rate. Typically, this temperature increases by about 0.6°C (1.1°F) per 100 m (328 ft) when the surface wet bulb temperature is 6°C (43°F) and 0.4°C (0.7°F) per 100 m (328 ft) when the surface wet bulb is 25°C (77°F) (Brake 2008). In an upcast shaft, the air decompresses as it travels up the shaft, resulting in a drop of both dry and wet bulb temperatures, a reduction in air density, and a decrease of

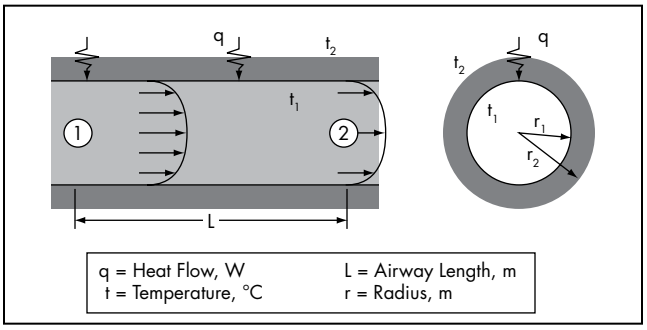


Figure 15.5-7 Heat flow from rock

atmospheric pressure. In locations with high humidity levels in the upcasting shafts and drifts, the decrease in temperature associated with decreasing pressure may result in the generation of fog as the conditions of saturation occur.

For a downcast shaft, the heat flow, q, can be estimated by

$$q = Q\frac{1}{V}(h_2 - h_1)\,\text{kW} \qquad (15.5\text{-}11)$$

where

Q = airflow rate, m^3/s
h_1 = enthalpy at state point 1 (shaft collar)
h_2 = enthalpy at state point 2 (shaft bottom)

Example 4. What is the heat load from autocompression when 100 m^3/s of dry air of t_{d1} = 30°C and t_{w1} = 20°C at sea level drops down a 1,500-m shaft?

Solution. Estimate the wet bulb temperature at the bottom of the 1,500-m shaft (t_{w2} = 26.8°C by interpolation); determine the specific volume of the air at shaft collar (V = 0.873 m^3/kg); and determine the enthalpies for t_{w1} and t_{w2} (h_1 = 57.2 kJ/kg and h_2 = 83.7 kJ/kg from Figure 15.5-1). Then, using Equation 15.5-11, q = 3,025 kW.

Electrical Equipment

Electrical motors, transformers, gearboxes, and other controls are not 100% efficient. Some of these inefficiencies are manifested in the form of sensible heat. All energy used in electrical equipment is transformed into two effects: work against gravity and heat. Conveyor belts, hoists, and pumps are examples where the electrical energy is transformed into potential energy and frictional heat. Other devices, such as rock breakers, fans, electric drills, and lights used on a horizontal plane, are examples where electrical energy is converted entirely into heat. Using Equation 15.5-12, for any given machine, the heat load, q, can be calculated as the difference in effective power and work done against gravity as follows:

$$q = PE - mgh\ \text{kW} \qquad (15.5\text{-}12)$$

where

P = input power, kW
E = total efficiency, decimal fraction
m = mass flow rate, kg/s
g = 9.81 m/s^2
h = elevation difference, m

Fans dissipate heat. However, this heat is detrimental only when the fan is located upstream of the main workings (either as blower or booster fans).

Example 5. A 1,500-m-long conveyor belt system transports ore at the rate of 720 t/h. The vertical lift is 280 m. The system consumes 2,500 kW of power at an efficiency of 85%. Determine the heat load along the belt line.

Solution. For an input power of 2,500 kW, an efficiency of 85%, m = 200 kg/s, and h = 280 m, Equation 15.5-12 yields q = 1,576 kW.

Diesel Engines

When diesel equipment is used in underground mines, in addition to smoke and fumes, a significant amount of heat is added to the air. The heat flow from a diesel engine is about 2.8 times the heat produced by an equivalent electric unit. About a third of this is added at the radiator, another third at the exhaust pipe in the form of smoke, and the remainder as useful work, which is finally also converted into heat. Each volume of fuel produces from three to ten volumes of water vapor. This results in increases of both latent and sensible heat of the air. The best way to determine the heat load is to estimate the fuel consumption, F, and the combustion efficiency, E. Using this information and the calorific value of the fuel (C = 34,000 kJ/L), the heat load, q, can be calculated using Equation 15.5-13.

$$q = FEC \tag{15.5-13}$$

Alternatively, it can be approximated by multiplying the effective engine power by 2.8 ($q = 3PE$).

Example 6. A 30-t truck (298 kW) is operated at 80% of its nameplate specifications in a development heading. How much heat does this engine produce when fully loaded?

Solution. From Equation 15.5-13, $q = 2.8 \times 298 \times 0.8 =$ 668 kW. This engine burns 668 kW of fuel to produce 298 kW of effective power.

Heat from Broken Rock

When freshly blasted rock is exposed to ventilation air and there is a temperature difference between them, there will be heat flow from the rock to the air. In most cases, the broken rock temperature will be less than the local VRT. However, the difference is small, and for rough estimations it can be ignored. Further, the rock temperature at the storage bin or exit point will be equal to the ambient air temperature. Based on these assumptions, the heat load from broken rock can be approximated using Equation 15.5-14 as follows:

$$q = mC_p(VRT - t_d)\ \text{kW} \tag{15.5-14}$$

where C_p is the specific heat of broken rock, kJ/kg°C.

Example 7. One hundred metric tons per hour of broken rock (granite) with a C_p = 0.82 kJ/kg°C is transported from a drawpoint where the rock temperature is 45°C to an ore bin where the rock has cooled down to 30°C. Estimate the heat transfer from the broken rock to the ventilation system.

Solution. For m = 27.78 kg/s and Δt = 15°C, Equation 15.5-14 yields q = 342 kW.

Heat from Groundwater

Mine water comes from two sources: natural fissures in the rock and human-induced sources from various applications, including water used for cooling. The fissure water has usually the same temperature as the VRT. However, the temperature of human-induced water can vary from below ambient temperature to more than 40°C (104°F). Ventilation air will pick up the heat from hot water in uncovered ditches. The heat flow from groundwater can be estimated by applying Equation 15.5-14, where the C_p(water) = 4.19 kJ/kg°C. Whenever possible, groundwater should be confined at the source and transported through pipelines or covered ditches.

Example 8. A mine produces 0.010 m^3/s of fissure water at the local VRT of 55°C. If the water exits the discharge system at 30°C, how much heat is added to the ventilation air?

Solution. For m = 10 kg/s and Δt = 25°C, q = 1,048 kW.

Other Sources of Heat

Other sources of heat include those from the oxidation of timber, sulfide ores, and carbon-based minerals. Oxidation of timber usually begins with the growth of bacteria in poorly ventilated areas (with less than 2% oxygen). Coal and pyritic sulfides will oxidize at normal ambient temperatures. The reaction is exothermic. If the heat is not removed, this can initiate a self-heating process and can develop into fires. Except for mines with extensive caved areas, this heat is usually swept out by ventilation and ignored in the estimation of the heat load.

Total Heat Load

For a given section, the heat load is determined as the sum of the heat loads from different sources. Once evaluated, this load is used to determine the ventilation or refrigeration requirements. The requirements are calculated by equating the wet bulb temperature at the exit of the section to the reject temperature. Psychrometric charts can be used to determine the size of fans and cooling devices.

Example 9. Air enters a section of a mine at t_d = 26°C, t_w = 20°C, and 101 kPa of absolute pressure. The heat load in the section is estimated at 3,000 kW. If the wet bulb temperature of the air leaving the section is not to exceed 28°C, what is the required quantity of air?

Solution. For V = 0.850 m^3/kg, h_1 = 57 kJ/kg, and h_2 = 89.5 kJ/kg, using Equation 15.5-11, q = 78.5 m^3/s.

MINE COOLING

When actual or projected heat loads exceed the ability of a mine's ventilation system to remove the heat, cooling methods must be considered. In principle, a mine cooling system is a heat-removal system. A substance such as air, water, or ice is sent into the mine at a low enthalpy state and removed at a higher one. Planning for existing or proposed mines generally follows the methods listed in this section (McPherson 1993; Howes and Hortin 2005).

Airflow Increase

It is generally less expensive to increase airflow than to install refrigeration. Primary fans can be replaced or sped up, booster fans can be installed, or new airways can be driven. However, for older mines, airflow increases are often impractical because of the cube relationship between fan power and airflow increase through a given resistance. Nor does increasing airflows help mines at or below the critical ventilation depth, which is the depth at which the intake shaft wet bulb temperature exceeds the design reject temperature.

Chilling the Service Water

This technique was developed in South Africa. All metal mines require service water to suppress dust whenever rocks are drilled or broken and to wash sidewalls and the face for ground support and geologic mapping. Chilled service water

is among the most flexible cooling methods available, because it is used where needed the most and then turned off and routed elsewhere. It intercepts rock heat before that heat enters the air. However, chilling the service water requires a refrigeration plant and insulated pipelines. Plate-and-frame evaporators can produce 1°C (34°F) water, whereas shell-and-tube heat exchangers are limited to about 3.3°C (38°F) because of potential tube rupturing. The effect can be wasted if chilled service water is used where cooling is not required. Chilled service water can be potable or nonpotable, but in any case it should be treated for microorganisms.

Bulk Cool Intake Air on Surface

This method is used at warm-climate mines to provide winter-like or better conditions year round. Surface refrigeration plants are almost always less expensive and easier to install and maintain than equivalent underground plants, are not limited in heat rejection, and are not subject to disruption following underground power outages. Ammonia machines can be used. Compressors are usually centrifugal or rotary screw. Multiple machines are often installed in parallel so that a single machine can be down for maintenance without disrupting cooling duty to any significant extent. Depending on conditions, evaporators and/or condensers on multiple machines can be installed in series to provide greater temperature difference (lift) to water temperatures. Mine refrigeration machines typically provide a lift of about 6° to 8°C (11° to 14°F). The lower positional efficiency and utilization over a year's time generally preclude surface bulk cooling from being used in cooler-climate mines.

Bulk Cool Air Underground

This practice has a higher positional efficiency and a higher utilization than do surface plants because it is closer to work areas. It also utilizes the full heat-removal potential of the ventilation system by rejecting heat to exhaust air via underground cooling towers. The refrigerant used must be nontoxic. Both centrifugal and rotary-screw compressors are used. As with surface plants, multiple machines are usually installed in parallel.

Installing Chillers

Also known as district cooling, midsize refrigeration machines of 350 to 1,250 kW(R) can chill water for a closed-loop system of cooling coils or spray chambers. These coils and chambers are located at the entrances of work areas. Coils can be located in a bank. Auxiliary fans draw air through the cooling units and blow it through a duct to work headings. Refrigeration machine condenser heat is preferably rejected to clean service water, if enough quantity is available. But heat is often rejected to mine sump water before being pumped out of the mine. Problems encountered with district cooling are coil-air side and condenser tube-water side fouling.

Spot Coolers

These units range from 70 to 350 kW(R) capacity. The units are portable and contain one or two direct-expansion cooling coils, one or two condensers, one or two compressors, and a fan. Cooled air is delivered through a duct to the heading. Condenser heat is usually rejected to the mine service water. This water typically discharges at temperatures between 35° and 45°C (95° and 113°F), and thus it should be piped to prevent it from contacting intake airflows. The service water flow rate is approximately 0.02 L/s per kW(R) of cooling at 21°C (70°F) water temperature, but can be less or more depending on temperature.

Spot coolers are popular with the work force. Although limited to moderate cooling duties, they are easily moved to positions close to advancing headings. Spot coolers cost more per unit of cooling but can be the only choice for development, exploration, or production workings on the periphery of operations. The service water system, mine dewatering system, and surface treatment or disposal system must be capable of handling the increased water flow required for condenser cooling. Some applications might employ air-cooled condensers. In the past, reciprocating compressors were used. More recently, scroll-type hermetic compressors have been employed because they are less expensive and can handle liquid slugging better than can reciprocating units. Larger spot coolers can use rotary-screw compressors. Current designs use non-ozone-depleting and non-global-warming refrigerants. As with water-cooling coils, direct expansion coils on spot coolers must be washed, the frequency depending on how much dusty air the unit receives.

Cooling Towers

Cooling towers utilize the full heat-removal capacity of a mine by rejecting refrigeration machine condenser heat to the exhaust air. Exhaust air typically exits the work areas at a temperature close to the design reject temperature, for instance, 28°C (82°F) wet bulb. A common misconception is that heat cannot be rejected to saturated air through evaporation, but the process in a cooling tower involves both sensible heating of the air by hot water followed by latent, or evaporative, cooling of the circulating condenser water. Latent heat is typically about 80% of total heat transfer. Water sprayed in the cooling tower is often about 45°C (113°F), which cools to about 35°C (95°F) before being sent back to the refrigeration room. Actual temperatures depend on the efficiency of heat transfer, both in tower and machine condensers, and on various design parameters, such as water flow rate and condensing temperature. Equation 15.5-15 calculates the total heat transfer on the water side, H_w, and Equation 15.5-16 calculates the total heat transfer on the air side, H_a, as follows:

$$H_w = M_w C_p \Delta t \text{ kW} \qquad (15.5\text{-}15)$$

where

M_w = mass flow of water, kg/s
C_p = specific heat of water, 4.18 kJ/kg°C
Δt = inlet water temperature minus the outlet water temperature, °C

$$H_a = M_a \Delta h \text{ kW} \qquad (15.5\text{-}16)$$

where

M_a = mass flow of air, kg/s
Δh = specific enthalpy of exit air minus the specific enthalpy of entering air, kJ/kg

Cooling tower analysis proceeds by equating H_w with H_a. Mines use the "factor-of-merit" method, developed in South Africa, for planning, designing, and analyzing cooling towers and spray chambers. The factor-of-merit for a direct-contact heat exchanger, such as a cooling tower or spray chamber, ranges between 0 and 1, where 0 implies no heat transfer takes place and 1 implies as much heat transfer as permitted by the second law of thermodynamics (Burrows 1982). In other words, the

fluid with the lower thermal capacity, either water or air, heats up or cools down to the entering temperature of the other fluid.

Normally the designer knows the exhaust air quantity and temperature available for the cooling tower through measurements or the mine plan. For a new mine, the tower entering air wet bulb temperature can be assumed to be about 1.5°C (2.7°F) lower than the design reject. The lower temperature is caused by leakage airflows entering exhaust. At most deep and hot mines, the air can be assumed to be saturated. Very roughly, 19.8 to 40.3 kW of heat can be rejected to 1 m³/s (2,100 cfm).

Cooling tower analysis is a complicated procedure. The designer must know or assume the heat to be rejected (the evaporator duty times the condenser heat-rejection factor), the entering air quantity and temperature, the tower factor-of-merit (from experience, based on accepted design criteria), and a first guess on the condenser water flow rate. Condenser water flow depends on the heat to be rejected and the refrigeration machine specifications. The exiting air temperature, the entering and exiting water temperatures, and the water makeup requirements can then be calculated via an iterative procedure. An example of cooling tower design for a 3.5-MW underground refrigeration plant is given in the 2007 ASHRAE applications handbook.

Large Spray Coolers

Large spray chambers are used to bulk-cool ventilating airflows that are used for mining operations. These chambers can be located on the surface at the intake shaft or underground. Chambers are usually two- or three-stage horizontal openings equipped with Vee Jet spray nozzles lining the chamber. Wave-fin mist eliminators at the exit keep water from blowing out of the chamber. More details are given in the 2007 ASHRAE applications handbook.

Small Spray Chambers

Small spray chambers are typically portable, enclosed, direct-contact heat exchangers for small duties 35 to 350 kW. An auxiliary fan draws or blows air through the chamber, which is then ducted to the work heading. Wave-fin or mesh-type mist eliminators keep sprayed water in the chamber. On the water side, about 0.018 L/s (2.85 gpm) per kW of chilled service water is required for operation. This water is dumped into the ditch and into the mine dewatering system. Or, in a closed loop system, the water is collected and pumped back to a district refrigeration machine. These chambers are efficient and inexpensive for small duties, especially those requiring auxiliary ventilation. Limited air scrubbing takes place. But they require the chilled water and mine dewatering infrastructure to support them.

Integrated Refrigeration Systems

Integrated or combination systems can cool both intake air and/or service water. The proportion depends on the season, with more bulk cooling in the summer and more water chilling in winter. If service water is recycled after being pumped out of the mine, it is often sent first to a precooling tower before being refrigerated. Recycling mine water may reduce treatment costs by reducing water that is rejected to surface drainage streams or ponds. Integrated systems come in a wide variety of cooling and energy recovery methods that utilize both surface and underground components. Most deep, hot mines employ some type of integrated refrigeration system.

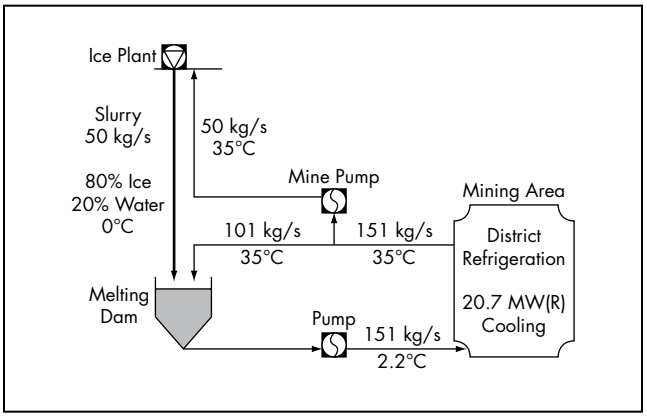


Figure 15.5-8 Basic ice cooling schematic with sample numbers

Ice Cooling

For deep, hot mines or mines already utilizing their full heat-removal capacity, ice cooling may be the answer. Ice can remove about 4.5 times the heat as an equivalent mass flow of chilled water. Ice is produced on the surface in chunk or slurry form and is sent underground to a melting dam, where the ice is melted by warm water returning from work areas. South African mines have been at the forefront of ice application. Problems with ice transport have been mostly solved. Currently, ice systems can compete on a cost basis with standard cooling techniques for deep mines. Figure 15.5-8 shows a basic ice cooling system.

Vests and Cabs

As mines become deeper, microclimate cooling should be considered for specific operations. Dry-ice (frozen carbon dioxide) or cold-pack cooling vests can be used by crews setting up ventilation in hot headings before permanent systems are in operation. These vests are heavy-duty "fire-fighter"-type jackets with internal pockets that are filled with ice slabs; the jacket is kept closed by a belt and/or straps, and the wearer is cooled as the ice melts. Vests are not popular with the work force—they are bulky and the cooling effect only lasts between 2 and 3 hours, then the vest must be recharged. Air-conditioned cabs on large load-haul-dump machines and trucks have proven themselves worthy of consideration. One of the biggest problems with diesel equipment is the heat produced. A diesel engine produces about 2.8 times the rated power in heat. Thus, a 100-kW loader operating at 80% utilization produces $100 \times 0.8 \times 2.8 = 224$ kW of heat. The design reject temperature can easily be exceeded in headings serviced by diesel vehicles. Air-conditioned cabs protect the operator from heat, noise, and dust. Although air-conditioned cabs are expensive, the cost is much less than trying to maintain the reject temperature in all headings serviced by diesel vehicles.

Energy Recovery

Most mines send water underground for various purposes, including wetting down muck piles and reducing dust. Hot mines also send down water solely for cooling. In deep, hot mines, water pressure must be periodically broken to prevent excessive hydrostatic heads in pipes. This is often accomplished by pressure-reducing valves or by using a cascading water-tank system. If service water flow is high enough,

thought should be given to a system that recovers the water's potential energy and converts it to useful work. A turbine—either a centrifugal pump designed to run in reverse or a Pelton wheel—can reduce the pressure and recover up to 80% of the energy required to pump the water back to the surface. Another benefit is preventing the conversion of water potential energy into heat; that is, 1°C (1.8°F) for every 427 m (1,400 ft) of drop. Other energy-recovery devices include the hydrochambers or hydrotransformers (popular in Europe). These are underground chambers equipped with two pumps in two separate circuits. The chambers are alternately filled with chilled water delivered from surface and hot water returned from the mine (McPherson 1993).

Controlled Recirculation

This technique is used with bulk air-cooling underground to reduce the fresh-air quantity needed to ventilate the mine. A portion of the mine exhaust air is cooled, scrubbed, and recirculated back to the workings. Advantages include higher air velocities in work areas; lower circuit fan pressure, because less air is drawn through the primary circuit; and a lower heat load due to autocompression. Autocompression can account for half the heat load at some mines. Controlled recirculation is usually applicable to deep, hot, mechanized mines, but because of the risk involved with recirculating air, it has not been used widely. Controlled recirculation systems must be monitored and controlled very carefully (Marks 1987). Fresh-air flow from the surface must be high enough to remove dust, blasting fumes, and diesel emissions. Controlled recirculation can be shut down during blasting.

Thermal Storage Systems

This innovative technique has been used to good effect in cold-climate mines. In winter, intake air is drawn through surface rubble piles or ice stopes where it is heated by the rock or by spraying water, which then freezes, releasing its latent heat into the air. In summer, the process reverses, with intake air being cooled by the rock rubble or melting ice (Stachulak 1989). This system is being successfully applied at Vale's Creighton mine in Ontario, Canada.

Maintenance

Mine cooling is not an install-and-forget system. Adequate maintenance must be planned from the start. Any cooling coil, direct expansion, or chilled water is subject to air side fouling. If at all possible, coils should be placed upstream of blasting. Coils receiving dusty and fume-laden air must be washed with a high-pressure spray (but not too high of a pressure or the fins will bend) at least every other day. Cooling coils should have a fin spacing no tighter than one fin per 4.2 mm (6 fins per inch) to limit fouling. If sump water is used for district refrigeration machine condenser cooling, the condenser tubes must be cleaned, either by an automatic reversing brush system with a flow diverter valve or by shutting the machine down periodically (depending on the water-fouling factor) and cleaning the tubes manually. Mines with extensive refrigeration systems (e.g., a large chiller plant or more than 10 spot coolers) should employ a mechanic specializing in refrigeration. This person can be assisted by apprentice mechanics. Another viable approach is to retain the services of factory mechanics or a local high-volume air conditioning and repair shop. Any such mechanics must be certified to handle refrigerants.

Selecting a Cooling System

Selecting a cooling system takes place after the mine cooling and ventilation requirements have been specified. The relationship between mine cooling and ventilation requires special attention, especially if mine exhaust air is expected to remove heat via underground cooling towers. Cost-benefit is the most common analytical technique. Each alternative requires certain expenditure and each provides a certain benefit. But just as important are hardware dependability, maintainability, dependency on outside factors, flexibility, safety, and technological level. An attempt must be made to quantify some of the more intangible aspects. Factors influencing selection of a mine cooling system include the following.

- **Seasonal surface ambient conditions.** As mentioned previously, warm climate mines tend to bulk-cool air on the surface close to the intake shaft.
- **Ore-body and mining methods.** The more massive the ore body, the more ideal bulk cooling becomes. Scattered stoping or continuously advancing headings lend themselves more to spot or district cooling. Large ore bodies amenable to bulk mining may use larger airflows and yet less air per ton of production.
- **Mining rate.** A fast mining rate prompts a high instantaneous heat load. But ironically, less total heat (and therefore lower costs) might be drawn from the wall rock due to stopes being quickly filled and sealed. One should leave as much heat in the wall rock as possible.
- **Size and condition of major airways.** In older mines beginning to experience heat problems, airflow increases might be prohibitively expensive due to small airways. Refrigeration might be needed earlier than at larger mines.
- **Heat sources.** The percentages of heat flow from various sources influences the selection of cooling methods. If autocompression is a major source, bulk cooling at the bottom of an intake shaft might be called for. If large quantities of heat come from diesel equipment, cabs might be appropriate. If more work takes place in the stopes or in the development and exploration headings than in the haulways, spot cooling might be best.
- **Costs.** Knowing the costs of labor, water, supplies, and power is critical for determining the optimum capital expenditure to control operating costs. For example, if power cost is high, a higher coefficient-of-performance system may be warranted.
- **Governmental regulation.** Some governments regulate heat stress. There are also laws prohibiting toxic refrigerants or combustible insulation underground. These factors can influence design of cooling systems.

VENTILATION AND REFRIGERATION PLANNING

Example 10 is presented to illustrate the various steps involved in developing a ventilation-refrigeration system for a hot deep mine. Decision parameters such as the critical depth, total heat load, and airflow requirements to create acceptable work conditions are highlighted. Once these parameters are made available, ventilation and climate simulators can be used to represent the mine openings and to predict the ventilation and air-cooling requirements.

CLIMSIM, a climatic simulator based on radial heat transfer from or to an airway, is used to predict the temperature and air-cooling profiles for a set of new workings, especially

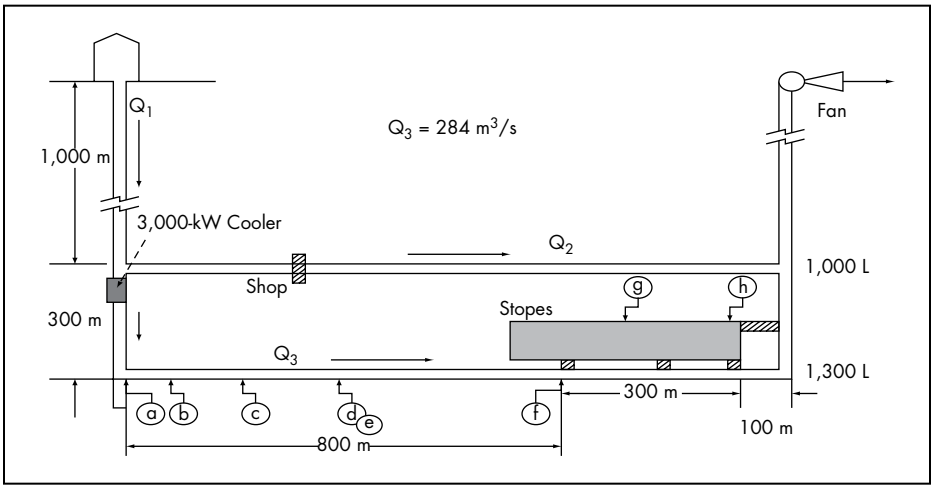


Figure 15.5-9 Mine ventilation schematic

for those subject to different types of heat loads (McPherson 1993). The profiles are then evaluated against preestablished standards, and the cooling requirements are determined. Wet bulb temperature and effective temperature are used as the target temperatures in the evaluation.

Example 10. A mining company extracts 1,200 t/d (1,320 stpd) of ore from an ore body located 1,000 m (3,280 ft) below the surface. Exploration work has shown that the ore body, a mineralized quartzite, extends to lower levels for at least 300 m (980 ft). This work has also shown a substantial increase in rock and groundwater temperatures with depth. Preliminary measurements have shown that, due to autocompression, the wet bulb temperature will increase by about 6°C (11°F) per 1,000 m (3,280 ft) of vertical depth and that the groundwater temperature will be 52°C (126°F) (VRT). The inflow of fissure water is estimated at 2.6 L/s (41 gpm). Currently, the company is preparing a prefeasibility study to deepen the mine to 1,300 m (4,265 ft) below the surface and to extract the remaining ore body at the rate of 1,500 t/d (1,650 stpd). The ore will be mined by conventional means of drilling and blasting. The required mining equipment includes seven jumbo drills (111 kW each), four production loaders (235 kW), and seven trucks (three 381 kW and four 135 kW), all diesel powered. In addition, the equipment includes a crusher-conveyor system (225 kW). The heat loads from the broken rock and groundwater are estimated at 352 and 262 kW, respectively. Figure 15.5-9 shows a schematic of the mine and the location of various heat sources. Table 15.5-2 summarizes the ventilation conditions that can be achieved with the current infrastructure, thermal properties of the rock, and heat loads for the resulting working areas. Based on this information, the problem is to determine the cooling requirement for this mine.

Solution. The problem was investigated using CLIMSIM for various heat load and ventilation conditions. During the analysis, two target temperatures were used: an average wet bulb temperature of less than 28°C (82°F) and a maximum wet bulb temperature of 32°C (90°F) at any working. When these targets were not met, the ventilation and cooling requirements were modified. Based on the psychrometric properties of the surface air, the depth of the openings, and an estimated VRT of 52°C (126°F), two scenarios were analyzed. The first simulated the mine before any air-cooling was installed and the second after an underground bulk air cooler was installed in the intake shaft.

Table 15.5-2 Airway, ventilation, and heat flow parameters for Example 10

Description	Parameter
Geometry of mine opening	
Length, m	1,200
Cross-sectional area, m^2	30.0
Perimeter, m	22.0
Airway friction factor, kg/m^3	0.04
Wetness factor	0.15
Ventilation at intake	
Airflow rate, m^3/s	284
Pressure, kPa	101.0*
Dry bulb temperature, °C	25*
Wet bulb temperature, °C	21*
Thermal properties (quartzite)	
Virgin rock temperature, °C	52.0
Geothermal step, m/°C	100
Thermal conductivity, W/m °C	5.50
Thermal diffusivity, $m^2\tau/s$	2.32
Heat sources†	
Ore bin, (x = 10 m), kW	352
Conveyor (x = 10 to 100 m), kW	100
Crusher (x = 100 m), kW	125
Drain water (x = 100 to 800 m), kW	262
Trucks (x = 100 to 800 m), kW	894
Loaders (x = 800 m), kW	940
Drills, controls (x = 1,000 m)	777
Booster fans (x = 1,100 m)	600

*Measured at intake shaft collar.
†x = Distance from intake.

For the first scenario, two sets of parameters were estimated: the required airflow rate to dilute the air contaminants and the heat generated at the workings. Based on the mining equipment, a flow requirement of 0.079-m^3/s (167 cfm) per kW of power, and a minimum quantity of 70-m^3/s (148,000 cfm) for two fixed facilities, the required airflow rate for the mine

Table 15.5-3 Airflow requirements for Example 10

Item	Utilization, %	Quantity, m^3/s
Loaders (4 × 235 kW)	80	59.4
30-t trucks (3 × 381 kW)	80	72.2
Other trucks (4 × 135 kW)	80	34.1
Jumbo drills (7 × 111 kW)	50	30.7
Crusher system (225 kW)	100	17.8
Underground shops	—	40.0
Other fixed facilities	—	30.0
Total quantity		284.2

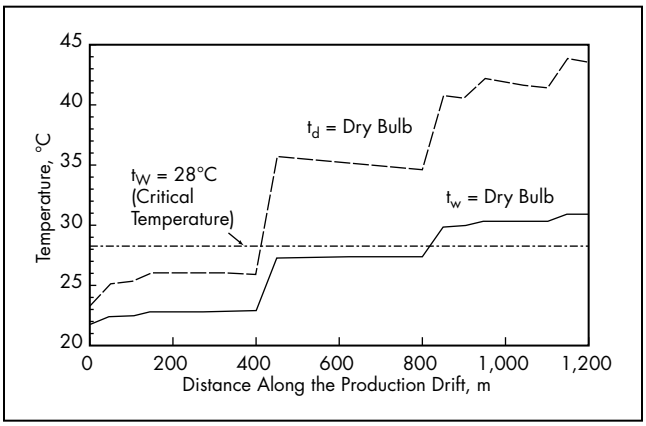


Figure 15.5-10 CLIMSIM output for Equation 15.5-10 (wet and dry bulb temperatures versus distance)

was estimated at 284 m^3/s (600,000 cfm) (Table 15.5-3). The heat loads and their distances from the intake shaft are those shown in Table 15.5-2 and Figure 15.5-9. The conveyor belt and the drainage water were represented as linear sources of heat. Once these values were input to the model and the simulator was run, the following results were generated: the wet bulb temperature at the bottom of the intake shaft was 26°C (79°F) and averaged 33.5°C (92.3°F) at the workings, and the dry bulb temperatures at active workings varied between 36° and 52°C (97° and 126°F). Both temperatures are above the acceptable target values, indicating that to maintain safe working conditions, the mine would require some form of cooling system.

For the second scenario, a 3,000-kW(R) refrigeration plant was installed in the intake shaft. Physically, this could be installed on the 1,000-m (3,280-ft) level, just upstream the intake shaft. Once this parameter was added to the model (as a negative heat source), the simulator generated the following results: the average wet bulb temperature would be equal to 26.5°C (79.7°F), with a maximum of 30°C (86°F) in the stopes; and the dry bulb temperature would average 33°C (91°F). These results show that the 3,000-kW plant would provide acceptable work conditions in the mine. Figure 15.5-10 shows two temperature profiles for the 1,200-m- (3,940-ft-) long production drift. The 3,000-kW refrigeration plant would reduce the wet bulb temperature at the bottom of the shaft to 22°C (72°F). This temperature would remain below 26°C (79°F) for 800 m (2,620 ft) where four 235-kW loaders are operated in four production stopes and two development headings. The heat and water vapor generated by these and other diesel units would increase the wet and dry bulb temperatures to 31° and 42°C (88° and 108°F), respectively.

This example shows the manner in which CLIMSIM can be used to formulate alternate solutions to heat flow problems and determine the capacity of the cooling system. It also shows the effect of autocompression in the ventilation air and allows the user to determine the critical depth, that is, the depth at which the mine would require some form of external cooling. In this example, such a depth is located at about 900 m (2,950 ft) below the surface.

CONCLUSIONS

In the United States, heat becomes an environmental concern in mines where workings are located at depths greater than 1,000 m (3,280 ft). At such depths, ventilation alone is not sufficient to control the heat load, and the mine must be supplemented by some form of air conditioning.

Since the introduction of the first heat stress index in 1923, more than 90 indices have been developed in various countries. In U.S. mines, heat stress is not regulated. Management must set and enforce the applicable limits. The most commonly used indices are

- Wet bulb temperature for continuous work: ≤27°C (80.6°F)
- Effective temperature for reduced workload: ≤28°C (82.4°F)
- Air velocity: ≥0.5 m/s (100 fpm)

In underground mines, heat is emitted from various sources, including autocompression, wall rock, mining equipment, groundwater, the oxidation of minerals, and metabolic heat. The estimation of the total heat load is crucial to the design of an air-cooling system. Once the individual heat loads are quantified, mine climate simulators such as CLIMSIM can be used to represent the mine openings and to predict the ventilation and air-cooling requirements.

Mine cooling is a heat-removal process in which a substance such as air, water, or ice is sent down the mine at a low enthalpy state and removed at a higher one. Depending on the amount of heat to be removed, several cooling methods can be used to create acceptable work conditions. These can range from upgrading the primary fans (increased ventilation) to cooling the mine air (refrigeration).

Ventilation is the least-expensive method to remove heat from mine workings. However, it has a limit—the critical depth. The critical depth is the depth at which the wet bulb temperature of the intake air exceeds the allowable wet bulb temperature.

Refrigeration plants used in hot mines range from spot coolers (for localized heat loads) to bulk air coolers (for widespread heat flow problems). Bulk air coolers can be installed on the surface, near the intake shaft, or underground, near the workings. Deciding at which stage a refrigeration plant is required is a difficult question to answer. Economics and site-specific conditions will dictate the optimal choice.

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CHAPTER 15.6

Radiation Control

Derek B. Apel and Zaher Hashisho

RADIOACTIVE DECAY OF URANIUM SERIES

Uranium is a naturally occurring radioactive element. It is found in rocks, soils, rivers, and oceans, and traces can be found in food and in the human body. Uranium is a common substance that is found throughout the earth's crust. It is more common than all of the precious metals and is more common than many nonprecious metals. Uranium, for example, is 40 times more common than tin. More than 200 minerals contain uranium, of which uraninite is the most common. Uranium concentrations vary from substance to substance and place to place.

Uranium is the heaviest of the naturally occurring elements and is made up of a number of different forms known as isotopes. Natural uranium is primarily made up of two isotopes, ^{235}U and ^{238}U. About 99.3% of natural uranium is ^{238}U and approximately 0.7% is ^{235}U, the "fuel" component of natural uranium. Because it is present virtually everywhere, it contributes to what is called the natural background radiation. Almost every backyard contains traces of naturally occurring uranium.

Radon is derived from the radioactive decay of uranium to radium then radon (see Figure 15.6-1). During the decay process, the unstable elements emit radiation. In essence, three different types of radiation are emitted: alpha, beta, and gamma.

Alpha radiation occurs when an alpha particle (the nucleus of a helium atom, which contains two protons and two neutrons) is emitted from the atom's nucleus. An alpha particle carries a relative electrical charge of +2, and when this particle is emitted from an element, the atomic number of this element is reduced by 2 and its atomic mass is reduced by 4. Usually, alpha radiation is not able to penetrate clothing and human skin, but if the alpha-emitting material is inhaled, it will cause damage to the lung tissue.

In simple terms, the beta radiation occurs when an electron is ejected from the nucleus of the decaying element. Therefore, this radiation has a relative electrical charge of –1, which causes the decaying element to gain a positive charge. This is shown on the periodic table as an increase by 1 atomic number of the new element. Beta radiation may travel a couple of meters in the air and can penetrate human skin. If beta-emitting materials are left on the skin for a longer period of time, they will cause skin injury.

Like visible light, radio waves, and X-rays, gamma radiation is a type of electromagnetic radiation. Gamma radiation can travel for several meters in the air and several centimeters in human tissue. Gamma rays can penetrate most materials. Dense materials (e.g., lead) are needed for shielding against gamma radiation.

Radon is a colorless, odorless, tasteless, radioactive gas that generally lacks activity toward other chemical agents. However, radon occasionally forms clathrate compounds with some organic compounds, and it may form ionic or covalent bonds with highly reactive elements such as oxygen or fluorine. Radon is the heaviest noble gas and exhibits the highest boiling point, melting point, critical temperature, and critical pressure of all noble gases. Radon is highly soluble in nonpolar solvents and moderately soluble in cold water. Radon's isotopes, all of which are radioactive, include mass numbers ranging from 200 to 226. ^{222}Rn formed in the ^{238}U decay chain is the most important isotope because of its relatively long half-life of 3.82 days. The short half-life beta- and gamma-emitting decay products of ^{222}Rn achieve equilibrium with the parent isotope within several hours. The density of radon is 9.73 g/L, and it has solubility in water of 22.2 mL/100 mL at 20°C.

Units of Radiation

The activity (rate of decay) of ^{222}Rn is expressed in units called curies. The curie is based on the rate of decay of 1 g of ^{226}Ra or 3.7×10^{10} disintegrations per second. The International System of Units (SI) measure of activity is becquerels per cubic meter (Bq/m^3). One becquerel equals one disintegration per second. The radiation unit most widely used by the uranium industry is the working level (WL), which is based on the measurement of energy released by ^{222}Rn and its progeny.

One WL means that the ^{222}Rn and its progeny released in 1 L of air will emanate 1.3×10^5 million electron volts (MeV)

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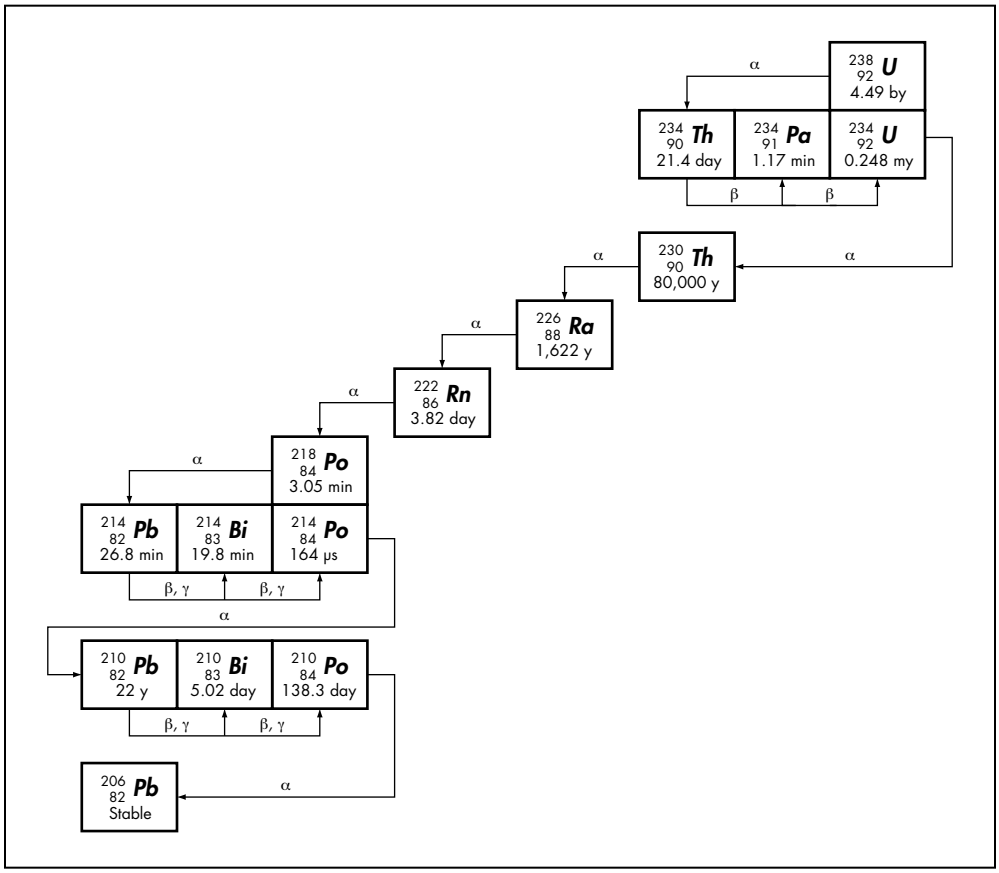


Figure 15.6-1 Uranium decay series (includes the half-life, atomic mass, atomic number, and emission type of each decay product)

of energy from alpha particles. The exposure to radiation at underground mines, which combines both exposure and time, is the working level month (WLM), which is defined as the exposure to one WL during 170 hours. For example, the WLM exposure for a miner who worked underground during a month for 120 hours and whose exposure was measured to be 0.9 WL has an exposure of 0.9 WL $\times$ 120 h/170 h = 0.64 WLM.

The joule-hours per cubic meter (Jh/m^3) is the SI unit of cumulative exposure to ^{222}Rn and its progeny. One WLM is equal to 3.5×10^{-3} Jh/m^3.

Occupational Exposure Limits

The exposure limit varies by regulating agency and type of worker. In the United States, the Mine Safety and Health Act regulates exposure to ^{222}Rn gas and ^{222}Rn progeny by underground miners. The act sets limits so that no employee can be exposed to air containing ^{222}Rn progeny in excess of 1.0 WL or 100 pCi/L (picocuries per liter) in active work areas. The act also limits annual exposure to ^{222}Rn progeny to less than four WLM per year (Field 1999).

EFFECT OF RADON PROGENY ON HUMAN HEALTH

The linkage between radon and the increased risk of cancer has been established throughout numerous studies (Frumpkin and Samet 2001). Even before the discovery of radon, many people blamed "mines air" for high mortality rates among underground miners. This was first described by Georgious Agricola (Agricola 1912) in his greatest work titled *De Re Metallica*, which was originally published in 1556. The study that most referenced in the literature linking radon exposure to lung cancer are findings summarized in a report titled *Health Effects of Exposure to Radon*, prepared by the National Research Council's Committee on the Biological Effects of Ionizing Radiation (BEIR) Report VI (NRC 1998). This report was based on data pooled from underground mines. The data show that of 60,000 miners who worked between 1941 and 1990 in different mines located in different countries, more than 2,600 developed lung cancer against only 750 cases based on background data. A study conducted by Kreuzer et al. (1999) reported that among 64,000 uranium miners in East Germany who worked a minimum of 180 days at these mines between 1946 and 1989, there were 1,436 deaths due to lung cancer.

The findings from these studies overwhelmingly show that radon progeny exposure causes lung cancer. Most of the radiation-hazard results from the inhalation of the short-lived radon progeny. Radon itself is of minor concern, because most of the inhaled radon is exhaled. Therefore, this is not radon itself but radon's progeny whose decay in lungs causes damage to the cells and DNA mutation that can lead to lung cancer. This was established by research conducted by Hursh et al. (1965). Additional work found that radon is removed from the body with a primary half-life of between 30 and 70 minutes (Crawford-Brown 1989).

EMANATION OF RADON AT UNDERGROUND MINES

The emanation of radon from uranium ore is the effect of radium decay. During this process, an alpha radiation is emitted from an atom of radium and its atomic mass is being reduced from 226 to 222. At the same time, the atomic mass drops from 88 to 86, and radium changes into radon. Radon can emanate to the mine atmosphere from the ore, rock walls, and mine water. The rate of radon emanation can be described as the number of radon atoms exhaled from a unit area of rock in a unit of time, and it is defined in units of picocuries per square centimeter per second (pCi/cm^2/s).

The emanation of radon from rock surfaces in underground mines (J) can be calculated (Bates and Edwards 1980) as follows:

$$J = C_\infty \sqrt{\lambda D \phi} \text{ pCi/cm}^2\text{/s} \quad (15.6\text{-}1)$$

where

C_∞ = radon concentration at infinite distance into rock, pCi/m^3
λ = radon decay constant (0.00755 disintegrations per atom or 2.1 $\times$ 10^{-6} Bq)
D = diffusion coefficient, m^2/s
ϕ = rock porosity (0.50 = 50% porosity)

If the emanation of radon from a unit volume of rock (B) is known, Equation 15.6-1 can be modified to the following form:

$$J = B\sqrt{\frac{D}{\lambda\phi}} \quad (15.6\text{-}2)$$

where B is sometimes called the emanating power, pCi/cm^3/s.

The random movement of the radon gas atoms mixed in the air results in a net migration of the radon gas toward the direction of its decreasing concentration in the air. This phenomenon is called molecular or atom diffusion.

Table 15.6-1 shows examples of various media and their coefficients of diffusion. The table illustrates that the coefficient of diffusion decreases with the density of the material.

Table 15.6-1 Coefficient of diffusion for radon in various materials

Material	Coefficient of Diffusion D, m^2/s
Dense rock	0.05 × 10^{-6}
Rock with porosity of 6.2%	0.2 × 10^{-6}
Rock with porosity of 12.5%	0.5 × 10^{-6}
Rock with porosity of 25%	3.0 × 10^{-6}
Unconsolidated soil material	3.0 × 10^{-6}
Compacted silty sands	3.2 × 10^{-6}
Compacted clayey sands	2.5 × 10^{-6}
Compacted inorganic clays	2.7 × 10^{-6}
Silty sandy clay	2.5 × 10^{-7}
Uranium mill tailings	6.0 × 10^{-8}
Concrete	(1.7–3.0) × 10^{-9}
Air	(10–12) × 10^{-6}
Water	1.13 × 10^{-8}

CONTROLLING RADON PROGENIES AT UNDERGROUND URANIUM MINES

The control of radon progenies at underground uranium mines should commence with the process of selecting the mining method, controlling water inflows, and designing a flexible ventilation system. In addition, each mine has to establish safety operating procedures specific for each operating mine. The latter is extremely important, as even the best ventilation system can malfunction because of a power outage, human error, or other unforeseen circumstances. When designing ventilation systems for underground uranium mines, deposits can be divided into two groups: low-grade deposits, usually ranging from 0.1% to 2% U_3O_8 (triuranium octoxide), and high-grade deposits, where the grade can exceed 20% U_3O_8.

Ventilation Systems

In the case of high-grade deposits, the radon emanation rate from the ore would make it practically impossible to dilute the radon daughters using flush-through ventilation, and in these cases the ore is mined using remote mining methods (e.g., raise boring or mining using water jets) while a minimum pillar thickness of 5 m of an inert rock is kept between the ore and the miners (Figure 15.6-2; Apel 2000).

Production areas where the emanation of radon is high can be ventilated using both flush-through ventilation and a secondary ventilation system that is used to capture radon at its source and then exhaust it to the restricted entry return air drifts. The new development headings shall be ventilated using a combination of positive and negative pressure ducting. The positive pressure ducting is used to flush the air at the heading, whereas the negative pressure rigid ducts are used to transfer the radon-contaminated air to the exhaust air drifts or shafts (see Figure 15.6-3).

All inactive areas must be sealed off by ventilation bulkheads, and the bulkheads should be periodically checked for radon leaks. In cases where a complete seal cannot be established by the ventilation bulkheads, it should be connected to the low-pressure negative ventilation system.

Experiments conducted in mines in the United States showed a reduction of radon concentration by using positive blowing versus the exhaust systems of up to 20% (Schroeder and Evans 1969; Franklin 1981). However, these experiments were conducted in relativly shallow mines, and it may not be feasible to achieve any decrease of radon production in deep mines where rock-in-situ stresses are much higher.

Water Control

Radon is soluble in water and is also readily released from it. Therefore, by limiting water inflows into mine openings, radon emanation into the mine air will be reduced. Controlling the water inflows can be achieved by using grout curtains ahead of the development or using ground freezing (Apel and Szmigiel 2006). The solubility of radon decreases with an increase in water temperature (e.g., 510, 230, and 169 cm^3/kg at 0°C, 20°C, and 30°C, respectively) (NRC 1998). Because not all the radon leaves water immediately after entering the underground openings, all the water sumps must be ventilated using negative pressure ventilation. The removal of radon from water can be done through aeration. Aeration simply forces radon from the water to the air and can be very effective. According to Dixon et al. (1991) and Kinner et al. (1993), bubble-plate aeration and diffused bubble aeration as point-of-entry units are capable of achieving removal efficiencies in excess of 99% at loading

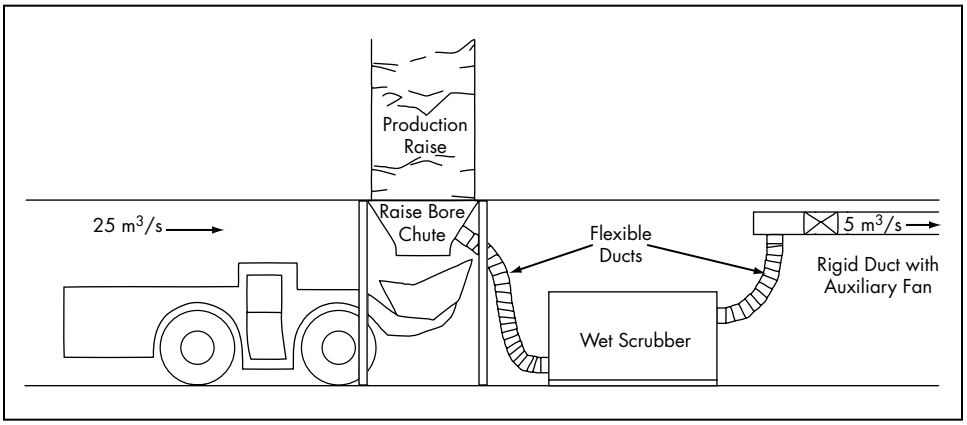


Figure 15.6-2 Ventilation setup for the raise-boring mining method

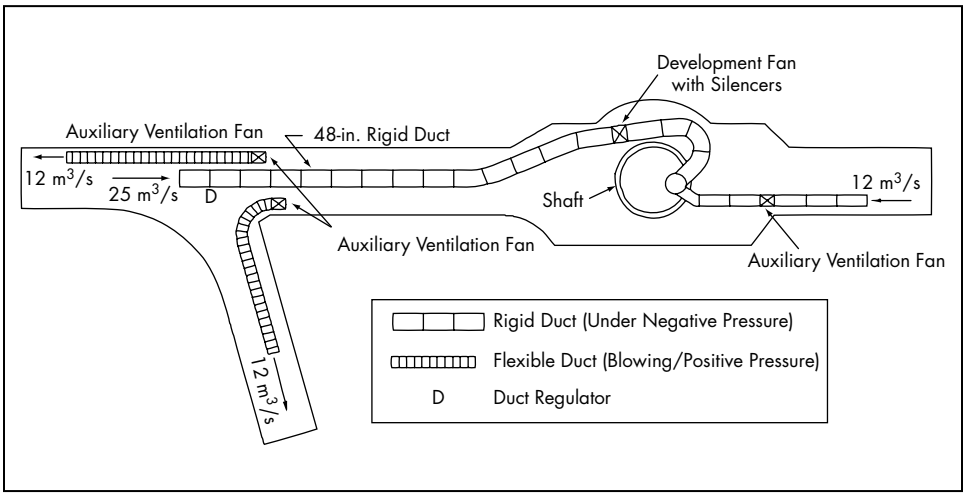


Figure 15.6-3 Ventilation setup during the development stage at a uranium mine

rates of 185 Bq/L or more, whereas the simple spray-jet aeration technique only removes between 50% and 75% of the radon.

Dust Control

Dust control is extremely important at uranium mines because radon progenies can get electrostatically attached to tiny particulates, such as dust, in air. If these particulates get inhaled, they will start decaying and releasing damaging alpha and beta particles. How much radon progenies get attached to the dust depends on factors such as the size and concentration of the airborne particles. The size and density of a particle determine whether or not the particle will reach the lung tissue. Particles with diameters of 10 μm are generally filtered by the nose and saliva during breathing and, therefore, should not reach the lungs. However, particles that range in diameters of from 2 to 10 μm have a good chance of being deposited in the bronchial region (Hinds 1999).

Obviously, selection of the mining method will impact how much dust will be created, but facts such as allowing longer periods after the blasts for dust to settle or spraying the muck with water after the blast will also limit the amount of air-bound dust created. Using personal protective equipment, such as an air respirator, is probably the best way to limit exposure to dust at uranium mines.

RADIATION MONITORING

Exposure of workers to radiation at underground mines can be controlled by adjusting the ventilation. The level of radiation at underground mines can change with further development of underground workings or changes in the uranium-ore production rates. Therefore, monitoring of radiation is very important in reducing the exposure of workers. Monitoring techniques at underground mines can be classified based on their ability to measure radon or its progeny. They can also be classified based on the measurement time frame into grab sampling, integrated sampling, or continuous monitoring and sampling. Radiation monitoring is not only limited to monitoring radon and its progeny, but it also includes monitoring of gamma-ray radiation.

Grab Sampling

Grab sampling consists of collecting samples over a short period of time, ranging from minutes to hours.

Radon Sampling with Scintillation Cells

A scintillation cell, such as the Lucas chamber, consists of a flask coated with silver-activated zinc sulfide on the inner surfaces, except for a clear window on one side to allow photons produced from alpha disintegrations to be counted. Sample air

is admitted to the evacuated cell, which is typically 100 mL to 2 L in size (Sengupta 1989). Usually, the air sample is left to age for 4 hours to allow for equilibrium between radon and its progeny. Radon concentration is proportional to the photon count obtained with a photomultiplier tube. Correction factors are used to account for the decay during the time between sampling and counting and for decay during counting (McDermott and Ness 2004). Scintillation cells offer a reliable technique for radon sampling. For short counting periods, a sensitivity of 5 to 10 pCi/L is attained, whereas the limit sensitivity can reach 0.1 pCi/L for long counting periods (≥100 minutes) (Sengupta 1989).

Radon Progeny Sampling

Kusnetz method. The Kusnetz (1956) method involves drawing a known volume of air (100 to 250 L) through an air filter using a vacuum pump during a short time frame, typically 5 minutes (McDermott and Ness 2004). Airborne radon progeny attached to dust particles are collected on the surface of the filter. The decay of these progeny results in emission of alpha particles. The alpha particles are counted over a typical duration of 5 minutes after a delay period of 40 to 90 minutes. The delay time is the time in minutes from the end of sampling to the middle of counting. For a given delay time, t_k, an appropriate Kusnetz conversion factor, K, can be calculated using the following equations (CNSC 2003):

$$K = 230 - 2t_k \quad \text{for } 40 \le t_k \le 70 \tag{15.6-3}$$

$$K = 195 - 1.5t_k \text{ for } 70 \le t_k \le 90 \tag{15.6-4}$$

Knowing the volume of sampled air and the alpha particles count, radon progeny concentration can be determined as WL. The original method assumes perfect equilibrium, which results in greater than a ±25% error (Vutukuri and Lama 1986). Modifications of the Kusnetz method compensate for the assumption of equilibrium by taking alpha-activity count at several occasions during the growth of the progeny activity and making corrections for nonequilibrium (Vutukuri and Lama 1986).

The original Kusnetz method measures alpha particle activity with a rate meter, resulting in a practical lower limit of measurement of 0.3 WL. In the modified Kusnetz method, a scaler is used instead of the rate meter, resulting in a 10-times improvement in the sensitivity (Sengupta 1989).

The concentration in working levels, WL, can be obtained with the following equation(DCEP 2008):

$$\text{WL} = \frac{C_S - C_B}{E \times K \times V \times t_c} \tag{15.6-5}$$

where

WL = concentration in working levels
C_S and C_B = gross and background alpha counts, respectively, recorded over the interval t_c
E = efficiency of equipment detection, fraction
K = factor to convert from counts per minute to working level
V = volume of air sampled, L
t_c = counting time, minutes

For the Kusnetz method, K is the Kusnetz conversion factor determined from Equations 15.6-3 and 15.6-4.

The right side of Equation 15.6-5 can be multiplied by a correction factor to compensate for the absorption of radon progeny on the sampling filter. This factor will probably be less than 0.02 for a cellulose ester filter, but it may exceed 0.1 for a glass-fiber filter (CNSC 2003).

A disadvantage of the Kusnetz method is that it is not suitable for atmospheres containing both ^{222}Rn and ^{220}Rn progeny, because the delay period allows for the ingrowth of ^{220}Rn progeny, which will contribute to the gross alpha count (DCEP 2008). In Canada, the Canadian Nuclear Safety Commission inspectors in underground mines rely on the modified Kusnetz method for sampling, counting, and computational procedures (CNSC 2003).

Modified Tsivoglou method. The modified Tsivoglou method is very similar to the Kusnetz method, except that the filter is counted three times after collection to measure the concentration of the individual radon daughters. Filter counting occurs at the following intervals after completion of sampling: 2 to 5, 6 to 20, and 21 to 30 minutes (McDermott and Ness 2004).

Based on a half-life for ^{218}Po of 3.04 minutes, the concentration of each of the radon decay products, that is, ^{218}Po, ^{214}Pb, and ^{214}Po, can be determined using the following calculations:

$$C_1 = \frac{1}{FE}\begin{pmatrix} 0.16921\ G1 - 0.08213\ G2 \\ +0.07765\ G3 - 0.5608\ R \end{pmatrix} \tag{15.6-6}$$

$$C_2 = \frac{1}{FE}\begin{pmatrix} 0.001108\ G1 - 0.02052\ G2 \\ +0.04904\ G3 - 0.1577\ R \end{pmatrix} \tag{15.6-7}$$

$$C_3 = \frac{1}{FE}\begin{pmatrix} -0.02236\ G1 + 0.03310\ G2 \\ -0.03765\ G3 - 0.05720\ R \end{pmatrix} \tag{15.6-8}$$

where

C_1, C_2, and C_3 = concentrations of ^{218}Po, ^{214}Pb, and ^{214}Po, respectively, pCi/L
F = sampling airflow rate, L/min
E = counting efficiency, counts per minute/disintegrations per minute
$G1$, $G2$, and $G3$ = gross alpha counts at the 1st, 2nd, and 3rd intervals, respectively
R = background counting rate (counts per minute)

The working level can be expressed using the following equation:

$$\text{WL} = 10^{-3}\ (1.028C_1 + 5.07C_2 + 3.728C_3) \tag{15.6-9}$$

Using an alpha-scintillation scaler instead of a rate meter results in a 10-times improvement in the sensitivity (Sengupta 1989).

Rolle method. The Rolle method is used for the determination of ^{222}Rn progeny concentration (DCEP 2008). This method is similar to the original Kusnetz method, except that the timing for filter counting after sample collection is different. In the Rolle method, the delay period is 5 to 10 minutes, allowing for faster measurement compared to the Kusnetz method. In addition, the short delay time minimizes the error in the radon progeny ratio allowing better measurement of the concentration of ^{222}Rn progeny (Sengupta 1989). The working level of ^{222}Rn progeny can be calculated using Equation

Table 15.6-2 Conversion factor for Rolle method

Sampling Time, minutes	Delay Time, minutes	Counting Time, minutes	Conversion Factor	Error, %
4	6.46	5	213	12
5	6.06	5	213	11
10	4.35	5	213	11
15	1.36	10	212	10
20	2.12	5	210	10

15.6-5 (i.e., the same equation used for the Kusnetz method) but with a conversion factor selected from Table 15.6-2 (DCEP 2008).

Rock method. The Rock method is mainly used for the determination of ^{220}Rn progeny concentration because of the extended delay time allowing for the ingrowth of ^{220}Rn progeny. The method is similar to the Kusnetz method, but the delay period is typically between 5 and 17 hours. An advantage of this method is that the long delay period allows sampling of remote locations and sending the samples for laboratory analysis. The concentration of ^{220}Rn progeny can be calculated using Equation 15.6-5, but with the conversion factor selected from Table 15.6-3 (DCEP 2008).

Cote method. The Cote method allows for the determination of the concentration of ^{220}Rn and ^{222}Rn progeny based on the selection of the delay period for the alpha count. The method consists of 10-minute dust sampling followed by several 15-minute alpha counts. The first count, conducted 1.2 minutes after the completion of sampling, is used to obtain the concentration of ^{222}Rn progeny, because the contribution of ^{220}Rn progeny is negligible at this time. A conversion factor of 218 is used in Equation 15.6-5 to obtain the WL concentration of ^{222}Rn progeny. The second count, conducted 155 minutes after sampling, provides the combined contribution of ^{222}Rn and ^{220}Rn progeny using a conversion factor of 14.1 in Equation 15.6-5. The final count, conducted 225 minutes after sampling, is used to obtain the ^{220}Rn progeny contribution using a conversion factor of 14.2 in Equation 15.6-5. A disadvantage of the Cote method is the need for a counting instrument to be placed on or very close to the sampling location (DCEP 2008).

Table 15.6-3 Conversion factor Rock method

Time After Sampling, hours	Conversion Factor
5	13.2
7	11.3
9	9.8
11	8.6
13	7.5
15	6.7
16	6.3

Integrated Sampling

Integrated sampling provides a single concentration value averaged over the duration of the sampling time, which can ranges from days to weeks, months, or up to a year.

Radon Sampling

Activated carbon canisters. Adsorption on activated carbon allows integrative sampling of radon gas over a period of up to 7 days. Because the adsorbed radon undergoes radioactive decays during the sampling period, this method does not uniformly integrate radon concentrations. Sampling can be active using a pump, or passive based on diffusion (Katz 1977; McDermott and Ness 2004). After sampling, the activated carbon is analyzed using a sodium iodide gamma-scintillation detector to count the gamma rays between 0.25 and 0.61 MeV emitted by the radon decay products on the activated carbon (Katz 1977; McDermott and Ness 2004). Alternatively, radon can be desorbed from the activated carbon and transferred to a counting cell for alpha counting (Katz 1977). For gamma counting of a diffusion-based activated carbon monitor, the lower limit of detection ranges from 0.1 to 0.2 pCi/L of radon for a sampling time of 4 days (George 1990). Radon sampling with activated carbon canisters is a simple and low-cost method. However, the results can be sensitive to humidity and temperature. Desiccant can be used to avoid interference from adsorption of water vapor.

Alpha track detectors. Alpha track detectors provide an integrated concentration of radon over the exposure period, which ranges from few months to a year. The detector consists of a plastic film, such as cellulose nitrate (LR-115, manufactured by Kodak-Pathé) or allyl diglycol carbonate (CR-39) films (Alter and Fleischer 1981). The films are enclosed in a container that allows only radon to diffuse through and prevent the penetration of radon progeny. Alpha particles emitted by the radon and its progeny in the container strike the detector and produce submicroscopic damage tracks. Following exposure, the detectors are treated with a caustic solution to visualize the damage tracks for manual counting with a microscope or automated counting with an optical counting system. Radon concentration is proportional to the net areal density of tracks per unit time. The advantages of alpha track detectors are their simplicity, cost-effectiveness, and the lack of need for external power sources. Their main disadvantage is the dependence of results on the type of film used. However, the variability in the results can be decreased by counting more tracks over a larger area of the detector (McDermott and Ness 2004).

Radon Progeny Sampling

Thermoluminescent dosimeter (TLD). TLD-based systems integrate radon progeny concentration over sampling periods ranging from few days to a few weeks. The system contains a detector assembly through which sample air is drawn. The detector assembly includes a filter and two TLDs. One TLD is used for measuring alpha particles emitted from radon progeny collected on the filter. The other TLD is shielded from the alpha particles emitted from the filter and is used for measuring background gamma radiation (Vutukuri and Lama 1986; McDermott and Ness 2004). Alpha particles emitted from the decay of the radon progeny are registered on a luminescent material such as calcium fluoride dysprosium or

lithium fluoride. Following sampling, the detector assembly is sent for analysis using a TLD reader. Light emitted from the TLD as a result of heating in the TLD reader is detected with a photomultiplier tube and is proportional to the radon progeny collected on the filter paper during sampling (George 1990). A TLD is accurate and suitable for long sampling periods; however, they suffer from their relatively high cost and clogging of filters (Vutukuri and Lama 1986). Instead of TLDs, optically stimulated luminescence (OSL) dosimeters are used for measuring exposure of mine workers to radiation. OSL dosimeters use luminescent material such as Al_2O_3:C that responds to gamma radiation (Figure 15.6-4). The OSL dosimeter is typically exposed for several months before it is analyzed. Unlike the case of TLDs, the readout method for OSL dosimeters is all optical, eliminating the need for heating of the samples. OSL dosimeters also provide a high-sensitivity, fast-readout process, as well as the possibility of multiple readings (McKeever 2001).

Electret detector and alpha track detector systems. A filter-based system similar to the one used with a TLD system can be used with other types of detectors, such as an alpha track detector or an electret detector. An electret is an electrostatically charged disk positioned in an ion chamber. Alpha particles generated from the decay of radon progeny collected on the filter result in negatively charged ions that are collected on a positively charged electret, resulting in a reduction in its surface voltage. Radon progeny concentration is proportional to the voltage reduction (McDermott and Ness 2004). Short-term electrets integrate measurements up to 7 days. Long-term electrets are less sensitive but integrate measurements for 1 to 12 months. The lower limit of detection for a short-term electret is 0.2 pCi/L of radon for a 3-day exposure (George 1990).

Continuous Monitors

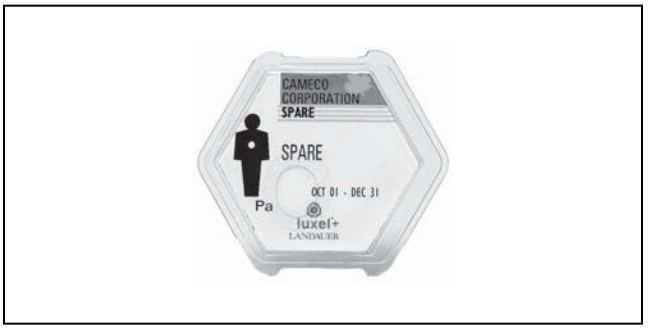
In continuous monitoring, sampling and analysis occur simultaneously and continuously.

Scintillation Cell

A variation of the scintillation cell used for grab sampling of radon can be used for continuous radon monitoring. One configuration consists of a flow-through scintillation cell in which air is continuously pumped through for a few minutes. Alpha particles from the decay of radon gas are counted as the gas is swept through. Another configuration consists of a periodic-fill cell, into which air is pumped once during each preselected time interval, and the alpha particles are counted in each interval. Passive scintillation cells in which radon diffuses through a filter area allow passive continuous monitoring of radon after a short diffusion time lag with a lower limit of detection of about 1 pCi/L for a 1-hour counting (George 1990; McDermott and Ness 2004).

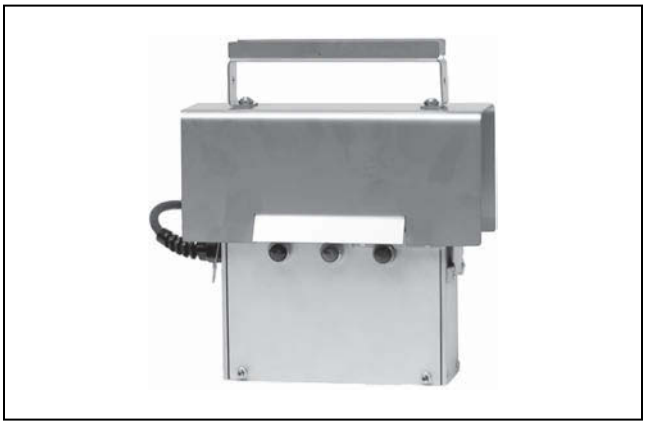
Diffusion-Based Ionization Chamber

A diffusion-based ionization chamber allows radon gas to diffuse into the chamber while radon progeny are removed electrostatically. As is the case with a passive scintillation chamber, there is a short time lag with this method. Inside the ionization chamber, the emitted alpha particles from radon decay generate bursts of ions that are recorded by an electrometer. The lower limit of detection is about 0.5 pCi/L for a 1-hour counting (George 1990).



Courtesy of K. Messer.

Figure 15.6-4 An optically stimulated luminescence dosimeter



Courtesy of K. Messer.

Figure 15.6-5 A Continuous working-level monitor intended for operation as a station monitor

Continuous Radon Progeny Monitors

Continuous radon progeny monitors use filters to collect radon progeny and an alpha detector to measure alpha particles generated from the decay of the radon progeny. Hence, they are similar to integrated radon progeny samplers, except that automated sampling of air is done on a continuous basis and alpha counting is integrated over short-time intervals ranging form several minutes to an hour (George 1990). Gross alpha counting is also used to provide WL radon progeny. One commonly used system, alphanuclear prism, has four alpha-energy counting windows that indicate WL concentrations of <0.1 WL, 0.1 to 0.25 WL, 0.25 to 0.5 WL, 0.5 to 1 WL, and >1 WL (Figure 15.6-5).

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CHAPTER 15.7

Noise Hazards and Controls

Gary E. Walter

INTRODUCTION

Noise is an inherent health hazard in the mining industry. The mining process is highly mechanized from the earliest ore removal to final processing, and heavy equipment is essential at virtually every stage of operation. Exposure to noise is a concern for workers who drive mechanized equipment as well as those who operate or work near stationary equipment such as haulage belts or crushing equipment. Exposure to noise levels above regulatory or recommended limits can result in hearing loss that adversely affects the daily lives of workers and negatively impacts society as workers so affected access health care services.

Worker exposure to noise is regulated by many government agencies. In the United States, these agencies include the Mine Safety and Health Administration (MSHA) and the Occupational Safety and Health Administration (OSHA). In the United Kingdom, the regulatory agency for worker health and safety is the Health and Safety Executive (HSE). The European Union (EU) has also enacted a directive on noise that must be implemented by member countries. Another significant contributor to the assessment of risk and establishment of industrial hygiene standards in the United States is the American Conference of Governmental Industrial Hygienists (ACGIH), which is not a government agency but rather a professional society that publishes recommendations for exposures to various hazardous agents to workers.

It is important to emphasize that most hearing loss is preventable. Prevention can be achieved by eliminating noise sources, substituting quieter equipment, installing appropriate engineering controls, implementing administrative controls, using personal protective equipment, and adopting effective hearing-conservation programs. Regulations in many parts of the world require noise-control and hearing-conservation programs. Many of these regulations were recently updated to reflect new information regarding the levels of noise that can cause hearing loss. This has resulted in reduced permissible exposure levels and enhanced requirements to conduct risk assessments and implement control measures.

DEFINITIONS

Action level (AL)—The level, established by regulation, at which action must be taken to protect the hearing of workers exposed to noise. The action required is country-specific.

- **Action level** (MSHA)—The 8-hour time-weighted average sound level (TWA_8) sound level of 85 dBA, or equivalently a dose of 50%, integrating all sound levels from 80 to ≥130 dBA (A-weighted). (Note: An A-weighted setting measures sound similarly to the way the human ear responds to sound. A C-weighted setting measures sound in a flat or linear response.)
- **Lower exposure action value** (HSE)—The lower of the two levels of daily or weekly personal noise exposure or of peak sound pressure that, if reached or exceeded, requires specified action to be taken to reduce risk; that is, a daily or weekly personal noise exposure of 80 dBA.
- **Upper exposure action value** (HSE)—The higher of the two levels of daily or weekly personal noise exposure or of peak sound pressure that, if reached or exceeded, requires specified action to be taken to reduce risk; that is, a daily or weekly personal noise exposure of 85 dBA.

Criterion level—The sound level that, if constantly applied for 8 hours, results in a dose of 100% of that permitted by regulation.

dB—The unit of measure for sound pressure, defined in two ways:

1. For sound-pressure levels measured by sound level meter (SLM), the decibel is 20 times the common logarithm of the ratio of the measured sound pressure to the standard reference sound pressure of 20 μPa, which is the threshold of normal hearing sensitivity at 1,000 Hz.
2. For hearing-threshold levels measured by audiometer, the decibel is the difference between audiometric zero (reference pressure equal to 0 hearing-threshold level) and the threshold of hearing of the individual being tested at each test frequency.

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dBA—The level of noise, measured by SLM or dosimeter, A-weighted to represent the way the ear processes sound.

Dosimeter—A personal monitoring device that integrates noise exposure. Dosimeters must be set to match the criterion, threshold, and exchange rate established by the relevant regulation.

Exchange rate—The amount of increase in sound level, in decibels, that would require halving of the allowable exposure time to maintain the same noise dose. The MSHA and OSHA exchange rates are both 5 dB (which the American Industrial Hygiene Association recommends lowering to 3 dB). The EU and HSE exchange rates are both 3 dB. The ACGIH threshold-limit value (TLV) exchange rate is also 3 dB.

Noise dose—The integrated noise exposure that reaches a worker's ear, measured by personal noise dosimeter, including all sound that is >80 dBA. The permissible exposure limit or the recommended criteria is exceeded when this value is >100%.

Permissible exposure limit (PEL)—The noise-exposure limit that must not be exceeded, established by government regulation. The MSHA PEL is 90 dBA.

Sound level meter—An instrument designed to react to noise much as does the human ear in order to measure and compare against appropriate noise standards.

Threshold—The cutoff level of sound used to measure and calculate noise dose. Sound below this level is considered zero in noise–dose calculations.

Threshold-limit value (ceiling), or TLV—An exposure that should not be exceeded, even instantaneously.

Threshold-limit value (time-weighted average) for physical agents—The time-weighted average exposure for an 8-hour workday and a 40-hour workweek.

NOISE REGULATIONS AND EXPOSURE LIMITS

Noise-exposure regulations vary by country, resulting in the existence of differing exposure limits, measurement procedures, compliance activities, and required controls. In the United States, workplace noise-exposure standards are governed by MSHA and OSHA. Whether a mine operation is regulated by these two entities determines which regulations the operator must meet. Some operations may be covered by both agencies for different parts of the operation. In general, most mine operations are regulated by MSHA. Some states have additional noise regulations. Environmental or community noise requirements established by the U.S. Environmental Protection Agency or by state and local governments may also affect operations near residential or public areas. The MSHA (2006) regulation for noise in mines is in the Code of Federal Regulations (CFR) as Title 30, Part 62, *Occupational Noise Exposure*. For OSHA-covered operations, the OSHA (2004a, 2004b) regulations are in the CFR as Title 29, Part 1910.95, *Occupational Noise Exposure* and *Methods for Estimating the Adequacy of Hearing Protector Attenuation*.

The European Union adopted Directive 2003/10/EC, *Minimum Health and Safety Requirements Regarding the Exposure of Workers to the Risks Arising from Physical Agents (Noise),* for application throughout Europe. Each member country must adopt this directive at a minimum but may adopt more stringent regulations. An example of a country adopting more stringent regulations is the United Kingdom, where the HSE issued *Control of Noise at Work Regulations 2005*, which applies to all U.K. industry sectors, including mining.

The ACGIH has for longer than 50 years adopted TLVs for chemical substances and physical agents including noise (ACGIH 2006). ACGIH members are industrial hygienists or other occupational health and safety professionals dedicated to promoting workplace health and safety. The ACGIH publishes guidelines regarding safe levels of exposure to chemical and physical agents found in the workplace. In many parts of the world where no regulations exist, ACGIH TLVs have become the standard reference for the protection of worker health.

Although present-day international standards for measuring noise and hearing are generally in agreement, there is some difference of opinion regarding what level of noise adversely affects human health. As a result, there are different acceptable levels of allowable exposure for workers and some differences in the actual application for regulatory standards. The following are MSHA, EU, HSE, and ACGIH standards for TWA_8 exposure (MSHA 2006; EU 2003; HSE 2005; ACGIH 2009).

- MSHA standards
  - AL: 85 dBA (50% dose); all sound >80 dBA is included
  - PEL: 90 dBA (100% dose); only sound >90 dBA is included
  - Threshold: AL is 80 dBA; PEL is 90 dBA
  - Exchange (doubling) rate: 5 dBA
- EU and HSE standards
  - Lower AL: 80 dBA
  - Upper AL: 85 dBA
  - PEL: 87 dBA
  - Threshold: 80 dBA
  - Exchange (doubling) rate: 3 dBA
- ACGIH recommendations
  - TLV: 85 dBA
  - Threshold: 80 dBA
  - Exchange (doubling) rate: 3 dBA

Note: A modern personal noise dosimeter can record and calculate both AL and PEL levels.

Using the EU (HSE) upper AL limit of 85 dBA and exchange (doubling) rate of 3 dB as the reference for worker exposure, the amount of time permitted at various noise levels can be calculated by adding 3 dBA to the 85 dBA limit and cutting the time in half from the permitted 8 hours. Table 15.7-1 shows the amount of time at various noise levels that is equivalent to 85 dBA for the time shown.

Table 15.7-2 shows the same information when the MSHA limits are used as the reference for worker exposure. The times are different partly because ACGIH uses the more conservative 3-dB exchange (doubling) rate whereas the MSHA standard uses 5 dB. The data in these tables can be used to calculate worker noise exposure using the formula given in Equation 15.7-1.

INSTRUMENTATION AND SURVEY TECHNIQUES

Noise levels must be measured before any control measures are undertaken. It is important to document worker exposures in order to meet regulatory requirements and make informed decisions about implementing hearing-conservation programs for exposed workers. These measurements are the basis for most decisions regarding the application of engineering controls and assist in determining the most effective control measures. Measurement of worker exposures as part of a risk-assessment process can help prioritize the application of engineering controls so that the greatest number of affected employees can be protected through implementation

Table 15.7-1 Noise-level equivalencies at various sound levels (EU reference standard)*

Noise Level, dBA	Maximum Permissible Time of Noise Exposure, h
85	8
88	4
91	2
94	1
97	0.5
100	0.25

Source: EU 2003.
*Maximum noise pressure level must not exceed 115 dBA. Peak level of sound pressure (in case of impulsive sound as well) must not exceed 140 dBC.

Table 15.7-2 Noise-level equivalencies at various sound levels (MSHA reference standard)*

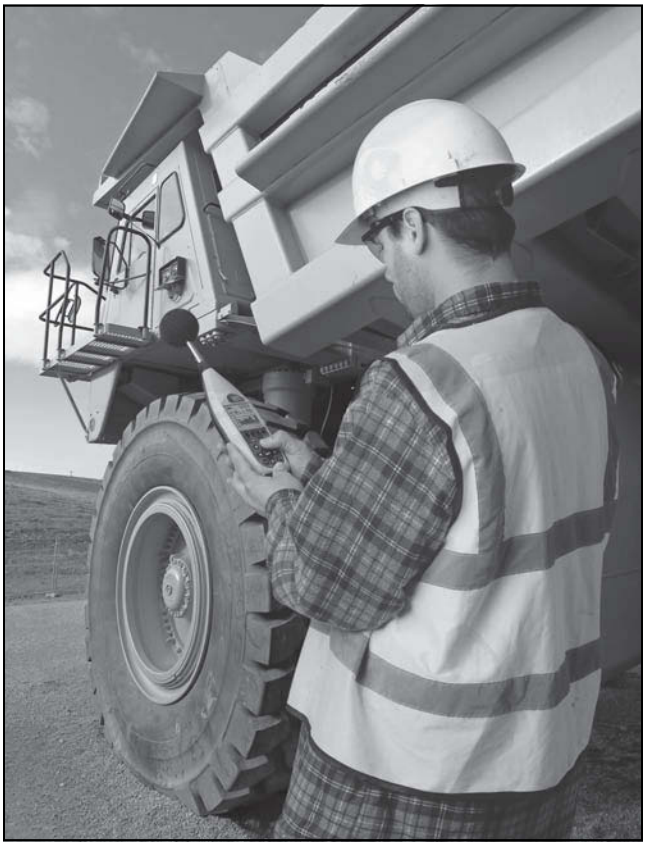
Sound Level, dBA	Reference Duration *T*, h	Sound Level, dBA	Reference Duration *T*, h
80	32.0	100	2.0
85	16.0	101	1.7
86	13.9	102	1.5
87	12.1	103	1.3
88	10.6	104	1.1
89	9.2	105	1.0
90	8.0	106	0.87
91	7.0	107	0.76
92	6.1	108	0.66
93	5.3	109	0.57
94	4.6	110	0.50
95	4.0	111	0.44
96	3.5	112	0.38
97	3.0	113	0.33
98	2.6	114	0.29
99	2.3	115	0.25

Source: MSHA 2006.
*At no time shall any excursion exceed 115 dBA. For any value, the reference duration T in hours is computed by $T = 8/2^{(L-90)/5}$ where L is the measured A-weighted, slow-response sound pressure level.

of engineered solutions or administrative or other control procedures. A good basic reference for understanding the measurement of noise is the *Handbook of Noise Measurement* (Peterson 1980).

Three types of instruments are used for measuring noise: the SLM, personal noise dosimeter, and octave band analyzer. Each has a specific purpose. Any instrument used for the measurement of noise must meet minimum standards to ensure accuracy. Figure 15.7-1 shows an SLM being used to measure noise levels near a haul truck. Figure 15.7-2 shows a miner wearing a newer personal noise dosimeter.

In the United States, SLMs must meet the standards set forth in American National Standards Institute (ANSI) S1.4-1983. Personal noise dosimeters must meet the standards set forth in ANSI S1.25-1991. In other parts of the world, international standard ISO 1999:1990 is the generally recognized standard for measurement of noise. In underground mines where explosive gases or atmospheres may be present,



Courtesy of Quest Technologies, a 3M company.
Figure 15.7-1 Sound level meter in use near a haul truck

electrical instruments must be rated intrinsically safe for use in explosive atmospheres. In the United States, MSHA maintains a list of permissible instruments and other equipment.

The SLM is the most commonly used instrument for identifying both stationary and mobile noise sources that may contribute to worker exposure. Most have both A-weighted and C-weighted settings. Readings are often used to assess machinery or community noise and to evaluate the attenuation of hearing protectors for use in high-noise environments.

A basic noise survey conducted with an SLM helps identify potential noise sources, determine sound levels in areas where workers may be exposed, and determine whether workers may be exposed to hazardous levels of noise. The entire area should be surveyed to identify all stationary sources so these sources and associated noise levels can be documented on a mine map or other plan. Mobile equipment must be measured while in service. SLM readings can help identify potential sources that require additional monitoring, often with more sophisticated equipment.

Although the noise dose that a worker receives can be calculated using SLM readings, in practice this is difficult to do with sufficient accuracy to determine compliance with regulations or standards. To use SLM readings, a worker must be followed throughout the work period to identify each source of noise, measure each source's noise level, and determine how much time the worker is exposed to each noise source and level. The worker's noise exposure is

$$\text{dose } (\%) = 100(C_1/T_1 + C_2/T_2 \ldots C_n/T_n) \qquad (15.7\text{-}1)$$

where C is the total duration of noise exposure at a specific noise level and T is the total duration of noise exposure permitted at that level. If the sum of the fractions exceeds 100, then the combined exposure exceeds the PEL if comparing against a regulatory limit or the TLV if comparing against the ACGIH recommended limit.

The personal noise dosimeter overcomes some of the disadvantages of the SLM. It is worn by the worker throughout the day and records all sounds exceeding the regulatory or recommended limit. At the end of the work shift, the dosimeter gives a readout of the equivalent dose in dBA or percentage of allowed exposure. Most contemporary dosimeters have software for downloading the collected data for individual time periods recorded, often for periods as short as 1 minute. The data can be printed out in table or graph form. Comparison of this data with observations about the work performed throughout the day can reveal which tasks or noise sources contributed to the overall exposure, allowing prioritization of noise sources for further study or engineering control.

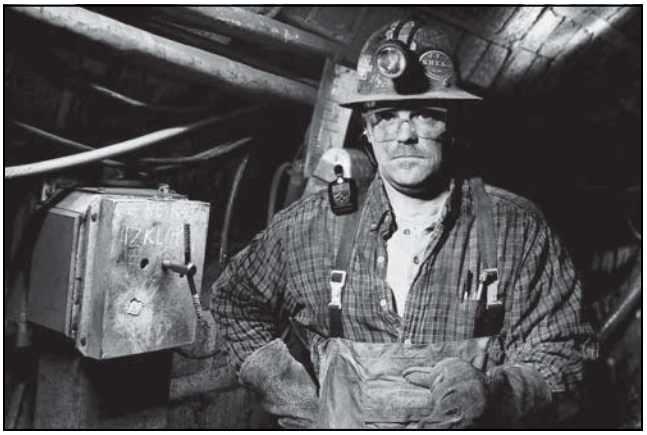
The octave band analyzer is a more sophisticated SLM. It is more accurate and has the asset of filters that measure noise at different frequencies. Table 15.7-3 lists the octave frequency bands that are commonly used for determining the frequency distribution of noise. Use of an octave band analyzer to obtain the frequency distribution of sound is important when designing engineering controls to reduce noise levels. As with other sound-measuring instruments, an octave band filter set and integrating SLM must meet specific standards (ANSI S1.43-1997 (R2002), ANSI S1.11-2004). Different frequencies may require different strategies for reducing worker exposure to noise. The information gained from this analyzer can also be used to calculate the protection factors of hearing-protective devices.

In general, noise surveys are of two types: (1) a health survey, to determine whether workers are exposed to noise levels above regulatory or recommended limits; and (2) an engineering survey, to identify sources of noise that contribute to worker exposure.

Health surveys, as noted previously, can be accomplished using either an SLM or a personal noise dosimeter. In conducting a health survey, it is important to ensure that an adequate number of workers are sampled so as to obtain a representative assessment of noise exposure. The best approach is to divide workers into homogeneous exposure groups (HEGs), each containing workers assigned similar tasks and thus expected to experience similar levels of exposure to noise. In deciding which workers to include in each HEG, the following must be considered: fluctuations in noise level, fluctuations due to working activities (variations in operating conditions, shutdowns, productions schedules), and variations in task performance among workers.

A number of samples for the same worker or multiple workers performing the same tasks over several different work periods is usually necessary to ensure that worker exposure is correctly measured. The number of samples required is based on statistical evaluation of the HEG and the variability of the noise levels. Six measurements for each HEG should be considered the minimum for a representative and statistically significant assessment. Samples should be as close to full-shift as possible and distributed over different days and shifts.

Software tools are available that can assist in obtaining statistically valid samples. After basic screening has been performed and potential workers and tasks that may be above regulatory or recommended limits have been identified, such tools can help determine whether the data is sufficient to determine compliance and the need for engineering controls. If in doubt, a competent, trained individual can be consulted to assist in the evaluation of worker exposure.



Courtesy of Occupational Health Dynamics.

Figure 15.7-2 Personal noise dosimeter worn by a miner

Table 15.7-3 Commonly used octave frequency bands (in Hertz)

Lower Band Limit	Center Band (Geometric Mean)	Upper Band Limit
22	31.5	44
44	63	88
88	125	177
177	250	354
354	500	707
707	1,000	1,414
1,414	2,000	2,828
2,828	4,000	5,656
5,656	8,000	11,312

Source: OSHA 1999.

Engineering surveys can be accomplished using an SLM or an octave band analyzer. The latter identifies specific frequencies that contribute to the total noise arising from individual noise sources. The engineering control solution often depends on the frequencies measured. Materials specified for controlling noise at the source must be matched to the frequency of the noise generated by the source.

It is important to remember that the decibel scale is logarithmic, so one cannot simply add decibel values when combining values from multiple noise sources. Two noises of exactly the same level, measured independently of one another, have a combined noise level that is 3 dBA higher than the individual values. The greater the difference between two individual noise sources, the lower the combined noise level.

Table 15.7-4 shows a simple way to determine combined noise levels. If more than two sources contribute to the overall noise level, then this process should be repeated iteratively to calculate the combined noise level. The *OSHA Technical Manual* (1999) contains a good description of this technique. When dealing with multiple noise sources, the best approach is to determine which source generates the most noise, and this source should be the first priority in deciding where to apply engineering controls.

Table 15.7-4 Adding decibels for two noise sources (in decibels)

Difference in dB Values	Add to Higher Level
0–1	3
2–3	2
4–9	1
≥10	0

Source: OSHA 1999.

Whether a survey is for health or engineering purposes, it is essential to document survey results so that one can understand the sources of noise and ensure that data conforms to regulatory requirements. Documentation must be kept in a form suitable for comparison with future survey results. Such comparisons are useful in evaluating engineering controls and equipment performance over time. In survey documentation, the following information should be recorded:

- Date and time
- Instrumentation details (model, serial number, settings, calibration)
- Workplace descriptions
- Task descriptions
- Unusual conditions

BASIC NOISE-CONTROL CONCEPTS

Noise control is complicated at best. The preferred method for reducing noise is to control it at the source so that it does not create an exposure hazard for workers. Mining operations present difficult challenges for the application of engineering controls, including underground equipment operation, special considerations for environment/climate, and type and age of equipment.

When obtaining new or replacement equipment, appropriate language must be included in the contract or purchase order ensuring that the purchased equipment will generate the lowest noise level that is technically feasible. Obtaining new equipment with noise-control features included reduces the need to retrofit controls in the work environment. The manufacturer is in the best position to reduce noise at the source by means of improved design and integration of noise controls.

As most mining equipment has a relatively long service life, noise controls may need to be added to current equipment to reduce exposure risk. Three basic types of control can be considered:

1. Substitution
2. Modification of the noise source
3. Modification of the sound wave

Substitution is a long-term effort to replace existing equipment or processes with alternatives that generate less noise. Although substitution may ultimately reduce noise levels, replacement schedules for most equipment in mining operations are too long to accommodate near-term improvements. Substitution should always be considered, but the need to protect workers and meet regulations means that alternative controls must be implemented sooner.

Modification of the noise source usually involves changing the way equipment works so as to reduce noise levels. Although some changes can be made to mobile equipment, changes may be more effective when made to stationary equipment such as screening and crushing equipment, motors, pumps, dryers, and air compressors. Typical control measures for stationary equipment include installation of isolation dampers and vibration-damping materials for vibrating equipment or surfaces, and use of exhaust mufflers and rerouting of intake/exhaust points on pneumatic equipment. Additional measures include improved bracing, increased stiffness, or application of damping materials to vibrating surfaces.

Modification of the sound wave is frequently the easiest method to implement. This can be accomplished by confining sound waves in acoustical enclosures, absorbing sound waves within a room or space, and absorbing sound along its transmission path. Effective enclosure design involves many factors, including the following:

- Selection of construction materials for the enclosure wall
- Selection of acoustical linings based on the frequency distribution of the noise
- Sealing of enclosure joints
- Mounting of the enclosure so as to minimize vibration
- Addition of access panels (doors) for maintenance and other operational needs
- Consideration of the effect that the enclosure design will have on the operating equipment
- Ventilation by means of inlets/outlets designed to prevent the escape of noise

It is important to focus on controlling the frequencies that most adversely affect human hearing in the speech range. Those frequencies are 500, 1,000, 2,000, 3,000, 4,000, and 6,000 Hz. The higher frequencies are usually easier to control.

If full enclosure is not possible, partial enclosures, noise barriers, or shields that separate the noise from the worker should be considered to reduce noise levels. Barriers are most effective for high- or medium-frequency noise. Barriers should be as high as practical and as close to the noise source as possible.

Ventilation ducts and conveyor systems can be important transmitters of noise. Ducts can be lined to attenuate sound traveling through them. Similarly, noise in conveyor systems can be attenuated by selection of appropriate lining materials or application of vibration-damping materials.

In addition to enclosing the noise source, it is also possible to reduce worker exposure to noise by enclosing the worker. This option is particularly effective for mobile-equipment operators. Many manufacturers offer original-equipment enclosures that reduce noise as well as minimize exposure to airborne contaminants and temperature extremes. Enclosed sound-reducing stationary-equipment control rooms can minimize worker exposure to noise at fixed locations. These enclosures can also serve as quiet areas where other workers can gain relief from noise during their work shifts.

It is important to consider the flammability of materials selected for engineering controls in mining operations. In the United States, MSHA references ASTM International's ASTM E162-87 in its regulations for product flammability in underground mines. Understanding the flammability of acoustical materials selected is particularly important where burning, welding, or other hot work is done in the vicinity of the selected materials.

More specific information regarding the design and construction of noise controls in mining is available from a variety of sources. In the United States, MSHA has a number of publications providing detailed guidance on applying engineering controls in mining operations (MSHA 1999a, 1999b, 1999c), many of which are available on the MSHA Web site at www.msha.gov. The U.S. National Institute for Occupational Safety

and Health (NIOSH) has specific programs for mining that are available on their Web site at www.cdc.gov/niosh/mining, including an area devoted to noise. The U.K. HSE lists case studies and sound solutions on their Web site at www.hse.gov.uk. Other countries may have similar information available. These resources can provide up-to-date information concerning control of noise in the mining industry.

Administrative controls are changes in work schedule or operations that reduce worker exposure to noise. Administrative controls should be used as adjuncts to—and not in lieu of—engineering controls. They may also have a place in minimizing worker exposure to noise while engineering controls are being designed and installed. As noise dose is a function of both noise level and duration of exposure, work can be organized into tasks that involve worker exposure to noise for only limited amounts of time during the work shift such that the noise dose remains below the regulatory or recommended limit. The balance of the work shift would include tasks where the noise is below the level that would contribute to the overall noise dose. Such rotational work schedules must be carefully designed so that worker exposures remain below acceptable levels. Other administrative controls could include remote operation of equipment or provision of quiet areas where employees can gain relief from noise. Improved maintenance of equipment and engineering controls can help to keep noise levels at the lowest level. In practice, however, administrative controls are of limited use because workers may not be allowed or able to switch among jobs, or work schedules can become too complicated to manage.

VIBRATION

Minimizing vibration, especially induced vibration, can contribute significantly to noise reduction. Many sources of noise that affect worker hearing are related to the vibration of equipment or surfaces. Vibration can be caused by a number of types of action, including rotation and reciprocation of devices. Examples of devices that tend to vibrate include compressors, pumps, fans, electric motors, pulleys, transfer belts, bin surfaces, gasoline and diesel engines, screening devices, and many more too numerous to mention. Vibration can result directly when rotating-equipment parts are unbalanced, misaligned, loose, or eccentric, or indirectly when induced in another piece of equipment.

The measurement and control of vibration serves two purposes: (1) prevention of premature wear and failure due to structural damage; and (2) reduction of noise levels. Measurement of vibration requires specialized equipment and experience in data interpretation. Many preventive-maintenance programs make use of vibration analysis to predict maintenance scheduling and equipment replacement.

Health effects beyond those related to hearing are also related to vibration. These additional effects are broadly considered to be ergonomics or human-factors issues related to whole-body vibration from activities such as operating mobile or stationary equipment. Both acute and chronic effects are of concern. Possible acute effects include loss of control of vibrating equipment due to mechanically induced decoupling from the hand. Possible chronic effects relate to the link between whole-body vibration and musculoskeletal problems including back pain and degenerative spinal disk disease.

The ACGIH, in its TLV recommendations and documentation, includes a TLV for whole-body vibration (ACGIH 2001). Whole-body vibration hazards and controls are not covered in this chapter but should be considered when vibration related to noise control is evaluated.

Vibration is controlled principally by reducing the mechanical disturbance that causes the vibration, isolating the disturbance from a radiating surface, and reducing the response of the radiating surface by applying damping or stiffening materials. Techniques such as balancing rotating equipment and installing isolation dampers may be obvious from initial evaluation. Increased maintenance to ensure that all equipment operates as designed and to observe early changes that may indicate increased vibration can help minimize vibration. Vibration control can be an iterative process involving incremental improvements as vibrating components are identified and managed.

PERSONAL PROTECTION

Even with the design and installation of engineering controls in mining operations, workers are still likely to be exposed to levels of noise that exceed regulatory or recommended limits. Even at levels below regulatory limits, noise-induced hearing loss can still occur. Most noise regulations established in recent years incorporate the concept of an action level. An action level is lower than the regulatory limit allowed for work-shift exposure, but still at a level where some hearing loss can occur. At this level, most regulations require that a number of measures be taken to monitor worker hearing and ensure that additional protective measures are implemented.

Most government regulations regarding noise in the workplace now require implementation of hearing-conservation programs for workers exposed above the action level. A hearing-conservation program should contain the following elements:

- Periodic assessment of worker exposure to noise (risk assessment)
- Health surveillance including audiometric testing
- Training
- Hearing protection

Periodic assessment of worker exposure to noise provides information about the continued effectiveness of installed engineering controls. Engineering controls over time may become damaged or require other maintenance to retain maximum effectiveness. Exposure data taken over time can also reveal changes in equipment noise levels caused by wear and tear. Continued compliance with regulatory or recommended exposure limits can be confirmed through periodic monitoring, and corrective actions can be taken as necessary. Worker use of engineering controls and hearing protection can be reinforced during periodic exposure assessments.

Health surveillance of workers exposed to high noise levels can identify workers with early signs of hearing loss so that further preventive measures can be taken. Usually health surveillance is conducted under the guidance of a physician who ensures that worker hearing is tested using approved audiometric test equipment and procedures and that test results are compared to normal hearing levels. If hearing loss is detected, additional testing can be undertaken and a specialist can be consulted to confirm the test results. If hearing loss is confirmed, the risk assessment should be reviewed and appropriate intervention taken for workers with confirmed hearing loss. The assessment should also explore whether workers have significant exposure to noise from activities outside of their employment.

Training, instruction, and information are key components to a successful hearing-conservation program. Most workers who understand the risks will take precautionary measures designed to prevent hearing loss. Information on the following subjects should be covered at a level at which the workers can understand:

- Nature of the hazards from exposure to noise
- Exposure limits compared to the exposure levels measured
- Risk assessment results
- Engineering controls in place
- Health surveillance program, including audiometric testing
- Purpose and value of wearing hearing protectors
- Types of hearing protection provided
- Government regulations as required

Computer software simulators have recently demonstrated the effects of noise-induced hearing loss. Use of these tools in training efforts can clearly demonstrate the effects of overexposure to noise. NIOSH has developed a hearing-loss simulator called HLSim, available from their Web site at www.cdc.gov/niosh/mining/products. The simulator demonstrates normal hearing and presents examples showing the effect of hearing loss on the ability to understand speech.

Hearing protection is critical for the prevention of worker hearing loss and is the last defense against the risks of noise. However, when properly selected and worn, hearing protectors are very effective in preventing hearing loss. Various types of protector are widely available, including ear plugs, ear caps, and ear muffs. The appropriate type for a particular situation depends on the level and frequency of noise measured during the risk assessment. In rare cases, double protection may be required. In the United States, MSHA has established a dual hearing-protection level of 105 dBA or greater for an 8-hour work shift. If necessary, a combination of ear plugs and ear muffs can provide added protection for very high noise exposure. Training programs should cover the various types of protector selected, the effectiveness of each type, and how to wear the protector properly.

Most hearing protectors have a noise-reduction rating (NRR) that indicates relative effectiveness. The NRR must be used with adjustment to compensate for variations in actual use. Several methods exist for calculating the adequacy of a protector. The easiest is to start with the NRR for a specific protector and calculate the attenuation value according to an approved method. NIOSH has published three methods for determining protector adequacy that vary by type of noise measurement data used (NIOSH 1975). OSHA has published a modified method for calculating adequacy based on the NRR value (OSHA 2004a, 2004b). Any of these methods can help ensure that the noise reaching the worker's ear has been reduced to protective levels.

SUMMARY

Noise exposure in mining operations remains a hazard, and continued diligence is required to protect worker hearing. Regulations to reduce the acceptable levels of exposure permitted for workers require the use of the following hierarchy of controls to reduce risk:

- Elimination
- Substitution
- Engineering controls
- Administrative controls
- Personal protective equipment

The controls at the top of this list are potentially more effective and protective than those at the bottom of the list, and result in the implementation of inherently safer systems that depend less on individual action. Thus, in the design of new processes and equipment, elimination and substitution, although the most difficult to implement, have the best chance for success. Administrative controls and personal protective equipment should be reserved for existing situations while engineering controls designed to remove the hazard are investigated and installed. If engineering controls do not reduce risk to acceptable levels, administrative controls and personal protective equipment may, by default, become long-term solutions.

The engineer plays an important role in the design and installation of engineering controls to reduce noise levels as much as reasonably achievable with the ultimate goal of eliminating noise as a risk in the work environment. There is broad recognition that this goal can be difficult to achieve. Understanding the risks of noise exposure and the importance of designing equipment and processes to minimize noise can help protect workers and minimize risk.

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PART 16

Environmental Issues

CHAPTER 16.1

Site Environmental Considerations

Michael G. Nelson

INTRODUCTION

Activities associated with mining have direct and lasting effects on both the physical and the human environments. The social, economic, and political effects in the latter category are often considered sustainability issues and are discussed briefly in this chapter. Also in the latter category are issues of worker health and safety, which are discussed in Part 15 of this handbook.

Mining includes the following activities, all of which have distinct impacts on the environment:

- **Exploration.** Economic deposits are identified and their characteristics are determined to allow recovery.
- **Development.** Preparations are made for mining.
- **Extraction.** Valuable material is removed for sale or processing.
- **Reclamation.** Disturbances caused by any of the preceding activities are corrected or ameliorated.
- **Closure.** Activity ceases and the area is abandoned or returned to another use.

This chapter will address the environmental considerations incumbent in the development and extraction in some detail, including the permitting process required (in almost all jurisdictions) before mining activities begin.

Environmental regulations vary among the countries of the world. Most large mining companies endeavor to follow uniform practices at all their mines, regardless of location, and have publicly committed to follow well-articulated standards of sustainability, such as those described under "Sustainable Practices" in this chapter. However, because of the variations in regulations, it is difficult to give a detailed description of environmental requirements and practices that applies worldwide. Thus, the focus of this chapter is to give a general description of environmentally sound practices and procedures. Some of the most important are illustrated by specific examples. The first example shows the approach to environmental risk assessment taken by CODELCO (Corporación Nacional del Cobre de Chile), the national copper corporation of Chile, as part of its environmental management plan under the International Organization for Standardization (ISO) Standard 14001. The second example gives a description of surface coal mining regulations in the United States, probably some of the most detailed and specific regulations in existence. The third example describes the permitting and approval process for the Safford mine, located in a historic mining area in southern Arizona (United States), and includes descriptions of the public input to this process. The fourth example describes a successful approach to early involvement of local communities in mine planning and permitting at the Diavik diamond mine, which operates in a pristine northern environment where no mining had previously occurred.

HISTORIC MINING PRACTICES

In the recent past, mining was done with little knowledge of its effects on the environment and, from a modern perspective, with little concern for the effects that were known. The consequences of this approach often resulted in significant damage to the natural environment, including (but not limited to)

- Unreclaimed mine pits, shafts, tunnels, and waste piles that may result in landslides and large amounts of blowing dust;
- Surface and groundwater that may be contaminated by solid particulates and chemical contaminants released by active and abandoned workings, or by waste piles;
- Abandoned pits, shafts, and tunnels that may create potential falling hazards to humans, livestock, and wildlife;
- Similarly, release of particulates and chemical contaminants from mine workings or waste and spoil piles that may cause direct injury to or damage the health of humans or animals;
- Erosion, with its consequent loss of soil and vegetation in and around unreclaimed workings, that may be a significant problem; and
- Underground workings, current or abandoned, that may cause surface subsidence, which can result in damaged surface structures, and in fissures or escarpments that are hazardous to humans and animals (Craig et al. 2001).

Examples of these kinds of mining-induced damages are readily found in any historic mining district. In fact, water

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contamination and abandoned waste piles left by ancient Roman mining activity in Spain resulted in rediscovery of the ore body on which Rio Tinto, one of the largest mining companies in the world, was founded (Raymond 1986).

In most cases, the environmental damage caused by mining was not well understood. There were some notable exceptions, such as the lawsuit in 1884 by downstream farmers against mining companies in California's "Mother Lode" district. By that time (well after the initial gold rush of 1849) those companies used a method called hydraulicking, in which entire hillsides were washed away with a powerful stream of water so the gold-bearing gravels could be processed. The resulting debris clogged streams and rivers and flooded meadows and fields, causing serious damage to agriculture (Hill et al. 2001). Other early environmental lawsuits related to the mining industry were directed at smelter operators by farmers who alleged that smelter gases were damaging their crops. Such suits were filed in England in 1865 (Brubaker 1995), and in the United States in Kansas in the 1880s (Junge and Bean 2006) and in Utah in 1903 (Lamborn and Peterson 1985). While these lawsuits may have been driven more by economic concerns than by a pure concern for the environment in the abstract, they certainly addressed what today would be considered environmental issues.

The public's expectations of the mining industry began to change in the 1950s, and by the end of the 1970s, governments in developed countries had enacted broad environmental laws that had direct bearing on all industrial activities, including mining (Kaas and Parr 1992). Although environmental standards still vary among countries, almost all major mining companies now state as policy that they will operate all their mines, regardless of location, to first-world standards of environmental protection and worker health and safety.

SUSTAINABLE PRACTICES

Leading mining companies have recently formulated, and pledged to follow, standards and principles for sustainable development of mineral resources worldwide. While not all of these principles relate directly to environmental practices, they represent a significant change in approach for the mining industry as a whole, a change that has already affected environmental practices in the industry. For that reason, the principles of sustainable development (as applied to mineral extraction) are discussed here.

In 1999, nine of the largest mining companies decided to embark on a new initiative intended to achieve a serious change in the way industry approached today's problems. They called this the Global Mining Initiative. It included a program of internal reform, a review of the various associations the companies belonged to, and a rigorous study of the societal issues they had to face. As a result, the International Institute for Environment and Development was commissioned to undertake the Mining, Minerals and Sustainable Development (MMSD) project.

Between 2000 and 2002, the MMSD project identified critical issues associated with development of mineral resources in four "spheres":

Economic sphere

- Maximize human well-being.
- Ensure efficient use of all resources, natural and otherwise, by maximizing rents.
- Seek to identify and internalize environmental and social costs.
- Maintain and enhance the conditions for viable enterprise.

Social sphere

- Ensure a fair distribution of the costs and benefits of development for all those alive today.
- Respect and reinforce the fundamental rights of human beings, including civil and political liberties, cultural autonomy, social and economic freedoms, and personal security.
- Seek to sustain improvements over time; ensure that depletion of natural resources will not deprive future generations through replacement with other forms of capital.

Environmental sphere

- Promote responsible stewardship of natural resources and the environment, including remediation of past damage.
- Minimize waste and environmental damage along the whole of the supply chain.
- Exercise prudence where impacts are unknown or uncertain.
- Operate within ecological limits and protect critical natural capital.

Governance sphere

- Support representative democracy, including participatory decision making.
- Encourage free enterprise within a system of clear and fair rules and incentives.
- Avoid excessive concentration of power through appropriate checks and balances.
- Ensure transparency through providing all stakeholders with access to relevant and accurate information.
- Ensure accountability for decisions and actions, which are based on comprehensive and reliable analysis.
- Encourage cooperation in order to build trust and shared goals and values.
- Ensure that decisions are made at the appropriate level, adhering to the principle of subsidiarity where possible.

In 2001, the board of the metals industry's representative organization, the International Council on Metals and the Environment agreed to broaden the group's mandate and transform itself into the International Council on Mining and Metals (ICMM).

In 2002, ICMM member companies signed the Toronto Declaration committing ICMM to continue the work started by the MMSD project and engage in constructive dialogue with key stakeholders, and in 2003, the International Council on Mining and Metals committed corporate members to implement and measure their performance against the ten principles shown in Table 16.1-1 (ICMM 2006).

Initially there was considerable debate in the mining community regarding the concepts of sustainability as applied to mineral extraction (NWMA 2002). Some argued that, because mineral resources are by nature finite, mineral extraction can never be truly sustainable. However, these concepts have in general been adopted by most of the mining industry, and they influence corporate practices in all areas of mining activity.

Table 16.1-1 ICMM's ten principles

1. **Implement and maintain ethical business practices and sound systems of corporate governance.**
   - Develop and implement company statements of ethical business principles, and practices that management is committed to enforcing.
   - Implement policies and practices that seek to prevent bribery and corruption.
   - Comply with or exceed the requirements of host-country laws and regulations.
   - Work with governments, industry and other stakeholders to achieve appropriate and effective public policy, laws, regulations and procedures that facilitate the mining, minerals and metals sector's contribution to sustainable development within national sustainable development strategies.
2. **Integrate sustainable development considerations within the corporate decision-making process.**
   - Integrate sustainable development principles into company policies and practices.
   - Plan, design, operate and close operations in a manner that enhances sustainable development.
   - Implement good practice and innovate to improve social, environmental and economic performance while enhancing shareholder value.
   - Encourage customers, business partners and suppliers of goods and services to adopt principles and practices that are comparable to our own.
   - Provide sustainable development training to ensure adequate competency at all levels among our own employees and those of contractors.
   - Support public policies and practices that foster open and competitive markets.
3. **Uphold fundamental human rights and respect cultures, customs and values in dealings with employees and others who are affected by our activities.**
   - Ensure fair remuneration and work conditions for all employees and do not use forced, compulsory or child labor.
   - Provide for the constructive engagement of employees on matters of mutual concern.
   - Implement policies and practices designed to eliminate harassment and unfair discrimination in all aspects of our activities.
   - Ensure that all relevant staff, including security personnel, are provided with appropriate cultural and human rights training and guidance.
   - Minimize involuntary resettlement, and compensate fairly for adverse effects on the community where they cannot be avoided.
   - Respect the culture and heritage of local communities, including indigenous peoples.
4. **Implement risk management strategies based on valid data and sound science.**
   - Consult with interested and affected parties in the identification, assessment and management of all significant social, health, safety, environmental and economic impacts associated with our activities.
   - Ensure regular review and updating of risk management systems.
   - Inform potentially affected parties of significant risks from mining, minerals and metals operations and of the measures that will be taken to manage the potential risks effectively.
   - Develop, maintain and test effective emergency response procedures in collaboration with potentially affected parties.
5. **Seek continual improvement of our health and safety performance.**
   - Implement a management system focused on continual improvement of all aspects of operations that could have a significant impact on the health and safety of our own employees, those of contractors and the communities where we operate.
   - Take all practical and reasonable measures to eliminate workplace fatalities, injuries and diseases among our own employees and those of contractors.
   - Provide all employees with health and safety training, and require employees of contractors to have undergone such training.
   - Implement regular health surveillance and risk-based monitoring of employees.
   - Rehabilitate and reintegrate employees into operations following illness or injury, where feasible.
6. **Seek continual improvement of our environmental performance.**
   - Assess the positive and negative, the direct and indirect, and the cumulative environmental impacts of new projects—from exploration through closure.
   - Implement an environmental management system focused on continual improvement to review, prevent, mitigate or ameliorate adverse environmental impacts.
   - Rehabilitate land disturbed or occupied by operations in accordance with appropriate post-mining land uses.
   - Provide for safe storage and disposal of residual wastes and process residues.
   - Design and plan all operations so that adequate resources are available to meet the closure requirements of all operations.
7. **Contribute to conservation of biodiversity and integrated approaches to land use planning.**
   - Respect legally designated protected areas.
   - Disseminate scientific data on and promote practices and experiences in biodiversity assessment and management.
   - Support the development and implementation of scientifically sound, inclusive and transparent procedures for integrated approaches to land use planning, biodiversity, conservation and mining.
8. **Facilitate and encourage responsible product design, use, reuse, recycling and disposal of our products.**
   - Advance understanding of the properties of metals and minerals and their life-cycle effects on human health and the environment.
   - Conduct or support research and innovation that promotes the use of products and technologies that are safe and efficient in their use of energy, natural resources and other materials.
   - Develop and promote the concept of integrated materials management throughout the metals and minerals value chain.
   - Provide regulators and other stakeholders with scientifically sound data and analysis regarding our products and operations as a basis for regulatory decisions.
   - Support the development of scientifically sound policies, regulations, product standards and material choice decisions that encourage the safe use of mineral and metal products.
9. **Contribute to the social, economic and institutional development of the communities in which we operate.**
   - Engage at the earliest practical stage with likely affected parties to discuss and respond to issues and conflicts concerning the management of social impacts.
   - Ensure that appropriate systems are in place for ongoing interaction with affected parties, making sure that minorities and other marginalized groups have equitable and culturally appropriate means of engagement.
   - Contribute to community development from project development through closure in collaboration with host communities and their representatives.
   - Encourage partnerships with governments and non-governmental organizations to ensure that programs (such as community health, education, local business development) are well designed and effectively delivered.
   - Enhance social and economic development by seeking opportunities to address poverty.
10. **Implement effective and transparent engagement, communication and independently verified reporting arrangements with our stakeholders.**
    - Report on our economic, social and environmental performance and contribution to sustainable development.
    - Provide information that is timely, accurate and relevant.
    - Engage with and respond to stakeholders through open consultation processes.

Source: ICMM 2006, © International Council on Mining and Metals.

Table 16.1-2 ISO 14001 standards

Systemic requirements
- a. Establish, document, implement, maintain, and continually improve an environmental management system in accordance with the standard.

Policy requirements
- a. Establish, define, document, implement, maintain, and communicate the organization's environmental policy.

Planning requirements
- a. Establish, implement, document, and maintain procedures to identify the environmental aspects of all activities, products, and services.
- b. Establish, implement, and maintain procedures to identify and clarify the legal and other requirements that apply to the organization's environmental aspects.
- c. Establish, implement, and maintain environmental objectives and targets.
- d. Establish, implement, and maintain programs to achieve environmental objectives and targets.

Operational requirements
- a. Personnel and resources
  - i. Provide the resources needed to establish, implement, maintain, and improve the environmental management system.
  - ii. Define, document, and communicate environmental management roles, responsibilities, and authorities.
  - iii. Appoint someone to assume the role of management representative.
- b. Training
  - i. Ensure competency of those who perform tasks that could have a significant environmental impact by identifying training needs, delivering training programs, and maintaining records of training activities.
  - ii. Establish, implement, and maintain a procedure to make people aware of the organization's environmental management system.
- c. Communication
  - i. Establish, implement, and maintain procedures to control the organization's internal and external environmental communications.
  - ii. Document the organization's environmental policy, objectives, and targets, and the main parts of its environmental management system.
  - iii. Describe how the parts of the organization's environmental management system interact.
- d. Documents
  - i. Control all environmental management documents and records required by the ISO 14001:2004 standard.
  - ii. Establish, document, implement, and maintain procedures to manage and control operational situations that could have significant environmental impacts.
  - iii. Establish, document, implement, and maintain procedures to control significant environmental aspects of goods and services provided by suppliers and contractors.
- e. Emergency situations
  - i. Establish, implement, and maintain procedures for potential emergency situations and accidents that could have an impact on the environment.
  - ii. Establish, implement, and maintain procedures to respond to actual emergency situations and accidents that have an impact on the environment.
  - iii. Test environmental emergency response procedures.
  - iv. Respond to actual environmental emergencies and accidents.
  - v. Prevent or mitigate the adverse environmental impacts that emergencies and accidents can and do cause.
  - vi. Review and revise environmental emergency preparedness and response procedures.

Compliance requirements
- a. Measurement and monitoring
  - i. Establish, implement, and maintain procedures to monitor and measure the operational characteristics that could have a significant impact on the environment.
  - ii. Use and maintain calibrated or verified environmental monitoring and measuring equipment.
  - iii. Keep a record of environmental monitoring and measuring activities.
- b. Documentation
  - i. Establish, implement, and maintain a procedure to periodically evaluate how well the organization complies with all relevant legal and other environmental requirements.
  - ii. Record the results of the organization's legal and other environmental compliance evaluations.
- c. Dealing with nonconformities
  - i. Establish, implement, and maintain nonconformance management procedures.
  - ii. Change documents when nonconformities make it necessary.
- d. Controlling records
  - i. Establish environmental records for the organization.
  - ii. Establish, implement, and maintain procedures to control the organization's environmental records.
- e. Internal audits
  - i. Establish, implement, and maintain an environmental management audit program and procedures.
  - ii. Conduct internal audits of environmental management system.
  - iii. Report internal audit results to management.
- f. Environmental management reviews.
  - i. Review the suitability, adequacy, and effectiveness of the environmental management system.
  - ii. Assess whether or not the organization's environmental management system, policy, objectives, or targets should be changed.
  - iii. Keep a record of all environmental reviews.

ENVIRONMENTAL MANAGEMENT SYSTEMS

ISO 14001, issued in 2004, provides standards by which a community or organization may put in place and implement a series of practices and procedures that, when taken together, result in an environmental management system (EMS).

An EMS is one part of an organization's larger management system. The EMS is used to establish an environmental policy and to manage the environmental aspects of the organization's activities, products, and services. A management system is a network of interrelated elements that include responsibilities, authorities, relationships, functions, processes, procedures, practices, and resources. A management system uses these elements to establish policies and objectives and to develop ways of applying these policies and achieving these objectives.

ISO 14001 is a detailed document comprising many pages. It describes systemic, policy, planning, operational, and compliance requirements. Table 16.1-2 outlines those requirements.

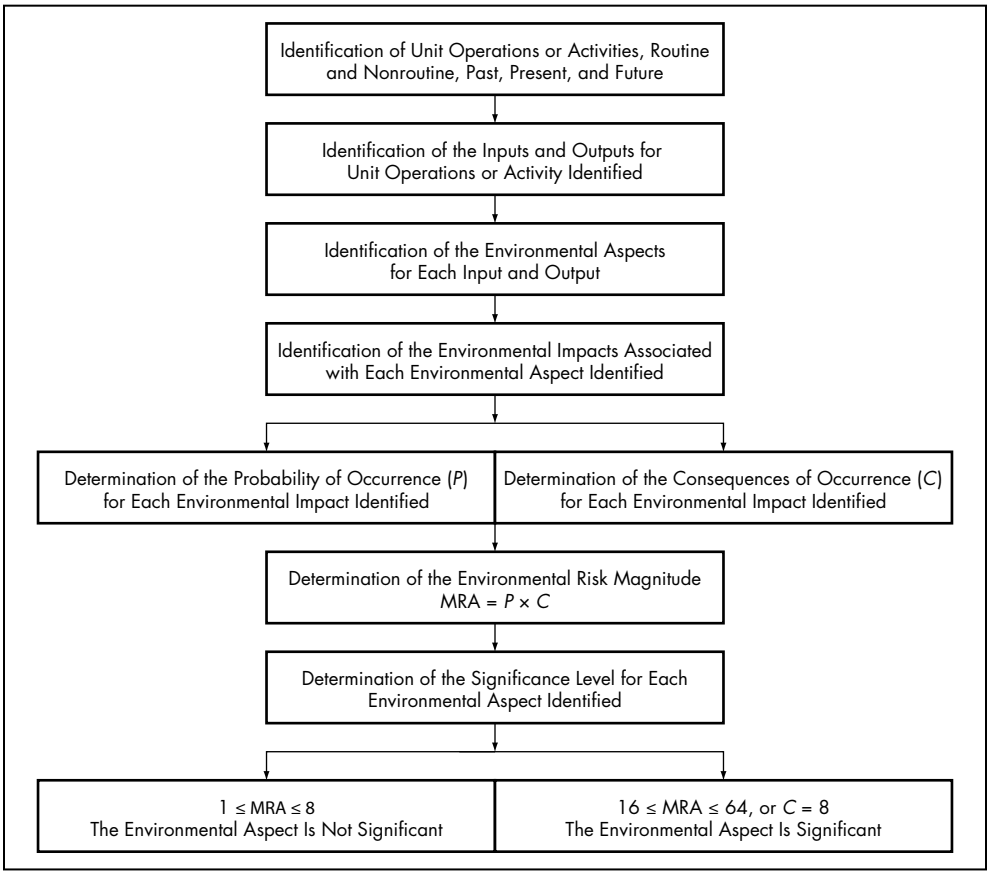
ISO 14001 is not a technical standard and thus does not in any way replace technical requirements embodied in statutes or regulations. It also does not set prescribed standards of performance for organizations. However, ISO certification or equivalent has become a critical component to acceptance by environmental agencies, nongovernmental organizations, and funding organizations.

The following example shows how a large mining company (CODELCO) complies with part of ISO 14001.

Example 1. Identification and Evaluation of Environmental Consequences

CODELCO adopted a sustainable development policy in June 2003 (CODELCO 2009). In December 2006, the company issued the Corporate Directive for Identification of Environmental Aspects and Evaluation of the Risk of Their Impacts (CODELCO 2006), providing a detailed and systematic method for assessing environmental risk, which is applied to all projects and activities in the corporation. The directive includes the following steps:

1. Identification of unit operations or activities, routine and nonroutine, past, present, and future.



Courtesy of CODELCO.

Figure 16.1-1 Process for identifying significant environmental aspects

2. Identification of all the inputs and outputs for each unit operation or identified activity.
3. Identification of the environmental aspects associated with the inputs and outputs of each unit operation or identified activity.
4. Identification of the potential associated environmental impacts of each identified environmental aspect.
5. Evaluation of the environmental risk, as determined by the probability of occurrence of each potential identified impact, as a function of its frequency, and the consequences of that impact on the environment, as a function of its reversibility and extent.
6. Determination of the magnitude of environmental risk (MRA, for the Spanish *magnitud del riesgo ambiental*), according to the probability and consequences, for each potential impact identified.
7. Determination of the degree of significance of each environmental aspect, according to the value of the MRA for that aspect.

The directive follows ISO 14001:2004 and distinguishes between environmental aspects and environmental impacts. An environmental aspect is an element of the activities, products, or services of one organization that can interact with the environment; environmental impact is any change in the environment, whether adverse or beneficial, wholly or partially resulting from environmental aspects of an organization.

The process begins with generation of the process map for each level in a given work center, using the environmental impacts as the main elements. This allows the direction of the process and any associated equipment to achieve common objectives aligned with the corporate standards. Here, the term *process* includes any activity of production, service, or administration and is defined as a sequence of activities that effect the transformation of certain inputs, with the use of certain resources, labor, goods, and services, to satisfy those who receive or use the process output. A process map is defined as a diagrammatic representation of a process in the abstract, showing connections among constituent activities and flows of material, energy, or information, and designed to facilitate understanding and analysis of the processes. Figure 16.1-1 shows a process map for the identification of significant environmental aspects.

In the identification of unit operations and activities, when there are areas or activities that are not defined within a specific organizational unit or may be subject to overlapping jurisdictions, the environmental management representative in charge of environmental affairs, together with the leaders of those areas or activities, define the allocation of responsibility for assessment of environmental risk.

In the identification of inputs and outputs, the responsible personnel examine each unit operation or activity (routine and nonroutine, past, present, and future). They identify all the inputs, including raw materials, resources, consumables, intermediate or recycled products, energy, services, and in general everything that the unit operation or activity receives from the preceding unit operation. Similarly, they identify all

Table 16.1-3 Evaluation criteria for the probability of occurrence of an environmental impact

	Criterion: Occurrence of the Impact, as a Function of Its Frequency	Value
Probability (*P*)	The impact always occurs in association with a given activity, or the impact happens often or frequently.	High (8)
	The impact is expected to occur sometimes, or the impact is known to have occurred on some occasions.	Medium (4)
	The impact is expected to occur on one occasion, or the impact is known to have occurred on one occasion.	Low (2)
	Occurrence of the impact is highly unlikely, or the impact has never occurred.	Negligible (1)

Courtesy of CODELCO.

Table 16.1-4 Evaluation criteria for consequences of the occurrence of an environmental impact

	Criterion: Description of the Impact Based on Its Reversibility and Extent	Value
Consequence (*C*)	The alteration of the original environmental conditions is considered irrecoverable, or the extension of the impact exceeds the limits for the area of direct and indirect influence of the project activity, or it significantly alters environmental conditions, and it cannot be attenuated.	High (8)
	The alteration of the original environmental conditions is considered to be 50%–90% recoverable, or the extent of the impact does not exceed the limit of the area of direct and indirect influence of the project of activity, or the impact significantly alters or has the potential to alter environmental conditions, but the effects can be mitigated by 50%–90%.	Medium (4)
	The alteration of the original environmental conditions is expected to be minor or transitory, or the possible alteration can, with appropriate control measures, be transitory or reduced to minor levels, or the environmental impacts occur only within the company property.	Low (2)
	The alterations in the original condition will not result in any noticeable changes to the environment, or it will be possible to minimize any alterations so they will be imperceptible, or controlled completely.	Negligible (1)

Courtesy of CODELCO.

Table 16.1-5 Values for environmental risk magnitude

		Consequences (*C*)			
	MRA	Negligible (1)	Low (2)	Medium (4)	High (8)
Probability (*P*)	Negligible (1)	1	2	4	8
	Low (2)	2	4	8	16
	Medium (4)	4	8	16	32
	High (8)	8	16	32	64

Courtesy of CODELCO.

Table 16.1-6 Significance criteria for environmental aspects

MRA = *P* × *C*	Classification of Environmental Impact	Significance of Environmental Aspect Leading to Environmental Impact
1–8	Insignificant	Insignificant
16–64	Significant	Significant
Consequence 8	Significant	Significant

Courtesy of CODELCO.

the outputs, including products sent to market, intermediate products, services to third parties, wastes and residues, and in general everything that the unit operation or activity delivers to subsequent unit operations or activities. The output of one unit operation or activity may often be the input to the next operation. In such cases, the line executive and the environmental management coordinators of both areas must define which operation is responsible for the identification of environmental aspects and the potential environmental impacts associated with this process flow, including the discharge, transport, required surge storage, and so on.

When identifying the environmental aspects, it is important to consider that different aspects can be associated with each operation or activity, not just for those performing the immediate task but for third parties. An example is the incorrect execution of a maintenance task, which may have effects on both the maintenance worker and on those who use the equipment after maintenance. The identification of these environmental aspects should consider (1) activities of all personnel who have access to the workplace, including contractors, suppliers, and visitors; (2) work in areas provided by the company and by third parties; and (3) activities that interact with other resources or protected areas, including tourist attractions, landscapes, or archaeological, anthropological, historical, or cultural heritage sites.

The probability of occurrence of an environmental impact, *P*, is evaluated according to Table 16.1-3, and the consequences of an environmental impact are evaluated according to Table 16.1-4. Next, for each identified environmental impact, the corresponding value in Table 16.1-5 is calculated as the product of the probability and the consequence for that impact.

Finally, the significance of the environmental aspects that give rise to each environmental impact is determined from Table 16.1-6. There are two conditions under which an environmental aspect is classified as significant: when the MRA is 16 or higher, or when the consequences of an associated environmental impact are high.

Figure 16.1-2 shows an example of a worksheet used to assess environmental risk by CODELCO's vice president of corporate projects, Office of Sustainable Development (E. Córdova, personal communication). The first column of the worksheet is prepared by listing every function or activity associated with a given project, in this case, the development of the Pilar Norte (the North Pillar) sector of the El Teniente deposit. The second column is completed with the environmental aspects and affected areas for each function or activity as identified by project personnel.

The sheet, with its first two columns completed, is circulated to experts within the company, each of whom assigns a value of 1, 2, 4, or 8 to the probability and the consequence of each environmental aspect in the second column. This procedure is similar to that used for hazard identification and risk assessment in the workplace (Chapanis 1986; Colling 1990). The product of the probability and the consequence agreed on for each aspect determines whether the aspect is significant or insignificant, according to the procedures and criteria described previously. Aspects with significant environmental aspects are addressed as priorities in project planning and management.

Function or Activity	Environmental Aspect	Affected Area	Probability	Consequence	Magnitude	Classification
Operation of diesel equipment	Particulate emissions	Work area	8	1	8	Not significant
	Gas emissions	Work area	8	1	8	Not significant
	Fuel consumption	Work area	8	1	8	Not significant
Handling hazardous materials	Potential for spills	Ground & air/Work area	4	4	**16**	**Significant**
	Generation of hazardous solid residues	Work area	8	2	**16**	**Significant**
	Welding gas discharge	Work area	8	1	8	Not significant
Equipment maintenance and repair	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	4	No Significant
Installation of workplace utilities and structures	Generation of non-process wastewater	Agua o Suelo	1	4	4	Not significant
	Generation of trash and other wastes	Work area	4	1	4	Not significant
	Generation of any solid residues	Work area	4	2	8	Not significant
	Potable water consumption	Natural resources	4	1	4	Not significant
Civil, mechanical, and electrical area control rooms; emergency drifts; main electrical room; transformer room; air shaft connections at level; extraction drift connections at level; hydraulic breaker foundations and control rooms; fan rooms	Welding gas discharge	Work area	4	1	4	Not significant
	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	4	Not significant
	Water consumption	Natural resources	8	1	8	Not significant
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Electical cables, cableways, and lights	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	4	Not significant
Installation and assembly of ventilation doors	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	4	Not significant
	Water consumption	Natural resources	8	1	8	Not significant
	Potential for spills of non-hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Installation of compressed air system	Generation of hazardous solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	4	Not significant
Foundations and other concrete structures for hydraulic breakers	Generation of any solid residues	Work area	8	2	16	Significant
	Electrical energy consumption	Natural resources	4	1	**4**	**Not significant**
	Water consumption	Natural resources	8	1	**8**	**Not significant**
	Potential for spills of non-hazardous materials	Ground & air/Work area	4	4	16	Significant
Installation of hydraulic breakers with hydraulic power systems and control panels	Generation of any solid residues	Work area	8	2	16	Significant
	Electrical energy consumption	Natural resources	4	1	**4**	**Not significant**
	Water consumption	Natural resources	8	1	**8**	**Not significant**
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	16	Significant
Installation of fire prevention system	Potential for spills of non-hazardous materials	Work area	4	2	**8**	Not significant
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
	Water consumption	Natural resources	8	1	**8**	Not significant
Installation of industrial water system	Emission of welding gases	Work area	4	1	**4**	Not significant
	Potential for spills of non-hazardous materials	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
Installation of water system for fire control	Welding gas discharge	Work area	4	1	**4**	Not significant
	Potential for spills of non-hazardous materials	Work area	4	2	**8**	Not significant
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
Installation of hydraulic power system	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Installation of ventilation fans	Potential for spills of non-hazardous materials	Work area	4	2	**8**	Not significant
Installation of steel arch-type roof supports	Potential for spills of non-hazardous materials	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
Modification and reconditioning of equipment washdown areas	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
	Water consumption	Natural resources	8	1	**8**	Not significant
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Construction of civil and electrical maintenance shops	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Installation of bridge crane	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
	Potential for spills of hazardous materials	Ground & air/Work area	4	4	**16**	**Significant**
Installation of water treatment (de-acidification) plant	Emission of welding gases	Work area	4	1	**4**	Not significant
	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Ground & air/Work area	4	1	**4**	Not significant
	Water consumption	Work area	8	1	**8**	Not significant
	Potential for spills of hazardous materials	Work area	4	4	**16**	**Significant**
Installation of air compressors and lubrication systems	Generation of any solid residues	Work area	8	2	**16**	**Significant**
	Electrical energy consumption	Natural resources	4	1	**4**	Not significant
	Potential for spills of hazardous materials	Work area	4	4	**16**	**Significant**

Courtesy of CODELCO.

Figure 16.1-2 Identifying and evaluating environmental effects of projects

ENVIRONMENTAL REPORTING

The conveyance of information is critical at all phases of a mining project. All of the people and institutions with an interest in how the project is conducted should be well informed of the ongoing plans for and progress with the project. MMSD addressed the issue of access to information in a report titled *Breaking New Ground* (MMSD 2002):

Information flow is essential in a sustainable stakeholder society. Information comes in different forms and is of variable quality. Information about a company and its operations is used by a range of actors, such as communities, investors, employees, lenders, suppliers, and customers, often through appropriate accounting and reporting procedures

based on defined indicators and measurement techniques. Within the industry, information is used by management to monitor performance efficiency and the impacts of operations. At the exploration phase, accurate geoscientific data and maps are crucial.

The working group that prepared *Breaking New Ground* held a workshop in 2001 in Vancouver. The workshop provided the opportunity to discuss the information used in preparing the report and examine how that information was viewed and handled by the mining industry. It was concluded that information often fails to flow to communities in a timely and transparent fashion, that disclosure practices often fall short of current best practice, and that one-size-fits-all systems of public reporting or a global reporting standard would be an extremely difficult initiative to develop. The distinct nature of specific mines, projects, companies, locations, and communities means that a different mix of indicators, metrics, and evaluations is needed. One effective way of scoping the need for information around any project is to ask the community what they need to know in considering project proposals.

The ISO 14001 standard includes disclosure as an outcome of the environmental auditing and management systems. The standard mainly addresses internal environmental management systems. Certain items within the standard refer in general terms to external communications as part of an organization's EMSs.

There are also voluntary initiatives to standardize the way corporations convey information. Most closely examined by the MMSD was the Global Reporting Initiative (GRI). It was convened in 1997 by the Coalition for Environmentally Responsible Economies "to make sustainability reporting as routine and credible as financial reporting in terms of comparability, rigour and verifiability" through "designing, disseminating and promoting standardized reporting practices, core measurements and customized sector specific measurements." The GRI Sustainability Reporting Guidelines suggest that reports include a CEO statement; key indicators; a profile of the reporting entity; policies, organization, and management systems; management performance; operational and product performance; and a sustainability overview (GRI 2009).

SOUND ENVIRONMENTAL PRACTICES

Past publications have provided detailed descriptions of environmental regulations and permitting procedures for a given location—often the United States. In this chapter, a more general approach is taken by describing the principles of sound environmental practice, which can and should be applied in any political jurisdiction. However, some examples of regulations and accepted practice are given for illustrative purposes.

Before discussing the individual activities involved in mining, it is important to point out that, in general, mining activities should be conducted with the future in mind. This will not only minimize the environmental effects of each activity, but will also result in significant cost savings. This approach will lead to some considerations that are very broad and apply to almost all projects. For example,

- Government officials and all residents in the area of the proposed mine should be well informed of and directly involved in all project plans from the beginning. Such involvement will help address the concerns of local people as they occur and may well obviate formal objections when permit applications are filed; and
- The overall mine plan should include consideration of the requirements of reclamation and restoration. Placement of waste piles, tailings ponds, and similar materials should be carefully planned to minimize rehandling of material.

At the same time, other considerations will be specific to a given project. For example,

- Design and location of roads developed for exploration should consider future needs for mining and processing. This will minimize unnecessary road building and decrease effects on the local ecosystem; and
- When containment ponds are required for drilling fluids or cuttings, consideration should be given to the future use of those ponds for tailings impoundment, storm water catchment, or settling ponds.

The mining operation should be thoroughly planned, in consultation with government agencies and people living in the region. Historic, cultural, and biological resources should be identified, and plans made for their protection. This is especially important in areas where indigenous people have little exposure to the technologies used in mining. Mining companies should take the appropriate steps to ensure that the indigenous people understand the mining plans. The indigenous peoples' values and land-use practices must be understood, and steps must be taken to protect them. In some locations, this will necessitate the involvement of anthropologists, sociologists, and other experts. It may also require a planning and approval process that differs from those to which the company is accustomed. For example, some indigenous peoples make decisions by group consensus, necessitating large community meetings that may last for several days.

Mine Development

Development is the preparation of the facilities, equipment, and infrastructure required for extraction of the valuable mineral material. It includes land acquisition, equipment selection and specification, infrastructure and surface facilities design and construction, environmental planning and permitting, and initial mine planning.

Hunt (1992) provided a useful checklist for mine development projects in the form of a questionnaire (see Table 16.1-7). While some aspects of this list are specific to operations in the United States, its breadth and detail make it a worthwhile resource for almost any mining project. It includes considerations for projects in areas where no mining has occurred, for projects in previously mined areas, and for expansion projects at operating mines.

Infrastructure and Surface Facilities Design and Construction

Infrastructure and surface facilities may include all of the following:

- Roads and railroads
- Power plants, electric power lines, and substations
- Fuel supply lines and tank farms
- Dams, diversions, reservoirs, groundwater well fields, water supply lines, water tanks, and water treatment plants
- Sewage lines and sewage treatment plants
- Maintenance shops
- Storage sheds and warehouses
- Office buildings, parking lots, shower facilities, and changehouses

- Worker accommodations (housing, cafeteria, infirmary, and recreation facilities)
- Houses for pumps, fans, and hoists
- Waste piles and impoundments
- Ponds for catchment of surface and groundwater, and mine drainage
- Industrial and household waste landfills

Six general environmental considerations apply to the design and construction of surface facilities and infrastructure:

1. Select locations of surface structures and infrastructure to minimize the effects of their construction and use on surface water, groundwater, plant and animal ecosystems, and nearby human habitation. Surface structures and infrastructure include roads, railroads, and power lines, for example.
2. Remove and store topsoil for future use from areas that will be later reclaimed.
3. Control runoff so that water exiting the boundaries of the permitted mine site is captured and treated as required to meet applicable discharge standards. Many jurisdictions permit mines as zero-discharge facilities, meaning that all water or solid waste is contained within the permitted mine area.
4. Protect surface water and wetlands. Because these features may exist directly over mineral reserves, their relocation may be necessary.
5. Revegetate disturbed areas. Recontour the land to a defined standard and revegetate with an approved mix of seeds and plantings.
6. Control emissions of dust and noise to meet the requirements of local regulations and the reasonable expectations of persons living nearby. In particular, if explosives are used, control air blast and ground vibration as required.

Specific considerations for some surface facilities or infrastructure include

- Maintenance shops should be designed to avoid contamination of soil and water by spilled fuel and lubricants;
- Surface facilities should be designed to minimize energy consumption, using solar water heating and alternative electric power generation wherever possible; and
- Surface facilities should also be designed architecturally to harmonize with the natural surroundings in the locale.

Environmental Planning and Permitting

The forward-thinking approach mentioned earlier is especially important in environmental planning and permitting. In the permitting process, careful planning is imperative, so that information is gathered efficiently, the applications submitted to regulators meet all requirements, including timely submission, and mine planning costs are minimized by avoiding redesign required after agency review. As pointed out by Hunt (1992), "One of the most serious deficiencies encountered by many operators in the permitting process is to fail to adequately plan the time required to collect baseline data and to provide for public and agency review of applications." Hunt goes on to list the major tasks and activities required during early phases of environmental planning:

- Identification of the regulatory agencies that will be responsible for permitting the proposed mining operation
- Acquisition of all necessary permitting forms and copies of applicable regulations
- Identification of possible legal or technical restrictions that may make approval difficult or require special attention in the permitting process
- Identification of environmental resources information that will be required and the development of a plan and schedule for collection of the required data

Development of a comprehensive permit plan during the initial stages of mine development is critical. The plan should identify predecessor information that must be generated at each stage in the permitting, thereby identifying critical path issues and potential bottlenecks. A Gantt chart is often used for this purpose. For example, if the groundwater discharge permit for the tailing impoundment takes 2 years, and the design must be submitted with the application, the mine may need to move the "design tailing impoundment" task up on the schedule. It is recommended that an environmental permitting specialist be tasked with preparing the comprehensive permit plan.

After preliminary planning, permit applications must be completed. Many jurisdictions require a detailed environmental audit, which is then described in a report, often called an environmental impact statement, or EIS. Depending on local regulations, the EIS may be issued for public review and comment. Topics covered in an EIS usually include the following 19 areas:

1. Permits required and permitting status
2. Ownership of surface and mineral rights
3. Ownership of the company or companies that will be operating the mine, including contractors
4. Previous history of the entities described in Item 3
5. Information on archaeological, historical, and cultural resources within the planned mine site and adjacent areas
6. Hydrology of the mine site and adjacent areas, including the presence of potential pollutants and plans for dealing with them
7. Potential for dust and air pollution, and plans for mitigation
8. Plans for use and control of explosives
9. Plans for disposal of mine waste, including methods for controlling dust generation, slope stability, and seepage of contaminated water
10. Plans for impoundments, and methods for preventing failure or overtopping
11. Plans for construction of backfilled areas during mining, and for backfilling of mine excavations when mining is finished, including methods for preventing slides or other instability
12. Potential for surface subsidence, both immediate and long term, and proposed methods of preventing subsidence or compensating for its negative effects
13. Potential for fires in mine workings, at outcrops, and in spoil piles, and methods for prevention and control of such fires
14. Planned use of injection wells or other underground pumping of fluids
15. Information on previous mine workings, plans for reclaiming previous workings, and description of how the planned workings will interact with previous workings
16. Socioeconomic impact analysis
17. Visual impact analysis
18. Noise impact analysis
19. Biological assessment and impact analysis

Table 16.1-7 Environmental audit checklist

Category	Checklist Items
Permits	☐ Provide a list of all permits existing, applied for, and in preparation for the mine. Indicate the current status of each permit. ☐ If the mine has National Pollutant Discharge Elimination System permits, air permits, waste permits, water withdrawal permits, or other permits, include a listing of each applicable permit. ☐ Are any additional permits required for the facility but not already obtained? ☐ Were any citizen protests or comments filed on any of the permits? If so, describe the issues raised. ☐ What is the status of the bond on each permit? Indicate the amount of bonding liability under each permit.
Surface and mineral rights information	☐ Is the surface property owned or controlled by the mining company? ☐ Does the mine have a specific written agreement with each surface owner pertaining to and providing applicable rights to the surface? ☐ If the mine is an underground mine, do the surface rights specifically include language that says the company has the right to subside? ☐ Has the mine ever received any complaints from a surface owner or any of the surrounding landowners? Indicate the nature of any complaint received. ☐ With respect to mineral rights, has there ever been a claim made suggesting that the mining company does not have complete and full mineral rights? ☐ Have any other entities been granted access rights to the property that may conflict with the rights to extract the mineral (e.g., oil wells, pipelines, utility easements, etc.)?
Ownership and violation information	☐ Is the mine permit held by any company not owned and controlled 100% by the operator? ☐ Is the mine permit being held by an independent contractor? ☐ Is the mine operation being conducted by any company not owned and controlled 100% by the permittee? ☐ Is the mine being operated by an independent contractor? Give a a brief summary of applicable relation. ☐ If the mine is permitted by or being operated by an independent contractor, has a review been completed of the contractor's violation history with regard to any other mines potentially affiliated with the contractor? ☐ If the mine is permitted by or being operated by an entity only partially owned, has a review been completed of the contractor's violation history with regard to any other mines potentially affiliated with the other parties? ☐ How is the relationship between mine personnel and the mine inspectors (excellent, good, poor)? If poor, please describe the problems encountered. ☐ Have there been any citizen complaints filed on the mine? If so, please describe the complaints filed.
Information on archaeological sites, cemeteries, etc.	☐ Within the permit area and adjacent areas, including the area over existing, past, and proposed underground workings, does the mine contain any of the following features? • Are there any federal lands? Indicate the type of federal land involved. • Is there any archaeological or historical site that has been previously identified? Indicate the type of site involved. • Are there public roads where there is not a specific written approval for mining through or under the public road by the agency that has jurisdiction over the road? • Are there occupied dwellings where there is not a specific written waiver from the owner of the occupied dwelling? • Are there any public buildings, schools, churches, community, or institutional buildings? • Are there any public parks? • Are there any public or private cemeteries?
Hydrology	☐ Has the mine obtained all necessary water discharge permits? ☐ Is the mineral being mined associated with potential pollutants, such as high-sulfur-bearing strata? ☐ Has the mine ever experienced any difficulties with poor-quality mine drainage? Briefly describe the situation encountered. ☐ Does any of the mine water need to be treated to meet effluent limitations or other applicable standards? ☐ Has any poor-quality mine drainage been observed on any part of the permit area, in any of the surrounding areas, whether associated with the mine or not? ☐ Have any of the surrounding landowners or occupants ever complained about loss of water or the mine contaminating their water supplies? ☐ Is the water used by surrounding landowners of poor quality? For example, is the water discolored, does it have a bad odor or bad taste, or are there other problems with water quality in the area? ☐ Has the mine ever had any complaints about the water level in streams increasing or decreasing, or the fish population in streams increasing or decreasing? ☐ Are there any in-stream treatment facilities or in-stream sediment ponds associated with the mine? ☐ Are there any wetland areas, wet areas with cattails, or similar aquatic vegetation that could be considered wetlands, within the vicinity of the mine?
Air pollution	☐ Has the mine obtained all the necessary quality permits? ☐ Have there been complaints about dust or air quality discharges from the mine? ☐ Are mine haul trucks covered with tarps to minimize dust and to prevent material from falling off? ☐ Is there a program for controlling dust and air pollution from the mine?
Explosives	☐ Has the mine had any complaints from landowners pertaining to blasting or the use of explosives? ☐ Is the mine access limited because of topographic conditions, or is it possible for local people to obtain unauthorized access to the mine site? ☐ If it is possible for unauthorized persons to obtain access to the site, what measures are currently taken to control this access? ☐ Are any personnel other than trained mine personnel allowed within the mine pit or the mine property in an area that may ever be subject to blasting?

(continues)

Table 16.1-7 Environmental audit checklist (continued)

Category	Checklist Items
Waste disposal	☐ Does the operation include a facility for the disposal of mine waste? ☐ Have there ever been any discolored or bad-quality water seeps from the waste pile? ☐ Does the groundwater indicate any levels of pollution from the waste pile? ☐ What is the worst-case chemical analysis for the waste material being disposed of at the mine? ☐ Have there ever been any signs of slumping or instability of the waste pile? ☐ Does the waste pile have any public roads, homes, occupied dwellings, or other structures downstream that could be affected by failure? ☐ Is the waste pile routinely inspected and certified by a professional engineer to verify that it is being built in accordance with the approved plan? ☐ Are there any abandoned mine waste piles within the vicinity of the mine, including the area overlying underground workings? ☐ Do any of the abandoned mine waste piles show any signs of instability or poor mine drainage? ☐ Does the disposal of any domestic or other nonmine waste occur within the property boundaries, including the disposal of garbage by mine or nonmine personnel? ☐ Has a survey been done to determine whether there is any nonmine waste on the property? Are there any abandoned nonmine waste piles within the mine property? ☐ What does the mine do with trash from the mine?
Impoundments	☐ Are there any large impoundments within the vicinity of the mine or included in the mine permit? ☐ Are there any public roads, facilities, dwellings, or other such facilities downstream of an impoundment that could be affected by the impoundment's failure? If so, briefly describe. ☐ Are all impoundments routinely inspected and certified by a professional engineer to verify that they are being built in accordance with the approved plans? ☐ Have any of the impoundments ever experienced overtopping?
Backfilling and slides	☐ Has the mine ever experienced a slide or instability of backfilled material or of outslope material? ☐ Does the mine have an excess spoil fill? Does the mine have a durable rock fill? Were any of the fills constructed in a location where there was a stream or running water prior to mining? ☐ If the mine is a surface mine, what is the maximum size pit and the size of the total pit currently open for the mine? How many cubic meters (cubic yards) of material would be required if the mine was closed today and the pit had to be backfilled? ☐ What is the maximum amount of highwall currently open on the operation, including total length and height? How many cubic yards (cubic meters) of material would be required if the mine was closed today and the highwall had to be completely eliminated?
Subsidence	☐ Does the mine include longwall or pillar removal mining? ☐ Has the mine ever identified any substance subsidence? ☐ Have there ever been any complaints by surface owners about surface subsidence? ☐ Does the mine have any significant problems with roof or floor control?
Mine fires	☐ Have there ever been any mine fires, including outcrop fires, identified on the mine property?
Underground injections	☐ Does the mine have any injection wells or pump any fluids or other material of any type underground? ☐ If the mine is an underground mine, has an evaluation ever been completed to determine what will occur after the mine is closed and the mine workings become flooded? If yes, briefly describe the resulting conclusion.
Previously mined areas	☐ Does the area affected by the mine include a previously mined area? ☐ Are there any of the following problems on the previously mined area? • Bad water drainage? • Unstable spoil? • Spoil on the outslope? • Refuse material on the outslope? • Unreclaimed areas where revegetation will be difficult? • Unreclaimed highwall?
Miscellaneous	☐ How close is the nearest house? ☐ Have there been complaints about the use of roads in the area? ☐ Has the mine ever received a closure or cessation order? ☐ Are there cleaning or other chemical solvents located at the mine? Have all necessary forms been filled out and provided in regard to "right to know" requirements for chemical exposure? ☐ Have all transformers or other electrical equipment been checked for polychlorinated biphenyl contamination? ☐ Is oil or fuel stored on the property? What types of facilities are provided for oil or fuel storage? Are there any underground storage tanks? Have all such tanks been tested? ☐ Are there any oil or gas wells, or similar facilities on the property that could conflict with mining?

Source: Hunt 1992.

Some countries require one comprehensive permit that addresses all environmental aspects, but some countries, such as the United States, require many individual permits covering myriad aspects. The following example illustrates the planning and permitting process for a surface coal mine in the United States.

Example 2. Premine Planning and Permitting for Surface Coal Mines in the United States

Table 16.1-8 describes reclamation requirements for surface coal mining based on requirements given in the Surface Mining Control and Reclamation Act (SMCRA) of 1977. Under SMCRA, specific requirements are given for obtaining a mining permit, and formal documents must be filed (CFR 2009).

A large amount of information is required for preparation of the reclamation and operations plan. Table 16.1-9 summarizes those requirements.

Government Agency Review of Permit Applications

The processing of a permit application begins with the submission of a completed application for a new permit, permit revision, or permit renewal to the regulatory authority and ends with the final decision to approve or deny the application. Agency review begins when the applicant officially files the permit application. For most permits, after the agency determines the application is complete, they begin the formal review process, notify affected parties, and begin a public comment period. In most cases, members of the public will be able to both submit written comments and request a hearing on the merits of the particular application. If the permit is granted, the operator may be required to post a bond. Upon approval of the bond, mining operations may commence.

Most agencies will conduct a detailed technical review of each application received to ensure that all applicable regulatory requirements are met. As part of this technical review, the applicant is notified of any deficiencies identified in the application as submitted and provided an opportunity to respond to the agency comments and to correct deficiencies. Failure to provide an adequate response or to correct any permit deficiency could result in denial of the application.

The time required for agency reviews will range from a few months to 2–3 years. This time can be even longer in areas where opposition to a project is well coordinated and well funded. The permitting for the Safford mine in Arizona, described in detail in this section, took 13 years. Thus it is extremely important that permit applications be prepared with great care and reviewed meticulously to minimize delays by the agencies.

In some countries, efforts have been made to speed up the permit review process. For example, in Chile, when environmental regulations were promulgated in 1994, specific approval times were specified. The maximum time for approval of an Environmental Impact Declaration is 90 days, not including the days required for response to questions from regulatory agencies. For the more complex environmental impact study, which requires establishment of baseline standards and community participation, the maximum is 180 days (República de Chile 2002).

Example 3. Freeport-McMoRan's Safford Mine

Exploration of the Dos Pobres/San Juan area of the Safford Mining District began in about 1940. The district is in Graham County, Arizona, about 12.9 km (8 miles) north of Safford. It comprises four porphyry copper deposits—Dos Pobres, San Juan, Lone Star, and Sanchez—located along the southwestern slope of the Gila Mountains north of the Gila River.

The Phelps Dodge Corporation (PD; now a part of Freeport-McMoRan) acquired an interest in the property in 1957 and began underground development in 1968. Underground activity stopped in 1982, and by 1992 interest had shifted to the oxide mineralization in the area. By 1994, PD owned most of these mineral deposits, but some extended onto public lands administered by the U.S. Bureau of Land Management (BLM). The portions of the deposits on BLM land were controlled by PD through mining claims filed under the U.S. General Mining Law of 1872 (Cooper 2008).

In an effort to consolidate its holdings in the Safford Mining District, PD proposed a land exchange to the BLM, in which PD would acquire public lands (referred to as the selected lands) within and adjacent to its existing private property in the mining district in trade for other lands (the offered lands) in Arizona currently owned by PD. Both parties signed an Agreement to Initiate (ATI), which began formal consideration of the land exchange.

The administration of public lands in the United States is often controversial, especially in the western states. In most areas, groups of citizens have organized to oppose almost any development on those lands. These groups are often supported by large, well-funded environmental organizations, such as the Sierra Club. Before making a decision about the land exchange, the BLM decided to prepare an EIS to comply with provisions of the U.S. National Environmental Policy Act (NEPA). After the parties agreed to formally consider a land exchange, public scoping took place in late fall of 1994 and baseline studies commenced.

Under the U.S. Clean Water Act, the U.S. Army Corps of Engineers (ACE) issues permits for the discharge of dredged or fill material into the navigable waters at specified disposal sites. Review of these permits usually requires several months after notice and the opportunity for public hearings. In late 1995, ACE stated that it would likely require an EIS as part of its environmental review for a Section 404 permit to implement the foreseeable mining uses if the land exchange was authorized. At about the same time, PD accelerated the planning and development schedules for the Dos Pobres/San Juan project.

PD submitted a mining plan of operations (MPO) in May 1996, which allowed the BLM, ACE, and EPA (U.S. Environmental Protection Agency), as cooperating agencies, to consolidate their respective environmental reviews for the proposed mining activities. The MPO alternative provided agency employees and the general public with more information regarding the potential effects of the foreseeable mining uses of the selected lands and also conformed to regulations of the Council on Environmental Quality and the guidelines published in *NEPA's Forty Most Asked Questions* (NEPA 2009).

In response to PD's submittal of an MPO, the BLM determined that the EIS should reflect the fact that an MPO was now the proposed action and made the land exchange proposal one alternative to the MPO. The BLM also involved ACE and EPA as cooperating agencies in the EIS process. The BLM then reinitiated the scoping process begun earlier because submittal of the MPO represented a significant change in scope from the original land exchange proposal. In December 1997, The U.S. Bureau of Indian Affairs (BIA) became a cooperating

Table 16.1-8 SMCRA reclamation requirements

Requirement	Description
Operations plan	• The type and method of coal mining, procedures, and proposed engineering techniques; anticipated annual and total production of coal, by tonnage; and the major equipment to be used for all aspects of those operations • The construction, modification, use, maintenance, and removal of the following facilities: retention dams, embankments, and other impoundments; overburden and topsoil handling and storage areas and structures; coal removal, handling, storage, cleaning, and transportation areas and structures; spoil, coal processing waste, and noncoal waste removal, handling, storage, transportation, and disposal areas and structures; mine facilities; and water and air • Existing structures proposed to be used in connection with or to facilitate the surface coal mining and reclamation operation • Blasting practices and blast monitoring system
Maps	Locations of • Lands proposed to be affected throughout the operation and any change in a facility or feature to be caused by the proposed operations • Buildings, utility corridors, and facilities to be used • The area of land to be affected within the proposed permit area, according to the sequence of mining and reclamation • Each area of land for which a performance bond or other equivalent guarantee will be posted • Each coal storage, cleaning, and loading area • Each topsoil, spoil, coal waste, and noncoal waste storage area • Each water diversion, collection, conveyance, treatment, storage, and discharge facility to be used • Each air pollution collection and control facility • Each source of waste and each waste disposal facility relating to coal processing or pollution control • Each facility to be used to protect and enhance fish and wildlife and related environmental values • Each explosives storage and handling facility • Each sedimentation pond, permanent water impoundment, coal processing waste bank, and coal processing waste dam and embankment, and fill area for the disposal of excess spoil
Air pollution control plan	Description of fugitive dust control practices and a monitoring program to provide sufficient data to evaluate the effectiveness of the fugitive dust control practices proposed to comply with federal and state air quality standards
Fish and wildlife information	All listed or endangered species, and all unusual habitats; protective measures to be undertaken during mining and reclamation; and plans for habitat enhancement
Reclamation plan	• A detailed timetable for the completion of each major step in the reclamation plan • A detailed estimate of the cost of reclamation of the proposed operations required to be covered by a performance bond, with supporting calculations for the estimates • A plan for backfilling, soil stabilization, compacting, and grading, with contour maps or cross sections that show the anticipated final surface configuration of the proposed permit area • A plan for removal, storage, and redistribution of topsoil, subsoil, and other material • A demonstration of the suitability of topsoil substitutes or supplements, based on analysis of the thickness of soil horizons, total depth, texture, percentage of coarse fragments, pH, and areal extent of the different kinds of soils • A plan for revegetation including, but not limited to, descriptions of the revegetation schedule; the species and amounts per acre of seeds and seedlings to be used; methods to be used in planting and seeding, mulching techniques, and irrigation (if appropriate); pest and disease control measures, if any; measures proposed to be used to determine the success of revegetation; a soil testing plan for evaluation of the topsoil results; and the handling and reclamation procedures related to revegetation • A description of the measures to be used to maximize the use and conservation of the coal resource • A description of measures to be employed to ensure that all debris, acid-forming and toxic-forming materials, and materials constituting a fire hazard are disposed of properly, and a description of the contingency plans that have been developed to preclude sustained combustion of such materials • A description, including appropriate cross sections and maps, of the measures to be used to seal or manage mine openings, and to plug, case, or manage exploration holes, other boreholes, wells, and other openings within the proposed permit area • A description of steps to be taken to comply with the requirements of the Clean Air Act, the Clean Water Act, and other applicable air and water quality laws and regulations, and health and safety standards • Description of the land use within the proposed mining area before mining, and of the proposed land use after mining and reclamation are complete • A general plan and a detailed design plan for each proposed siltation structure, water impoundment, and coal processing waste bank, dam, or embankment within the proposed permit area, and of the plans for removing each such structure
Hydrology	• Sampling and analysis methodology • Baseline information • Groundwater information, including the location and ownership for the permit and adjacent areas of existing wells, springs, and other groundwater resources; seasonal quality and quantity of groundwater and usage. Water quality descriptions shall include, at a minimum, total dissolved solids or specific conductance, pH, total iron, and total manganese. • Surface water information, including the name, location, ownership, and description of all surface water bodies such as streams, lakes, and impoundments; the location of any discharge into any surface water body in the proposed permit and adjacent areas; and information on surface water quality and quantity sufficient to demonstrate seasonal variation and water usage • Baseline cumulative impact area information • Alternative water source information • Probable hydrologic consequences determination • Cumulative hydrologic impact assessment • Hydrologic reclamation plan • Groundwater monitoring plan • Surface water monitoring plan

(continues)

Table 16.1-8 SMCRA reclamation requirements (continued)

Requirement	Description
Geology	• Sufficient detail to assist in determining the probable hydrologic consequences of the operation on the quality and quantity of surface and groundwater in the permit and adjacent areas, including the extent to which surface water and groundwater monitoring is necessary; all potentially acid- or toxic-forming strata down to and including the stratum immediately below the lowest coal seam to be mined; and whether reclamation can be accomplished and whether the proposed operation has been designed to prevent material damage to the hydrologic balance outside the permit area, including, at a minimum, the following: A description of the geology of the proposed permit and adjacent areas down to and including the deeper of either the stratum immediately below the lowest coal seam to be mined or any aquifer below the lowest coal seam to be mined that may be adversely impacted by mining. The description shall include the areal and structural geology of the permit and adjacent areas, and other parameters that influence the required reclamation and the occurrence, availability, movement, quantity, and quality of potentially impacted surface and groundwaters. • Analyses of samples collected from test borings, drill cores, or fresh, unweathered, uncontaminated samples from rock outcrops from the permit area, down to and including the deeper of either the stratum immediately below the lowest coal seam to be mined or any aquifer below the lowest seam to be mined that may be adversely impacted by mining. The analyses shall result in the following: (a) logs showing the lithologic characteristics including physical properties and thickness of each stratum and location of groundwater where occurring; (b) chemical analyses identifying these strata that may contain acid- or toxic-forming or alkalinity-producing materials and to determine their content, except that the regulatory authority may find that the analysis for alkalinity-producing materials is unnecessary; and (c) chemical analyses of the coal seam for acid- or toxic-forming materials, including the total sulfur and pyritic sulfur, except that the regulatory authority may find that the analysis of pyritic sulfur content is unnecessary.
Diversions	Stream channel and other diversions to be constructed within the proposed permit area, including maps and cross sections
Protection of area	Measures that will be taken to protect any publicly owned parks and historic places within the proposed permit area
Public and private interests	Description, including appropriate maps and cross sections, of measures that will be taken to ensure that the interests of the public and landowners affected are protected if during mining the operator undertakes the relocation or use of public roads
Disposal	Description, including appropriate maps and cross-sectional drawings, of methods for disposal of excess spoil, including the proposed disposal site, the design of the spoil disposal structures, the removal, if appropriate, of the site and structures, and any requisite geotechnical investigation
Roads	Description, including plans and drawings, for each road to be constructed, used, or maintained within the proposed permit area

Source: CFR 2009.

agency in the preparation of this EIS, specifically to contribute its expertise in the areas of Indian trust resources and tribal consultation. Protection of groundwater resources was of particular interest to the Indian (indigenous) people in the project area.

In September 1998, the draft EIS was published. It was more than 400 pages long (not including seven appendices), and included 108 tables and 73 figures. It addressed the scoping issues shown in Table 16.1-10. The EIS also identified the laws, regulations, and other government actions applicable to the project, including nine federal acts, five federal executive orders, six federal land-use or management plans, and twelve state and local laws.

A groundwater model and the Model, Monitor, and Mitigate program for groundwater protection were prepared. Both documents were reviewed at length by the BLM's hydrologist, the BIA, and a consultant retained by the BIA. However, the agencies disagreed on these and other issues, and the BIA withdrew as a cooperator in June 2000, without facilitating consultations with Indian tribes regarding potential impacts to trust resources. In April 2001, BLM reinitiated direct consultations with the Gila River Indian Community and the San Carlos Apache Tribe, which continued throughout the preparation and evaluation of the EIS.

Publication of the draft EIS was followed by a 60-day public comment period that was extended twice, and during which public and tribal open house meetings were held in four locations. Notices were published in the *Federal Register* and in the *Eastern Arizona Courier.*

The draft EIS informed readers that the 60-day comment period would end on November 25, 1998. During this period, BLM received three requests by electronic or regular mail for an extension of the comment period. BLM notified these three requesters via electronic mail and the public at large by a notice published in the *Federal Register* on November 30, 1998, that the comment period was being extended until December 18, 1998. The public was notified of a second extension of the comment period until January 29, 1999, through a notice printed in the *Federal Register* on December 18, 1998. Thus, the public comment period lasted a total of 127 days.

Three public open house meetings lasting 4 hours each were held at BLM field offices in Safford, Tucson, and Phoenix on, respectively, October 27, 28, and 29, 1998. Representatives from the BLM, ACE, and EPA were present in each meeting. A total of 212 members of the public attended the three meetings. A tribal open house meeting was held on December 4, 1998, on the San Carlos Apache reservation and was attended by 23 persons representing the tribe. Graphic depictions of the mine layout and selected and offered lands, a summary of the impacts of the project, and self-addressed comment sheets were available to the public at these meetings.

Within the comment period, BLM received 269 letters on the draft EIS including two letters originally sent to ACE. Of the total, 127 letters were from private individuals and 142 were from persons representing organizations, groups, businesses, or agencies. The BLM identified 650 comments in these letters.

All comments made in the letters were addressed by BLM. Those comments, and the responses to them, were

Table 16.1-9 Information needed to prepare reclamation and operations plans for surface coal mining in the United States

Natural Factors		Cultural Factors	
Topography	Geology	Location	Type, intensity, and value of use
Relief	Stratigraphy	Accessibility	Agricultural
Slope	Structure	Travel distances	Forestry
Climate	Geomorphology	Travel times	Recreational
Precipitation	Chemical nature of overburden	Transportation networks	Residential
Wind patterns and intensity	Coal characterization	Site size and shape	Commercial
Humidity	Soils	Surrounding land use	Industrial
Temperature	Agricultural characteristics	Current	Institutional
Climate type	• Texture	Historical	Transportation/utilities
Growing season	• Structure	Land-use plans	Water
Microclimatic characteristics	• Organic matter content	Zoning ordinances	Population characteristics
Altitude	• Moisture content	Land ownership	Population
Exposure (aspect)	• Permeability	Public	Population shifts
Hydrology	• pH	Commercial or industrial	Population density
Surface hydrology	• Depth to bedrock	Private or residential	Age distribution
• Watershed considerations	• Color	Type, intensity, and value of use	Number of households
• Flood plain delineations	Engineering characteristics	Agricultural	Household size
• Surface drainage patterns	• Shrink–swell potential	Forestry	Average income
• Runoff amounts and qualities	• Wetness	Recreational	Employment
Groundwater hydrology	• Depth to bedrock	Residential	Educational levels
• Groundwater table	• Erodibility	Commercial	
• Aquifers	• Slope	Industrial	
• Groundwater flow amounts and qualities	• Bearing capacity	Institutional	
• Recharge potential	• Organic layers		
	Terrestrial ecology		
	Natural vegetation		
	• Characterization		
	• Uses and survival needs		
	Crops		
	Game animals		
	Resident and migratory birds		
	Rare and endangered species		
	Aquatic ecology		
	Aquatic animals		
	Aquatic plants		
	Aquatic life systems		
	• Characterization		
	• Uses and survival needs		

Source: Adapted from Ramani et al. 1990.

organized into 14 categories. Discussion of the categories and the responses to the letters required 95 pages in the EIS. A review of that material showed that those letters came from the following sources:

- Fifteen from other government agencies, requesting clarification or further response to specific issues
- Seven from individuals, objecting to specific aspects of the project or to the project in general
- Five from nongovernmental organizations, objecting to specific aspects of the project or to the project in general
- Five from Indian tribes or related organizations, objecting to specific aspects of the project or to the project in general
- One from an Indian tribe stating no objection to the project
- One from PD, requesting clarification or further response to specific issues

The remaining 235 letters expressed support for the project or for the proposed land exchange.

In addition to the 269 letters received during the comment period, 24 letters were received after the close of the extended comment period on January 29, 1999. These letters are not included in the official administrative record for this project; however, all were read by BLM and analyzed for comments. The BLM determined that all the issues and comments raised in the late letters had already been raised by other commenters and were being considered in the preparation of the final EIS.

In August 2001, PD submitted an updated MPO that addressed several concerns raised during the draft EIS public comment period. Specifically, PD modified the crushing, pretreatment, and material handling elements of the MPO to reduce impacts and increase the efficiencies of the ore production processes. As a result of PD's continuing optimization efforts, the projected water usage, truck haulage and associated air emissions, tank storage, and sulfuric acid truck deliveries were meaningfully reduced.

In July 2004, the BLM issued a Record of Decision (ROD) in favor of the land exchange, allowing the project to move forward (BLM 2004). In August the San Carlos Apache Tribe and Western Mining Action Project (WMAP) claimed that the land exchange between BLM and PD did not benefit the public interest and should not take place, and both parties filed protests to the ROD. WMAP claimed that BLM failed to meet its responsibility to ensure that springs and water holes will not be dewatered, instead saying, "Groundwater pumping and aquifer drawdown from the proposed mine will significantly reduce and/or eliminate a number of springs and water holes in the area" (Jones 2004a). The same groups also questioned the validity of PD's mining claims on selected lands

Table 16.1-10 Scoping issues addressed in the Dos Pobres/San Juan Environmental Impact Statement

Land Use	Physical Resources	Biological Resources	Cultural Resources	Socioeconomic Resources	Indian Trust Resources
Public lands management	Climate	Vegetation	Archaeological resources	Population and demographics	Indian trust assets
Access and recreation	Air quality	Wildlife resources	Traditional cultural properties	Local and regional economy	
Encumbrances	Geology	Special interest species/critical habitat		Infrastructure	
Agriculture and grazing	Soil	Biodiversity		Transportation	
Mineral rights	Groundwater quality/ quantity				
Surface water rights	Surface water quality/ quantity (including legally constituted surface waters of the United States)				
Noise and vibrations					
Visual resources					
Hazardous materials					

Source: Data from BLM 2004.

in the Gila Mountains, saying the validity of the claims was assumed by BLM when it prepared the EIS and the ROD. The WMAP protest accused BLM of basing the EIS and ROD on the assumption that the mine was going to be built "without determining if the projects would comply with federal law" (Jones 2004b).

By September, BLM had responded to the protests and allowed the ROD to stand. PD proceeded to file for air quality and aquifer protection permits and began the final feasibility study for the project (Stockton 2005). The feasibility study was completed in 2006. Construction at the site began in June of 2007, and the first cathode copper was produced on December 31, 2007 (Cooper 2008).

Public Participation in the Review of Permits

The environmental regulations in most jurisdictions provide an opportunity for public participation in the permitting process. Notice of the submittal of the permit application is usually advertised locally, with information provided on the schedule for public comment, the availability of the application for public review, and the location where comments may be submitted and hearings requested.

Interested persons or organizations, particularly those who may be adversely affected by a proposed mine, have the right to submit comments and objections to the issuance of the permit during the public comment period. Requests for hearings or conferences on the application must also be submitted within the time provided for public comment. In most jurisdictions, affected persons who file comments on the application, and the applicant if the permit is denied, may appeal the agency's decision through an administrative or judicial hearing. In an administrative hearing, evidence will be accepted, and the administrative law judge or review panel will render a decision to uphold or overrule the agency determination. Additional appeals from a final administrative decision can be brought to appropriate state or federal courts if desired by an adversely affected party.

Recently, there has been an increase in opposition to mining projects in developing countries, from individuals and environmental organizations in developed countries. These organizations are well funded and expert in using legal maneuvering to delay or prevent projects they oppose. Legal delays are but one tactic used to prevent development projects (Olpin 1991).

One example of these organizations is Earthworks, which is "a non-profit organization dedicated to protecting communities and the environment from the destructive impacts of mineral development, in the U.S. and worldwide" (Earthworks 2009). Earthworks is the sponsor of the No Dirty Gold campaign, which has organized opposition to mines proposed in Canada, Costa Rica, Ghana, Papua New Guinea, Peru, Romania, and the United States (No Dirty Gold 2009). The No Dirty Gold campaign has been particularly effective in persuading prominent retail jewelers to pledge not to use gold produced from mines that the campaign judges to be noncompliant with its "Golden Rules," which are as follows:

1. Respect basic human rights outlined in international conventions and law.
2. Obtain the free, prior, and informed consent of affected communities.
3. Respect workers' rights and labor standards, including safe working conditions.
4. Ensure that operations are not located in areas of armed or militarized conflict.
5. Ensure that projects do not force communities off their lands.
6. Ensure that projects are not located in protected areas, fragile ecosystems, or other areas of high conservation or ecological value.
7. Refrain from dumping mine wastes into the ocean, rivers, lakes, or streams.
8. Ensure that projects do not contaminate water, soil, or air with sulfuric acid drainage or other toxic chemicals.
9. Cover all costs of closing down and cleaning up mine sites.
10. Fully disclose information about social and environmental effects of projects.
11. Allow independent verification of the above (No Dirty Gold 2009).

Clearly, these rules are quite similar to the principles of sustainability adopted by ICMM. However, some of the rules are particularly difficult for mine operators. For example, Rules 4, 5, and 6 offer no consideration of the fact that ore deposits are located without respect to political conditions, protected areas, or human communities. Similarly, the independent verification in Rule 11 may be difficult to attain. Career environmentalists are often mistrustful of

government agencies and mining company personnel, so "independent verification" implies inspection and approval by someone selected by the environmental organizations. Unfortunately, these organizations do not get increased financial support by giving approval to mining and other resource development projects; quite the contrary, a mining company may be reluctant to allow "independent verification" of this type.

The direct involvement of organizations like Earthworks, Greenpeace, Oxfam, and others in the permitting process makes it even more important for mining companies to form good relationships with local people and governments well before permit applications are filed. An excellent example of a project where this was done successfully is the Diavik diamond mine, described in the following example.

Example 4. Diavik Diamond Mines Inc.

The Diavik diamond mine operates in the Northwest Territories of Canada, one of the most untouched and ecologically sensitive environments in the world. The mine is surrounded by tundra and is home to bears, wolverines, and migrating caribou. The water of nearby Lac de Gras is exceptionally pure and hosts many species of fish and birds.

Development of the Diavik mine required about 10 years. The first claims were staked in 1991 and 1992. Sampling and exploration continued, and Diavik Diamond Mines Inc. was formed in 1996. In 1997, baseline environmental studies were completed. As required by Canadian law, a project description was submitted in March 1998, and the environmental assessment report followed in September of the same year. Regulatory and financial approvals were given in 1998 and production began in 2003.

In planning the mine, Diavik consulted extensively with local communities about its operation and effects. Diavik is committed to providing training, employment, and business opportunities to residents in local communities of the Northwest Territories and the West Kitikmeot region of Nunavut, ensuring that it leaves a legacy of economically and socially stable local communities in the region. The process followed in developing agreements with indigenous peoples is described by Prest (2002):

> At the start, we developed a shared vision with our communities with regards to project development. As part of that vision, right from the beginning we integrated social and environmental dimensions into our project plans.
>
> We had regular meetings with our all neighboring aboriginal and other communities. We incorporated traditional knowledge learned from community elders and members. We are proud to say that information learned from our aboriginal communities has definitely led to a better designed project.
>
> As a third step in the engagement process, we entered into a number of formal agreements for the project. There are three legs to this framework, the Environmental Agreement, the Socio-Economic Monitoring Agreement, and, separate Participation Agreements with each of our five aboriginal groups.
>
> [Diavik's] Participation Agreements are legally binding, private contracts which contain mutually beneficial provisions for both the company and individual aboriginal communities. Parts of these agreements address shared responsibility for training, employment and business participation.

The negotiations with indigenous communities were conducted according to the traditions of those communities. Diavik's representatives met with each community and discussed all the issues until a consensus was reached. Reportedly, this sometimes took several days (R. Davey, personal communication).

Diavik's commitment to community development and investment is evident in the Diavik Socio-Economic Monitoring Agreement (SEMA), which was formalized in 1999 between Diavik Diamond Mines Inc. and the Government of Northwest Territories. The agreement was ratified by local indigenous communities and outlines Diavik's commitment to provide training, employment, and business opportunities to northerners and, more specifically, indigenous northerners. Diavik individualized these commitments through participation agreements negotiated with each of the same five aboriginal groups (Diavik 2009b):

1. Tlicho Government
2. Yellowknives Dene First Nation
3. North Slave Metis Alliance
4. Kitikmeot Inuit Association
5. Lutsel K'e Dene First Nation

Commitments under SEMA are reported publicly twice a year. Diavik has consistently exceeded expectations outlined in SEMA:

- Diavik committed to supporting a 40% northern work force during construction. At its conclusion, Diavik had reached 44% northern employment.
- Diavik committed to reaching levels of northern purchasing of 38% during construction. At its conclusion, Diavik had actually reached 74%, representing about Can$900 million (US$853 million) in contracts with northern companies, of which approximately $600 million was with indigenous companies.
- During operations, Diavik had committed to purchasing 70% of its annual requirements for goods and services from northern companies.
- For operations, Diavik committed to 66% northern employment and 40% indigenous employment for its operations. In 2007, Diavik's work force averaged 785 people, 67% of whom were northern, with one-half being indigenous.

The mine meets its environmental protection commitments through a comprehensive management system. This system is ISO 14001:2004 certified and is responsible for managing environmental protection activities, ensuring that all employees are properly trained, anticipating and avoiding environmental problems, ensuring regulatory compliance and due diligence, and ensuring consistency with the corporate environmental policy of the company (Diavik 2009a).

Diavik has several active programs that contribute to environmental protection and sustainable development values. In some cases, the measures taken in these programs go beyond the strict legal requirements that apply to the project area. The programs include the following:

- **Caribou management.** A small portion of the Bathurst caribou herd passes through the Lac de Gras region during

spring and fall migrations. To protect caribou passing near the mine, haul roads display advisory signs to ensure that caribou and other wildlife have the right of way. Diavik annually monitors the caribou within the region, with the assistance of elders from local indigenous communities.

- **Water quality management.** During early meetings with local communities, the indigenous people emphasized the importance of preserving water quality in the lakes and drainage. Diavik constructed an extensive water collection system to help protect the surrounding lake waters. Sumps, pipe networks, storage ponds, and reservoirs collect runoff water, which is used in processing and can then be treated before release to the environment. In addition, the Aquatic Effects Monitoring Program ensures that lake water is regularly sampled and analyzed at set locations over the complete range of depth, in all seasons of the year.
- **Fish habitat.** The mine constructed rock-fill dikes to access the three ore bodies under the lake, so that mining operations occupy less than 0.5% of the area of Lac de Gras. However, even this small area removed some fish habitat, so Diavik constructed a coarse rock-fill area on the outside of one dike to provide new fish habitat. During mining, rock shoals are constructed inside the dike area; when the mine closes, these will become fish habitat. The area behind the dikes will be flooded, the dikes will be breached to locate islands, and the area returned as part of Lac de Gras. Thus there will be no net loss of fish habitat, as required by Canadian fisheries regulations.
- **Progressive reclamation.** The progressive reclamation practiced at Diavik uses knowledge gained through community consultation to prepare the mine site for eventual closure. Some examples include the contouring of country rock piles to create smooth hills that allow caribou safe access, and the creation of new fish habitat. Revegetation studies and waste rock management projects are also being conducted.
- **Closure security.** Diavik is committed to complete closure of the mine site. It has committed a substantial amount of financial security to guarantee full closure when mining ceases, as established in the environmental agreement. The value of this security is revised regularly to adjust for the projected environmental liability and its associated cost of closure versus efforts the company has made to progressively reclaim the site.
- **Sustainable development research.** Diavik is cooperating with Canadian universities and researchers in numerous scientific studies of the environment and geology at the mine site. Recent studies have included research into effects of mine blasts on fish, evaluations of potential plant species for reclamation, and monitoring of dust distribution using lichen as a bioindicator. The results of these research projects are published regularly and are available on the Internet (Diavik 2009b). Research results are routinely incorporated into operating and reclamation practices at the site. Diavik is also examining the effect of the arctic climate on diamond waste rock piles, an area of which little is known. This research will facilitate long-term protection of the environment and has long-term applications throughout the world.

Mine Planning

All of the development activities described to this point constitute *mine planning*. However, the term as used here refers specifically to the planning of the mine workings, including gaining access to the mineral deposit, removing the valuable mineral, and handling any waste material produced. The activity as comprehended here also includes tasks described as *mine design*. Although the details of these processes are described in the next section, planning for them is begun in the development stage, and is thus discussed at this point.

In mine planning, the specific day-to-day operations of the mine are set out, analyzed, and documented. Mine planning continues throughout the life of every mine to account for changing geologic and economic conditions. The objective of mine planning at each point in time is to determine the manner in which the mine should be operated to optimize the goal(s) of the operating entity, usually the return on investment. Although mine planning varies with site-specific conditions, the following points must be determined in the initial plan, and continually reevaluated throughout the life of the mine:

- The extent of the mineral deposit
- The market for anticipated products of the mine
- The manner in which the deposit will be accessed for extraction
- The equipment and personnel that will be used for extraction
- The operating sequence for extraction
- The disposition of extracted material, both waste and valuable mineral
- The interactions of mine workings with adjacent or overlying areas and properties

Initial mine planning will obviously take place concurrently with the environmental planning and permitting, and the consideration of environmental consequences of the initial mine plan will be included in this process.

Extraction

Mining methods may be classified according to a variety of schemes. Here it is convenient to distinguish surface mining, underground mining, aquatic or marine mining, and solution mining. Solution mining is the removal of valuable minerals from in-place deposits by dissolving the mineral in a suitable liquid that is then removed for recovery of the desired constituent. It may also be called in-situ mining or in-situ leaching. Solution mining uses techniques similar to those used in the extraction of petroleum and natural gas, and is thus not discussed further in this chapter. The operational details and environmental considerations of surface mining, underground mining, and aquatic or marine mining are discussed separately.

Surface Mining

Surface mining is the removal of material from the earth in excavations that are open to the surface. In some cases, material is removed directly from the earth's surface; for example, sand and gravel may be removed directly from deposits of those materials put in place by ancient lakes or rivers. In other cases, material that has no economic value (overburden) covers the valuable material and must be removed.

A typical operational cycle for surface mining, described in detail by Saperstein (1992), is as follows:

1. Install erosion and sedimentation controls.
2. Remove topsoil from areas to be mined.

3. Prepare the first drill bench by leveling the bench with a bulldozer, inspecting and scaling the highwall as required, and laying out the blastholes.
4. Drill the blastholes.
5. Blast the rock.
6. Load the fragmented material.
7. Haul the fragmented material—waste to the waste dumps and product to the load-out for sale or to subsequent processing.
8. Manage the waste dumps as required by contouring waste piles to a stable configuration or returning waste to mine workings for use in reclamation.
9. Prepare mine workings for reclamation by
   a. Recontouring to the original contour, or to another approved and stable configuration.
   b. Returning the stored topsoil to the mine site and spreading it uniformly on the recontoured surfaces.
10. Reclaim the prepared surfaces by
    a. Revegetating with an approved mixture of seeds and plantings.
    b. Irrigate and maintain revegetated areas as required until a stable condition is reached.
11. Remove temporary drainage controls and stream diversions.

The environmental considerations that must be addressed during surface mining include the following:

- Control drainage from mine workings to avoid contamination of surface water, groundwater, and the existing ecosystem. Treatment may be required to remove particulates or chemical contamination.
- Control inflow or surface water and storm runoff so these waters are not contaminated by passing through the mine workings.
- Carefully analyze the groundwater regime in and around the mine workings, and provide monitoring wells as required. In some cases, it may be advisable to minimize the interaction of the mine workings with groundwater by pumping out the aquifers in and around the mine. There are many potential uses for this water. For example, it might be used in the processing of ore, discharged directly to surface drainages or discharged to constructed wetlands on the surface.
- Consider the formation of "pit lakes" when mined-out surface workings fill with water. This water, which may originate from surface runoff or flow from underground aquifers, may become acidified or otherwise contaminated, and the effects of such contamination on the environment (including wildlife, plant life, and humans) must be managed.
- Design and construct waste dumps and tailing impoundments to minimize erosion and protect surface and groundwater.
- Control emissions of dust and noise to meet the requirements of local regulations and the reasonable expectations of persons living nearby. In particular, if explosives are used, control air blast and ground vibration as required.
- When mining is done at night, consider the effects of artificial light on wildlife and humans living near the mine.
- Control slope stability and landslides. Of course, this is also necessary for successful mining operations, but inadequate control of slope stability may also lead to environmental damage. Slope failure may block or contaminate streams, damage wildlife habitat, and cause flooding that endangers wildlife and humans living near the mine.
- Consider the effects of increased traffic to and from the mine site, especially when product haulage will result in a marked increase in heavy truck traffic.

Underground Mining

Underground mining is the removal of material from the earth in excavations below the earth's surface. Access to such underground workings may be gained through a drift or adit, a shaft, or a slope. Drifts and adits are horizontal tunnels, usually in a hillside, that connect to the mineral deposit. In the case of minerals like coal that occur in seams (near-horizontal deposits of fairly uniform thickness and considerable areal extent), the drift may be developed in the mineral itself. Travel in drifts is by rail or rubber-tired vehicles. Shafts are vertical tunnels developed from the surface to access mineral bodies below. Travel in shafts is by cages or cars, which are lowered and raised by a mechanism on the surface, similar to elevators in tall buildings. Slopes are tunnels neither vertical nor horizontal and may be lineal or spiral. Travel in lineal slopes may be by rubber-tired vehicles or hoists but rarely by rail. Travel in spiral slopes (also called ramps) is by rubber-tired vehicles.

Saperstein (1992) has described in detail a typical operational cycle for underground mining in which material is fragmented by drilling and blasting, as summarized:

1. Enter the workplace after the previous blasting round is detonated.
2. Ensure that the workplace is in good condition and safe for continued work by checking that ventilation is adequate and that blasting fumes have been removed; providing for dust suppression; checking for the presence of hazardous gases; and inspecting for and removing loose material.
3. Load the fragmented material.
4. Haul the fragmented material to the appropriate location.
5. Install ground support as required.
6. Extend utilities as required: ventilation, power (electricity or compressed air), and transportation.
7. Survey and drill blastholes for the next round.
8. Load the explosives and connect the detonation system.
9. Leave the workplace and detonate the round.

This cycle is typical for the mining of narrow, steeply dipping veins (typical of metal ores), and for massive deposits of limestone, salt, and dimension stone, where the vertical extent of the valuable mineral is 6 m (6.6 yd) or higher.

The operational cycle for methods in which material is mechanically fragmented and removed in a continuous process may be similarly summarized:

1. Enter the workplace after the required ground support has been installed.
2. Ensure that the workplace is in good condition and safe for continued work by checking that ventilation is adequate and that blasting fumes have been removed; providing for dust suppression; checking for the presence of hazardous gases; and inspecting for and removing loose material.
3. Cut and load the fragmented material until the limit for advance is reached.
4. Concurrently, haul the fragmented material to the appropriate location.

5. Remove fragmentation, loading, and hauling equipment from the workplace.
6. Install ground support as required.
7. Extend utilities as required: ventilation, power (electricity or compressed air), and transportation.
8. Survey as required to ensure that the mined opening is maintaining the required dimensions and directional orientation.

This cycle is typical for mining in near-level seams of relatively soft material that have considerable areal extent and are thinner than 6 m (6.6 yd), such as coal, trona, phosphate, and potash.

Seven environmental considerations must be addressed during underground mining:

1. Locate ventilation fans and hoist houses to minimize the effects of noise on wildlife and humans living nearby.
2. Whenever possible, dispose of mine waste in underground mine workings.
3. When waste dumps are constructed on the surface, design them to minimize erosion and protect surface and groundwater.
4. Control drainage from mine workings to avoid contamination of surface water, groundwater, and the existing ecosystem. Treatment may be required to remove particulates or chemical contamination.
5. Control inflow or surface water and storm runoff so these waters are not contaminated by passing through the mine workings.
6. Carefully analyze the groundwater regime in and around the mine workings, and provide monitoring wells as required. In some cases, it may be advisable to minimize the interaction of the mine workings with groundwater by pumping out the aquifers in and around the mine, and discharging the pumped water to constructed wetlands on the surface.
7. Analyze and predict the subsidence likely to result from mine workings. Design workings to minimize the effects of subsidence on surface structure, utilities, and important natural features such as lakes, streams and rivers, and wildlife habitat.

Aquatic or Marine Mining

Aquatic or marine mining is the removal of unconsolidated minerals that are near or under water, with processes in which the extracted mineral is moved by or processed in the associated water. This type of mining may also be referred to as alluvial mining or placer mining. The two major types of aquatic or marine mining are dredging and hydraulicking.

Materials typically recovered by aquatic or marine mining have usually been deposited in fluvial, aeolian, or glacial environments, and are thus unconsolidated. They include aggregate (sand and gravel) and materials deposited because of their relatively high specific gravities. The latter deposits are called placers and include native (naturally occurring) precious metals, tin (as the oxide cassiterite), heavy mineral sands (oxides of zirconium, hafnium, titanium, and others), and precious stones. In some cases, placer deposits are mined by methods described in the section on surface mining; in such cases, the environmental precautions given for those methods apply.

Dredging is the use of a powered mechanism to remove unconsolidated material from a body of water. The mechanism is almost always a type of bucket or shovel. In the simplest case, it may be a metal bucket moved by chains or steel cables that are attached to a pole. The bucket is dropped through the water and into the solid material on the bottom. As the bucket is retracted, its weight and trajectory force its leading edge into the solid material, and the bucket fills. When the bucket comes to the surface, it is emptied. This type of dredge is usually installed on the shore near a body of water that covers a valuable mineral deposit.

More complex dredge mechanisms attach several buckets to a wheel or a ladder. A ladder is a structure designed to support a series of buckets attached to a chain, which moves continuously in a loop. Both mechanisms are usually installed in a floating vessel, the dredge. While moving, the bucket wheel or the dredge ladder is lowered into the unconsolidated material below the surface, picking up that material and returning it to the dredge where it is either further processed or transferred to the shore for further usage. The dredge vessel may operate in a natural body of water or in an artificial body called a dredge pond. A typical operational cycle for dredging is comprised of the following steps:

1. If dredging in a dredge pond, complete subitems a–g; if not, proceed to Step 3.
   a. Locate a source of water for filling the pond.
   b. Install erosion and sedimentation controls, and divert surface water as required.
   c. Prepare dikes or dams required for the dredge pond. The perimeter of the dredge pond must extend beyond the extent of the mineral to be recovered to allow for deposition of overburden or tailings removed in the first pass of the dredge.
   d. Fill the dredge pond.
   e. Remove vegetation.
   f. Remove and stockpile topsoil.
   g. If necessary, remove overburden with excavating equipment, and place overburden in stable piles.
   h. Proceed to Step 3.
2. If dredging from the shore, prepare the site for installation of the dredge mechanism.
3. Install the dredge.
4. Remove by dredging any overburden that was not removed in Step 1g.
   a. When dredging in a natural body of water, the overburden will be deposited under the water in an area from which the valuable mineral has already been removed or in an area under which there is no valuable mineral.
   b. When dredging in a dredge pond or from the shore, the overburden will be deposited beside or behind the dredge in an area from which the valuable mineral has already been removed or in an area under which there is no valuable mineral. When mining in a dredge pond, the overburden will for a time be at least partially under water; when mining from the shore, it will not. However, in both cases, the overburden will over time drain and be left exposed. It will thus require reclamation as described in Steps 12 and 13.
5. Remove the valuable mineral by dredging.
6. Process the valuable mineral as required to concentrate the valuable constituent(s). This processing is almost always done on the dredge and is integrated with the dredging of the material.

   a. When dredging in a natural body of water, the tailings will be deposited under the water in an area from which the valuable mineral has already been removed or in an area under which there is no valuable mineral.
   b. When dredging in a dredge pond or from the shore, the tailings will be deposited beside or behind the dredge in an area from which the valuable mineral has already been removed, or in an area under which there is no valuable mineral. When mining in a pond, the tailings will for a time be at least partially under water; when mining from the shore, it will not. However, in both cases, the tailings will over time drain and be left exposed. They will thus require reclamation, as described in Steps 12 and 13.
7. Remove the valuable constituent(s) for sale or further processing off-site.
8. Deposit the tailings under the water in an area from which the valuable mineral has already been removed or in an area under which there is no valuable mineral.
9. Continue dredging until reaching the limits of the dredge's cables and other connections, the limits of the pond, or the boundary of the deposit.
10. If dredging in an artificial pond, prepare and fill the next pond as in Step 1, and transfer the dredge. If dredging in a natural body of water, move the dredge to the next location.
11. Begin dredge operation in the new location by returning to Step 3. Simultaneously, and in accordance with the approved reclamation plan, reclaim overburden and tailings piles from previous work, according to Steps 12 and 13.
12. Prepare mine workings for reclamation:
   a. Recontour to the original, stable contour, or to another, approved and stable configuration.
   b. Return the stored topsoil to the mine site, and spread it uniformly on the recontoured surfaces.
13. Reclaim the prepared surfaces:
   a. Revegetate with an approved mixture of seeds and plantings.
   b. Irrigate and maintain revegetated areas as required until a stable condition is reached.
14. Remove temporary drainage controls and stream diversions.

There may be considerable variation in the cycle, depending on how the dredge is transported between sites.

Hydraulicking is a method of mining placer deposits that was used extensively in the past but has fallen into disfavor because of the potential for serious effects on the stability of the remaining surface and on nearby surface waters.

Hydraulicking required a large deposit of auriferous alluvium and a source of water that would provide sufficient volume and pressure head. Overburden and pay gravel were both removed by high-pressure water that flowed through a monitor—a large nozzle that could be rotated horizontally and vertically. Material removed by the flow from the monitor moved to the bottom of the valley, where the flow was controlled so that overburden passed directly into the stream channel and pay gravel flowed through a sluice or similar recovery device for recovery of the ore. Tailings from the sluice also flowed into the stream channel.

As mentioned earlier, hydraulicking of the placer gold deposits in the famous California Gold Rush of 1849 led to severe contamination of the rivers that drained the area of the gold deposit. The solids from the hydraulicking operations filled rivers with solids, which were eventually deposited downstream with serious consequences for agriculture in the downstream areas.

Hydraulicking is still a very inexpensive method for moving large volumes of unconsolidated material and is still used on a limited basis with five precautions:

1. Material discharge from a hydraulicking operation must be captured and treated to remove solids, including, in particular, any fine, suspended solids. This will require a settling pond, and chemical flocculants may also be necessary. Even water that appears to be clear must be tested by appropriate methods and treated on the basis of those tests.
2. Tailings, overburden, and deactivated settling ponds must be reclaimed appropriately, usually by recontouring and revegetation. It may be necessary to remove and stockpile topsoil for use in reclamation. Coarse gravel, which may accumulate in separate piles, should receive special attention to ensure it is reclaimed properly.
3. The area of the mining operation should be isolated from the flow of surface streams and runoff to prevent contamination of those waters.
4. Discharge of water from the hydraulicking operation should be managed to prevent interference with the existing flow regimes in the drainage. The quantities and velocities of discharge should not modify the existing stream flow in a manner that will cause erosion, undercutting of banks, or flooding.
5. Highwalls and embankments produced by hydraulicking should be reclaimed, again by recontouring and revegetation. Even when topsoil is removed and stockpiled, there may not be enough topsoil for the reclamation required. In such cases hydroseeding will likely be required.

CONCLUSION

Activities associated with mining have direct and lasting effects on both the physical and the human environments. In the recent past, mining was done with little concern for its effects on the environment. The consequences of this approach often resulted in significant damage to the natural environment. However, as political and cultural norms changed, and new legal requirements were enacted, almost all major mining companies adopted rigorous policies and procedures for sustainability, community engagement, and environmental risk assessment and mitigation. These companies apply such policies throughout their operations, many of which are worldwide. In addition, many mining companies work actively to remediate environmental damage caused by historic mining operations in areas where they have past or current operations.

Thus, environmental considerations are an important part of the modern mining industry. These considerations include preproject planning, permitting, compliant operation, reclamation, and closure. These considerations must be included in all project planning, and feasibility studies must account for the influence of environmental considerations on project schedules and costs.

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CHAPTER 16.2

Mining and Sustainability

R. Anthony Hodge

INTRODUCTION

The 1987 publication of the report, *Our Common Future* by the United Nations (UN) World Commission on Environment and Development (WCED), brought the concept of sustainable development into the limelight. Chaired by the former Prime Minister of Norway, Gro Harlem Brundtland, and following hearings held across the world, the commission proposed an agenda for world development that would enhance security and reduce North–South disparities. It would be development "which meets the needs of the present without compromising the ability of future generations to meet their own needs" (WCED 1987).

Since then, a rich debate has ensued about what this means in practical terms. Though many other sets of words have been suggested for defining the phrase *sustainable development*, the Brundtland Commission definition has stood the test of time and remains the anchor. For a rich discussion, see the "definitions" portal of the International Institute for Sustainable Development, or IISD (SD Gateway n.d.)

In recent years, the word *sustainability* has also found its way into common use. The idea is simple. Sustainability is the persistence over a long time—indefinitely—of certain necessary and/or desired characteristics of both human society and the enveloping ecosystem (Robinson et al. 1990). These characteristics range from primary needs such as air, water, food, clothing, shelter, and basic human rights to a host of conditions that would collectively be called *quality of life*, not only for people but for other life forms as well.

It is here that the definitional issue becomes difficult for some, because the choice of which characteristics are to be sustained and the degree to which they will be sustained depends on the particular values that are applied. In turn, these depend on who is doing the applying. In other words, it is not a closed definition. What a company CEO chooses as important may be different than a politician, doctor, or librarian; what a Mexican chooses may be different from, for example, a Tanzanian or Australian. Because of these potential differences, a fair and effective process of interaction and seeking consensus is critical to the practical application of these ideas. Herein lies the rationale for why it is essential to always address not only the *what*, or the *substantive* part of human action, but also the *how*, or the *process* part. In other words, in the practical application of sustainable development concepts, not only what we do is important but how we do it.

Thus, the ideas of sustainable development and sustainability are different but synchronous. *Sustainability* is a more general term that captures the idea that we need to maintain certain important aspects of the world over the long term. *Sustainable development* is the human or action part of this set of ideas: As a society, we want to make choices about our actions that allow us to provide for the present without undermining the possibility for future generations to provide for themselves.

Together, these ideas are very appealing. However, their translation to practical action remains much debated. This is not surprising. Human society is complex. There are about 10,000 cells in the standard industrial classification—our way of classifying human activities within the market economy. This does not account for many more activities outside the market economy. There are about 200 countries across the world, and the global ecosystem is complex and not fully understood.

For its part, the mining, minerals, and metals industry has been a particularly active locus of sustainability-related policy and practice innovations because

- The potential implications—both positive and negative—of mining activities and the minerals and metals that result are significant;
- Many interests are touched by mining;
- The role of many of these interests in decision making is growing (e.g., communities and indigenous people);
- The nature of contemporary communications systems has brought the often dramatic nature of mining operations into the public eye; and
- Industry, governments, civil society organizations, and the public, in general, are all anxious to ensure mining makes a positive contribution that is fairly shared.

Importantly, the concept of sustainable development has not disappeared like so many "flavor of the month" ideas. Rather,

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it has grown in prominence and is now deeply entrenched in legislation, government, and corporate policy and practice. Sustainability is the subject of university curricula, given as a label to vice presidents and departments of mining companies, incorporated into the names of service providers, and included as an element of key performance indicators.

This chapter provides an overview that links the ideas of sustainable development and sustainability to the mining, minerals, and metals industry. To do so, a particular template is used for organizing the many interlinked bodies of knowledge that must be brought together: the Seven Questions to Sustainability (7QS) (MMSD North America 2002).

This template is pragmatic though much informed by the theoretical foundation of systems theory. It recognizes that to bring sustainability ideas into practice for the mining, minerals, and metals industry, the hard (well-defined) and the soft (ill-defined) systems of the real world must both be addressed, as well as the objective (independent of judgment) and the subjective (dependent on judgment). In doing so, all must be treated, if not exactly scientifically (which is not always possible), at least in a way that is characterized by intellectual rigor (see discussion of systems theory and sustainability in Hodge 1995 and 1996). The 7QS template weaves together ideas from many disciplines but recognizes that deeper exploration is often warranted depending on site-specific conditions. In this context, the objective of this chapter is to open a door as a first step to practical application.

MINING AND METALS INDUSTRY RESPONSE TO SUSTAINABLE DEVELOPMENT

The 1970s and 1980s were a time of reaction to dramatic change for mining. Echoing increasing concern for the environment across society, the late 1980s saw a number of leading mining companies publish "state-of-environment" reports related to their operations. Taking another important step, 30 leading mining and metals companies from across the world came together in 1991 to create the International Council on Metals and the Environment (ICME). ICME would give the industry an international voice on environmental matters.

Also in the early 1990s, many of these same companies joined with senior governments, labor unions, aboriginal peoples, and environmental nongovernmental organizations (NGOs) in a broad review of mining practices in Canada. The Whitehorse Mining Initiative turned out to be a precursor of a number of initiatives convened to bring sustainability ideas to practical application around the world. The resulting Leadership Accord (Whitehorse Mining Initiative Leadership Council Accord 1993) is a summons to change, framed within the context of a commitment to social and environmental goals. It seeks a sustainable mining industry within the framework of an evolving and sustainable society. The ideas it champions and the multi-interest process it uses elegantly capture sustainable development in practice.

For the mining industry, the decade of the 1990s was a bleak period. Commodity prices dropped while public criticism skyrocketed, much driven by a civil society that was quick to take advantage of newly available and quickly evolving computer-based communications. As a whole, the industry found itself under attack and in a defensive posture. Its *social license* to operate was threatened (though that particular label was to come later). In the late 1990s and faced with growing concern about access to capital, land, and human resources, the chief executive officers of nine of the world's largest mining companies took an unprecedented step.

Working through the World Business Council for Sustainable Development, they initiated the Global Mining Initiative (GMI). As part of GMI, they commissioned the International Institute for Environment and Development (London) to undertake a global review that would lead to the identification of how mining and minerals can best contribute to the global transition to sustainable development. The resulting project, Mining, Minerals, and Sustainable Development (MMSD), sparked a large and rich literature, including the project's final report, *Breaking New Ground: Mining, Minerals and Sustainable Development* (MMSD 2002).

Before the GMI was completed, participants moved to create an organization that would carry the resulting recommendations forward to implementation. Thus, in 2001 and building on the foundation established by ICME, the International Council on Mining and Metals (ICMM) was created. Many of the ideas summarized in this chapter have emerged from or been refined through subsequent work of ICMM.

Almost simultaneously with these events, NGO pressure on the World Bank Group led to the initiation in 2001 of a multi-interest review of the group's involvement in extractive industries. The Extractive Industries Review sought to test whether or not industry projects could be compatible with the World Bank Group's goal of sustainable development and poverty reduction. In its final report, *Striking a Better Balance* (Salim 2003), the review concluded in the affirmative but only if three enabling conditions were in place:

1. Public and corporate governance advocacy for the poor, including proactive planning and management to maximize poverty alleviation through sustainable development
2. Much more effective social and environmental policies
3. Respect for human rights

The resulting refocused World Bank policy emphasizes strengthened governance and transparency, ensuring that benefits reach the poor, mitigating environmental and social risks, protecting human rights, and promoting renewable energy and efficiency to combat climate change. This refocusing has in turn influenced mining approaches to implementing sustainable development on the ground. In sum, this process has served to elucidate and reinforce the concepts of sustainable development addressed in this discussion.

MINING'S CONTRIBUTION TO SUSTAINABLE DEVELOPMENT

At the base of the interlinked ideas of sustainability and sustainable development lies the simple idea that any human activity—including mining—should be undertaken in such a way that the activity itself and the products produced together provide a net contribution to human and ecosystem well-being over the long term.

From an engineering design perspective in general and a mine design perspective in particular, this simple idea gives rise to an overarching two-dimensional design criterion. That is, mining activity (or any human endeavour for that matter) should be designed to achieve (through the activity itself and the products that result) a net contribution to both human and ecosystem well-being over the long term. The achievement of design success should, in turn, be tested against this design criterion.

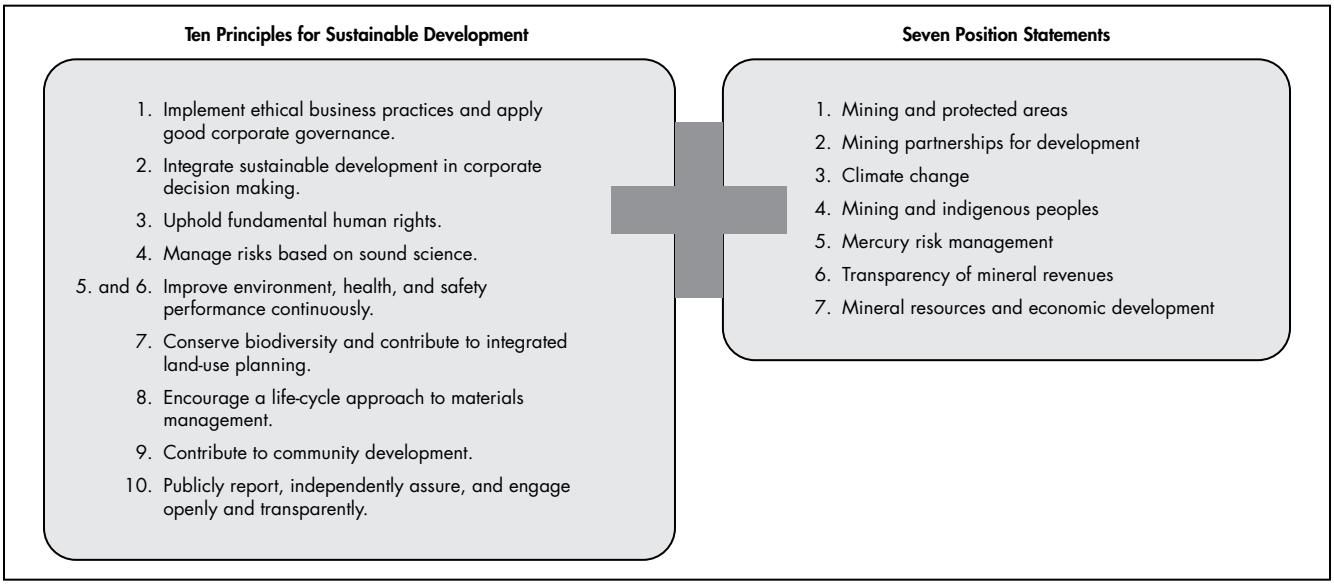


Figure 16.2-1 Principles and position statements that comprise the sustainable development framework of ICMM

Thus, the focus here is not on how mining can be sustainable—any mining project as a discrete activity cannot continue indefinitely—but on how mining, minerals, and metals can contribute to sustainable development. This is a conceptual shift away from a singular analysis and mitigation of *impacts* to a more comprehensive analysis and encouragement of *contribution*.

The focus on contribution is a tougher but fairer approach. It demands consideration of both the good and the bad. (The idea that the mining and metals industry should be designed for and tested against its contribution to the well-being of people and the environment—to sustainable development—was first proposed by Professor Robert Gibson in 2000 and later taken up in development of the 7QS template [MMSD North America 2002; Hodge 2004].) That a mining activity might be challenged to make a positive contribution to the ecosystem over the long term may seem a tough, even impossible challenge to some. However, articulating explicit engineering design criteria in this way sets a design objective that is essential if mining and metals-related activity and the resulting products are to achieve the sustainable development contribution that is being demanded by society.

Application of these ideas is not simply a *greening* phenomenon; it is related as much to well-being and security of people as the environment. And, interestingly, the mining industry's capacity to deal with the environmental aspects is currently stronger than its ability to address the full range of social aspects. This is likely because many environmental issues can be addressed through *hard* scientific and technical solutions, whereas social issues often require *soft* behavioral-type solutions, which can be much harder to design and implement—and which often fall outside the engineer's training.

These ideas also veer sharply away from thinking in terms of a trade-off between human and ecosystem well-being—it is not a balancing act that pits people against the environment. There are obviously many small trade-offs in any practical application: between interests, between components of the ecosystem, across time, and across space. This is particularly the case for mining and the related process of concentrating and refining. However, in an overarching sense, the ideas of sustainability and sustainable development call for both human and ecosystem well-being to be maintained or improved over the long term. Doing one at the expense of the other is not acceptable because, either way, the foundation of life is undermined.

PRINCIPLES AND FRAMEWORKS

The nature of applied sustainability is evident from the many attempts to articulate sustainable development principles. For a listing of more than 100 such principles, see the the "principles" portal of the IISD (n.d.). In short, there is no one-size-fits-all approach to defining, framing, and characterizing the ideas of sustainable development and sustainability. One set of principles of particular relevance to the mining industry is the 10-part set developed by the ICMM, along with its complementary set of position statements. ICMM brings together 19 of the largest mining companies of the world and, on an annual basis, member companies assess performance against these commitments using a procedure that includes third-party independent assurance (ICMM 2010a).

The ICMM principles and position statements as of 2010 are summarized in Figure 16.2-1. Just as many definitions and principles have been proposed, so too have many organizing frameworks been designed to bring theory to action. A comparative analysis of about 30 such characterizations or frameworks is found in Hodge 1997.

Almost three decades before the popularization of sustainable development ideas, geographer Walter Firey pointed out that three broad groupings of knowledge were pertinent to natural resource use (Firey 1960; and see discussion in Hodge 1997):

1. Ecological (environmental)
2. Ethnological (social/culture)
3. Economic

In the late 1980s, Firey's three-part model of natural resource use was adopted by a number of those attempting to operationalize the concept of sustainable development (Mitchell 1991). Since then, the three-part environmental–social–

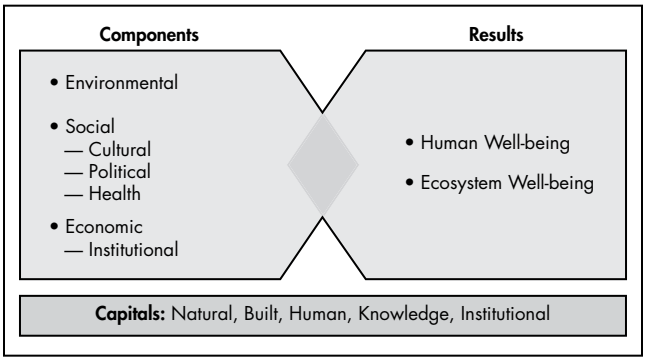


Figure 16.2-2 Different characterizations of sustainable development

economic characterization of sustainable development has gained broad usage. A significant limitation of this approach, however, arises because of the lack of common treatment for the social element.

Many other frameworks have also emerged that make sense for the particular application: population health, healthy communities, sustainable communities, foreign aid, urban design, agriculture, forestry, fisheries, indigenous peoples' needs, and so forth. From one particularly important perspective, the World Bank Group has used the lens and insights of economics to offer a "capitals" model to apply sustainable development ideas: natural, built, human, knowledge, and institutional. Figure 16.2-2 is a synthesis that brings a number of the alternatives together.

Another perspective that offers practical insight comes from an analysis of mine closure options that was undertaken for the Faro mine project in the Yukon Territory, Canada (Hodge and Merkhofer 2008). In this case, a sophisticated multi-attribute utility analysis—driven by a multistakeholder process—was used to assess the alternatives. Multi-attribute utility analysis is a form of decision analysis in which a set of objectives is articulated and each alternative's performance is assessed against those objectives. The underlying principle is that the alternative that best performs against the objectives is the best one. The foundation of this kind of analysis is the articulation of objectives that can be translated to a scale, which can be assessed in terms of a direction (e.g., more is better, less is worse) and magnitude (e.g., how much better or worse).

The rigorous scaling and multi-interest process of assessing and judging are complex. In the Faro analysis, two time horizons were used in the assessment: (1) short, 15–40 years; and (2) long, 500–1,000 years. Of relevance to this discussion is the framework of closure objectives, because it was set to reflect an overarching government policy of sustainable development. Eight objectives were used for assessing closure alternative at the Faro mine (Yukon Territory, Canada):

1. Maximize public health and safety
2. Maximize worker health and safety
3. Maximize restoration, protection and enhancement of the environment
4. Maximize local socioeconomic benefits
5. Maximize Yukon socioeconomic benefits
6. Minimize cost
7. Minimize restrictions on traditional land use
8. Minimize restrictions on local land use

As with all sustainable development frameworks, a broad range of topics is captured. However, this example is particularly useful for another reason. It is clear that the simultaneous treatment of all objectives sets up tensions that must ultimately be resolved: In a comparative analysis of alternatives, different value sets might well judge performance differently and, at the same time, place greater weight on certain objectives. This is a poignant example of applied sustainability in practice. Here again the process of finding common ground can be seen as critical to practical implementation of sustainable development ideas.

From a different but important perspective, as part of the World Bank Group response to the Extractive Industries Review, the International Finance Corporation (IFC) prepared and adopted a sustainability framework that consists of (1) the Policy on Social and Environmental Sustainability, (2) the Policy on Disclosure of Information; and (3) a set of performance standards on social and environmental sustainability.

In its performance standards, the IFC addresses eight topics:

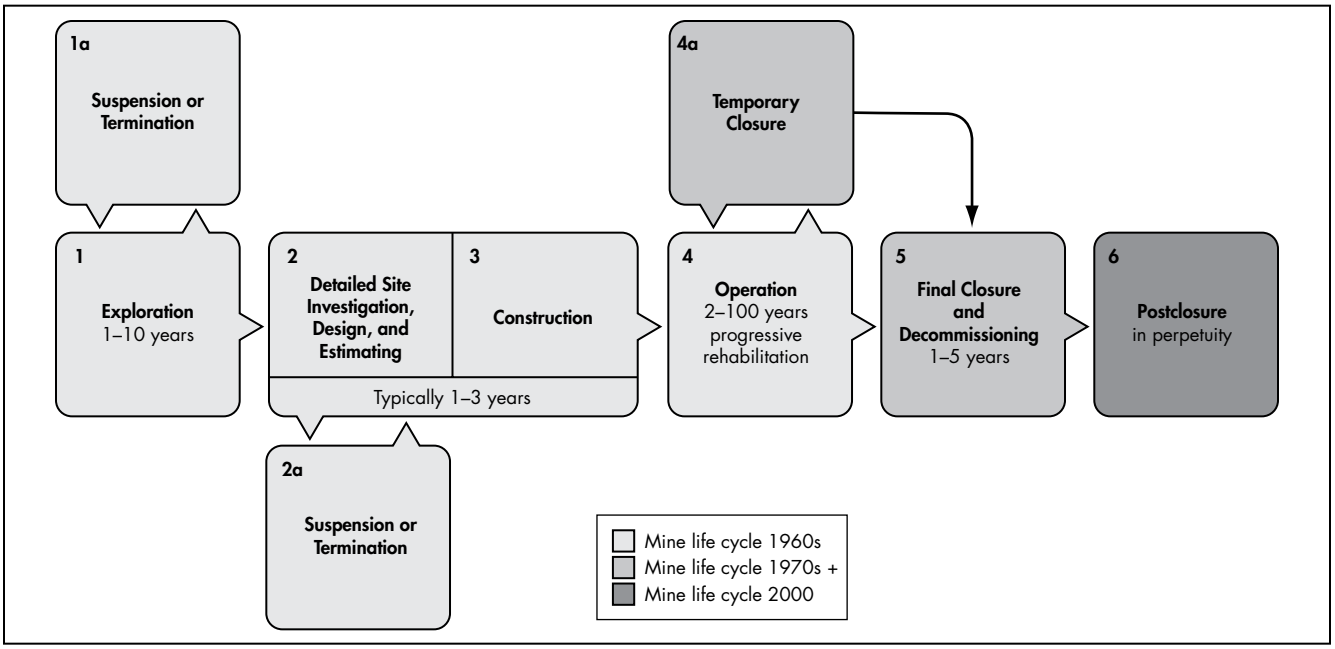
1. Social and environmental assessment and management systems
2. Labor and working conditions
3. Pollution prevention and abatement
4. Community health, safety, and security
5. Land acquisition and involuntary resettlement
6. Biodiversity conservation and sustainable natural resource management
7. Indigenous peoples
8. Cultural heritage

Here again is another definition of what is appropriate to include in addressing application of sustainability concepts to mining. The IFC sustainability framework is currently under review with the expectation that a revised package is targeted for release in 2011. The IFC has also published a number of Good Practice Notes/Handbooks (IFC 2006) relevant to the practical issues of sustainability. Most importantly, the performance standards along with the World Bank Group's Environmental, Health, and Safety Guidelines form the basis of the Equator Principles, which have been adopted by 67 lending institutions worldwide (the Equator Principles lending institutions), many of whom provide financing for mining activities. The Equator Principles are a voluntary set of standards for determining, assessing, and managing social and environmental risk in project financing. They are considered the financial industry gold standard for sustainable project finance.

Another important framework is the Global Compact, a principle-based framework for businesses, which was established by the United Nations in 2000. The Global Compact is the world's largest voluntary corporate citizenship initiative and states 10 principles in the areas of human rights, labor, the environment, and anticorruption. Members are committed to aligning their operations and strategies with the 10 principles (United Nations Global Compact n.d.).

For their part, the Millennium Development Goals are eight international development goals that all 192 UN member states and at least 23 international organizations have agreed to achieve by the year 2015:

1. Eradicating extreme poverty and hunger
2. Achieving universal primary education



Source: Adapted from John Gadsby (personal communication) and MMSD North America 2002.
Figure 16.2-3 Mine project life cycle

3. Promoting gender equity and empowering women
4. Reducing child mortality
5. Improving maternal health
6. Combating HIV/AIDS, malaria, and other diseases
7. Ensuring environmental sustainability
8. Developing a global partnership for development

Each of the approaches to capturing what sustainability means in terms of principles and frameworks is useful for the relevant driving application. There is no single panacea that applies in all cases. However, taken together they capture the breadth of issues and topics that must be considered.

NONRENEWABLE NATURE OF MINING, MINERALS, AND METALS

Much of the literature through the 1980s and 1990s focused on renewable resource management and the idea of living off the interest of a continuing core stock. For some, nonrenewable resource-related activities such as mining simply did not fit into the sustainability concept, although the hat might be tipped toward recycling and reuse of nonrenewable materials as helpful strategies.

One result of this early emphasis was the marginalization, to a great extent, of concerns and perspectives about nonrenewable resources. Interestingly, the fact that minerals are nonrenewable (or stock) resources and, in some sense, fixed in absolute quantity turns out to be relatively unimportant from a sustainability perspective—at least at the macro scale (MMSD North America 2002). The nonrenewable character of minerals received a great deal of attention in the literature from 1950 into the 1970s. However, the long statistical record of continued output at relatively constant prices, together with growing understanding first of environmental issues and then of sustainability, has served to deemphasize this concern. At the same time, recognition is growing that the products of mining are needed both to provide for the world's population and, even more so, to support approaches that will allow society to walk more lightly on the earth. For example, all the strategies needed for development, transmission, and use of renewable energy sources depend on mined metals and minerals. Similarly, strategies to move to a carbon-reduced economy are only possible through creative uses of mined minerals and metals.

Thus, the sustainability-related focus is now appropriately on mining as an activity and its implications for the communities and ecosystem within which minerals are embedded. At any given site, whether a mining, smelting, refining, primary metals manufacturing, or recycling operation, there is a beginning and end: No mining/mineral activity can be expected to have an indefinite life span. However, the implications of that activity (not only as a direct result of the activity but also through the product that is produced) go on indefinitely.

In that sense, mining/mineral activities serve as a bridge to the future. The sustainability challenge is to ensure that the implications of mining activities and the products that result are net positive for people and ecosystems over the long term: It is the well-being of human society and the enveloping ecosystems that need sustaining. Limited-term mining projects can serve sustainability objectives if they are designed and implemented in ways that ensure they meet that challenge.

BOUNDARY CONDITIONS FOR APPLYING SUSTAINABILITY TO MINING

Bringing sustainability ideas into mine design has a significant impact on mine design boundary conditions. Four aspects of boundary conditions apply in this case, each of which has implications for setting the time and space dimensions and helps to identify the system components that must be considered:

1. Mine project life cycle (Figure 16.2-3)
2. Mineral life cycle (Figure 16.2-4)
3. Time horizon (Table 16.2-1)
4. Communities of interest

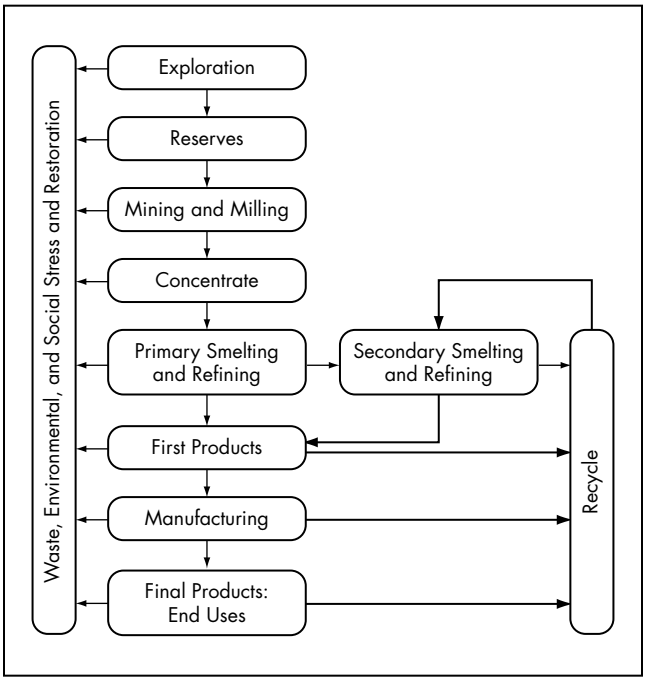


Figure 16.2-4 Mineral life cycle

First, the mine design process must take into consideration the full life cycle of a mine project. Many of the greatest problems facing the mining industry today stem from the fact that this has not been done in the past. In the 1960s, the mine design process was limited to the end of operations. In the 1970s and driven by rising environmental consciousness, the need for mine reclamation gave rise to concerns about closure, which at the time focused on cleaning and grooming a site, with revegetation being a key activity. It is only in the last decade that the realization has set in that a postclosure phase can in some cases extend indefinitely because of the geochemical processes at work in waste rock piles, tailings, and exposed workings. These give rise to potential liabilities that must be factored into annual financial statements and the calculation of share value.

Second, the mine design process must consider the mineral life cycle. The mining industry has come to learn that it must not only consider the production of minerals and metals but also their use. Too often in the past, the mine design process has limited its perspective in time and space to the immediate mine operation. In fact, many significant implications extend across space and time in a kind of ripple effect (Figure 16.2-5). Some of these—both positive and negative—are significant. And all must be taken into consideration to ensure that the full contribution is accounted for.

Third, the time horizon to be used needs particular mention. Table 16.2-1 lists different perspectives on the time horizons relevant to applied sustainability. This element of the boundary conditions for the mine design, operation, and closure processes has changed dramatically in the last few decades. The mine project life cycle shown in Figure 16.2-3 illustrates how the time horizon evolved after the 1970s to eventually take into considerations the postclosure phase, which can sometimes stretch environmental and social obligations of mining operations into perpetuity. This is new terrain for all interests—industry, government, host communities, and civil society organizations—and is not yet fully understood or appreciated.

Table 16.2-1 Perspectives on the time horizons for applied sustainability

Perspective	Typical Time Horizon
Financial/tax cycle	Annual or quarterly
Election cycle	2, 4, or 5 years
Standard engineering design life	Up to 50 years
Social time horizon (seven generations)	Up to 200 years
Environmental time horizon	200 to 10,000+ years

Source: Adapted from Freeze 1987.

The fourth and final aspect of defining the boundary conditions for applying sustainability relates to identifying the communities of interest that must be considered in the mine design, operation, and closure process. In times past, a company and government would simply come to a bilateral agreement on conditions that would govern mine activities. Today that is no longer the case. Many interests play active roles, and the definition of those roles itself requires great care and attention. Interests important to mining are

- Industry (investors, employees, industry associations, other companies);
- Support services (financial, consultants, contractors, suppliers);
- Government (federal, state/provincial, county/regional district, local);
- Indigenous people and their organizations;
- Organized labor;
- Mining-affected communities (by economic, social, and/or environmental [e.g., watershed] dependency);
- NGOs or civil society organizations; and
- Academic, learning, and research and development support (universities, technical schools, private and public research centers).

SEVEN QUESTIONS TO SUSTAINABILITY

One product of the mining industry's MMSD project was a template aimed at assessing the compatibility between mining operations and sustainability criteria (MMSD North America 2002; Hodge 2004, 2006). The 7QS offers seven queries for consideration in the mine design and assessment process. Each question is the interrogative form of a goal statement. Seen in another light, these seven goals define the application of sustainability concepts. The technique of using the interrogative form in this way is drawn from the accounting profession's approach to auditing and assuring the validity and accuracy of financial statements.

The focus on applying the 7QS approach is not on how a given mine can be sustainable—mining as a discrete activity cannot continue indefinitely—but on how the process of mining and the products it produces can best contribute to sustainable development. Thus it enshrines the conceptual shift away from analysis and mitigation of *impacts* to analysis and encouragement of *contribution*.

The seven questions are summarized in Figure 16.2-6. In this section, the 7QS template is used to systematically organize the parts of applied sustainability, rendering their application more practical. (The material presented in this section is modified from MMSD North America 2002.) In applying this

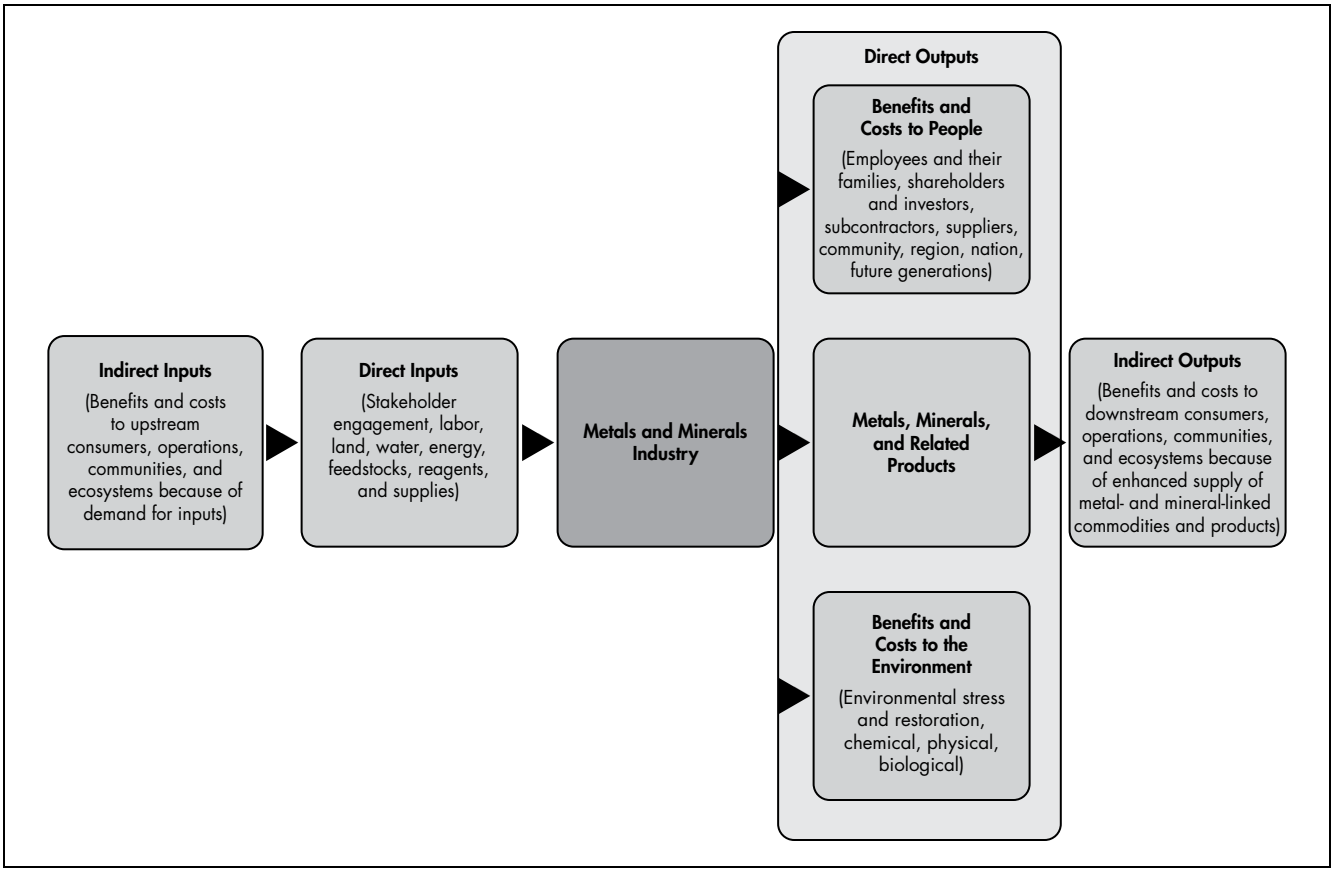


Figure 16.2-5 Ripple effect

template, attention must be paid to the compilation and analysis of both quantitative and qualitative insights. Ultimately in assessing effectiveness, efficiency, and progress, four distinct steps are essential (Hodge 2007):

1. Drawing the qualitative insight that comes through understanding the **story** that is relevant to the object(s) of the assessment
2. Undertaking and compiling the relevant quantitative **measurement**
3. Applying a systemic approach to synthesizing, setting criteria, and **judging** significance
4. Effectively **communicating** the results to different key interests

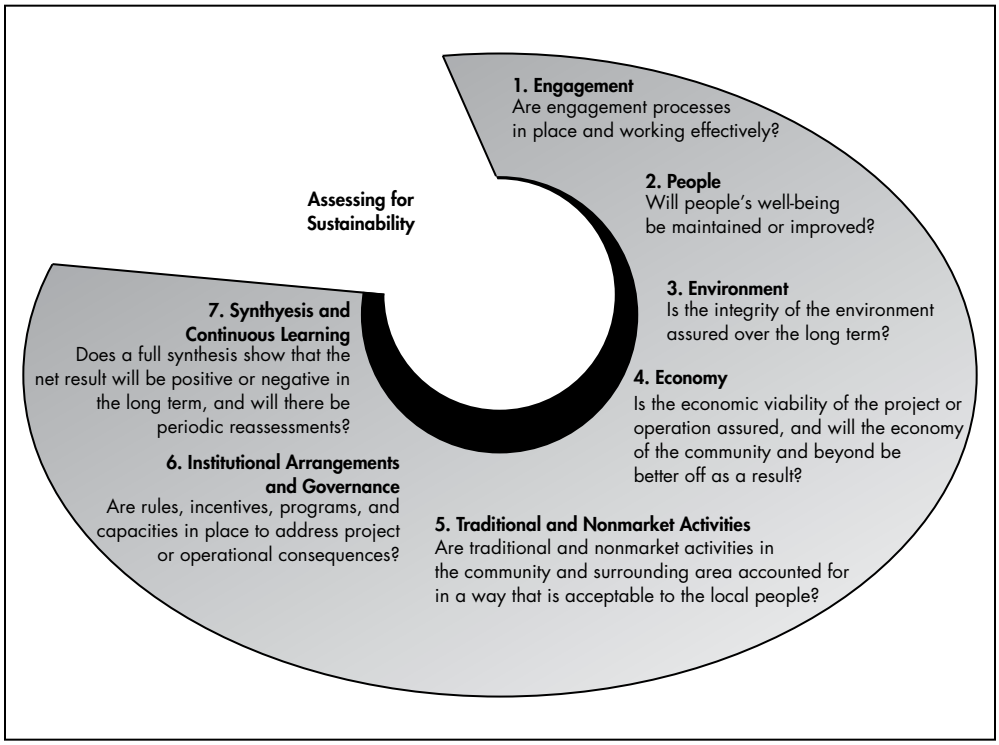
The 7QS approach encompasses four categories of insight:

1. **Relationships.** Question 1 in Figure 16.2-6 (engagement) deals with the state of relationships that are important to any given project or within any region that is being assessed (see Thomson and Joyce 2000 for a succinct discussion of this topic). A key issue facing many projects is the sense that the distribution of costs, benefits, and risk is unfair. This sense can cause stakeholder reactions that range from feelings of discontent to outright civil disobedience and damage to persons and property. Although detailed procedures will vary from site to site, the 7QS approach calls for addressing this issue early and in a way that facilitates constructive relationships achieved over the full project life cycle.
2. **Ends.** Questions 2 (people) and 3 (environment) focus on the end results that must be achieved and against which the success of any project must be tested—human and ecosystem well-being over the long term.
3. **Means to achieving ends.** Questions 4 (economy), 5 (traditional and nonmarket activities), and 6 (governance and institutions) cover the various means of achieving human and ecosystem well-being.
4. **Feedback.** Lastly, Question 7 (synthesis and continuous learning) provides the feedback mechanism that allows managers and others to ensure accountability and to learn from the inevitable mistakes, adapt and improve designs as necessary, and celebrate the successes, giving credit where due.

Effective Engagement

If relationships with those important to a mining/minerals/metals project are unhealthy, the chance of achieving a successful project—one that contributes to sustainability—is greatly reduced. Although this is a simple idea and one that is key to many successful nonmining businesses around the world, application of this idea is only now gaining momentum across the industry.

At any point in time, mining activities must align with the norms and values of society as a whole. When it does, a social license to operate is the result—an unwritten approval. If that alignment is not apparent, the social license will be challenged. Within the mining industry, elucidation of this concept has been led by Ian Thomson and Susan Joyce (2008).



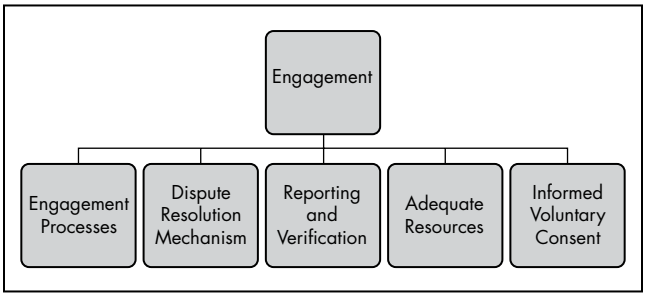
Courtesy of the International Institute for Sustainable Development.

Figure 16.2-6 Seven Questions to Sustainability

Thomson and Joyce (2006) have also played a lead role in recognizing the central role of the explorationist and junior companies (that are the first to enter an area) in either creating lasting positive relationships or a climate of long-term tension. Working to entrench such acceptance is an insurance policy that includes both the ability to recognize meaningful change and the ability to consider accommodating these changes.

Building a constructive engagement with the local community involves a series of challenging steps to (1) identify key interests, (2) learn how to listen to each interest's concerns amid all the noise of existing pressures, and (3) develop a way forward for mining and metals operations based on mutual respect, trust, and integrity. It requires effort and resources, sometimes just as much as many technical aspects of a project. And today, with the changing and growing role of many interests in society, if care is not taken to build the needed relationships, implications for proceeding effectively and efficiently on a project can be seriously undermined. At worst, physical conflict can occur.

A leading example of addressing community relationship issues has been completed by Newmont Gold Corporation (2009). In this case, concerns over conflict led an ethical investor to ask for a complete review of the relationships between Newmont operations and its host communities. The initial work, completed over 2 years, included an independent review panel. Building on existing and past attempts to build effective relationships, it led to a series of company–host community interactions that, in turn, changed Newmont's own internal management system. One result was that, in 2010, Newmont was ranked 16th in *Corporate Responsibility* magazine's 11th Annual List of 100 Best Corporate Citizens in the United States (*Corporate Responsibility* 2010), the only mining company in the top 20 and joining such businesses as Hewlett-Packard, Intel, Gap, IBM, and Microsoft.



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-7 Assessing the effectiveness of relationships between operations and other interests

A particularly important issue for the mining industry is building effective relationships with indigenous people. A useful overview of issues relevant to this topic can be found in Render 2006. A current perspective on how to best achieve effective working relationships with indigenous people is provided in ICMM 2010b. This is especially relevant for countries where the legal system does not provide strong protections for indigenous peoples. In the last decade, indigenous peoples' concerns have led to the emergence of special impact benefit agreements that formally entrench special arrangements for participation. A current tool kit addressing such agreements is now available (Gibson and O'Faircheallaigh 2010).

Considerations important to building effective relationships are shown in Figure 16.2-7. Table 16.2-2 is modified from the 7QS work and lists the ideal answer to the question,"Are engagement processes in place and working effectively?,"and offers example indicators that can be considered for compilation in order to assess how close a given mining or metals

Table 16.2-2 Engagement: Are engagement processes in place and working effectively?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Are processes of engagement committed to, designed, and implemented that • Ensure all affected communities of interest (including vulnerable or disadvantaged subpopulations due to, for example, minority status, gender, ethnicity, or poverty) are well informed and have the opportunity to participate in decisions that influence their own future; • Are understood and agreed upon by implicated communities of interest; and • Are consistent with the legal, institutional, and cultural characteristics of the community and country where the project is located?	Satisfactory processes of engagement have been designed and implemented that • Ensure all affected communities of interest (including vulnerable or disadvantaged subpopulations due to, for example, gender, ethnicity, or poverty) are well informed and have the opportunity to participate in the decisions that influence their own future; and • Are understood, agreed upon, and consistent with the legal, institutional, and cultural characteristics of the community and country where the project is located.	Input ⇓ Output ⇓ Result
	As indicated by:	
	Engagement processes. Engagement processes are in place for all phases of the project/operation life cycle to serve as a mechanism for • Collaboratively identifying desired objectives, best approaches for gathering evidence in support of achieving objectives (quantitative and qualitative), assessment criteria, trade-offs and the bases for judging trade-offs; and • Overseeing the application of the approach to assessing the contribution to sustainability articulated here.	• Comprehensive mapping of interests completed. • Design of engagement strategy completed, including guidelines that are agreed upon by all interests • Full and satisfactory disclosure of project-related information • Effective implementation as signaled by participant satisfaction
	Dispute resolution mechanism. An agreed upon, affordable dispute resolution mechanism (or set) exists and is understood by and accessible to all communities of interest.	• Dispute resolution mechanism(s) • Effective mechanisms as signaled by participant satisfaction
	Reporting and verification. Appropriate systems of reporting and verification are in place.	• Systems in place • Systems working effectively from perspective of various interests
	Adequate resources. Adequate resources have been made available to ensure that all communities of interest can effectively participate as needed. Note: Responsibility for ensuring that this capacity is in place rests with a mix of government, company, and the local community itself. The exact distribution of this responsibility should be worked out collaboratively.	• Adequate resources available • Satisfaction with level of support • Effective participation achieved as assessed by company, community, indigenous peoples, and government
	Informed and voluntary consent. The informed and voluntary consent of those affected by the project or operation has been given. Note: Inclusion of this factor does not imply that consent be given as a requirement for a project to proceed. The responsibility for approval lies with the relevant regulatory agency that is mandated by the laws of the country. Rather, this factor is included as a means to assess the extent of concurrence of those affected by a project. If that concurrence is high, the potential for achieving a net positive contribution to sustainability is greatly enhanced. In contrast, if negative feeling toward a project or operation is extensive, that potential is greatly reduced.	• Broad community support

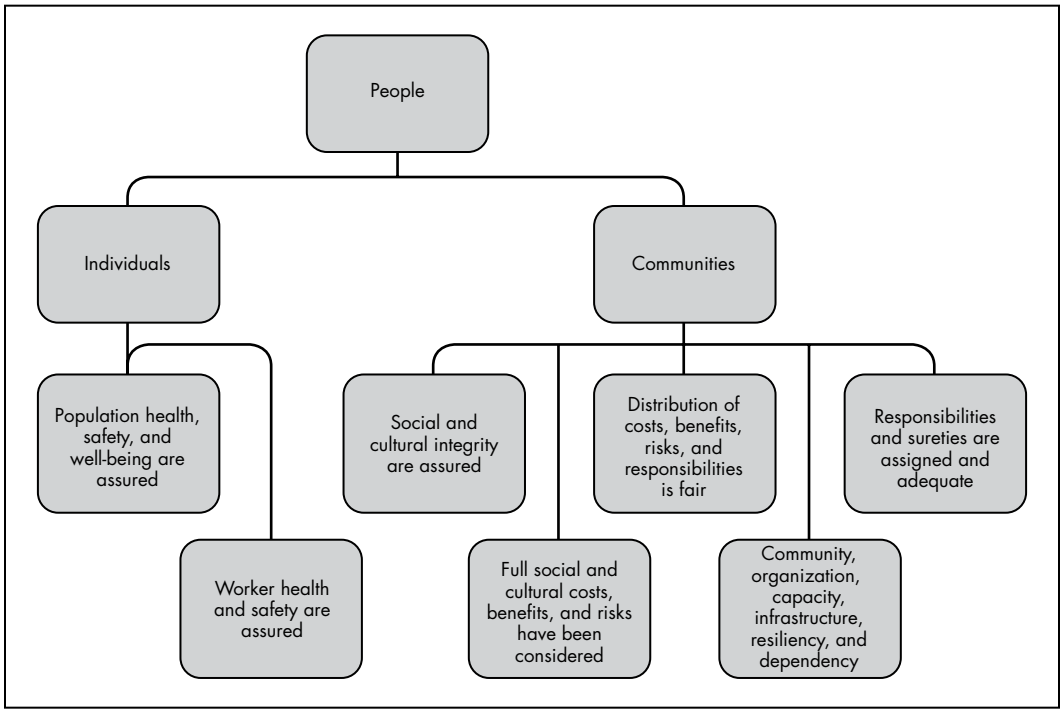
Source: Adapted from MMSD North America 2002.

operation is to achieving the ideal answer. In turn, the breadth of knowledge that Table 16.2-2 spans provides an indication of what any mining operation might be expected to take into consideration when addressing this aspect of sustainable development.

Contributing to Human Well-Being

Most inside the mining industry take it for granted that mining activities contribute to society in general, to investors that risk their capital, to management and workers who are gainfully employed as a result, and to the many host communities who experience the secondary and tertiary benefits that ripple out from any mining activity. However, following hard on the heels of the rise of the contemporary environmental movement came close examination of the social and economic implications of mining activities, particularly in emerging nations. One observation that emerged was that examples of developing economies suggested a link between mining activity and ongoing poverty. In these instances, was the presence of natural resources in fact a "resource curse" resulting from a combination of poor governance, corruption, and civil war? There is a large amount of literature on this subject (e.g., see Davis 2009, Crowson 2009, Auty 1993, and Sachs and Warner 1995).

More recently, work completed by the ICMM (McPhail 2008) has demonstrated that the resource curse needn't occur if appropriate collaborative action is taken. The ICMM work has identified six key areas of focus: (1) poverty reduction,



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-8 Example factors to consider in assessing the contributions of mining and/or metals operations to human well-being

(2) revenue management, (3) regional development, (4) local content (the use of local labor and locally derived services and supplies), (5) social investment, and (6) dispute resolution. The key to success is the use of collaborative approaches involving host governments, companies, civil society, and donor agencies.

One spin-off of the emergence of sustainability ideas starting in the mid-1980s, the resource curse debate of the 1990s and 2000s, and the ongoing challenging of the mining industry by NGOs and in some cases host governments is a realization that much greater effort is required to understand, capture, and share the nature and extent of mining and metal's contribution to human well-being. No longer can it simply be taken for granted.

A key issue requiring attention in emerging economies is a documented gender bias in mining whereby benefits (such as employment, income, training, education, and health care) accrue largely to men and the local elite while most risks (such as family and social disruption, domestic violence, alcoholism, HIV/AIDS, increased prostitution, loss of gardens for subsistence agriculture, pollution, and water losses) fall on the poorer women, the less advantaged, and the families they care for (see Eftimie et al. 2009a and 2009b). A company purchasing the bulk or all of its produce needs locally can be beneficial for local farmers and landowners who see prices increase for their produce, but this can also have the unintended effect of worsening the situation for the poor community women who may be unable to afford the higher food prices. These are complex issues that require careful management approaches for any mining company.

Figure 16.2-8 offers example factors to be considered in answering the question—"Will people's well-being be maintained or improved" by a given mining, minerals or metals operation? These factors draw on the foundation provided by socioeconomic impact analysis but use the lens of contribution. Table 16.2-3 then provides both the ideal answer as well as examples of the kind of indicators that might be considered in assessing how close a given operation is to meeting the ideal answer. As in the previous section, the range of topics provides a sense of what sustainable development means to mining from this particular perspective.

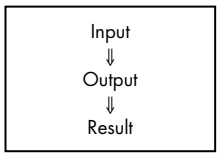
Contributing to Ecosystem Well-Being

The relationship between a mining operation and its host environment is the focus of the environmental impact assessment and the resulting environmental management plan and related environmental management systems. The approach used here builds on this foundation.

Figure 16.2-9 shows one way of conceptualizing the environmental implications of a mining operation. It looks at chemical, physical, and biological implications from both the generation of environmental stress and the linked potential for ecosystem restoration. The concept of restoration that is used here is *not* the idea that everything needs to be returned to a chemical, physical, and biological premining state. Rather, a robust ecosystem is sought that is naturally reproducing and sustainable. This is the domain of the emerging science and art of *restoration ecology*.

A chemical, physical, and biological characterization of environmental implications is traditional and helpful from several perspectives, including (1) often regulatory requirements are organized in this manner; (2) the professional expertise that must be drawn on is often organized in this way; and (3) traditional education is offered in this way and therefore management's understanding is often aided by this approach.

Table 16.2-3 People: Will people's well-being be maintained or improved?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Will the project/operation lead directly or indirectly to maintenance of people's well-being, preferably an improvement • During the life of the project or operation, or • In postclosure?	The project or operation will lead directly or indirectly to the maintenance or improvement of people's well-being • During the life of the project or operation or • In postclosure As indicated by:	
	Community organization and capacity. Effective and representative community organization and capacity (knowledge, skills, and resources) are in place in the local community, including representation of women and the disadvantaged in community leadership and decision making. Note: Responsibility for ensuring this capacity is in place rests with a mix of government, company, and the local community itself. The exact distribution of this responsibility should be worked out collaboratively.	• Presence of an organizational structure that links and represents the community in project-related decision-making processes • Training facilities in place • Local education/skills level to serve project needs and provide basis for postclosure activities • Community access to the information and expertise needed to ensure that properly informed decisions are made
	Social/cultural integrity. All communities of interest have a reasonable degree of confidence that social and cultural integrity will be maintained or preferably improved in a way that is consistent with the vision and aspirations of the community. Involuntary resettlement and other interventions will be undertaken in such as way as to maintain or preferably improve social and cultural integrity. Note: This category is particularly dynamic and will change as a project proceeds.	• Existence of community and regional visions expressed explicitly in development and land-use plans • Presence of key indicative social structures and their states • Sense of satisfaction signaled by all interests that social and cultural integrity will be maintained or improved (including separate consultations with community women and representatives of disadvantaged groups in the community) • Social and cultural indicators identified as significant by the community • Preparation and implementation of community-supported and project-induced migration management plans to help manage the impacts of the inflow of outsiders that occurs when a mining development is expected or announced
	Worker and population health, safety, and well-being. Improvement of indicators of worker and population health, safety, and well-being are maintained or improved. Note: Responsibility for gathering this data and information lies with a mix of company (in terms of workers), community, and government. However, statistics on population health, training and education, jobs, income, poverty, debt, community resiliency, and community dependency typically fall to government.	• Baseline studies completed that include basic demographics to track population change (birth rate, infant mortality, morbidity rates, in/out-migration), household incomes, and so forth, including gender disaggregated data • Worker health and safety • Population health • Training and education • Jobs, income, poverty, debt • Crime and security • Community resiliency • Community dependency
	Availability of basic infrastructure. The infrastructure to meet basic needs is available to workers and residents.	• Water supply, sewage and wastewater treatment, power, communications, transportation, education, health services
	Consideration of all direct, indirect, and induced or diffuse effects. All communities of interest have a reasonable degree of confidence that all direct, indirect, and induced or diffuse effects have been considered and addressed. Note: Requirements will change through the project.	• Direct, indirect, and induced or diffuse economic, social, and cultural effects of project • Changes in social behavior as a result of the project
	Full social/cultural costs, benefits, and risks. All communities of interest have a reasonable degree of confidence that the full costs, benefits, and risks to people have been identified and factored into project or operation-related decision making (as it applies throughout the full project or operation life cycle).	• Satisfaction that all social/cultural costs, benefits, and risks found across the full life cycle from exploration through postclosure have been identified and addressed

Source: Adapted from MMSD North America 2002.

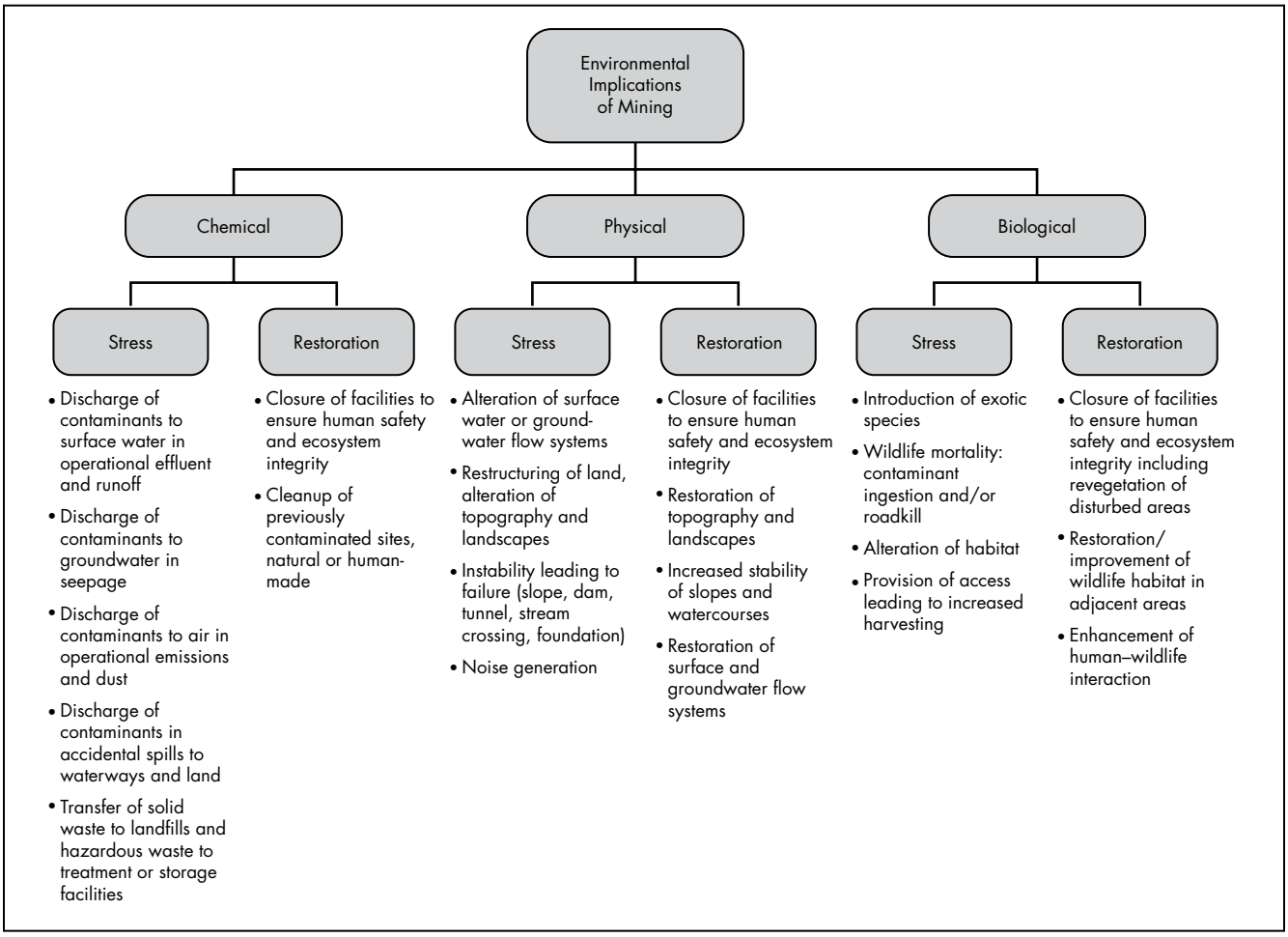


Figure 16.2-9 Chemical, physical, and biological implications of mining/metals operations

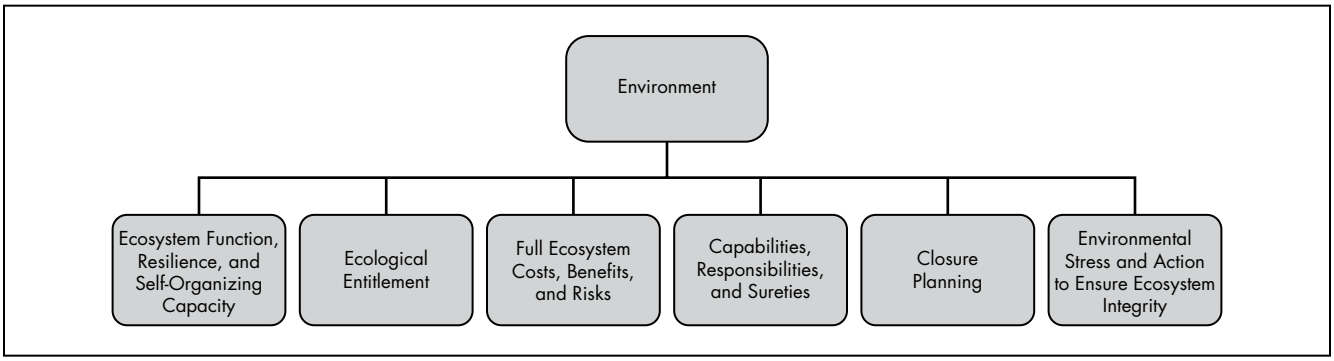
However, the ecological system does not function in such a compartmentalized way, and those expert in ecology in general and ecological restoration in particular have offered alternative ways of addressing this challenge. Figure 16.2-10 offers such an approach, and Table 16.2-4 addresses the question—"Is the integrity of the environment assured over the long term?"—and provides both an ideal answer to this question as well as an example set of indicators that, if compiled, would serve to facilitate an assessment of how compatible a mining/metals operation would be with sustainable development An increasingly important emerging issue is understanding the full life-cycle environmental contribution of the metals. Much remains to be learned about the technique of life-cycle analysis.

Economic Viability: Project, Community, Nation

Understanding the economics of a project lies at the heart of successful mining from a management and investment perspective. Two key factors govern project economic stability: (1) the licensing and fiscal regimes under which mining takes place—that they are efficient, noncorrupt, and result in security of tenure; and (2) an equitable sharing of costs, benefits, risks, and responsibilities between the host country and the investor.

For any project to be successful, the deposit, the company, and industry must all be economically viable. The typical economics of a mine is such that the first few years (generally, years 1 to 5) are usually very profitable, which is necessary to serve investors who require a return on invested capital. Middle years are moderately profitable (generally, years 6 to 15), and older mines are usually marginally viable. Company managers strategize from this perspective because it is their legal responsibility to protect the investors. If a large number of operating mines within their system are in the older phase, a company can be vulnerable. However, from the perspective of an employee or service provider, the mine is meaningful as long as it operates and provides employment. And the concern from the local, regional, or national government is that the mine leads to a lasting contribution—however that is achieved. These different perspectives all factor into the sustainable development equation.

However, they do not imply that a project/company should be assuming responsibility for the local, regional, or national economy. Rather, it is often possible that by working in collaboration with in-country partners, benefits can be achieved for the host community, region, or country that involve little cost to the company but have significant benefits for the host. Further, if unintended economic consequences related to this broader perspective arise and are not recognized, they may result in a liability to the company over the long term that can have grave consequences, including the possibility of the national/local business environment deteriorating to the point



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-10 Elements for assessing the contribution of mining/metals operations to the environment over the long term

Table 16.2-4 Environment: Is the integrity of the environment assured over the long term?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Will the project or operation lead directly or indirectly to the maintenance or strengthening of the integrity of biophysical systems so that they can continue in postclosure to provide the needed support for the well-being of people and other life forms?	The project or operation will lead directly or indirectly to the maintenance or strengthening of the integrity of biophysical systems, as indicated by:	Input ⇓ Output ⇓ Result
	Ecosystem function, resilience, and self-organizing capacity. All communities of interest hold a reasonable degree of confidence that ecosystem function, resilience, and self-organizing capacity will be maintained or improved over the long term.	• Baseline studies completed • Monitoring systems in place • Projected effects of project on indicator species of aquatic and terrestrial flora and fauna (identified through both scientific assessment and traditional ecological knowledge studies) • Projected long-term well-being of water systems and renewable resources in the area of the mine/mineral project • Tracking rapid geological change
	Ecological entitlement. All communities of interest have a reasonable degree of confidence that the capacity of project-affected renewable and nonrenewable resources will be maintained or improved such that the needs of both present and future generations will be met.	• Degree of confidence
	Full ecosystem costs, benefits, and risks. All communities of interest have a reasonable degree of confidence that the full costs, benefits, and risks to the ecosystem have been identified and factored into project/operation-related decision making (as it applies throughout the full project/operation life cycle).	• Full or total cost-accounting tools to assess the implications of the project • Satisfaction that all social/cultural costs, benefits, and risks related to the full life cycle from exploration to postclosure have been identified and addressed
	Capabilities, responsibilities, and sureties. All communities of interest have a reasonable degree of confidence that the responsibilities and sureties for ensuring both short- and long-term ecosystem well-being have been fully and fairly assigned and accepted and that those responsible have the capacity and willingness to fulfill their roles (including responsibilities attached to company, community, government, or NGO).	• Financial sureties and mechanisms to address potential present and future environmental liabilities and effort required to ensure the continuing integrity of biophysical systems • Satisfaction that sureties and mechanisms will provide adequate bridging to postclosure state.
	Closure planning. All communities of interest are satisfied that closure plans meeting best industry practice have been completed; mine design and implementation are driven by closure requirements to ensure that long-term financial, environmental, and social costs are minimized.	• Closure plans to best industry practice are in place and fully integrated into mine design and implementation

(continues)

Table 16.2-4 Environment: Is the integrity of the environment assured over the long term? (continued)

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Will the project or operation lead directly or indirectly to the maintenance or strengthening of the integrity of biophysical systems so that they can continue in postclosure to provide the needed support for the well-being of people and other life forms?	**Environmental stress and action to ensure ecosystem integrity.** Physical, chemical, and biological stress imposed on the ecosystem by the project or operation does not threaten the function, resilience, and self-organizing capacity of the biophysical system; and appropriate actions are taken to ensure the continuing integrity of biophysical systems.	• Material inputs/flows—water (surface, ground), energy (by form and source), reagents (e.g., cyanide), fuels, solvents, lubricants, other supplies • Recycling – Material recovery, oil, solvents, lubricants, batteries, tires, etc. – Waste – Hazardous and solid waste generated and discharged • Water quality – Surface water: effluent chemistry, ambient water quality downstream, stream sediment storage and load – Groundwater: contaminant plume, plume chemistry, ambient quality downstream • Water quantity—maintenance of surface and groundwater supplies • Acid rock drainage and metal leaching—tailings, waste rock, workings; short- and long-term treatment required • Soils – Chemical, biological, and physical change – Erosion • Rapid and long-term landscape change – Deep open pits, mountaintop removal – Landslides and avalanches • Air quality – Emissions chemistry – Indoor air quality – Greenhouse gas emissions • Noise, on- and off-site • Environmental incidents, on- and off-site • Reclamation and restoration, on- and off-site • Environmental effects monitoring system in place

Source: Adapted from MMSD North America 2002.

where the company can no longer operate efficiently or, in the worst case, not at all. These are significant risks.

Figure 16.2-11 is a simple characterization of the economic aspects that require consideration. Table 16.2-5 responds to the question—"Is the economic viability of the project assured; will the community and broader economy be better off as a result?" It then offers an ideal answer to that question and a set of example indicators that can be compiled to assess how close a given operation is to the ideal condition.

Traditional and Nonmarket Activities

Since the 1980s, recognition has been growing of the significance of traditional and nonmarket activities—human activities that are not part of the market economy and therefore not captured in the accounting systems that underlie our systems of taxation or the national accounts.

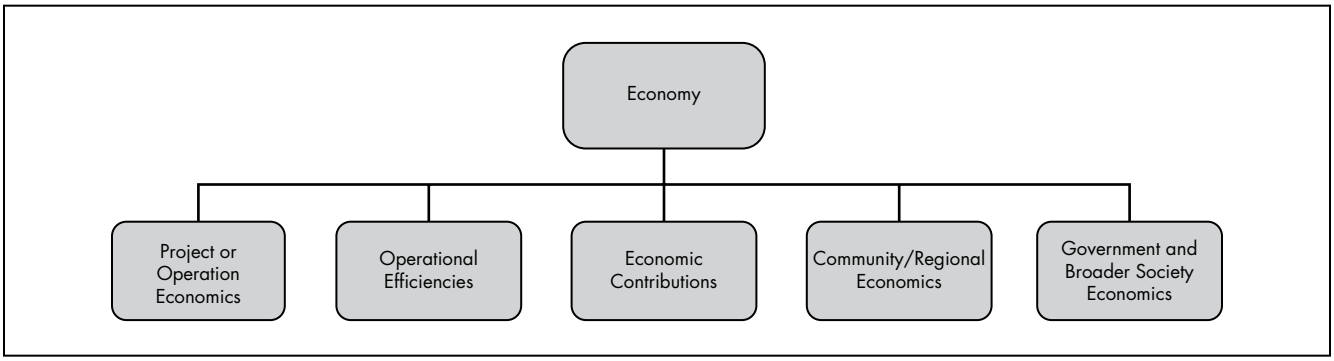
National accounts are used to track the flow of goods and services in the economy. Contributing data are organized in an indtustry classification system, which provides a useful indication of human activity. The compiled data are used to calculate such indicators as national income and gross domestic product or gross national product. However, many activities important from a sustainability perspective lie outside the formal market economy. One way of getting a sense of these is to consider the vast array of organizations of civil society, those engaged in not-for-profit work. Table 16.2-6 is a listing of the general categories of a classification of nonprofit institutions, which was developed at Johns Hopkins University and subsequently adopted by the United Nations (2003).

Often, reacting to public criticism, mining companies focus on environmental or social advocacy groups. A more strategic approach would be to consider all aspects of civil society that are important in a given community, ensuring that the contribution of the mining/metals activity to nonmarket and traditional activities is as efficiently and effectively pursued as possible. Traditional culture-based activities of indigenous people lie within this aspect of the sustainability equation and are of particular importance to mining/metals operations in many parts of the world.

Figure 16.2-12 is a simple characterization of the traditional and nonmarket aspects that require consideration. Table 16.2-7 responds to the question—"Are traditional and nonmarket activities in the community and surrounding area accounted for in a way that is acceptable to the local people?" It then offers an ideal answer to that question and a set of example indicators that can be compiled to assess how close a given operation is to the ideal condition.

Institutional Arrangements and Governance

This category addresses the effectiveness of the formal and informal rules that society creates to govern activities such



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-11 Elements for assessing the economic viability of mining/metals operations over the long term

Table 16.2-5 Economy: Is the economic viability of the project assured?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Is the financial health of the project/company assured and will the project or operation contribute (through planning, evaluations, decision making, and action) to the long-term viability of the local and regional economy in ways that will help ensure sufficiency for all and provide specific opportunities for the less advantaged?	The financial health of the project/company is assured and the project or operation will contribute (through planning, evaluations, decision making, and action) to the long-term viability of the local, regional, and global economy, as indicated by:	Input ⇓ Output ⇓ Result
	Project or operation economics. Project or operation economic success targets are achieved.	• Demonstrated positive project economics as calculated in the feasibility study; economic targets met as the project proceeds
	Operational efficiencies	• Mining efficiency • Processing efficiency • Refining efficiency
	Economic contributions: annual/total	Contributions • To the local economy • To the regional economy • To the national economy • To the global economy • Total
	Community/regional economics. Community and adjacent regional economic success targets are met, including direct employment, indirect employment, and small and medium business development.	• Demonstrated positive project economics as calculated in the feasibility study; economic targets met as project proceeds • Distribution of economic benefits resulting from the project between the company, government, and local community, and within the local community between men and women and between the wealthy and the disadvantaged
	Government/broader society economics. A net economic contribution to governments and broader society is achieved.	• Net financial gain for government as the project proceeds and meets economic targets, which contributes to economic growth and increased standards of living

Source: Adapted from MMSD North America 2002.

as mining/mineral projects and operations. This is a complex area that includes internal corporate management systems as well those found in broader society. Although some rules are written, others are unwritten and simply accepted as part of culture—internal corporate culture and culture of the host community and country.

Internal Management Systems

Adaptation of internal management systems to sustainability is well under way. A useful overview of many related issues is provided by Botvin 2009. An early foundation was provided by the work of the International Organization for Standardization (ISO), which has developed widely accepted quality management systems (ISO 9001), environmental management systems (ISO 14001), and risk management principles and guidelines (ISO 31001).

Reporting and Assurance

Since the mid-1990s, there have been a number of significant developments regarding how a mining/metals company publicly reports its sustainability-related performance and how that reported progress is assured using independent third-party assurance. Three initiatives are particularly relevant to the mining industry:

Table 16.2-6 United Nations classification of nonprofit institutions

Group	Subgroups
1. Culture and Recreation	Culture and Arts, Sports, Other
2. Education and Research	Primary and Secondary, Higher, Other, Research
3. Health	Hospitals and Rehabilitation Nursing Homes, Mental Health and Crisis Intervention, Other
4. Social Services	Social Services, Emergency Relief, Income Support and Maintenance
5. Environment	Environment, Animal Protection
6. Development and Housing	Economic, Social and Community Development, Housing, Employment and Training
7. Law, Advocacy, and Politics	Civic and Advocacy Organizations, Law and Legal Services, Political Organizations
8. Philanthropic Intermediaries and Volunteerism Promotion	Grant-Making Foundations, Other
9. International	International Activities
10. Religion	Religious Congregations and Associations
11. Business and Professional Associations, Unions	Business Associations, Professional Associations, Unions
12. Not elsewhere classified	Not elsewhere classified

Source: United Nations 2003.

1. Mining and Metals Sector Supplement (MMSS) of the Global Reporting Initiative (GRI)
2. ICMM Reporting and Assurance System
3. Towards Sustainable Mining (TSM) system of the Mining Association of Canada (MAC)

All of these initiatives recognize that for the industry to improve its reputation and credibility, improved performance linked to an alignment with public values is essential. From a company perspective, there is a close link to risk management. All three of these initiatives are cognizant of the need to synchronize.

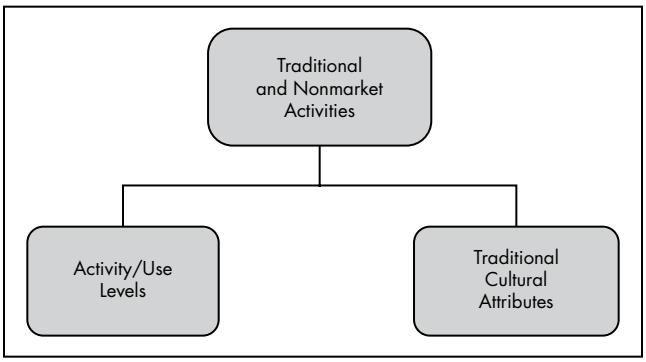
GRI MMSS. The GRI is a network-based organization that over the past decade has pioneered the development of the world's most widely used sustainability reporting framework. The GRI framework sets out principles and indicators that organizations can use to measure and report their economic, environmental, and social performances. At its heart sits the generic 2006 G3 Guidelines that refer to all applications. In addition, sector supplements and national annexes are being developed over time. As of 2010, 15 sector supplements have been developed, one of which is focused on mining.

Following 6 years of intense multistakeholder development, the GRI MMSS was released in March 2010 (ICMM 2010c). MMSS addresses the following nine main sector topics:

1. Biodiversity
2. Emissions, effluents, and waste
3. Labor
4. Indigenous rights
5. Community
6. Artisanal and small-scale mining
7. Resettlement
8. Closure planning
9. Materials stewardship

ICMM's reporting and assurance system. ICMM's system serves as a means to demonstrate that members have delivered on the commitments contained in the 10 ICMM principles and the related position statements (Figure 16.2-1). Members must meet GRI reporting requirements, and independent third-party assurance is required on an overall company system basis.

MAC's TSM initiative. MAC describes TSM as "a strategy for improving the mining industry's performance by aligning its actions with the priorities and values of Canadians; and a process for finding common ground with our communities of interest, to build a better mining industry today and in the future" (MAC 2010). TSM focuses on company management systems and provides a means of evaluating their quality, comprehensiveness, and robustness. As of 2010, four performance elements were finalized: (1) tailings management, (2) energy use and greenhouse gas emission management, (3) external outreach, and (4) crisis management planning. In addition, work on aboriginal relations, biodiversity conservation, safety and health, and mine closure is in varying states of implementation.



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-12 Elements for considering implications of mining/metals operation on traditional and nonmarket activity

Nation-Based Mining Law

In all cases, host countries own the natural resources. Thus complex systems of laws and regulations are put in place to guide exploration, production, closure, use, reuse and recycling. Every company is responsible for developing an understanding of the particular regulatory regime that is in place where it is active.

Important International Agreements

A large number of international agreements have significant implications for the mining, minerals, and metals industry. These range from the transport of hazardous materials to human rights and chemicals management. Here, too, it is the responsibility of companies to understand this complex part of the operating environment.

Table 16.2-7 Traditional and nonmarket activities: Are traditional and nonmarket activities accounted for in a way that is acceptable to the local people?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Will the project/operation contribute to the long-term viability of traditional and nonmarket activities in the implicated community and region?	The project/operation will contribute to the long-term viability of traditional and nonmarket activities in the implicated community and region, as indicated by:	Input ⇓ Output ⇓ Result
	Activity/use levels. Maintenance of activity/use levels is designated by the community in question.	• Baseline study of traditional and nonmarket activities completed • Dependency levels on traditional and nonmarket activities
	Traditional cultural attributes. Maintenance of traditional cultural attributes is designated by the community in question.	• Use of indigenous language • See also Table 16.2-3

Source: Adapted from MMSD North America 2002.

Assessing the Effectiveness of Institutional Arrangements and Governance

Elements that could contribute to an assessment of the effectiveness of institutional arrangements and governance are shown in Figure 16.2-13 and are described in greater detail in Table 16.2-8.

Synthesis and Continuous Learning

Many companies employ an internal system of planning, decision making, implementation, performance assessment, and adjustment as a matter of course. This element of the 7QS framework calls for the same kind of approach to be taken but using the lens of sustainability as the driver. Example elements that could contribute to an assessment of the effectiveness of an overall integrated evaluation and capacity for continuous learning and improvement are summarized in Figure 16.2-14 and described in greater detail in Table 16.2-9.

PERSPECTIVES BY PROJECT LIFE-CYCLE PHASES

Application of sustainability concepts to the mining, minerals, and metals industry requires attention paid to the full project and mineral life cycles (see earlier discussion in the "Boundary Conditions for Applying Sustainability Ideas to Mining" section). A cursory summary of phase-by-phase observations follows. (This discussion is modified from MMSD 2002.)

Exploration

Exploration is the starting point of the entire mineral project life cycle. However, only 1 out of 1,000 exploration projects ever evolves to the next level of activity (often more exploration). Explorationists generally have the objective of getting into and out of areas quickly and moving on to the next evaluation. Exploration is highly competitive and therefore secretive by design. Without positive relationships with the local community, serious tensions can occur. For their part, local residents want to know what is happening and how their lives might be affected. Left to ferment, anger and/or unintended expectations can grow—either of which can subsequently cause difficulty. Although the extent of the human and environmental implications is less than in later phases of activity, they can be profound. Oftentimes concerns center on access routes, which can open up large areas of untouched territory.

Exploration is the first time that the desirability or appropriateness of mining in a particular area is considered. Because exploration sets the stage for all that follows, the nature of the relationship established between the exploration team and affected communities of interest sets the tone going forward. Only in the last few years have models for appropriate engagement during exploration begun to emerge (e.g., see Thomson and Joyce 2006 and 2008).

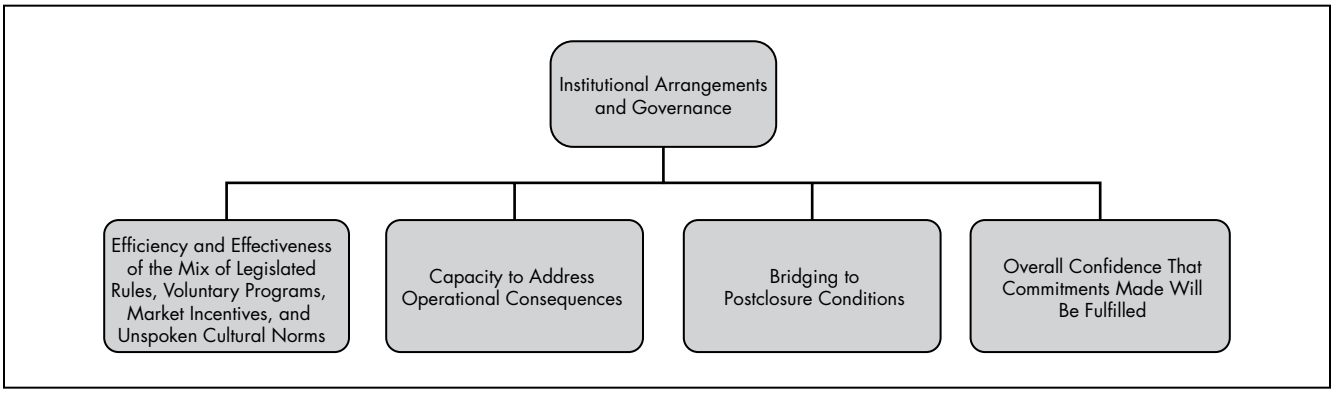
The leading node of guidance on exploration and sustainability comes from the Prospectors and Developers Association of Canada (PDAC) and its e3 Plus Framework for Responsible Exploration (PDAC 2010). The program is aimed at helping exploration companies continuously improve their social, environmental, and health and safety performances and to comprehensively integrate these three aspects into all their exploration programs. It includes a set of overarching principles and explanatory guidance notes; and three comprehensive tool kits addressing social responsibility, environmental stewardship, and health and safety. For their part, the principles deal with

- Responsible governance and management,
- Ethical business practices,
- Human rights,
- Project due diligence and risk assessment,
- Engagement with host communities and other affected and interested parties,
- Contributing to community development and social well-being
- Protecting the environment, and
- Safeguarding the health and safety of workers and the local population.

The e3 Plus program is currently examining options for reporting and assurance that will echo those described. A significant driver is the demand by senior firms that the conditions they might inherit through acquisition of advanced exploration projects take into consideration the full range of sustainability components, in particular the need for effective engagement with host communities.

Design and Construction

Relative to other phases of activity, the design and construction phase is short. However, this intense pulse of activities and related social and environmental implications can be destructive if not carefully managed. This phase of activity



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-13 Example elements for assessing the effectiveness of institutional arrangements and governance

Table 16.2-8 Institutional arrangements and governance: Are capacities in place to address project or operational consequences?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Are the institutional arrangements and systems of governance in place to provide a reasonable degree of confidence that the capacity to address project or operation consequences will continue to exist through the full life cycle, including postclosure?	Satisfactory institutional arrangements and governance mechanisms are in place, as indicated by:	Input ⇓ Output ⇓ Result
	Mix of rules, market incentives, voluntary programs, and cultural norms. An effective mix of legislated rules, market incentives, voluntary programs, and cultural norms is in place for governing project activities.	• Satisfaction with mix from the perspective of the various communities of interest, including company, community, indigenous peoples, NGOs, government
	Capacity. A reasonable degree of confidence is held by all communities of interest that the capacity to address project or operation consequences is in place now and will continue to exist throughout the full project/operation life cycle, including postclosure.	• Capacity of community support infrastructure to meet the needs of residents and workers of the region • Monitoring and enforcement programs in place with adequate resources committed for the full project life cycle
	Bridging. A reasonable degree of confidence is held by all communities of interest that sufficient local capacity and commitment will be developed and an adequate level of resources will be accumulated and set aside throughout the life of the project/operation to ensure a smooth transition to an acceptable postclosure condition (ecological, social, cultural, economic) for the community that remains.	• Existence of community-based *nonmining* economic and social development initiatives
	Confidence that commitments made will be fulfilled. A reasonable degree of confidence is held by all communities of interest that commitments that have been made will be fulfilled.	• System of publicly reported reporting and assurance in place • Level of funding to cover rehabilitation/reclamation costs during operations and at closure

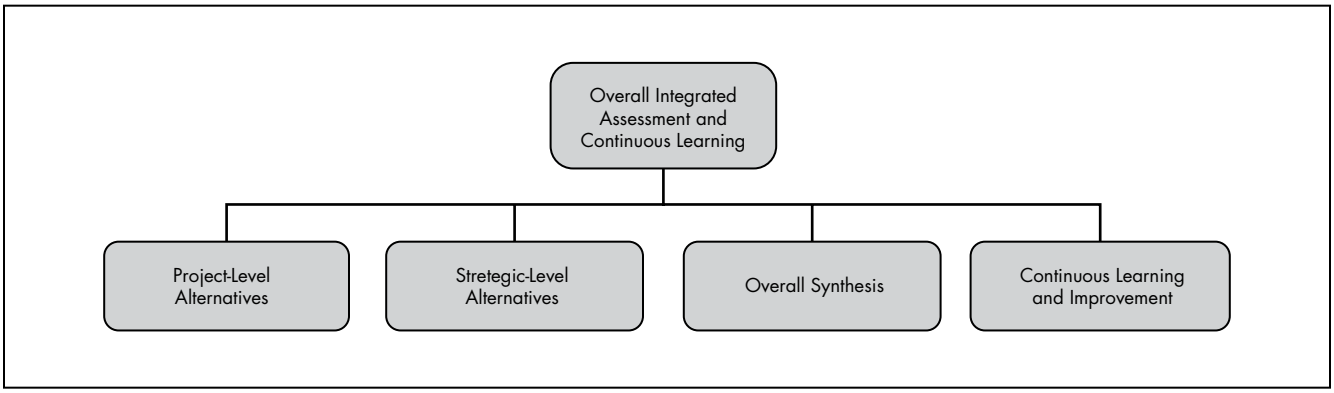
Source: Adapted from MMSD North America 2002.

also marks the time that formal approvals are sought and received. During approvals processes, key opportunities exist for engagement with communities of interest (see previous discussion of effective engagement in the "Effective Engagement" section).

The influx of outsiders seeking employment and new opportunities can create significant changes within the community—often for the worse—and preparation and implementation of a project-induced migration management plan that is acceptable to the established community population can help manage the impacts. Many of the activities are undertaken by subcontractors who may or may not follow the same practices as the project owner or manager. Effective systems (policy, management, oversight, incentives) for ensuring good practices are essential.

Operation

The operation phase typically receives the greatest amount of attention. When the general public conjures an image of a mining/mineral project or operation, it is the operation phase that is imagined, whether the activity be mining, smelting, refining, metals manufacturing, or recycling. Economic, social, cultural, and environmental implications are relatively well understood, although their treatment is uneven in practice.



Courtesy of the International Institute for Sustainable Development.

Figure 16.2-14 Synthesis and continuous learning—elements to consider

Table 16.2-9 Synthesis and continuous learning: Does synthesis show that the net result will be positive or negative?

Question (Goal)	Ideal Answer (Objectives)	Example Indicators
Has an overall evaluation been made and is a system in place for periodic revaluation based on • At the project level, consideration of all reasonable alternative configurations and designs (including the no-go option in the initial evaluation); • At the overarching strategic level, consideration of all reasonable alternatives for supplying the commodity and the services it provides for meeting society's needs; and • A synthesis of all the factors raised in this list of questions, leading to an overall judgment that the contribution to people and ecosystems will be net positive in the long term?	An overall evaluation been made and a system is in place for periodic revaluation based on consideration of • At the project level, consideration of all reasonable alternative configurations including the no-go option; • At the overarching strategic level, consideration of all reasonable alternatives for supplying the commodity and the services it provides for meeting society's needs; and • Synthesis of all the factors raised in this list of questions, leading to an overall judgment that the contribution to people and ecosystems will be net positive in the long term; As indicated by:	Input ⇓ Output ⇓ Result
	Project-level alternatives. All reasonable project alternatives have been considered.	• All key alternatives considered, for example: – Access – Transportation – Energy supply – Water supply – Local infrastructure – Tailings and effluent management – Mineral processing options
	Strategic-level alternatives	• Strategic level review that confirms project need
	Overall synthesis. A synthesis has been completed, and the system is in place to for periodic reassessment.	• Synthesis undertaken
	Continuous learning and improvement. A commitment to continuous learning and improvement is held by all interests, including company, community, government, and others.	• Mechanism(s) and resources in place to periodically repeat the overall sustainability assessment and report the results publicly

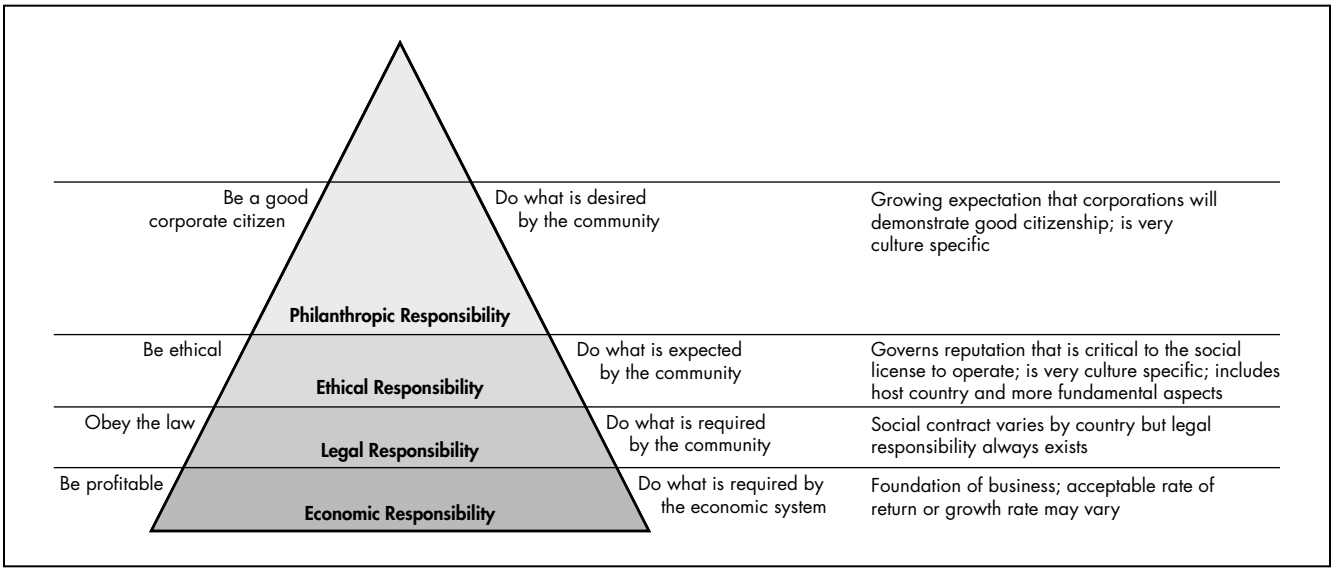
Source: Adapted from MMSD North America 2002.

Closure: Temporary, Final, Post

Temporary closure (due to changing metal prices, accident, disaster, or labor strife) rarely receives the pre-event planning and consideration that it should. Though often a temporary closure can run a few short days or weeks, sometimes so-called temporary closures can stretch into many years' duration.

While concepts of *design-for-closure* date from the 1970s (J. Gadsby, personal communication), concepts of sustainability now demand *design-for-postclosure*. Huge benefits can be realized by ensuring that closure and postclosure activities are conceived of and implementation is begun during the design and construction and operational phases of activity. Today, design-for-closure includes (1) physical environmental aspects, (2) social and community sustainability aspects, and, importantly, (3) financial assurance that resources will be available for needed closure and postclosure activities.

Design-for-postclosure involves a significant increase in the time horizon governing project design criteria, whether the focus be social or environmental in nature. Furthermore, successful design-for-postclosure identifies a need for involvement of those affected by postclosure conditions from the earliest phases of any project. Fortunately, in the case of closure and postclosure, research and experience have produced a number of successful models. One thing emerges from all



Source: Carroll 1991.

Figure 16.2-15 Carroll's pyramid of corporate responsibility and performance

of them: *succeeding custodians*, the people who will remain nearby long after mining operations cease, need to be at the table. It is only with their presence that their values can be factored into project implementation and the bridging role of a mine/mineral project or operation can be realized.

One delicate issue that the industry is now examining, which emerges because of the longer time horizons now being considered combined with the societal demand for greater transparency, is the whole question of estimating long-term closure liability and related financial assurance.

CORPORATE SOCIAL RESPONSIBILITY

Application of sustainability concepts to the mining, minerals and metals industry evokes yet another perspective that is growing in importance—the issue of corporate social responsibility (CSR). Establishing the nature and extent of a company's social responsibility is of deep importance in today's world. CSR provides a critical and natural overarching supplement to sustainable development and sustainability ideas and provides another direct connection to values through the addition of an ethical lens. At the same time, contemporary ideas of applied sustainability provide a foundation for CSR; they are closely linked but not the same.

Values are "something which is prized as of great worth and desirability: that which is respected and which motivates action...concepts which motivate actors in a general way to more specific goals" (Chadwick 1978). The described treatment of applied sustainability starts with a values-based proposition, one that says that we care for both people and the environment. In contrast, *ethics* is the science of right or wrong. It provides rules of behavior that signal what is considered decent and honest as opposed to misleading, dishonest, or unscrupulous.

Values and ethics combine to establish the boundaries of appropriate CSR. Over the last decade, there has been growing emphasis on CSR. The events in the financial services industry that sparked the 2008 recession have added fuel to the debate. Carroll (1991) offers an interesting and helpful pyramid of corporate responsibility and performance (Figure 16.2-15) and serves to illustrate a kind of hierarchy that applies to corporate responsibility.

Revenue Transparency and the Extractive Industries Transparency Initiative

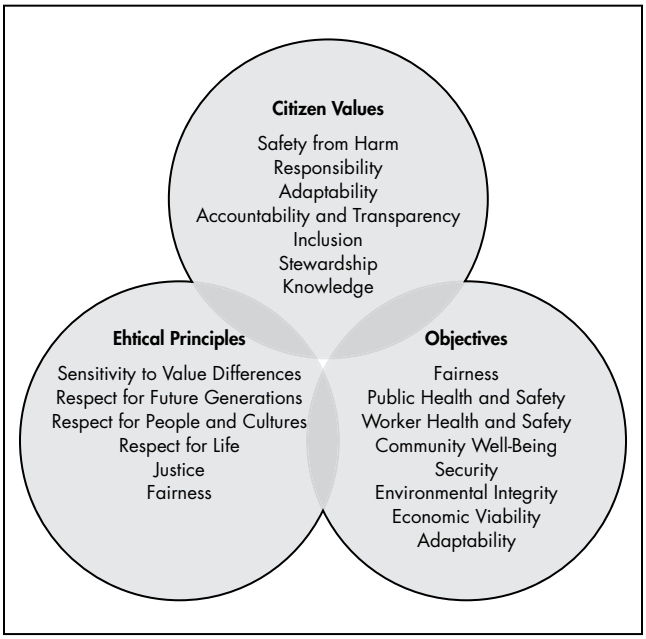
One dimension of CSR that has been in the limelight has been the question of the revenues that flow from an active operation to the government. Concerns that these revenues were being intercepted for personal gain led to the creation of the Extractive Industries Transparency Initiative (EITI). Launched at the World Summit on Sustainable Development in Johannesburg, South Africa, in September 2002, EITI has grown to become a broad coalition of governments, major organizations of civil society, institutional investors, international organizations, international financial institutions, and companies (oil and gas, mining). EITI is active in 47 countries in which mining plays a significant role in the economy.

EITI represents a global standard for (1) extractive companies to publish the revenues they pay to governments in implementing countries (taxes, royalties, licenses, fees); (2) national governments to disclose what they receive; and (3) the payments and receipts to be reconciled in a third-party assured process. Each national application is overseen by an in-country tripartite multistakeholder committee consisting of representatives of industry, government, and civil society.

Bridging Issue of Human Rights

Respecting human rights is clearly part of corporate responsibility. For the 19 corporate members of ICMM and the 30 affiliated mining associations that, in turn, link to another 1,500 companies, respect for human rights is a key aspect of sustainable development.

In 2005, the UN secretary-general appointed John Ruggie, Berthold Beitz professor in human rights and international affairs at the Kennedy School of Government, Harvard University, as his special representative on business and human rights. Five thousand companies are now engaged worldwide in this initiative (Business and Human Rights Resource Centre n.d.). Ruggie played a central role



Source: Data from Joanna Facella, Nuclear Waste Management Organization, Toronto, Ontario, Canada.

Figure 16.2-16 Toward an ethical lens for mining

in preparing the UN Millennium Summit in 2000, including drafting the summit's final "Declaration," which adopted the Millennium Development Goals that have brought renewed energy and focus to the fight against global poverty. In 2008, Ruggie produced the now well-accepted "Protect, Respect, and Remedy" framework for corporate responsibility, which outlined in clear terms the distinctive but complementary roles of government and corporations regarding human rights: While governments have the primary responsibility for the protection of human rights, companies have a responsibility for respecting them. The reporting and assurance systems described previously include treatment of company respect for human rights.

Toward an Ethical Lens for Mining

It is inevitable that the material described in this chapter, taken together, signals that the world is moving toward developing a kind of *ethical lens* for assessing the mining, minerals, and metals industry. During the period 2002–2005, a Canadian initiative undertook a comprehensive review of alternatives for managing used nuclear fuel over the long term. Figure 16.2-16 shows the results of an extensive discussion with Canadians about the values (Watling et al. 2004), ethical principles (NWMO 2005), and related objectives that were used to assess options for long-term management of used nuclear fuel (NWMO 2004). Together they serve as an innovative ethical lens that is as applicable to mining in general as it is to the management of nuclear waste.

LOOKING FORWARD

Ongoing Issues

A number of issues remain to be fully addressed (modified from MMSD North America 2002 and Hodge 2004):

- **Fair distribution:** Detailed site-specific procedures for describing and addressing the distribution of costs, benefits, risks, and responsibilities among governments, men and women, rich and poor, and present and future generations.
- **Decision making, trade-offs:** Designing and implementing optimum decision making systems and approaches that effectively and fairly address trade-offs in any given site application.
- **Need and alternatives:** Effectively and fairly assessing the need for a given project and/or commodity in light of considerations and alternatives that span local to global implications.
- **Achieving a whole system perspective:** Seeing, understanding, and factoring in a sense of the whole system, not just the small component parts.
- **Uncertainty, precaution, and adaptive management:** Dealing with uncertainty using an appropriate level of precaution and an adaptive management approach.
- **Attribution problem and dealing with cumulative impacts:** Effectively addressing the common situation where a project is one of a number of contributors to social, cultural, economic, and environmental change or stress—establishing the cumulative implications, apportioning responsibility, and deciding who should take responsibility for the analysis.

These issues are not new. Together, they are a testament to some of the complexities that must be faced in bringing the ideas of sustainability from theory to practice. In addition to the previous list, the following practical issues remain to be resolved to improve the contribution of the mining industry to sustainability:

- **Capacity building:** Creating the necessary national and local governance systems that can facilitate effective and principled engagement among industry, government, and civil society.
- **Voluntary codes of conduct:** Increasing participation in and enhancement of values-based, principled codes of conduct that lift broad industry performance above legislated minimums.
- **Climate change:** Establishing how the mining industry can most effectively contribute to addressing the issue of global warming.
- **Artisanal and small-scale mining:** Establishing how such mining can be best managed in various countries and within the context of sustainable development, taking into account environmental and social implications as well as links to large-scale mining (for a comprehensive treatment of these issues, see CASM n.d.).
- **Consistent application of performance standards:** Achieving consistency in the application of sustainable development ideas within countries, between countries, and across the industry from juniors to majors.
- **Effective human resource management:** Effectively creating the foundation of education, training, and support that is required to ensure the needed flow of skilled personnel (see Freemen and Miller 2009).

All the issues listed are critical to effective application of sustainable development concepts to the mining and metals industry.

New Perspectives

In current public debate, many challenge the consumptive nature of modern society. Professor Saleem Ali has offered an

alternative perspective on this divisive topic that has significant implications for the mining, minerals, and metals industry (Ali 2009). He argues that simply disavowing consumption of materials is not likely to help in planning for a resource-scarce future, given the extent of inequity in today's world, development imperatives, and our goals for a democratic global society. Rather than suppress the creativity and desire to discover—which he calls "the treasure impulse"—Ali proposes a new environmental paradigm, one that accepts our need to consume "treasure" for cultural and developmental reasons but warns of our concomitant need to conserve.

In evaluating the impact of treasure consumption on resource-rich countries, he argues that a way exists to consume responsibly and alleviate global poverty. Ali's thesis brings a much needed additional perspective relevant to applying sustainability ideas to the mining industry.

Change and Change Management

The role of metals and minerals in providing for both human and ecosystem well-being is critical, and the mining and metals industry enjoys a deep and rich heritage. However, this same heritage brings with it an innate resistance to change that can impede needed change. There are many contributing factors to this resistance, including

- Tradition: "We've always done it this way and it has worked, why change now?" or "If it's not broken, why fix it?";
- A number of economic factors that tend to favor risk-averse behaviors, including the volatility and uncertainty inherent in commodities markets, the scale and complexity of operations, the highly competitive nature of the business, and the resulting thin margins (Peterson et al. 2001); and
- A range of factors that can come into play at a personal level and apply not only in mining but in any industry. Examples include uncertainty, lack of training, lack of understanding about how change might affect a given job or activity, loss of job status, loss of security, and timing (summarized from Stanislao and Stanislao 1983).

To overcome this resistance requires well-thought-out and appropriate change strategies. Sometimes this calls for a catalytic role, sometimes a role in bringing people together, sometimes the provision of resources, and sometimes the design of a new and innovative solution. The key is to be strategic.

Since the 1980s, a number of factors have emerged to provide a vital and exciting climate of change in the mining industry. These factors include

- Increased public scrutiny of mining activities and the rise in the activity level, sophistication, and significance of civil society organizations;
- Public recognition of the significance of environmental and social risks and liabilities associated with mining practices and, in both regulatory and financial services industry practices, a shift that clearly assigns those liabilities to project owners and proponents;
- Changing practices in the financial services industry that are increasingly linking access to capital to a demonstration of corporate responsibility using sustainability criteria in the assessment process—a factor much reinforced by the 2008–2009 financial crisis;
- Consolidation among both operating firms and technology suppliers in response to commodity-price pressures and the desire to achieve economies of scale;
- Globalization of mining activities and the increased role of developing regions in mineral production and of mining in the developing countries' economies;
- Conflict in weak governance zones;
- Increasing worldwide consumption of key mined commodities with increasing demand in emerging economies;
- Increased dependence on metals to support a shift to a green economy; and
- Growing formal recognition that no one party can do it alone.

These drivers of change are not going to disappear; if anything, their significance will increase. The result is that the mining industry is embracing change in an unprecedented manner, and the concepts that have emerged to guide that change in a strategic and systematic way are those of sustainability and sustainable development.

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CHAPTER 16.3

Impacts and Control of Blasting

Charles Dowding

Control of blast effects near critical rock masses or constructed facilities depends on two main considerations:

1. The amount of explosives detonated per volume of rock and the shot pattern must be adjusted to ensure adequate fragmentation and relief.
2. Shot designs must reduce the amount of explosives detonated at any instant and adjust the initiation sequence to reduce resulting ground and airborne disturbances.

Therefore, the initiation sequence must be simultaneously concentrated appropriately in space but spread out in time. There is an optimum design that achieves both objectives: adequate fragmentation and control of disturbances. This optimum can be reached only through an understanding of the physics, rock mechanics, and geology of rock masses, structural and rock mass response to blast disturbance, and the interaction between rock fragmentation and shot design.

This chapter summarizes the state of the art in vibration measurement and structural response to facilitate optimum blast design. It also summarizes shot design for fragmentation. In addition, it applies the principles of earthquake engineering and nuclear blast protective design to blast-vibration monitoring and summarizes scientifically acceptable experimental observations of mining-induced ground motions and structural response. These technologies are integrated with the summary of the state of the art to mitigate several trends described in the following paragraphs.

There has been a general downward trend in regulatory limits on allowable blast-induced vibrations. This drift can be attributed, in part, to the tendency to take the limit of the last study and divide it in half to be safe. Unfortunately, too many studies whose limits were divided were themselves only summaries of past work that had also been divided past limits. This chapter presents the background of and references to original experiments conducted to determine safe blasting controls and therefore enables the reader to set appropriate limits based on original past work within the framework of existing regulations.

Another trend is the misapplication of peak particle-velocity limits that were determined for cosmetic cracking of residential structures. These limits have been applied to concrete tunnel liners, welded steel pipelines, radio towers, slabs on grade, and the curing of concrete, to name just a few instances. This chapter draws attention to studies that determined limits specifically for these and other cases. Where no studies exist, it presents methods based on response spectra or ground strains that allow setting of appropriate criteria or limits.

Frequency of vibration and ground strain form the foundation for the presentation. The importance of frequency cannot be overemphasized, as it is as critical as peak particle velocity in determining the response of aboveground structures. For belowground structures, frequency, in combination with propagation velocity, controls the response of the rock mass and embedded structures. In both cases, cracking results from induced strains, where particle velocity is used as an index of strain.

In addition, this chapter describes computerized monitoring instrumentation. Computerization simultaneously increases monitoring efficiency and decreases costs, both original capital costs and costs associated with record keeping. The labor-saving efficiencies associated with automated record keeping continue to be undervalued by many mining operations.

RANGE OF BLAST EFFECTS

Blast effects on surrounding earth materials and structures can be divided into two categories: permanent and transient displacements. Although the focus of this chapter is transient displacements, effects of permanent displacements are presented because they are associated with significant transient effects at relatively small distances.

Permanent Degradation and Displacement of Adjacent Rock

Permanent effects, with the exception of fly rock, are described within only a few hundred feet. They can be divided into two categories: degradation and displacement.

Degradation

Degradation is normally described by cracking intensity. Such blast-induced cracking has been observed experimentally to

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vary with hole diameter and rock type (Siskind and Fumanti 1974; Holmberg and Persson 1978). Small-hole-diameter construction blasting has induced cracking at distances of 3.5 to 7 ft (1 to 2 m). Likewise, large-hole-diameter mining blasting has induced cracking at distances of 35 to 70 ft (10 to 20 m). Careful blast design can dramatically reduce these maximum distances.

Displacement

Displacement can be produced by either delayed gas pressures (those that accumulate during detonation) or vibration-induced shaking. Delayed gas pressure has dislocated blocks as large as 1,300 yd^3 (1,000 m^3) during construction blasting (Dowding 1985). Such movement is unusual but is associated with isolated blocks, gas leaks along open joints, and poor shot design with large burdens. Vibratory or shaking-induced displacement is normally associated with unstable blocks in rock slopes and can occur wherever static factors of safety are low and ground motions produce permanent displacements that are greater than the first-order asperity wavelength (Dowding and Gilbert 1988). Gas-pressure-related displacement can occur out to several hundred feet.

Two special cases of permanent displacement exist:

1. **Fly rock.** Fly rock is a special case of permanent displacement of rock by explosive expulsion from the top of the blasthole. Rock has been observed to have been propelled as far as 330 to 3,300 ft (100 to 1,000 m) (Roth 1979). Statistical studies have shown that the probability of these extreme events is quite low under normal circumstances, 1 in 10,000,000 at 2,000 ft (600 m) (Lundborg 1981). Since the probability increases with decreasing distance, blasting mats are required in any construction blasting in an urban environment to prevent all fly rock.
2. **Soil densification or compaction.** Another special case of permanent displacement is the vibratory densification of a nearby mass of loose, clean sand. The propensity for such densification is a function of the soil's density, mineralogy, grain-size distribution, and saturation. Soils that are densifiable are loose sands with <5% of silt-size particles found below the water table. For example, such sands were densified out to distances of 70 ft (20 m) after detonation of single 11-lb (5-kg) charges within the loose sand mass itself (Ivanov 1967). Soils that are slightly cemented, contain >5% fines, or are above the water table are less subject to vibratory densification from typical ground motions.

Transient Structural Response

Transient effects result from the vibratory nature of ground and airborne disturbances that propagate outward from a blast. In this discussion, it is assumed that no permanent displacements are produced. Thus the only effects are those associated with the vibratory response of facilities in or on the rock or soil mass surrounding the blast. Transient means that the peak displacement is only temporary and lasts <1/100th of a second, and the structure returns to its original position afterward.

Transient structural effects can be arranged to reflect the expected distance from a blast. Beginning with the closest, transient effects are in this order: structural distortion, faulted or displaced cracks, falling objects, cosmetic cracking of wall coverings, excessive instrument and machinery response, human response, and microdisturbance. The first four effects, those that relate to structural response, are normally grouped together for experimental observation as structural response and do not normally occur when vibration levels are regulated to prevent cosmetic cracking.

Excessive structural response has been separated into three categories (Northwood et al. 1963; Siskind et al. 1980b). In order of declining severity and increasing distance of occurrence, beginning with effects that occur closest to the blast, the categories are as follows:

1. Major (permanent distortion): Seriously weaken the structure (e.g., large cracks or shifting of foundations or bearing walls, major settlement resulting in distortion or weakening of the superstructure, and walls out of plumb)
2. Minor (displaced cracks): Are surficial, and do not affect the strength of the structures (e.g., broken windows, loosened or fallen plaster, hairline cracks in masonry)
3. Threshold (cosmetic cracking): Open old cracks and formation of new plaster cracks, dislodging of loose objects (e.g., loose bricks in chimneys)

These specific definitions of response should not be described collectively as "damage." To do so blurs the distinction between threshold or cosmetic cracking and major response or structural distress.

Regulatory controls in North America are based on the occurrence of threshold cracking of plaster and gypsum wallboard in residential structures (Siskind et al. 1980a; Dowding 1985, 1996). Observed cracking is cosmetic in nature and does not affect structural stability. These cosmetic cracks are hair-sized and are similar to cracks that occur during the natural aging of structures. In fact, they are indistinguishable from cracks that result from natural aging. Control limits are based on direct observations of test homes immediately before and immediately after blast events, to avoid confusion with similar cracks that might occur from natural processes. These controls do not apply to engineered structures that are constructed of steel and concrete, buried structures, or adjacent rock.

Distinction Between Blast-Induced Cracking and Natural Cracking

Control of blast-induced transient effects to prevent threshold or cosmetic cracking reduces blast-induced displacement or strains in structures to or below that caused by everyday human activities and weather changes (Stagg et al. 1984; Dowding 1988). These cosmetic cracks in many cases are smaller than cracks caused by other natural or occupant-initiated processes that are active in all constructed facilities. Thus blast-induced threshold cracks can be scientifically observed only with visual inspection immediately before and after each blast. Observations made under less stringently controlled conditions have little scientific merit because of the high probability of environmentally produced cracks occurring between or before visual inspections.

Several institutional references present excellent summaries of the multiple origins of cracks (BRE 1977; Association of Casualty and Surety Companies 1956; Thoenen and Windes 1942). Basically, cracks are caused by the following:

- Differential thermal expansion
- Structural overloading
- Chemical changes in mortar, bricks, plaster, and stucco
- Shrinkage and swelling of wood
- Fatigue and aging of wall coverings
- Differential foundation settlement

Table 16.3-1 Strain levels induced by daily environmental changes and household activities with their corresponding blast levels

Loading Phenomena	Site*	Microstrain Induced by Phenomena, μin./in.	Corresponding Blast Level†	
			in./s	mm/s
Daily environmental changes	K_1	149	1.2	30.0
	K_2	385	3.0	76.0
Household activities				
Walking	S_2	9.1	0.03	0.8
Heel drops	S_2	16.0	0.03	0.8
Jumping	S_2	37.3	0.28	7.1
Door slams	S_1	48.8	0.50	12.7
Nail pounding	$S_{1,2}$	88.7	0.88	22.4

Source: Stagg et al. 1984.
*K_1 and K_2 were placed across a tape joint between two sheets of gypsum wallboard.
†Blast equivalent based on envelope line of strain vs. ground vibration.

Over time, all of the causes listed are likely to crack walls, whether or not blasting occurs. This list has three important implications: (1) Structures expand and contract preferentially along existing weaknesses (cracks). Seasonal expansion and contraction along these cracks return patching and repainting to the original cracked state within several years. This persistent cracking is annoying to owners who are unaware of the difficulty of patching existing cracks of any kind. (2) The distortion that caused the cracking also creates stress concentrations that may lower the resistance of a wall covering to vibration cracking; however, current regulatory limits already implicitly include these distortion effects as explained in the paragraphs that follow. (3) Natural cracks continue to occur over time. Therefore, any postblast inspection at low vibration levels is likely to find new cracks from natural aging unless preblast inspection is conducted immediately before the blast.

Response of Structures to Everyday Activities

A comparison of relative levels of strain gives perspective to the observation of cracking at low particle velocities. Table 16.3-1 compares strain levels from daily environmental changes (temperature and humidity) and from household activities measured in the U.S. Bureau of Mines (USBM) test house, and lists their corresponding blast levels; a door was slammed adjacent to the wall on which the strains were measured.

Apparently, in the course of daily life, an active family produces strains in walls similar to the strains produced by blasting vibrations of 0.1 to 0.5 in./s (2.5 to 12 mm/s). Most astonishing are the measurements, in a wood-framed house, of relatively enormous strains from daily changes in temperature and humidity. These strains alone are great enough to crack plaster.

Blast-Induced Air Overpressures

Blast-induced air overpressures are the air pressure waves generated by explosions. The higher-frequency portion of the pressure wave is audible and is the sound that accompanies a blast; the lower-frequency portion is not audible, but it excites structures and in turn causes a secondary and audible rattle within a structure.

Overpressure waves are of interest for three reasons: (1) the audible portion produces direct noise; (2) the inaudible portion by itself or in combination with ground motion can produce structural motions that in turn produce noise; and (3) they may crack windows, although air-blast pressure alone would have to be unusually high for such cracking. Response noise within a structure is the source of many complaints. Blast-induced air overpressures often generate sounds by means of the response of components such as loose windows and doors or highly flexible structural panels. Delicately balanced loose objects may rattle or fall at low levels of motion, generating even further concern.

Human Response

Humans are quite sensitive to motion and noise that accompany blast-induced ground and airborne disturbances. Therefore, human response is significant in the reporting of blast-induced cracking. Motion and noise from blasting can be startling and lead to a search for some physical manifestation of the phenomena. Frequently, a previously unnoticed crack provides such confirmation of the event. Furthermore, if a person is worried and observes a crack that was not noticed before, the crack's perceived significance increases over that of one noticed in the absence of any startling activity. These concerns are real and, in the mind of the observer, are sincere.

In typical mining situations, significant blast-induced inaudible air overpressure and audible noise immediately follow the ground motion and intensify human response. Both ground and airborne disturbances excite walls, rattle dishes, and together tend to produce more noise inside a structure than outside. Thus both the audible noise and the wall rattle produced by inaudible pressures contribute to human response. To complicate matters even more, inaudible air overpressures can vibrate walls to produce audible noise at large distances, and are inaccurately reported by occupants as ground motions because the audible effects are so similar.

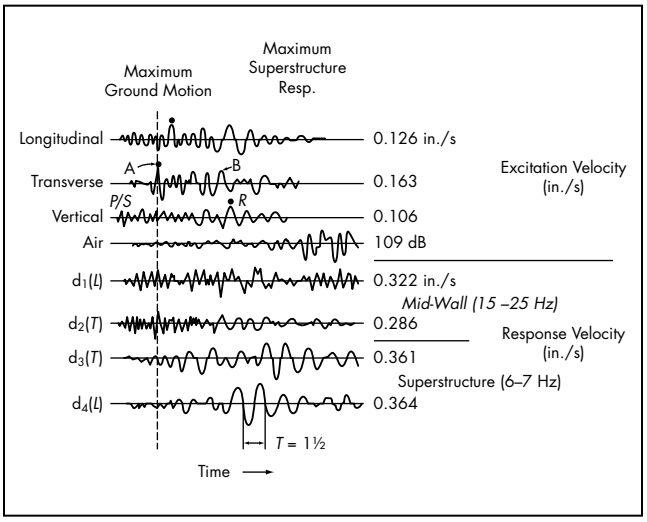
CHARACTER OF BLAST EXCITATION AND STRUCTURAL RESPONSE

Figure 16.3-1 shows blast excitations by ground and airborne disturbances and the response of the walls and superstructure of a residential structure. Measurements were made about 2,000 ft (600 m) from a typical surface coal-mining blast. Both ground and airborne disturbances (the upper four time histories) produce structure responses (the lower four time histories). Because of the importance of frequency, the full waveform or time history should be recorded. When a critical location is known, blast response is best described by the strain at that location. Alternatively, particle velocity (shown in the figure) can be measured outside the structure of concern, as many recent cracking studies have correlated cracking with excitation particle velocity measured in the ground.

Ground Motion

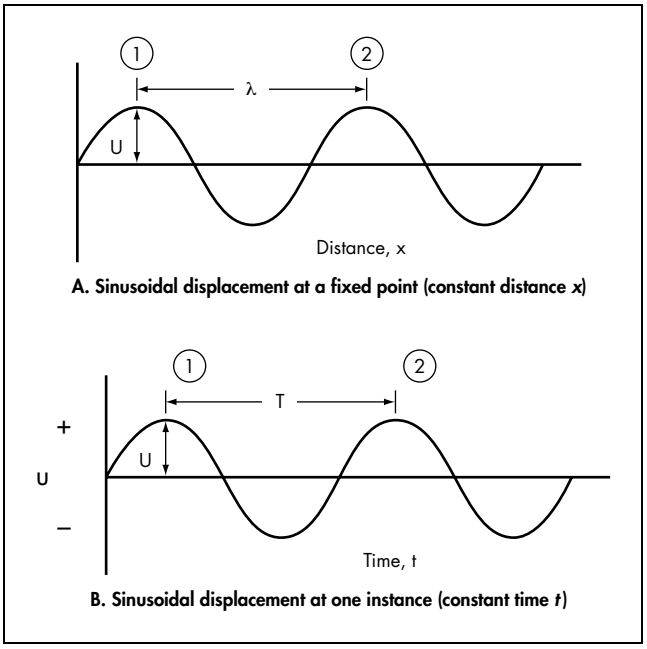
Ground motion can be described by three mutually perpendicular components: longitudinal (L), transverse (T), and vertical (V) (Figure 16.3-1). The L and T directions are oriented in the horizontal plane with L directed along the line between the blast and recording transducer. When a study focuses on structural response, axes can be labeled H1, H2, and V, with H1 and H2 oriented parallel to the structure's principal axis.

Variation of peak motions in each component (L, V, and T in Figure 16.3-1) has led to difficulty in determining which is most important. Horizontal motions seem to control the horizontal response of walls and superstructures; vertical motions



Source: Adapted from Dowding 1996.

Figure 16.3-1 Blast excitations by ground and airborne disturbances and the response of the walls and superstructure of a residential structure



Source: Dowding 1985.

Figure 16.3-2 Sinusoidal approximations

seem to control the vertical response of floors. In an absolute sense, the peak ground motion is actually the maximum vector sum of the three components, which usually occurs at the largest peak of the three components, the dashed line in the figure. This *true* maximum vector sum is not the *false* maximum vector sum calculated with the maxima for each component (dots in the figure), no matter their time of occurrence. The false maximum vector sum may be as much as 40% greater than the true maximum vector sum, which is normally 5% to 10% greater than the maximum single-component peak.

In general, experimental observations of threshold or cosmetic cracking, which form the basis of blasting controls in North America, have been correlated with the maximum single component regardless of direction. Therefore, use of the false maximum vector sum, for control, provides a large unaccounted-for factor of safety.

Two wave types are produced by blasting, *body* and *surface*, and are illustrated by the ground motion in Figure 16.3-1. Body waves travel through earth materials; surface waves travel along surfaces and interfaces of earth materials. The most important surface wave is the Rayleigh surface wave, denoted *R* on the vertical trace in the figure. Body waves can be further subdivided into *compressive* (compression/tension) or soundlike waves and *distortional* or *shear* waves, denoted *P*/*S* on the vertical trace in the figure. Explosions produce predominantly body waves at small distances, which propagate outward in a spherical manner until they intersect a boundary such as another rock layer, soil, or the ground surface. At this intersection, shear and surface waves are produced. Rayleigh surface waves become important at large transmission distances, as shown in the vertical trace by the relatively large amplitude of *R* compared to that of *P*/*S*.

Sinusoidal Approximations

Typical blast vibrations, no matter the wave type, can be approximated as sinusoidally varying in either time or distance along the radial or longitudinal line, as shown by the time variations in Figure 16.3-2. This approximation is useful because it makes calculations for strain and for acceleration from particle velocity much simpler than they are for irregular pulses. Ground motion from a blast is similar to the motion of a floating cork caused by a passing water wave produced by dropping a stone in a pond. Displacement of the cork from its at-rest position is similar to the displacement u of a particle in the ground from its at-rest position. Similarly, the cork's velocity u as it bobs up and down is analogous to that of a particle in the ground; hence the term *particle velocity.*

The water wave that excites the cork can be described by its wavelength γ, which is the distance between wave crests; the wave speed or propagation velocity c at which it travels outward from the stone's impact; and the frequency f, which is how many times the cork bobs up and down in 1 second. Frequency f is equal to $1/T$, or the reciprocal of the period (time that it takes the cork to complete one cycle of motion). Frequency is measured in cycles per second or Hertz (Hz). Propagation velocity c should not be confused with particle velocity $\mathring{u}$, as c is the speed at which the water wave passes by the cork and $\mathring{u}$ is the speed at which the cork moves up and down while the wave passes. In the same fashion as for the water wave, blast-vibration waves can be described by their wavelength, propagation velocity, and frequency.

Kinemetric Relationships of Ground Motion

The general form for the sinusoidal approximation is best understood by beginning with the equation for sinusoidal displacement u:

$$u = U \sin(2\pi ft) \qquad (16.3\text{-}1)$$

where

U = maximum displacement
f = frequency
t = time

The relationship between maximum particle displacement u, maximum particle velocity $\mathring{u}_{max}$, and maximum

acceleration $\ddot{u}_{max}$ is also greatly simplified by the sinusoidal approximation and is found by differentiation with respect to time whenever the sin/cos function maximizes at l:

$$\begin{aligned} u_{max} &= U \\ \mathring{u}_{max} &= U\,(2\pi f) = 2\pi f u_{max} \\ \ddot{u}_{max} &= U\,(4\pi^2 f^2) = 2\pi f \mathring{u}_{max} \end{aligned} \qquad (16.3\text{-}2)$$

Usually, acceleration is normalized (i.e., divided by gravitational acceleration, 386.4 in./s^2 [9,814 mm/s^2]). Therefore, an acceleration of 79 in./s^2 (2,000 mm/s^2) is 2,000/9,814 = 79/386 = 0.2 g, or two-tenths that of gravity.

Kinemetric relations among particle displacement, velocity, and acceleration for complex waveforms are exactly related by integration or differentiation of any of the waveforms. For instance, integration of an acceleration time history gives a velocity time history; integration of the latter in turn gives a displacement time history. Even though a particle-velocity record can be differentiated to find acceleration, it is not recommended, as the procedure is sensitive to small changes in the slope of the velocity time history. Further discussion of the inaccuracies of differentiation and integration can be found in Dowding (1985) and in texts devoted to interpretation of time histories (e.g., Hudson 1979).

Transient Nature of Blast Motions

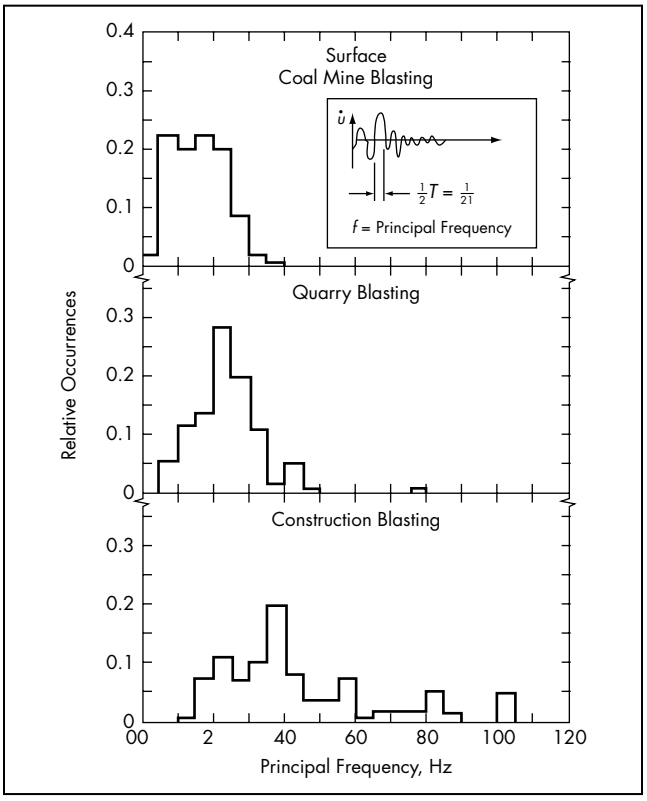
Great care should be taken not to confuse the effects of steady-state, single-frequency motions with those of transient blast motions. Most vibration studies conducted by personnel trained in mechanical and electrical engineering and in geophysics implicitly assume that the motions are continuous (last many cycles) and steady-state (have constant frequency and amplitude). However, as is evident in Figure 16.3-1, blast-induced motions last only one or two cycles at a relatively constant amplitude and frequency. Such conditions are not similar enough to steady-state motion to allow specific application of steady-state approximations such as resonance.

Estimation of Dominant Frequency

The use of frequency-based vibration criteria has made the estimation and calculation of dominant frequency an important concern. Dominant frequency can be estimated by visual inspection of the time history or calculated from Fourier frequency spectra or, alternatively, response spectra.

The accuracy or difficulty of visually estimating the dominant frequency depends on the complexity of the time history. The easiest type of time-history record for estimating frequency is one with a single dominant pulse like that shown in the inset of Figure 16.3-3. This dominant frequency can be determined by hand measurement of the time of the two zero crossings on either side of the peak. The difference between these times is one-half of the period, which is the inverse of twice the frequency of the dominant peak, as show in the inset.

Figure 16.3-3 shows that the relatively large blasts produced by surface coal mining, when measured at typically distant structures, tend to produce vibrations with lower principal frequencies than do construction blasts. Construction blasts involve smaller explosions, but the typically small distances between a structure and a blast, as well as rock-to-rock transmission paths, tend to produce high frequencies. The high-frequency motions associated with construction blasts are less apt to crack adjacent structures (Dowding 1985).



Source: Adapted from Siskind et al. 1980b.

Figure 16.3-3 Dominant-frequency histograms of blasts for three types of industry, measured at structures adjacent to the blast (the inset defines dominant frequency)

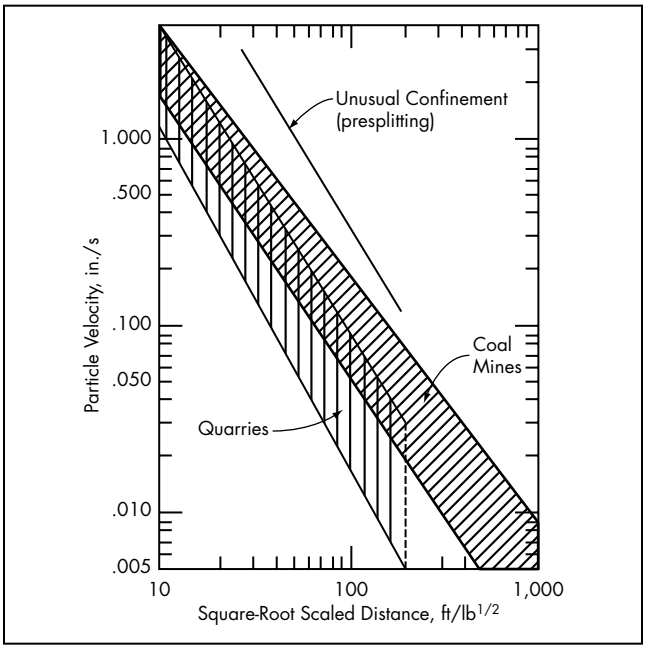
The most difficult type of record to interpret is one that contains nearly equal peaks at two dominant frequencies, such as that shown Figure 16.3-1. The two dominant frequencies are the initial 15- to 20-Hz portion (peak A) and the later 5- to 10-Hz portion (peak B). The initial portion produces the highest wall response; the second portion produces the greatest superstructure response. For the best frequency correlation of both types of response, both frequencies should be calculated.

The best computational approach to determining the dominant frequency involves the response spectrum. The response spectrum is preferred over the Fourier frequency spectrum because it can be related to structural strains (Dowding 1985). A compromise approach is to calculate the dominant frequency associated with each peak by the zero-crossing method described previously.

Since histories do not often contain as broad a range of dominant frequencies as that in Figure 16.3-1, most approaches require only the calculation of the frequency associated with the maximum particle velocity for blasts that produce small particle velocities. More complex frequency analyses are needed only when peak particle velocities approach control limits.

Propagation Effects

Ground motion always decreases with increasing distance. The effects of constructive and destructive interference and geology are included in the scatter of data for the mean trend of the decay of amplitude with distance. Although this scatter is



Source: Adapted from Siskind et al. 1980b.

Figure 16.3-4 Attenuation relationships showing scatter from geological and blast design effects as well as high expected velocities from confined shots such as presplitting

large, the associated decay with distance is observed in all blast-vibration studies. Typical examples of this decay are shown in Figure 16.3-4 where maximum particle velocity is plotted as a function of square-root-scaled distance from the blast.

Square-root scaling, or plotting peak particle velocity as a function of the distance R divided by the square root of the charge weight, $R/W^{1/2}$, is more traditional than cube-root scaling, which incorporates energy considerations (Hendron 1977). Both square- and cube-root scaling can be used to compare field data and predict the attenuation or decay of peak particle velocity; however, square-root scaling is more popular.

Several square-root attenuation relationships used in the United States are shown in Figure 16.3-4. They are banded to reflect scatter, which is typical of blasting operation. The upper (unusual confinement) line should be used for presplitting, cratering, and beginning new bench levels. It is also the basis for U.S. Office of Surface Mining Reclamation and Enforcement (OSM) regulations for conservative shot design when monitoring instruments are not used.

Dominant frequencies tend to decline with increasing distance and increasing importance of surface waves. At larger distances typical for mining, higher-frequency body waves begin to have relatively lower peak amplitudes than do lower-frequency surface waves, as shown in Figure 16.3-1. Since lower frequencies can induce greater structural response (Medearis 1976), OSM scaled-distance limits decline with increasing absolute distance.

Blast-Induced Air Overpressures

As for ground motions, the waves from blast-induced air overpressures can be described with time histories as shown in Figure 16.3-1. The higher-frequency portion of the pressure wave is audible sound. The lower-frequency portion, although not audible, excites structures, causing a secondary and audible rattle within the structure. Air-blast excitation of the walls is evident from a comparison of the last one-quarter of the time histories of air blast and wall response in Figure 16.3-1. Unlike ground motions, air overpressures can be described completely with only one transducer, since at any one point air pressure is equal in all three orthogonal directions.

Propagation of blast-induced air overpressures has been studied by numerous investigators and is generally reported with cube-root-scaled rather than square-root-scaled distances. Peak pressures are reported in terms of decibels (dB), defined as

$$\text{dB} = 20 \log_{10} (P/P_0) \tag{16.3-3}$$

where

P = measured peak sound pressure
P_0 = reference pressure of 2.9×10^{-9} psi (20×10^{-6} (P_a)

Figure 16.3-5 summarizes the effect of two important instrumentation and shot variables:

1. The effect of the weighting scales in the sound-pressure-level meter is dramatically evident. C weighting greatly reduces the recorded peak pressure at any scaled distance. This does not mean that the peak is reduced by changing instruments, but rather that the C weighting system does not respond to low-frequency pressure pulses. However, low-frequency pressure peaks excite structures and occupants whether or not they are sensed by measurement instruments. The other (5- and 0.1-Hz) labels denote the lower-frequency bounds of the recording capabilities of "linear" systems.
2. The effect of venting caused by inadequate stemming is also evident from the higher average pressures produced by the parting shots at any scaled distance. Parting shots are detonated in thin rock layers between coal strata in surface mines. Consequently, there is less hole height available for stemming, and these shots frequently eject the stemming and thereby produce abnormally high air overpressures. The unconfined relationship should be used for the demolition of structures after modification for the effects of weather and ground reflection.

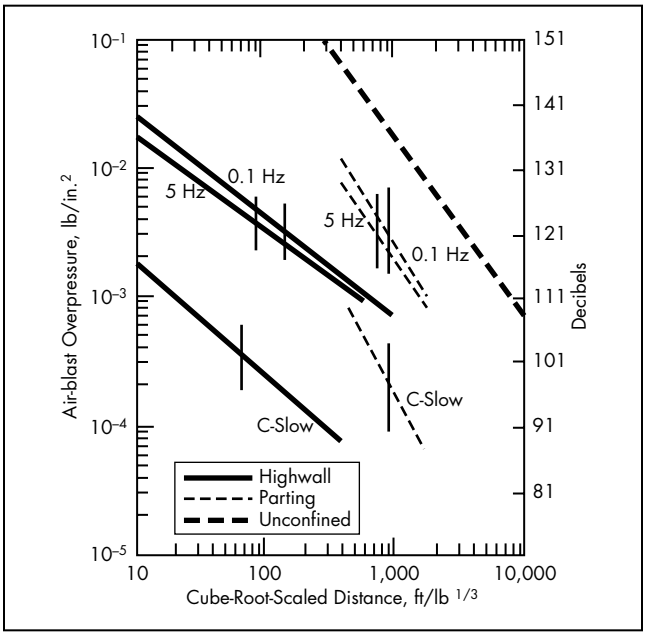
Various effects of wind have been reported and should be added to the average relations presented in Figure 16.3-5. Wiss and Linehan's (1978) study of air overpressures produced by surface coal mining show that, in moderate winds, the typical 7.7-dB reduction for each doubling of distance is reduced by

$$7.7 - 1.6\, V_{\text{mph}} \cos\theta \text{ dB} \tag{16.3-4}$$

where

V_{mph} = wind velocity, in miles per hour
θ = the angle between the line connecting the blast and transducer and the wind direction

An air-temperature inversion causes the sound-pressure wave to be refracted back to the ground and at times to be amplified at small 16-acre- (0.65-km^2-) sized locations. Such inversions occur when the normal decrease in temperature with altitude is reversed because of the presence of a warmer upper layer. Schomer et al. (1976) showed that, for propagation distances of 2 to 40 mi (3 to 60 km), inversions produce zones of intensification of up to 3 times the average attenuated or low air overpressures at those (but not smaller) distances;



Source: Adapted from Siskind et al. 1980a.

Figure 16.3-5 Attenuation relationships for air overpressures produced by confined (highwall) and partially confined (parting) surface coal-mining blasts and unconfined blasts

the average increase is 1.8 times (5.1 dB). At distances of <2 mi (3 km), where high air overpressures are likely to occur, his measurements show no inversion effects.

Structural Strains vs. Particle Velocity

Although particle velocity is the traditional measurement of choice, structural strains control cracking. They should be measured directly from relative displacements on structures or within rock masses when critical locations are known, and can be obtained with a variety of strain and relative displacement gauges (Stagg et al. 1984). Unfortunately, critical locations may be either unknown or too many in number to measure economically. Therefore, some means of estimation is necessary.

Ground-motion and air-overpressure time histories can be used to calculate the relative displacement of structural components with a knowledge of the responding structure's dynamic-response characteristics (Dowding 1985, 1996). These relative displacements can in turn be used to calculate strains. The accuracy of these estimates is limited by the degree to which the structure behaves as a single-degree-of-freedom system and the accuracy of the estimate of dynamic response characteristics.

Measurement of Particle Velocity

Although any of the three kinemetric descriptors (displacement, velocity, or acceleration) can be used to describe ground motion, particle velocity is the most preferable. It has the best correlation with scientific observation of blast-induced cracking, which forms the basis of vibration control. Furthermore, it can be integrated to calculate displacement. If acceleration is desired, it should be measured directly to avoid differentiation of the particle-velocity time history.

The location for measurement varies throughout the world. In North America, excitation or ground motion is measured on the ground adjacent to the structure of interest. In Europe, it is measured on the structure's foundation. The difference stems from historical precedent and location of transducers during scientific observation of cracking rather than from difference in philosophy. In North America, it is frequently impossible to place transducers on adjacent property owned by a party not involved in the project. Excitation motions should be measured outside of and not on the structure. Structural-response motions should be measured on the most responsive structural members, which are not the basement or foundation walls because of the restraint provided by the ground.

Time histories of the three components of motion should be measured because of the importance of excitation frequency. Recording of peak motions alone does not yield information about the dominant frequency and time history details that control structural response. Peak motions and dominant frequency can be used to describe low-level, noncritical motions. Therefore, machines used to monitor critical motions should be capable of recording time histories of selected critical motions. Machines that record only peak motions can be used with those that record time histories to provide redundant measurement where frequency content does not vary widely.

USE OF MEASUREMENT INSTRUMENTS

This section describes the characteristics of instruments that measure ground motions (acceleration, velocity, displacement) and air blasts (air overpressure). Since there are many excellent sources for information on instruments, the principal characteristics of available systems are summarized rather than exhaustively reviewed. The most complete single reference for detailed instrumentation information is the *Shock and Vibration Handbook* (Harris and Crede 1976).

Almost all field-portable vibration-monitoring systems are now built around a digital central processing unit (sometimes called a data logger) and an analog-to-digital (A/D) converter. Geophone or air-pressure transducers convert time-varying velocity ground motion or air pressure into an analog (or continuous) voltage signal. The A/D converter translates this analog signal into digital (or discrete) form at a constant rate of sampling (typically 1,000–2,000 samples per second) and saves the data to a file in system memory. The onboard computer or central processing system analyzes the data and generates reports for download to and display on a portable computer. Typical A/D converters subdivide the transducer voltage range into 2^{12} (4,096) or more divisions; some new systems are capable of 2^{24} subdivisions. Every year, sampling speed increases and A/D converters become more precise. Some of these systems are self-contained and are the size of a brick. Others connect to exterior transducers and are the size of a thick notebook computer. In development are systems the size of a deck of playing cards that may eventually replace current instruments, provided that smaller batteries become available.

Transducers

Transducers are one of the weaker links in the measurement system because they must translate kinemetric motions or pressures into electrical signals. The remaining components transform electrical signals or light beams and are not restricted by mechanical displacement. The main characteristics of transducers that affect their performance are sensitivity and frequency response.

For energy-converting transducers (i.e., those that do not require an energy source), instrument sensitivity is the ratio of

the electrical output to the kinemetric displacement, velocity, and acceleration or overpressure. Since allowable limits are specified in terms of ground-particle velocity, all blast monitors come equipped with velocity gauges.

Frequency response is the frequency range over which electrical output is constant with constant mechanical motion. This constancy is normally expressed in terms of decibels. For instance, "linear within 3 dB between 5 and 200 Hz" means that the transducer produces a voltage output that is constant within 30% between 5 and 200 Hz, and up to 450 Hz on some European machines. Generally, it is better to look at the transducer's response spectra (such as those shown in Figure 16.3-6) to determine the frequencies where this difference occurs. For example, the difference occurs at low frequencies for the velocity transducers in the figure. At 70% of critical damping, the velocity transducer in the figure is ±3 dB (±30%) down at 1 Hz.

One of the most critical aspects of vibration monitoring is how the transducer is mounted in the field, which depends on the particle acceleration of the wave train being monitored. When the measurement surface is horizontal and the vertical maximum particle acceleration is <0.3 g, the type of mounting is not particularly critical, as the possibility of rocking is small and the transducer needs no device to hold it. When the measurement surface consists of soil and vertical maximum particle acceleration is in the range 0.3–1.0 g, the transducer should be buried completely (Johnson 1962). When the measurement surface consists of rock, asphalt, or concrete, the transducer should be fastened to the measurement surface with double-sided tape, epoxy, or quick-setting cement (Hydrocal or other gypsum-based cements set in 15–30 minutes). If these methods are unsatisfactory or accelerations exceed 1.0 g, only cement or bolts are sufficient to hold the transducer to a hard surface.

All transducers mounted on vertical surfaces should be bolted in place. Air-overpressure transducers should be placed at least 3 ft (1 m) aboveground, pointed downward to prevent rain damage and fitted with a windscreen to reduce wind-excitation-induced false events.

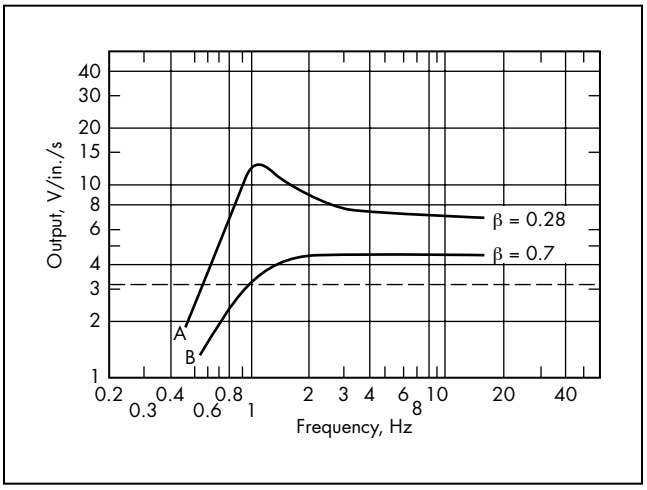
Instrument Calibration

It is obvious that the entire vibration-measurement system should be calibrated, as it is futile to record data if they cannot be exploited because of a lack of reference. Manufacturers supply calibration curves with their instruments that are similar to the response spectra for transducers shown in Figure 16.3-6. Recalibration or checking requires special vibrating platforms where frequency and displacement are controlled, and, in the field, a calibrating circuit to pulse the magnetic core of the geophone (Stagg and Engler 1980).

Number of Instruments

Although the obvious irreducible number of instruments for each blast is one, two instruments provide more thorough documentation of the spatial distribution of effects. If only one instrument is used, it should be located at the nearest or most critical receiver. This single instrument should be what we term a Type I instrument and should record time histories of the three axes of particle velocity as well as air overpressure. Since it must monitor continuously, it must trigger (begin recording) automatically and be capable of monitoring even while printing or communicating results.

When blasting will occur at more than one general location (i.e., will involve nearest structures separated by



Source: Dowding 1985.

Figure 16.3-6 Sample response spectra for velocity transducers with differing percentages of damping

hundreds of feet or meters), then minimally two and optimally four instruments should be used, including a spare to ensure continuous coverage in case of failure.

Additional instruments other than the spare can be Type II, which provide a lesser level of service. They must at least continuously record the peak particle velocity in one axis and may or may not measure air overpressure. The best axis is the vertical, since no horizontal-direction decision is required and surface waves usually involve a significant vertical component regardless of the direction of the maximum horizontal component. These instruments should be located at distances greater than the nearest structure to monitor a large area.

The spare instrument can be either Type I or II. Where air overpressures are problematic or frequencies are critical, the spare should be Type I. The spare instrument can also be used to monitor sites where complaints develop. Such public-relations work is essential in North America where lawsuits arise even when all blast effects comply with regulatory guidelines.

This approach describes the least number of instruments. Applicable regulations and mining schedules may require a larger number.

Instrument Deployment During Test Blasts

When blasting projects begin, when geological conditions change radically, or when new initiation systems are introduced, test blasts should be conducted to minimize the number of instruments necessary to monitor production blasts. The test blasts should produce project-specific attenuation relationships for both air overpressures and ground motion. Such relationships vary from project to project because of changes in geology and blasting practices. Additionally, the test blasts should enable determination of the frequency of motions at different scaled and absolute distances. Frequency is important in estimating structural response by response-spectrum analyses.

The attenuation relationship is not solely a site property. Although it depends on geology, it also depends heavily on blast geometry and timing. For instance, with the same charge per delay, a blast with a larger burden produces an attenuation relationship with a similar slope or decay with distance but a larger intercept. Furthermore, differing initiation timing changes the time history in terms of both length and frequency.

During test blasts, at least four instruments should be used to measure peak particle velocities at widely differing scaled distances for the same blast. Therefore, for any one blast design, parameters and initiation sequences are constant, and the resulting attenuation relationship shows the effect of only distance, direction, and/or geology. Seismographs and/or transducers should be placed along a single line with constant geology to best determine the attenuation relationship, or at all critical structures to determine the effects of direction and variable geology. Ideally, linear orientation should be along a path with constant soil thickness and not cross any large geologic discontinuities such as faults. If geology changes radically, then two such attenuation lines are necessary, but not necessarily with each blast.

EVALUATION OF MEASUREMENTS

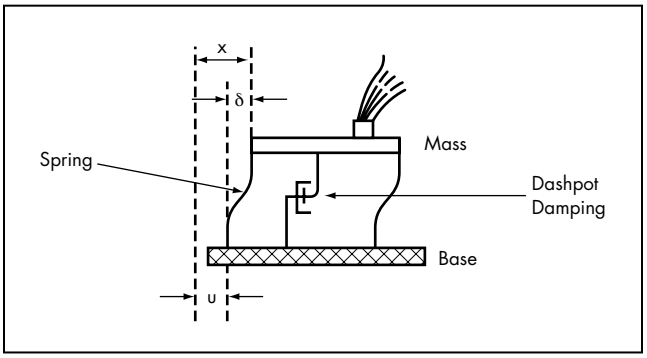
Documentation of blast effects involves two radically different endeavors: measurement of ground and air disturbances as well as observation of cosmetic cracking. Measurement can now be accomplished remotely with computers to eliminate completely human interaction, whereas scientific observation must involve meticulous human inspection immediately before and after a blast. While the focus of this section is instrumental monitoring, the alleged appearance of cracking by neighboring property owners is nonetheless a very serious consideration.

Principal problems in the evaluation of measured effects involve (1) accounting for geologic and weather effects on the overall attenuation with a small number of instruments and (2) incorporating structural response and frequency effects. Principal problems with the observation of blast-induced cracks involve (1) discriminating between blast-induced cracks and environmentally and human-induced cracks and (2) reducing the enormous amount of time necessary for direct observation. Observational problems are normally overcome by using instrumentally measurable blasting controls at levels that are low enough to avoid the threshold of cosmetic cracking even for old, degraded structures or to eliminate cracking altogether. Otherwise blast-induced cracks can be observed only by immediate before-and-after blast inspection.

Structural Response and Frequency Effects

Structures respond to both ground and airborne disturbances, as shown by the bottom four time histories in Figure 16.3-1. Walls respond more to the higher-frequency (15–20 Hz) waves that occur early during ground motion; the superstructure or overall skeleton of the structure responds more to the lower-frequency (5-Hz) waves that occur late during ground motion. Walls are again excited by the arrival of the air-pressure wave. Structural response can be calculated from ground motion if the natural frequency and damping of structural components are known or estimated.

Langan (1980) showed that measured structural response has a higher correlation with calculated single-degree-of-freedom (SDF) response than with peak ground motion. Therefore, structural motions can be estimated more accurately by assuming that they are proportional to response-spectrum values at a particular structure's natural frequency than by assuming that they are proportional to the peak ground motion. This improved correlation is largely a result of the consideration of frequency in the response spectrum, which is calculated from the SDF response.



Source: Dowding 1985.

Figure 16.3-7 Single-degree-of-freedom model of a house, showing the relative displacement of the walls δ, and the analogy between model mass and roof mass, model stiffness, and wall stiffness

Origin of the SDF Model

One of the critical structural response factors is the amount of differential displacement (δ in Figure 16.3-7) that occurs between or along structural members, because it is proportional to strain, which, in turn, causes cracking. Such displacements can be computed by mathematical idealization of the SDF model shown in the figure.

To make calculations practical, it is necessary to simplify most structures. The fundamental characteristics of a structure that govern its behavior under vibratory or dynamic loading are (1) the masses of the main components (analogous to floor and roof masses), (2) the spring stiffness of the main components (analogous to wall stiffness), and (3) the amount of damping or energy dissipation (analogous to differential movement in cracks, joints, and connections). The behavior of one- and two-story buildings is directly analogous to the behavior of an SDF system when movement in only one direction is considered. For multiple-story structures, it is necessary to model the structure as multiple-degrees-of-freedom systems. However, even such systems can be idealized as SDF systems to enable calculation of the fundamental mode of response.

If a structure's damped natural frequency f_d and its fraction of critical damping β are known, the values of dynamic properties f_d and β can be accurately measured from a free-vibration time history of the building response. These measured parameters automatically account for factors that are difficult to quantify, such as the degree of fixity of the columns and the damping coefficient. As shown in Figure 16.3-l, these parameters can be measured from the structure's free response. The time between peaks is the period $T = 1/f_d$ and the decay of free oscillation is proportional to the damping β.

Estimation of Dynamic Response Properties

The fundamental natural frequency f_d of the superstructure of any tall building can be estimated from compilations of work in earthquake engineering (Newmark and Hall 1982):

$$f_d = 1/0.1N \qquad (16.3\text{-}5)$$

where N is the number of stories. Substitution for N of 1 and 2 for residential structures yields f values of 10 and 5 Hz for one- and two-story structures, respectively, which compares favorably with the results of actual measurements.

Table 16.3-2 Natural frequencies for unusual structures

Type of Structure	Height, m	*f*, Hz
Radio tower	30	3.8
Petroleum distillation tower	21	1.2
Coal silo	60	0.6
Bryce Canyon rock pinnacle	27	3

Source: Medearis 1975; Dowding and Kendorski 1983.

Damping β is a function of building construction and, to some extent, the intensity of vibration. Thus it cannot be simplified as easily as can the natural frequency. Measurements reveal a wide range of damping values for residential structures, with an average of 5% (Dowding et al. 1981). This value is appropriate for initial estimates for taller engineered structures. Further details for engineered structures are available in Newmark and Hall (1982).

Walls and floors vibrate independently of the superstructure and have their own but similar fundamental frequencies of vibration in the range 12–20 Hz and averaging 15 Hz (Dowding et al. 1981). Floors in office buildings with large floor spans tend to have natural frequencies that are lower than those of floors in residential structures, but similar to those of walls in residential structures.

Dynamic-response properties of some unusual tall structures cannot be estimated with the 1/0.1*N* equation (Equation 16.3-5). Field-measured natural frequencies for these types of structures are given in Table 16.3-2.

Response Spectra

The pseudo-velocity response spectrum of a single ground motion, such as that of the seven-delay quarry blast in Figure 16.3-8, is generated from the relative displacement δ_{max} values of a number of different SDF systems when excited by that motion. In the figure, the time history is operated on by the SDF equation to produce a computed relative displacement δ, which is then multiplied by the circular natural frequency ($2\pi f$ or $2\pi 10$ for point 1) to produce the pseudo-velocity response *PV*.

Consider two different components of the same structure: a 10-Hz superstructure and a 20-Hz wall. If the ground motions $\mathring{u}(t)$ of the quarry blast are processed twice by the SDF equation with $f = 10$ and 20 Hz and β is held constant at 3%, two δ_{max} values result. For the 10-HZ system, the associated values are

$$p = 2\pi(f) = 2\pi(10)$$

$$\delta_{max} = 0.25 \text{ mm } (0.01 \text{ in.}) \qquad (16.3\text{-}6)$$

$$PV_{10} = \delta_{max} = 2\pi(10)(0.25) = 15.7 \text{ mm/s } (0.62 \text{ in./s})$$

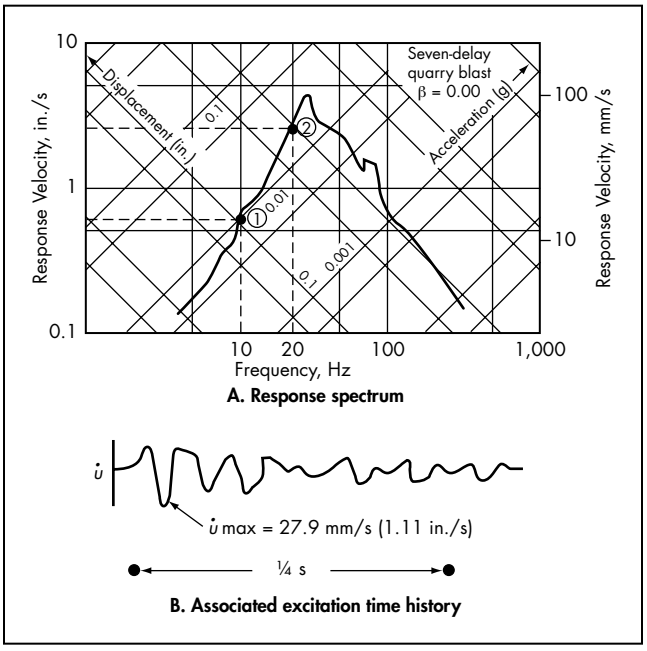
For the 20-Hz system, the associated values are

$$p = 2\pi(\text{f}) = 2\pi(20)$$

$$\delta\text{max} = 0.5 \text{ mm } (0.02 \text{ in.}) \qquad (16.3\text{-}7)$$

$$\text{PV20} = 2\,\pi(20)(0.5) = 63.5 \text{ mm/s } (2.5 \text{ in./s})$$

The *PV* values for the two systems are shown in Figure 16.3-8A as points 1 (PV_{10}) and 2 (PV_{20}). If the same ground-motion time history in Figure 16.3-8B is processed a number of times for a variety of *f* values and β is held constant, the resulting pseudo-velocities form the solid line in Figure 16.3-8A.



Source: Dowding 1985, 1986.

Figure 16.3-8 Construction of a pseudo-velocity response spectrum

Computers enable easy calculation of spectra from time histories. The two most commonly used spectra are the Fourier frequency and the pseudo-velocity response spectra. Although they are essentially different in meaning and typical use, they are similar for undamped response where the maximum motion occurs near the end of the time history (Dowding 1985; Hudson 1979). Since response spectra are calculated for damped response and peaks normally occur in the middle as well as the beginning of the time history, the two spectra are not usually the same.

Only the pseudo-velocity response spectrum can be used to directly calculate structural response. Because of the similarity of Fourier frequency and response spectra, either can be used to determine the dominant frequency in the ground motion.

Case Histories

Figure 16.3-9 compares time histories and response spectra from the longitudinal components for an urban construction blast and a surface coal mine blast. Although the peak particle velocities are similar—0.15 in./s (3.8 mm/s) for the construction blast (Figure 16.3-9A), and 0.13 in./s (3.3 mm/s) for the coal-mine blast (Figure 16.3-9B)—the response spectra in the frequency range 5–20 Hz differ greatly. This difference is greatest in the range of natural frequencies for residential structures and their components, 5–20 Hz. In this range, surface-mining motions produce response velocities that are 10 times greater than those for construction motions.

Urban Construction Blast

Ground motion induced by the construction blast was produced by a much smaller shot than that induced by the surface-mining blast. Five holes (12 ft [3.6 m] deep, 1.5 in. [38 mm] in diameter) were arranged in a single row. Each hole was charged with a stick of gelatin dynamite and initiated

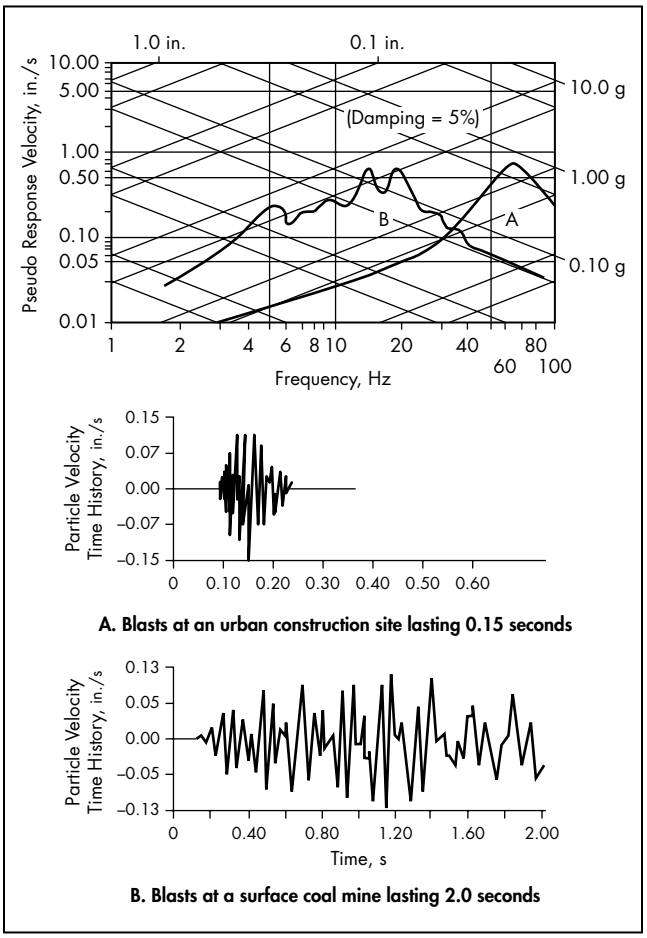


Figure 16.3-9 Time histories and response spectra

separately after a delay held constant at 25 ms. The burdens and hole spacings were small, approximately 2–3 ft (0.6–0.9 m). The total charge was 20 lb (9 kg), and the maximum charge per delay was 5 lb (2.3 kg).

The structure of concern, a historic theater, and the recording transducers were located 50 ft (15 m) away. The rock being fragmented consisted of granitized biotite schist. The shot produced peak particle velocities along the L, T, and V axes of 0.15, 0.16, and 0.28 in./s (3.8, 4.1, and 7.1 mm/s), respectively, with dominant frequencies between 75 and 125 Hz.

Surface Mine Blast

Ground motion induced by the surface-mining blast was produced by a multiple-row blast. Some 60 holes (83 ft [25 m] deep, 15 in. [380 mm] in diameter) were arranged in a four-row pattern. The burden between rows was 20 ft (6.1 m) and the hole spacing was 25 ft (7.6 m). Each hole contained four decks (charges that are detonated after an interval held constant at ≥17 ms). The ammonium nitrate and fuel oil (ANFO) charge weight per deck ranged from 100 to 130 lb (45 to 60 kg). Therefore, the largest charge per delay was 130 lb (60 kg), and the total charge was 27,700 lb (12,600 kg).

The geology between the blast and the transducer, which was located at the nearest residence, consisted of sedimentary rock with 10–30 ft (3–10 m) of overlying silty glacial till. A small gully (33 ft [10 m] deep) was located 1,300 ft (400 m) north of and between the blast and the transducer. The soil at the transducer was 10–15 ft (3–4 m) deep. Some 2,750 ft (825 m) separated the shot from the transducer, where the longitudinal velocity time history in Figure 16.3-9 with a peak of 0.13 in./s (3.3 mm/s) was recorded. The accompanying transverse and vertical peak particle velocities were 0.18 and 0.09 in./s (4.6 and 2.3 mm/s) with dominant frequencies between 11 and 13 Hz. Other similarly designed shots with distances between 1,900 and 2,700 ft (580 and 825 m) produced peak particle velocities between 0.17 and 0.23 in./s (4.3 and 5.8 mm/s) with dominant frequencies between 13 and 17 Hz.

These particle velocities are high at 2,700 ft (825 m), according to scaled-distance relationships. The high values may result from unusual confinement (too large a burden) of delay overlap. For instance, if two delays overlap so as to add, then the maximum charge per delay doubles and the square-root-scaled distance declines by 30%. Particle velocities measured at smaller-scaled distances are closer to expected levels.

Restrained Structures and Rock Masses

Aboveground structures such as homes and rock pinnacles, due to their capacity for free response, can selectively amplify incoming ground motions. In contrast, buried or restrained structures such as pipelines and rock masses cannot respond freely. Regardless of whether the response to strain is restrained or free, however, cracks are still initiated by strains. Strains in freely responding structures are proportional to the relative displacement between the ground and the superstructure, as shown in Figure 16.3-7; strains in restrained structures such as pipelines are usually those of the surrounding ground and can be approximated as the strain produced by plane-wave propagation (Dowding 1985, 1996):

$$\varepsilon = \mathring{u}_c/c_c \text{ and } \gamma = \mathring{u}_s/c_s \qquad (16.3\text{-}8)$$

where

ε = axial strain
$\mathring{u}_c$ = maximum compressive-wave particle velocity
c_c = compressive-wave propagation velocity
γ = shear strain
$\mathring{u}_s$ = maximum shear-wave particle velocity
c_s = shear-wave propagation velocity

For cases involving one critical location along a pipeline, pipe strains should be measured directly on the metal. For cases involving tunnels and/or cavern liners, critical strains can be estimated by calculation of the relative flexibility of the rock and liner (Hendron and Fernandez 1983).

CONTROL OF BLAST EFFECTS

When discussing blast design, it is important to mention the significance of burden. The two most familiar blast design parameters, charge per delay and powder factor, do not optimize the absolute burden between holes. Charge per delay (or instant of time) distributes the detonation process in time; powder factor distributes the detonation process in space. Unfortunately, powder factor alone cannot be used to calculate the maximum or optimum burden, which often controls external blast effects such as ground motion. Burden is often "designed" by general rules such as "*k* times borehole diameter." Proportioning maximum burden by borehole diameter tends to promote larger-diameter holes for a given powder factor, since the larger the hole diameter, the larger the burden

and thus the fewer the number of holes per unit volume and the lower the drilling cost. Unfortunately, large burdens for a given geology and rock type can lead to too large a separation of blastholes, which can lead to excessive back break and vibration.

One simple factor that can be used to check shot designs for excessively large burdens is the so-called drill factor, equal to (holes per unit surface area times hole depth)/unit volume. Experience with pipeline trench blasting reveals that shots designed with powder factors of 5.4 and 3.4 lb/yd^3 (3.2 and 2.0 kg/m^3), which does not seem like much of a difference, can produce vastly different results. These two shots involved 25 and 5 lb (11.4 and 2.3 kg) of explosive per running foot (0.3 m) of 16- and 10-ft- (5- and 3-m-) deep trenches and 76 and 10 lb (34.5 and 4.5 kg) of explosive per hole, respectively. The larger shot cratered the rock and damaged pipe 21 ft (6.4 m) away, whereas the more optimally designed shot did not crater or damage pipe 9 ft (2.7 m) away. The drill factors for these two shots are 1.1 and 3.4 ft/yd^3 (1.3 m/m^3), respectively.

Direct regulation or specification of effects, rather than shot design, is the most effective control from a regulatory viewpoint because effects are so dependent on the details of shot geometry and initiation sequence. Such dependency renders control impossible by simple regulatory specification of two- or three-shot design parameters. For instance, consider control by specification of the maximum charge weight detonated per instant at given distances from the nearest structure. Even with such detailed specification, intended vibration limits at the structure may be exceeded because of poor choice in the location of holes and/or the relative times of initiation.

Present regulatory control limits in many countries are below the levels at which cosmetic cracking may appear. There are two principal reasons for such tight restrictions. First, regulatory limits are influenced heavily by human response to blast-induced vibration and noise. Since humans are approximately 10 times more sensitive than structures to vibration, low regulatory limits are understandable. Second, many regulations appear to have been adopted without the documented scientific experimentation necessary to determine the vibration levels that cause cracking.

Statistical Analysis of Data by Pre- and Postblast Inspection

Unmeasurables in observation can be taken into account indirectly by considering the appearance of cosmetic cracks as a probabilistic event. As an aid in determining the effects of certain data sets on overall conclusions, the probabilities of cracking at various different particle-velocity levels have been calculated several times (Siskind et al. 1980b; Siskind 1981). These studies involve both immediate pre- and postblast inspection of walls in residential structures, many of which were old and distorted, and whose walls were covered with plaster. The definitions of observed cracking in each study are described previously in the "Range of Blast Effects" section.

Data from various sets of observations were analyzed with cracking points and the assumption that every cracking point excludes the possibility of noncracking at a higher particle velocity (Siskind et al. 1980b). If the probability of cracking is calculated as the percentage of points at lower levels of velocity, the result is the log-normal scaled plot of the probability of cracking vs. particle velocity in Figure 16.3-10, where threshold damage is the occurrence of hair-sized cosmetic cracks similar to those caused by natural expansions and contraction. This approach seems conservative as low particle-velocity observations do not count noncracking at higher levels.

According to Figure 16.3-10, there appears to be a lower limit of particle velocity of 0.5 in./s (12 mm/s) below which no cosmetic or threshold cracking (extension of hairline cracks) has been observed from blasting anywhere in the world. This observation includes the data with unusually low frequencies that were collected by Dvorak (1962). His data are those that tend to populate the lower region of the figure. High-frequency data (>40 Hz) show that a 5% probability of minor cracking does not occur until particle velocities reach 3 in./s (75 mm/s) (Siskind et al. 1980b).

The admissibility of Dvorak's data has been questioned because of the absence of time histories; some other studies, such as that by Langefors et al. (1958), are plagued by the same problem. To resolve this difficulty, only new USBM observations have been included in a recomputation of probabilities in Figure 16.3-11. The observations include low-frequency motions associated with surface mining. Again, there is a particle velocity, 0.79 in./s (20 mm/s), below which no blast-induced cracking was observed.

Comparison of Blast and Environmental Effects

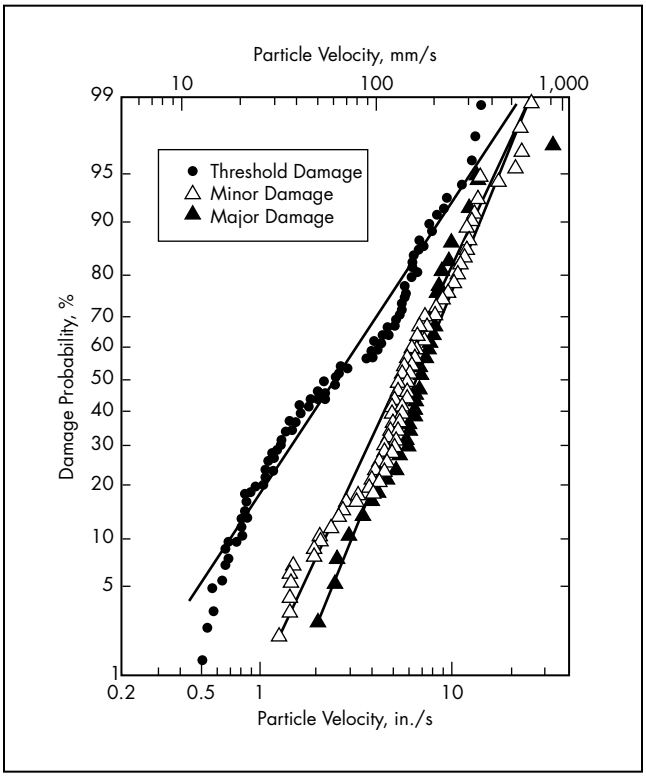
Changes in crack width caused by ground motions of <1 in./s (25 mm/s) are smaller than those caused by the passage of weekly weather fronts (Dowding 1988). This conclusion was reached after measuring, for 8 months, the displacement response on a poorly built house located close to surface-coal-mining vibrations. Displacements were measured at 10 different wall positions that included cracked and uncracked wall coverings. Weather- and blast-induced crack displacements across the most dynamically responsive wall-covering cracks are compared in Figure 16.3-12. The continuous and highly cyclical curve is that of displacements produced by environmental changes. The small circles are the maximum zero-to-peak dynamic displacements recorded by the same gauge. Even though the maximum recorded particle velocity is as high as 0.95 in./s (24 mm/s), the maximum weather-induced displacements are three times higher than those produced by blasting. On other gauges, weather changes produced displacements that were 10 times higher than those produced by blasting. This crack response is typical of those for the more than 30 cracks in over 20 studies reviewed by Dowding (2008).

Special Considerations

Statistically determined control limits are too low for basement walls and engineered structures. They are based on the response of residential structures and on lower-limit cases involving cracking of aboveground plaster or gypsum wallboard coverings in older distorted structures.

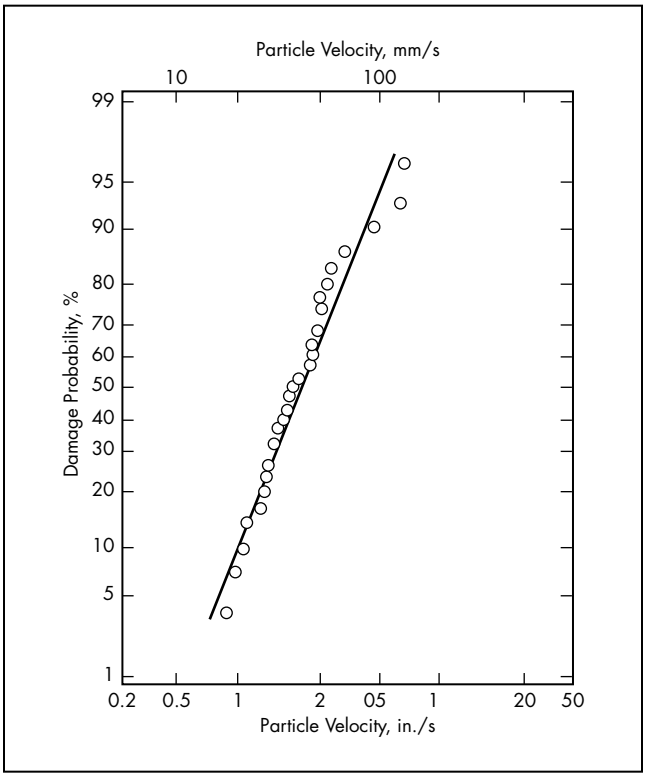
Engineered Structures

Concrete is much stronger than plaster. Therefore, engineered structures constructed of concrete can withstand maximum particle velocities of at least 4 in./s (100 mm/s) without cracking (Crawford and Ward 1965). Furthermore, buried structures such as pipelines and tunnel linings are not free to respond as are aboveground residential structures whose response provides the data from which most limits are chosen. Therefore, underground structures are able to withstand even greater excitation motions (Dowding 1985).



Source: Siskind et al. 1980b.

Figure 16.3-10 Probability analysis of worldwide blast cracking data



Source: Siskind 1981.

Figure 16.3-11 Probability analysis of blast-induced threshold cracks observed by the USBM

Specific engineered structures should be ana1yzed in terms of the strain that can be withstood by critical elements and the strain should be measured. This approach is particularly appropriate for singular structures with isolated portions near the blast source, such as buried pipelines. Often the ground and pipe strains can be confirmed by directly measuring the strains on the buried pipes, as was done by the USBM (Siskind et al. 1994). Their measurements confirmed that welded steel pipelines can withstand ground motions considerably above the typical control limit of 50 mm/s. Oriard (2002) believes that even the high allowable peak particle velocities reported by Siskind et al. are conservative for welded steel pipelines. Recently, use of measured strains at critical locations within engineered structures has been adopted for blasting immediately beneath steel-frame structures. Strain gauges attached to the steel H-beam cores of the building columns confirmed that high peak particle velocities were not causing high strains.

Fatigue or Repeated Events

Since current regulatory limits are so low as to restrain blast-induced displacements below those caused by the passage of weekly weather fronts, the question of repeated events becomes moot. Weather by itself over the years produces greater repeated-event effects than does blasting.

A repeated-event experiment conducted at the USBM test house (Stagg et al. 1984) confirms that current regulatory controls for fatigue cracking are low enough. The test house was framed in wood and had paper-backed gypsum-board interior walls. When continuously vibrated at an equivalent ground particle velocity of 0.5 in./s (12 mm/s), no response

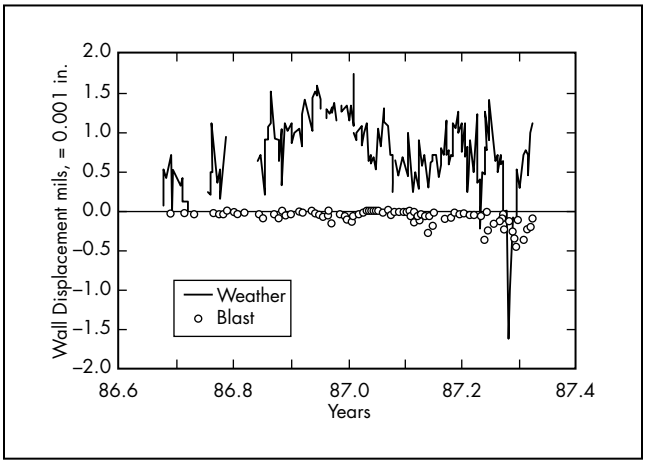


Figure 16.3-12 Displacements produced by weather-induced changes in humidity and temperature (continuous line) and by surface-coal-mine-induced ground motions (circles)

was observed until 52,000 cycles, when a taped joint between sheets of gypsum board cracked. These taped joints are the weakest and most compliant zones in a house with paper-backed gypsum-board walls.

Rock Mass Displacement and Cracking

Cracking rock immediately adjacent to a blast can be controlled by limiting particle velocities to 27 in./s (700 mm/s) in the volume of rock to be protected (Holmberg and Persson 1978).

Rock displacement by forces produced from delayed gas pressure cannot be controlled by specifying an allowable particle velocity. Fortunately, these displacements occur only very close to a blast, within 100 to 165 ft (30 to 50 m) and are associated with blocks that are unconstrained by other surrounding rock.

The sliding instability of individual rock blocks must be evaluated on a case-by-case basis. Each block must have an adequate factor of safety to prevent static failure (Dowding and Gilbert 1988). Since wavelengths of blast-induced shearing motions in rock might be 100–200 ft (roughly 30–60 m) for a 50-Hz wave, only blocks 23–46 ft (7–14 m) in size (~one-fourth the wavelength) can accelerate uniformly. Thus large slopes are unlikely to be destabilized by blasting where the peak particle velocity is high.

Frequency-Based Control with Dominant Frequency

Figure 16.3-13 shows the limit adopted by the OSM, based on a suggested but not rigorously validated proposal by the USBM (Siskind et al. 1980b). Corner 1 of the figure is the lowest particle velocity at which USBM personnel observed cosmetic cracking; corners 2 and 3 are unconfirmed. The dotted line shows the limit for safe construction blasting near engineered structures; the dashed line shows the limit for safe construction blasting in urban areas near older homes and historic buildings.

The dominant frequency that is consistent with Figure 16.3-13 is the frequency that is associated with peaks in the time history whose amplitudes are >50% of the peak or maximum particle velocity. The frequency of these peaks was calculated by the zero-crossing method, as shown in the inset for Figure 16.3-3. The frequency associated with the peak-particle-velocity pulse is a good first approximation and eliminates the need for sophisticated Fourier frequency analysis or response-spectrum analysis. Response-spectrum analyses are the most precise approach to account for the frequency effects of structural response and should be used in singular cases where exacting analysis is required.

Regulatory Compliance for Air Overpressure

Although broken glass is normally associated with excessive air-blast overpressures, limits in the United States are based on the wall response necessary to produce wall strains equivalent to those produced by surface-coal-mining-induced ground motions with a peak particle velocity of 0.75 in./s (19 mm/s). These limits are listed in Table 16.3-3. If a wall-strain level equivalent to that produced by a particle velocity of 1.0 in./s (25 mm/s) measured in the ground were chosen, the allowable overpressure would increase by 3 dB. Most cases of broken glass are reported to have been observed at air overpressures of 136–140 dB (as measured with a linear transducer). Wiggins (1969) reports that sonic-boom air overpressures of >140 dB were necessary to produce cosmetic cracking in military barracks and windows. This observation underscores the conservatism of the 133-dB regulatory limit for cosmetic cracking.

Because of the different sound-weighting scales used by monitoring instruments, the recommended levels in the table differ by instrument system. Since structures are most sensitive to low-frequency motions and the greatest air pressures occur at these inaudible frequencies, A-weighted scales cannot be used at all. Since C-weighted scales are least sensitive at low frequencies, their use requires the most restrictive limits.

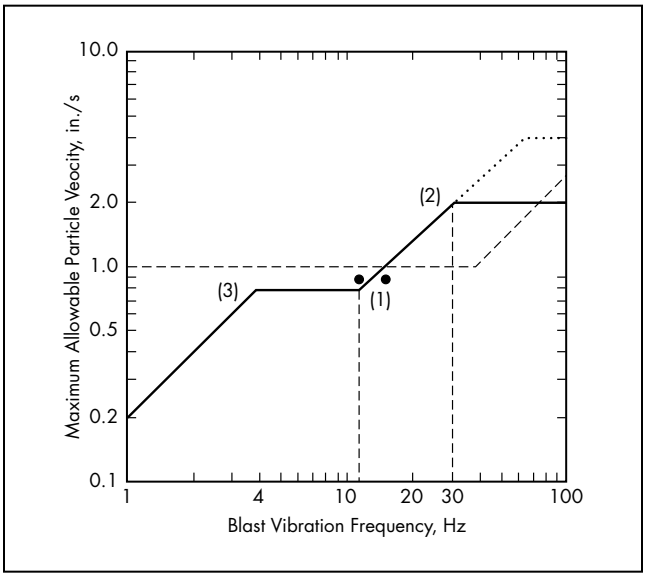


Figure 16.3-13 Frequency-based blast-vibration control limit to protect residential structures, modified from suggestion by USBM

Table 16.3-3 Air-overpressure control limits for various instrument lower-frequency limits

Lower-frequency limit, Hz (–3 dB)	Maximum level, dB
≤0.1-Hz flat response*	134 peak
≤2-Hz flat response	133 peak
≤6-Hz flat response	129 peak
C-weighted slow response*	105 peak dBC

Source: Stachura et al. 1981.
*Only when approved by the regulatory authority.

DIGITAL-AGE IMPROVEMENTS IN BLAST CONTROL

Significant digital-age improvements in blast control have accrued from the use of electronic detonators and signature hole analysis.

Electronic Detonators

Electronic detonators contain a microprocessor that improves the precision of firing times by almost two orders of magnitude (Bartley et al. 2000). This increase now makes it possible to precisely control the firing time of each blasthole, even for shots with several hundred holes. Although electronic detonators are more expensive than pyrotechnic detonators, they offer a number of advantages because of their increased control of detonation times as well as increased safety and flexibility of timing.

The increase in firing-time precision is dramatic. Several decades ago, single batches of pyrotechnic detonators had variations (coefficient of variation or 1 standard deviation divided by the average) in delay times as high as 5% to 10% (Winzer et al. 1979); today these values are on the order of 2%–5% (Cunningham 2000). Although 5% may not be important for shots with a small number of holes (<30), it is important for a four-row, 100-hole shot, designed, for example, with surface delays of 17–25 ms between holes and 42 ms between rows. If a nonelectric system is used for this shot, then all of the downhole delays must be at least 500 ms to allow the surface signal

to traverse all of the holes and reach the last hole before the first hole detonates. Thus the last hole will fire between 475 and 525 ms 64% of the time, between 450 and 550 ms 95% of the time, and between 425 and 575 ms 99% of the time. Thus 64% of the delay times fall within ±1 coefficient of variation (5%) of the expected or average delay time of 500 ms. For every shot, there is a 5% possibility of a delay overlap with three (i.e., 50/17) of the previous holes. These increases in unintended external effects occur if overlap results in simultaneous detonations (less than about 8 ms apart), detonation of a rear-row hole before the front-row holes complete, or detonation of holes out of sequence along a row. Electronic detonators now eliminate these potential overlaps and associated increases in back break and peak particle velocity.

The downhole microprocessor in the electronic detonator allows complete freedom in programming firing times. It is now no longer necessary to carry an inventory of preprogrammed delays for all anticipated blast designs. This freedom allows designs to accommodate changes in inter-row and inter-hole delay times as a function of the changing free face at corners and other challenging shot geometries.

Electronic detonators not only reduce peak particle velocities, they also improve safety and in-hole diagnostics (Bartley and McClure 2003). They offer increased immunity to radio-frequency transmissions, electromagnetic interference, and stray current. They are completely testable and can be automated to test and disarm. Most important, in these days of threats of terrorism, they require a specific blasting machine as well as specific identification and password.

Signature-Hole Analysis

Production of synthetic seismograms (ground-motion time histories) by superposition of time-lagged waveforms is another product of the digital age. This superposition is popularly called *signature-hole analysis*, as it requires a waveform measured at the position of interest generated from a single (i.e., a signature) hole at the blast location. The time history recorded for the signature hole (which automatically accounts for the geological irregularities in the travel path) is superposed and time-lagged for each hole's designed detonation time (and travel path in more sophisticated versions) (Anderson 2008). Like electronic detonators, this analytical technique has benefited from the digital age, as it would not be possible without digital waveforms and computerized manipulation.

Signature-hole analysis provides a means to estimate the peak-particle-velocity waveform that results from a change in blast timing. Thus it enables estimates of both the resulting peak particle velocity and the frequency content to match frequency-dependent control limits. Scaled-distance relationships enable only the peak particle velocity, but not the frequency content, to be estimated. Regardless of the genesis of the changes in design, be it the need to improve fragmentation or ground motions or both, signature-hole analysis can be helpful in predicting the changes in amplitude and frequency of the ground-motion time history produced by blasting.

Signature-hole analysis has been the subject of both support and skepticism over the years. It was hampered in its early days by wide differences between design and actual delay firing times for pyrotechnic delays. Yet even its skeptics believe that, when applied correctly, it is a useful design tool. Increased precision of electronic detonators as described previously now eliminates mistiming as a cause of differences between values that are predicted by signature-hole analysis and values that are measured.

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CHAPTER 16.4

Water and Sediment Control Systems

Christopher D. Lidstone and Abby Korte

INTEGRATING ENVIRONMENTAL COSTS

Watershed plans have been integrated into public policy in response to the complex requirements of water resources and land-use planning. The permitting of a new mine and the expansion of an existing mine both face public scrutiny, especially in relation to the effects of mining on the local watershed or groundwater basin. Geographic information system (GIS) databases, which are available on numerous public Web sites, typically identify the presence of a mine as a potential point source for pollution.

The mining industry is responsible for much of this perception because of its historic failure to address drainage pollution of all types and because of its historic failure to include environmental costs as part of its public stewardship of the land. In the United States, new sand and gravel operations are often first reviewed by local planning and zoning boards. These planning and zoning boards consist of volunteers or individuals serving by local appointment rather than technical experts or regulators experienced with mining and the impacts related to mining. It is this local board of citizens that first addresses a mine's potential to have an adverse impact on area water wells or nearby flood levels.

The quality and quantity of the surface water runoff that leaves a mine site is often the public's first indication of the presence of a mining operation and the environmental behavior of the corporate citizen. Unlike industrial pollution, which is often generated from an undesirable or unforeseen by-product of the manufacturing process and which is short lived, water pollution from mining can continue long after the mine is shut down and the mineral resource has been fully exhausted. Underground mining can impact groundwater by changing its flow direction, by mixing multiple groundwater sources, and by changing water quality through the physiochemical processes of oxidation and reduction. Runoff from inadequately controlled or poorly reclaimed surface mine disturbances can result in large volumes of sediment-laden waters exiting the mine site and delivering toxic and acid-forming substances to off-site drainages. The act of surface mining will disrupt vegetation, parent material (bedrock), and soil beds. As a surface drainage system trends towards geomorphic instability, excessive erosion will occur and deliver sediment and turbid water off-site.

Cost Benefit and Probability of Failure

The design and implementation of an adequate drainage system before, during, and after surface and underground mining operations is essential to minimize adverse and costly environmental impacts. In addition, the mining industry is faced with the challenge of designing hydrologic and hydraulic control under a myriad of climatic conditions worldwide. For example, surface mines in Sumatra, Indonesia, need to control 3 m (10 ft) of rainfall over a 6-month period during the wet season, while surface mines in western New South Wales, Australia, need to control 0.3 m (12 in.) of rainfall in 60 minutes. Both situations create hydrologic challenges, and one universal truth can be identified—the larger the rainfall volume or the greater the rainfall intensity, the larger and more costly the hydrologic control structure.

Life of the Structure

It can be said that all engineered systems are designed to fail; it is the probability of failure that governs the degree of environmental protection. Each mining company must make this design decision either in response to regulatory criteria or as a cost–benefit analysis made during the planning stages. The probability of failure relationship can be simply stated as

$$P = 1 - q^n$$

where

P = probability of failure
$q = 1 - (1/T_R)$, where T_R is the return period of the design storm
n = anticipated life of the structure

For the purposes of this handbook, the four classes of structures (Class A through Class D) shown in Table 16.4-1 should be considered.

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Table 16.4-1 Design life of structures

Type and Life of Structure	Return Period of Design Storm, T_R
Class A: 10 years	25 years or less
Class B: 11 to 20 years	50 years or less
Class C: Greater than 20 years	100 years or less
Class D: Enhanced protection need	Greater than 100 years

Many temporary haul roads, with a design life of less than 10 years, would dictate the use of a Class A structure (e.g., culverts or diversions). Railroad access structures that will be present for the life of the mine may dictate the use of Class C or Class D structures. A tailings dam or a diversion around a tailings dam will certainly dictate the use of a Class D structure design, including a structure designed for a probable maximum flood (PMF).

DESIGN CONSIDERATIONS RELATED TO MINING

Geomorphology

Surface drainage systems represent a delicate balance between supply and demand. There exists a delicate balance between the amount of material to be carried by a particular stream system and the amount of power available to move that material downstream. The relation can be presented as follows (Lane 1955):

$$QS \propto Q_s D_{50}$$

where

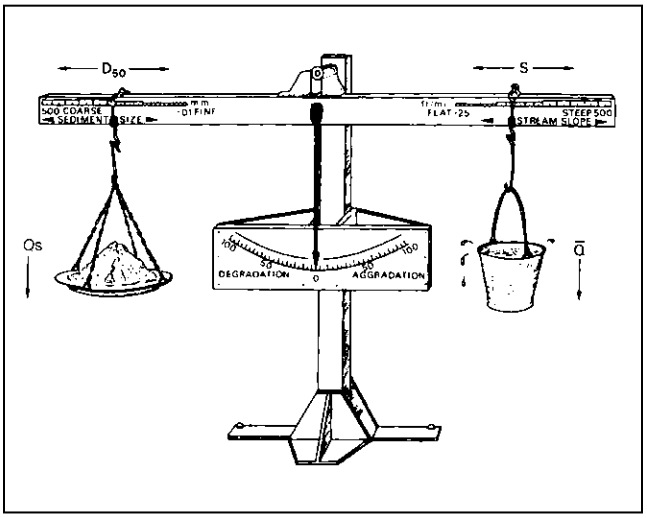
Q = discharge
S = channel slope
Q_s = sediment discharge
D_{50} = median grain size of the bed material

This equation shows that the discharge, Q, and channel slope, S, are proportional to the sediment discharge, Q_s. This relationship is presented graphically in Figure 16.4-1 and can be used to predict how a channel system might respond to changes in various parameters (sediment load, channel geometry, channel slope, discharge, etc.).

Whether constructing a diversion around a surface mineral mine or determining the appropriate setback distances for a floodplain sand and gravel mine, knowing how that system behaves geomorphically is critical. Historic aerial photo analysis can be a useful tool in determining a river system's channel migration zone and determining the risk of avulsive (versus accretionary) changes in the channel bank location. In addition, it is important to understand how a natural system behaves when designing a surface water diversion so as to avoid additional degradation or aggradation of the channel.

Groundwater Hydrology

It is critical for the engineer or mining company to put wells into the ground to evaluate both groundwater hydrology and chemistry before opening up the ground to extraction activities. Putting wells in the ground allows for the collection of data relating to groundwater levels, groundwater chemistry (oxidizing versus reducing environment), and the direction and rate of groundwater flow. If a proper evaluation of site conditions is not completed prior to the commencement of mining, the site can be inundated with more inflow than the pumping system can handle, resulting in flooding and possibly environmental excursions as mine personnel must handle higher than anticipated levels of "dirty" water.



Source: Lane 1955, with permission from the American Society of Civil Engineers.

Figure 16.4-1 Lane's balance diagram

Water and Sediment Management

Diversion and Interception of Undisturbed Runoff

It has been said that an ounce of prevention is worth a pound of cure. Treating disturbed sediment or nutrient-laden runoff from a mine site can be a complicated and costly endeavor. Diverting undisturbed runoff or upgradient groundwater around the mining operation via a constructed channel or drain will eliminate the need for postprocess water treatment. Diversions take on many design forms, depending on the quantity of runoff and the slope of the channel. Classically, as the quantity of runoff increases, there is a progression from bare earthen to grassed waterways to rock riprap channels. In the design of diversions, the development of a stable channel for the design storm appears to be the primary consideration. Often diversions can accommodate the large design storm event, but failure to consider potential sediment deposition from intermediate events and the subsequent loss of conveyance capacity sometimes leads to failure of the control structure.

If undisturbed runoff cannot be diverted around a mine site, it can sometimes be detained in one or a series of interceptor ponds. Interceptor ponds are designed to store and infiltrate all runoff associated with a given design storm, so that undisturbed runoff does not interact with land disturbances.

Treatment of Disturbed Land Runoff

In the event that surface runoff can be neither diverted around the mine site nor stored and infiltrated back into the groundwater via the use of interceptor ponds, then all runoff, which mixes with the mining process, must be treated to prevent the contamination of downstream water resources. One of the primary pollutants associated with surface mine operations is sediment. Because there are other chapters in this handbook dedicated to the treatment of acid mine drainage and various other chemical constituents, this chapter will deal exclusively with the removal of sediment from disturbed runoff.

Increased sediment inputs to stream systems can have a series of adverse ecological and economic impacts, ranging from increased sediment loading in reservoirs, degradation of fish-spawning habitat, and increased nutrient loading that can lead to eutrophication in stagnant water bodies.

The basis for sediment removal is settlement under quiescent flow conditions. Stokes' law describes the settling velocity of a spherical object in a viscous liquid as a result of the pull of gravity. The settling velocity of a small spherical particle (i.e., sediment) can be described by the following equation:

$$V_s = \frac{2(\rho_p - \rho_f)}{9\mu} g R^2$$

where

V_s = downward settling velocity, m/s
ρ_p = mass density of the particles, kg/m^3
ρ_f = mass density of the fluid, kg/m^3
μ = fluid's dynamic viscosity, Pa s
g = acceleration due to gravity, m/s^2
R = radius of the object, m

When the required sediment settling velocity is determined, a sediment detention pond can be designed such that turbid water is retained long enough for the design particle size to settle out of solution.

In some cases, the detention of turbid waters is not enough to remove clay-size particles, which are too small to be removed by settling. In these cases, the treatment of disturbed runoff by the use of flocculants may be necessary. Most flocculants are composed of heavy organic polymers with a strong positive charge that attracts the negatively charged clay particles in a "bundle," allowing them (collectively) to settle out faster than single particles might settle.

Flocculants tend to be cost prohibitive and are generally used only when at least one of the following conditions are present: there is a limitation on the amount of space and, therefore, time available to allow particulates to settle out; there is a particularly large proportion of the sediment load that is silt or clay in size; or the water quality of the receiving water body is of a pristine nature. If a flocculant is used, the most important component of the distribution system is to ensure that the flocculant is well mixed with the discharge.

SURFACE WATER HYDROLOGY

An accurate analysis of surface water hydrology and runoff processes is a critical first step in the design and management of water and sediment control systems. It is important not only for the water resources professional to determine whether the project cost, timing, and scope warrant the collection of surface water data, but also to recognize the limitations and errors associated with the collection and analysis of a time-limited data set. Often on-site data must be augmented with data from published sources. For any given project location and scope, there is no absolute best method. However, the only universal truism in surface water hydrology is that, if available, there is no substitution for locally collected data.

Selection of Analysis Method

Many methods are available for estimating design rainfall depths and for constructing synthetic rainfall distributions or a rainfall hyetograph. The construction of the rainfall hyetograph leads to the calculation and simulation of a runoff hydrograph. For the design of some surface water controls like culverts and some diversions, a peak runoff rate is sufficient. The design of storage components such as detention ponds and certain conveyance channels require the determination of the entire storm hyetograph and runoff hydrograph, as the design must take into account conveyance and residency time calculations.

Design Rainfall Event

The design rainfall event is typically mathematically generated from volume-duration-frequency data or intensity-duration-frequency data for a specific region or project location (Seybert 2006). The design rainfall event is often established by local permitting regulations and is the foundation of the runoff modeling and design calculations for hydraulic structures associated with surface mining operations. The most comprehensive source of rainfall data for the United States and its territories is provided by the Hydrometeorological Design Studies Center and the Office of Hydrological Development of the National Oceanic and Atmospheric Administration's National Weather Service through its Precipitation Frequency Data Server, which can be accessed through the Internet at www.weather.gov/ohd/hdsc/ (NOAA 2009). Through the listed Web site, one can access state-specific rainfall distribution data in the form of rainfall atlases (NWS 1961) and distribution tables (Arkell and Richards 1986). International organizations maintain similar rainfall information databases, and these can be researched in advance of mine planning. If it is important to know the temporal distribution of rainfall for a particular event, and real-time local data are unavailable. The Soil Conservation Service (SCS) and local agricultural services provide synthetic rainfall distribution curves (SCS 1973).

Peak Flow Determination

Rational Method

The rational method is the most commonly used technique for determining peak flow rates. The rational method formula is

$$Q_{PK} = CiAk$$

where

Q_{PK} = peak flow rate, m^3/s (ft^3/s)
C = dimensionless runoff coefficient
i = rainfall intensity, mm/h (in./h)
A = drainage area, ha (acres)
k = conversion factor equal to 0.00278 in metric units (1.008 in English units)

The dimensionless runoff coefficient, C, is designed to incorporate differences of rainfall interception/infiltration, surface detention, and/or antecedent moisture conditions. It is nearly impossible for a single factor to incorporate all of these. Therefore, more commonly, the C factor is based on land-use type and imperviousness. A simplified list of C factor values is shown in Table 16.4-2. The C factor can also be area weight averaged throughout the basin of study (Haan and Barfield 1978). The rational method is a simple peak flow calculation based on several assumptions. Included in the assumptions is that the rainfall occurs uniformly throughout the basin and that the rainfall frequency equals the runoff frequency. Typically, this can be successfully used for smaller basins. The method is well described in numerous publications, including Haan and Barfield (1978) and Simons, Li and Associates (1982b).

Table 16.4-2 Simplified table of rational Method *C* factors

Groundcover	Runoff Coefficient, *C*
Lawns	0.05–0.35
Forest	0.05–0.25
Cultivated land	0.08–0.41
Meadow	0.1–0.5
Parks, cemeteries	0.1–0.25
Unimproved areas	0.1–0.3
Pasture	0.12–0.62
Residential areas	0.3–0.75
Business areas	0.5–0.95
Industrial areas	0.5–0.9
Asphalt streets	0.7–0.95
Concrete streets	0.7–0.95
Roofs	0.75–0.95

SCS Curve Number Method

An alternative is the SCS curve number (SCS-CN) method, which was first published in the *National Engineering Handbook* (SCS 1982) in 1956 by the Soil Conservation Service, now known as the Natural Resources Conservation Service. The SCS-CN method was developed from extensive field investigations and has been revised several times. The SCS-CN formulas are as follows:

$$Q = \frac{(P - I_a)^2}{(P - I + S)}$$

$$I_a = 0.2S$$

$$S = \frac{1{,}000}{\text{CN}} - 10$$

where

Q = runoff, in.
P = precipitation, in.
I_a = initial abstraction, in.
S = potential maximum retention, in.
CN = curve number

Initial abstraction generally consists of interception losses, surface storage, and soil infiltration. Because of the dimensional incongruity of these formulae, there is no direct coefficient conversion between English and metric units, and each variable must be converted separately. This method requires the water resources professional to collect information pertaining to land use, hydrologic condition, and hydrologic soil type. A more detailed explanation of the development and use of the SCS-CN method is beyond the scope of this chapter, but many excellent texts and papers outline the methodology and provide the necessary tabular data (Seybert 2006).

Time of Concentration

The time of concentration, t_c, is the length of time it takes for precipitation to travel from the most remote part of a watershed to the point of interest, often the basin outlet. Several methods exist for estimating t_c, including, but not limited to, the SCS upland method and the kinematic wave method (Kirpich 1940; Ragan and Duru 1972). The Ragan and Duru equation is

$$t_c = \frac{C_1 (nL)^{0.6}}{(i_e^{0.4} S^{0.3})}$$

where

t_c = time of concentration, min
C_1 = a constant equal to 1.977 in metric units (0.928 in English units)
n = Manning's roughness coefficient
L = length, m (ft)
i_e = excess rainfall intensity, mm/h (in./h)
S = slope (vertical/horizontal), m/m (ft/ft)

This equation is only valid when the product of i_e and L is greater than 152 m (500 ft).

The Manning's roughness coefficient, n, can be assigned as a single unit for the reach of interest or can be composited to better represent a channel with varying degrees of resistance to flow. Several sources of information are available for Manning's n values, including books with photographs of channel sections with measured n values (Chow 1959; Acrement and Schneider 1984).

The Kirpich equation for estimating t_c is

$$t_c = C_2 L_1^{0.77} \left(\frac{L_1}{H}\right)^{0.385}$$

where

C_2 = a constant equal to 0.0195 in metric units (0.0078 in English units)
L_1 = maximum flow length, m (ft)
H = elevation difference between the hydraulically most remote point in the watershed and the point of interest, m (ft)

SURFACE WATER HYDRAULICS

Designing a diversion ditch or reclamation channel requires that the mining professional must have a good understanding of the hydraulic and sediment transport characteristics, as well as channel geometry and morphometric characteristics of the native system. Flow conditions in open channels are complex. The position of the free water surface and channel geometry can change with respect to time and space. Depth of flow, velocity, discharge, and slope of the channel bottom and water surface are interdependent.

The basic hydraulic theories that relate flow velocity, depth, discharge, and slope are as follows:

- Conservation of mass (continuity)
- Conservation of linear momentum
- Conservation of energy

Conservation of mass states that matter can neither be created nor destroyed. Based on this it can be concluded that flow discharge in each section of a river reach is identical. Conservation of momentum concludes that the summation of all external forces (pressure, body, and shear forces) acting on the fluid is equal to the rate of a flux of momentum through the boundary of the fluid. The law of conservation of energy (the first law of thermodynamics) states that energy can neither be created nor destroyed. For steady flow, the total energy (kinematic and potential energy) at any point in a stream is equal to the sum of the total energy at any downstream point and the loss of energy between the two points.

The continuity, momentum, and energy equations can be used to determine the elevation of the water surface in an open channel for a given discharge. For steady flow, the energy equation is written in the following form to compute the water surface profile:

$$\delta L = \frac{H_2 - H_1}{S_o - S_f}$$

where
δL = distance between sections 1 and 2, m (ft)
H = specific head, m (ft)
S_o = bed slope (vertical/horizontal), m/m (ft/ft)
S_f = friction or energy slope (vertical/horizontal), m/m (ft/ft)

The specific head, H, is defined as

$$H = \frac{V^2}{2g} + y$$

where
V = average velocity, m/s (ft/s)
g = acceleration due to gravity, m^3/s (ft^3/s)
y = flow depth, m (ft)

The energy slope, S_f, is the slope of the energy line representing the elevation of the total energy of flow. It is related to the velocity and depth by the following empirical relationship, which is known as Manning's equation:

$$Q = VA = \left(\frac{k}{n}\right) AR^{2/3} S^{1/2}$$

where
Q = flow rate, m^3/s (ft^3/s)
V = velocity, m/s (ft/s)
A = flow area, m^2 (ft^2)
k = conversion factor equal to 1.00 in metric units (1.486 in English units)
n = Manning's roughness coefficient
R = hydraulic radius, m (ft)
S = channel slope (vertical/horizontal), m/m (ft/ft)

R is calculated by the cross-sectional area of the channel divided by the wetted perimeter. Basic channel hydraulics is well summarized in Simons, Li and Associates (1982b).

SEDIMENT PRODUCTION

Soil Loss Equations

The Universal Soil Loss Equation (USLE) calculates sediment production via a combination of sheet and rill erosion and the associated runoff for the landscape profile. The soil loss is an average erosion rate for the landscape profile. Erosion can vary widely, even on a uniform slope, depending on slope position and configuration of the slope profile. USLE does not estimate the amount of sediment leaving a field or watershed but estimates soil movement at a particular site. The quantity of sediment eroded from a slope depends on the erosive power of rainfall, soil characteristics, slope length, and gradient, as well as type and amount of soil cover and conservation practices. These parameters have been combined into the USLE (Wischmeier and Smith 1978).

$$A = R \times K \times LS \times C \times P$$

where
A = soil loss, kg/ha (st/acre)
R = rainfall factor
K = soil erodibility, t/ha (st/acre) per unit R
LS = dimensionless length–slope factor that accounts for the actual length and slope compared to a standard slope of 9% and 22.1 m (72.6 ft) in length
C = a factor that accounts for the effectiveness of vegetative cover, mulches, etc.
P = a factor that accounts for the effectiveness of conservation practices such as terraces or fallow cover

Values for these parameters (i.e., R, LS, C, and P) may be found in many sources and have been modified for the mining industry (Chow 1959; Wischmeier and Smith 1978).

Contemporary to the USLE, the Revised Universal Soil Loss Equation (RUSLE) computes sheet and rill erosion and the associated runoff from untilled lands, as the original USLE was derived and intended for use in agricultural settings. RUSLE contains several updates and changes from USLE, including new and improved soil erodibility indexes for some areas, time-variant soil erodibility to reflect freeze–thaw cycles, new equations to account for slope length and steepness, and additional C and P factors to account for cover and conservation practices on untilled lands.

Regression Studies

In some cases, particularly when there are anthropogenic or human-related sources of sediment present, it is possible to pull real data on sediment contributions from previously conducted field research. In many cases, real data have been collected on sediment production rates from given land-use types, and regression (i.e., retrospective) analyses have been performed to model the rate of sediment production as a function of several landscape variables. For example, several studies have been completed quantifying sediment production from various unsealed road surfaces. One such study has quantified and modeled sediment production from all sources, focusing on forest roads in the southern Sierra Nevada, California, United States (Korte and MacDonald 2005).

Other Methods

In the international setting, predictive models developed in the United States may not be applicable. International mining efforts typically require the site-specific application of empirical methods and observations. Denudation rates can be calculated from observation of past mining practices. One technique that is uniformly applicable includes the volumetric measurement of accumulated sediment within a pond over a known period of time. In an effort to develop an appropriate empirical model, it is important for the mining professional to evaluate observed conditions within the watershed that delivered water and sediment to that pond.

HYDROLOGIC STRUCTURES

Hydrologic control structures are an integral part of any mining operation. Such structures are generally referred to in the context of surface mines and their associated disturbances, but are becoming increasingly more important for both underground and in-situ recovery (ISR) mining operations. All mining

operations include such surface disturbances associated with roads, buildings, power lines and substations, stockpile areas, loadout and water treatment facilities. Such facilities require water and sediment control during mining and reclamation at the end of mining. Surface mining operations are further complicated by the associated disruption of the existing topography to extract the ore or mineral commodity from the ground. Such operations will require a more sophisticated drainage control and a far more complicated final reclamation plan than underground or ISR operations.

To adequately control pollution, it is necessary to establish a complete drainage design plan for the entire permit area using sound engineering practices. It is often important to separate the hydrologic control plan on a temporal basis, that is, short term (1 to 5 years) and long term or "life of mine," depending on the size and dynamic nature of the operation. The short-term plan will include design details, engineering calculations, and likely all required construction permits from the various regulatory agencies. The long-term or life-of-mine plan is simply a concept plan, which will dictate the progression and type of hydrologic control features but will allow flexibility as mine production and mining methods inevitably change. This long-term plan is completed without actual engineering design or regulatory permitting.

In addressing the overall hydrologic control plan, it is essential that the design engineer recognize that the drainage basin or basins within the mining area are only part of a larger, more complex system. Mining within a portion of the drainage basin will elicit a complex response both upstream and downstream from the area of actual disturbance. The design engineer must consider the active mine controls and, even more importantly, the postmining restoration of this drainage system, such that the reclaimed land surface will properly function as part of a larger system. Hydrologic control structures are summarized in Table 16.4-3.

Table 16.4-3 Hydrologic control structures

Type of Structure	Design Life	Purpose of Structure
Diversion channels and ditches	Temporary or permanent	Divert undisturbed runoff around the mining operation in a stable fashion.
Culverts	Temporary	Divert runoff below mine access or haul road.
Sediment ponds	Temporary	Treat disturbed land runoff.
Interceptor ponds	Temporary or permanent	Prevent undisturbed runoff from entering a mine pit or mine area. Can serve as a stock pond or detention pond.
Small area sediment controls (best management practices)	Temporary	Treat disturbed land runoff from small acreage disturbance.
Dry wells	Temporary or permanent	Capture and treat disturbed land runoff.

Diversion Channels and Ditches

Diversions are used throughout mining to intercept and convey runoff. Typically they are used to convey runoff from undisturbed areas around active mining sites and preclude mixing of runoff waters and reduce the quantity of runoff requiring treatment. In some cases, diversions are necessary to maintain water flow for downstream water users, including farmers, recreation, wildlife, and fisheries. Because the construction of a diversion of an intermittent or perennial stream is generally a high capital-cost item, mine management must closely address the question of the proposed design life and determine value engineering. As the size of the drainage basin increases, it can be assumed that the means to stabilize each diversion will transition from a simple earthen cut/fill at a graded slope to a well-established vegetated channel to a channel stabilized with geotextile, concrete, or riprap. Drop structures are typically constructed at locations where the design or as-constructed channel gradient is too steep, resulting in excessive flow velocities and the need for channel protection (Jones and Lidstone 1996). A diversion structure that will remain in place after mining is completed must also incorporate geomorphic stability and may include aquatic habitat features (e.g., pools, riffles, large woody debris, and riparian vegetation).

The diversion hydrology is determined after the design life is established. Permanent diversions or life-of-mine features are typically designed for the 100-year discharge event, with additional capacity to accommodate freeboard. Regulatory programs may dictate the use of the PMF or some proportion of the PMF, where mine tailings or hazardous wastes are involved. The diversion hydraulics can be simplified using normal depth procedures (Manning's equation) or using more complicated software programs such as SEDCAD, from Civil Software Design LLC, and HEC-RAS and HEC-HMS, which were developed by the Hydrologic Engineering Center of the U.S. Army Corps of Engineers. For the purposes of this section, conveyance capacity of a diversion can be summarized by

$$Q = VA$$

where

Q = design discharge, m^3/s (ft^3/s)
V = velocity, m/s (ft/s)
A = cross-sectional area, m^2 (ft^2)

Design hydraulics can be simply calculated using Manning's equation as follows:

$$V = (1.49/n)R^{2/3}S^{1/2}$$

where

V = average velocity, m/s (ft/s)
n = roughness coefficient of the channel, which is empirically determined
$R = A/P$, or the cross-sectional area of the channel divided by the wetted perimeter, m (ft)
S = slope (vertical/horizontal) of the channel or hydraulic gradient, m/m (ft/ft)

The channel must be proportioned to carry the design runoff at average velocities less than or equal to the maximum permissible velocity. This is accomplished by applications of the Manning's formula with iterative adjustments in channel parameters until the maximum permissible velocity is achieved.

Numerous studies address the maximum permissible velocity of a channel, and the most commonly accepted is in Chow (1959). Channel type or channel lining must be established in the determination of maximum permissible velocity. Noncohesive (sand bed) channels are governed by initiation

of motion and upper limits of 0.6 to 0.9 m/s (2 to 3 ft/s). Well-vegetated channels can withstand velocities of 1.5 to 2.1 m/s (5 to 7 ft/s) before erosion will take place. Cohesive bed (clay) channels are typically not governed by the principles of initiation of motion but rather by tractive force as follows:

$$\tau_c = \gamma DS$$

where

τ_c = critical shear
γ = unit weight of water, which is 1,000 kg/m^3 (62.4 lb/ft^3)
D = flow depth, m (ft)
S = channel slope (vertical/horizontal) or hydraulic gradient, m/m (ft/ft)

Allowable shear is dependent on the strength characteristics of the clay, plasticity, and electrical conductivity or salinity. Typical critical shear, τ_c, limits range from 0.73 to 3.12 kg/m^2 (0.15 to 0.64 lb/ft^2).

Where erosion problems are anticipated in design, the engineer will need to employ hydrologic control structures such as riprap or gabion drop structures to reduce overall channel slope; riprap or geotextiles to protect the underlying soils from erosion; or interlocking concrete blocks to stabilize the problematic reach of the channel. Numerous publications assist in the design of such structures, including Haan and Barfield (1978); Simons, Li and Associates (1982a); Ferris (1996); and Jones and Lidstone (1996). The design and construction of a riprap drop structure or grade control is an important means to reduce the overall channel slope and thus the erosive energy within a stream channel or diversion.

Permanent diversions require forward planning and a dedication of design and construction resources. Such diversions, which are used for the reestablishment of a drainage system affected by surface mining, must perform adequately without the assured benefit of periodic maintenance. The designer must be familiar with the inherent stability of the upstream and downstream undisturbed watersheds, including channel geometry, channel slope, channel bed and bank materials, and geomorphic characteristics. Although the designer may not necessarily want to duplicate these natural conditions, in recognition that the act of mining has permanently changed the landform, such preexisting conditions will lend themselves to design concepts. Channel design should incorporate a pilot channel to convey the annual channel forming event, typically the 1.5- to 2-year flood event, and a floodplain that at a minimum can convey the 100-year/24-hour flood event. Channel meanders, pools, and riffles may be incorporated into the ultimate channel design. Implementation of final design must incorporate biological and riparian considerations. Typically, this includes the presence of randomly placed boulders, large woody debris, and bank stabilization with woody vegetation. Numerous publications can assist in the design of such permanent diversions (Simons, Li and Associates 1982b).

Diversion ditches are a smaller-scale diversion structure, but they have become increasingly important in areas disturbed by underground mines and ISR-type mining operations. Diversion ditches can protect a well field from overland flow, as well as protecting the face-up or main portal area from off-site erosion. The design event for such ditches are governed by the life of the structure to be protected or, in some cases, the length of time for the site to be successfully revegetated and become stable in its own right. The ditches control a small basin area and often deliver water to a culvert and ultimately to native, undisturbed receiving water. Similar to the drainages, adequate freeboard must be allowed, and the water should be passed via the diversion ditch in a nonerosive fashion.

Culverts

Culverts are used throughout mining to intercept and convey runoff under access roads, stockpile areas, and structures. The selection of a properly sized culvert is dependent on the life of the structure (design discharge), type of culvert, entrance configuration, pipe length and slope, headwater constraints (the available elevation above the top of pipe to pond water), and tail-water condition (the degree to which the downstream portion of the culvert is submerged). Design procedures have been somewhat simplified by the development of culvert-capacity charts or nomographs. Culvert capacity, Q, is governed by the following general equation:

$$Q = \frac{a\sqrt{2gH}}{\sqrt{1 + K_e + K_b} + K_c L}$$

where

a = cross-sectional area of the pipe, m^2 (ft^2)
g = gravitational acceleration, which is 10 m/s^2 (32 ft/s^2)
H = head loss through the pipe, m (ft)
K_b, K_c, and K_e = coefficients that reflect various head loss coefficients that are pipe dependent
L = length of the pipe, m (ft)

Most manufacturers provide design nomographs for their type of pipe, and the Federal Highway Administration (FHWA) presents an extensive culvert-design publication called HEC-5. Publicly available software, HY-8, can be downloaded from the FHWA Web site (FHWA 2009).

Several types and shapes of culverts are commonly used in modern mines. These include concrete pipe, reinforced concrete pipe, corrugated metal pipe, corrugated steel pipe, corrugated plastic pipe, and smooth-walled plastic pipe. Each pipe type has certain advantages in cost, transportability, loading characteristics, and durability. The shape of the culvert will often dictate its capacity and, given modern environmental concerns, its ability to allow sediment and fish to pass. Barrel culverts, which are often the most common, will typically maintain the highest velocities, but they can be a barrier to fish. Box culverts provide greater flow capacity, but can require more frequent maintenance and may not provide the depth of flow to allow fish passage. Arch culverts and, more preferably, bottomless arch culverts (i.e., those where there is neither a steel underbody nor an engineered lower section) can be used. Bottomless arch culverts can be founded on bedrock or on the natural alluvial material, which comprises the actual streambed. The use of such culverts can balance flow-capacity demands while allowing fish passage with a natural substrate. The development of a natural substrate within the floor of the culvert will allow the development of a low-flow channel for the movement of fish. However, such culverts can be sediment sinks, given that the floor of the culvert will often behave as a natural channel. The geomorphic position of bottomless arch culverts is a critical design consideration.

A fundamental mistake at many mine sites is the use of an undersized culvert or the placement of a culvert at too flat

a grade. Under no circumstances should a culvert less than 0.45 m (18 in.) in diameter be used. The ability of the culvert to pass sediment and debris must be considered in all designs. Where regular maintenance is available, the use of a trash rack can prevent culvert clogging. In an area of high sediment yield, too flat a culvert grade will result in plugging. In this case, culverts should be set at a minimum grade of 2%. Where erosion may occur at the inlet or outlet or if a road overtopping design must be considered, the placement of erosion protection is a critical design concern. A fundamental error in culvert design and placement is the purchase of too short a pipe length. Where erosion control is required at the inlet or outlet portion, the pipe must project a sufficient length to allow the adequate construction of erosion control. Usually this requires an additional 3 to 6 m (10 to 20 ft) of culvert length, depending on the burial depth and diameter of the culvert. This erosion control could be riprap, gabion structures, or a concrete headwall. Riprap is the most commonly used. Riprap should be placed on a slope no steeper than a 1.5:1 (horizontal/vertical) slope. So, if a 1.2-m- (4-ft-) diameter culvert is placed under a haul road with 0.6-m (2-ft) burial depth, then the culvert will need to project 2.7 m (9 ft) on the upstream and downstream sides. For a typical 7.4-m- (24-ft-) wide haul road, this would require the purchase of a 13-m- (42-ft-) long culvert. Tail-water protection is equally important to ensure that the culvert is not undermined by erosion or, in the case of overtopping flows, that adequate protection is available. The tail-water protection is generally loose riprap and should extend a minimum of 3 m (10 ft) beyond the end of the culvert.

Water and Sediment Control Ponds

Water and sediment control ponds have been used extensively by the mining industry for stormwater management and sediment control. For simplicity purposes, three basic types of ponds are used in mining: sediment ponds, interceptor ponds, and infiltration basins. Sediment ponds are used to reduce peak flow, trap sediment, and meet effluent limits (Ferris 1996). In many countries, mine discharge must meet certain governmental standards. In the United States, these standards are dictated by both state and federal programs under the guise of the National Pollutant Discharge Elimination System, Water Pollution Control Facility, and/or a general stormwater permit. The key distinction in treatment requirements is to determine whether or not the source water mixes with process water or has the potential for being contaminated by some other pollutant besides soil. In general practice, it is important to keep stormwater from commingling with the following: truck traffic (oil and grease), process or wash water, and chemical leachate sources.

For this reason, interceptor ponds may be constructed. Interceptor ponds are used to prevent runoff from entering the mining pit and impacting mining operations. Many mine operators will construct a pond on the upstream side of a mine pit to ensure that at least a portion of the native stormwater is captured prior to entering the mining operations. This water can then be diverted or pumped around the mining operation or can be left to evaporate or infiltrate away. The design capacities of such ponds are not dictated by regulatory or treatment requirements but are more likely based on mining economics. A larger pond (100-year/24-hour capacity) can provide the pit with a high degree (0.01 annual probability) of protection. A smaller pond will provide incrementally less protection. Interceptor ponds can also include small, elongated ponds, which are constructed on mine benches to capture groundwater and/or surface water. Such ponds will prevent stormwater from mixing with the mining operation, thereby eliminating the need for water treatment.

Finally, an infiltration basin is a depression in the earth that is designed to percolate the water to the subsurface. Such basins are designed on either highly pervious soils/rock, or alternatively, they may be overexcavated and filled with pervious rock. Infiltration basins are most common on aggregate mines, rock quarries, and placer mines where such pervious soils or fractured rock are most likely to occur. The advantage of such basins is that there will be a minimal water-handling requirement, and water treatment regulations typically do not apply.

Design Considerations

Sediment pond design must address placement or location standards, hydrology, sediment production from the upstream watershed, and the upstream watershed's efficiency to trap sediment. The location of this sediment pond within the watershed must consider the size and location of the mine-related disturbance, sizing criteria to ensure that the dam height and volumetric capacity will meet regulatory criteria, pond shape to ensure sediment trapping efficiency, and overall shape to allow sediment cleanout as required.

Sediment ponds are generally located downstream of the mining disturbance, so as to capture and treat disturbed land runoff. The mining engineer must compute design hydrology to permit the sizing of the structure and, as applicable, its embankment and spillway. Design hydrology is directly related to the discussion of the design life of the structure (see the "Life of the Structure" section earlier in this chapter). In the United States, design hydrology may be dictated by federal or state regulation. For example, a coal mining company in the United States must establish sediment control "so as to prevent to the extent possible additional contributions of suspended solids to stream flow, or runoff outside the permit area, but in no event shall contributions be in excess of requirements set by applicable State or Federal law" (30USC1265). Various state guidelines interpret sediment pond containment for the water and sediment yield for the 10-year/24-hour storm event. The volume associated with the design event is directly related to the position of the structure within the watershed and the runoff of the soils, as discussed earlier in the "Surface Water Hydrology" section. A larger water or sediment containment volume could translate into a larger embankment structure and may dictate a more complex permitting/design or construction process. Therefore, the selection of the embankment's location within the watershed (i.e., upstream contributing watershed) may be critical in an effort to avoid such complexities.

In the case of process water ponds, where the generation of sediment and dirty water is the result of material processing operations, basin or pond shape is critical. These ponds are often required to discharge, and as such they must meet discharge standards. The length-to-width ratio is important in that the longer the flow length, the greater the opportunity to deposit or settle suspended solids. Such ponds may incorporate baffles to increase flow length (settling time) and to allow periodic cleaning of the facility. Where sedimentation rates need to be accelerated and additional flow length cannot be achieved, flocculants can be added to water to enhance the sedimentation process.

Sediment pond design is well documented (Haan and Barfield 1978; Simons, Li and Associates 1982a; Ferris 1996).

Dams and Incised Structures

A variety of dam/impoundment types have been utilized for these structures, including embankment dams and excavated basins. The most common pond consists of a dam embankment that may range from 3.6 to 6 m (12 to 20 ft) in height and a collection basin. The dam height requirement of 6 m (20 ft) may be dictated by certain regulatory requirements. For example, the Mine Safety and Health Administration's Safety of Dam requirements will influence the determination of a sediment-pond dam height in the United States (30CFR 77.216; MSHA 1975). On an international basis, dam height consideration may be governed by local rules, but more importantly, dam height is governed by the talents of the designer and by quality control during construction. Large dams that exceed 6 m (20 ft) should be designed by an experienced engineer and will require foundation preparation, material selection and compaction control, among other requirements. The safest ponds are those with no embankment. Such pond basins are constructed as incised or below-grade basins. These basins can be of an unlimited depth and may include the mine pit itself. Such basins are not subject to rules and regulations dictating dam stability, but they can be subject to rules that require thorough drainage or passage of water for downstream users.

Inlet Structures

Inlet structures are an important design consideration in all sediment ponds. Such structures should be located to provide the hydraulically longest flow path between the inlet and outlet. This will help to reduce short-circuiting and provide a higher potential for mixing the inflow with the permanent pool, thereby reducing sediment concentration through dilution and providing a longer residence time needed for settling sediment.

Stabilizing the inlet structure is particularly important in an excavated basin where the sediment pond is incised below the elevation of the natural ground surface. In this case, the inlet structure conveys water from the natural stream's bed elevation to the bottom of the pond or to the elevation of the permanent pool. Without an effort to stabilize the inlet structure, erosion and excess generation of sediment will occur. Inlet structures can be stabilized by riprap, concrete, vegetation, or numerous erosion-control products.

Sediment Storage Capacity

Sediment storage can be addressed by numerous methods, including computer programs or a guideline such as *x*-acre-feet of storage per disturbed contributing watershed acreage (Ferris 1996). The most common method is an adaptation of the USLE or one of the numerous variations available in the literature, discussed previously in the "Sediment Production" section and later in the "Small-Area Sediment Controls" section. Depending on cleanout opportunities, each sediment pond should be designed to hold a minimum of 3 years of sediment accumulation.

Permanent-Pool Storage Capacity

The size of the permanent pool affects peak flow reduction, sediment trap efficiency, and effluent concentration. A permanent pool can be generated from groundwater inflow or surface water inflow where seasonal precipitation and runoff exceeds evaporation and infiltration. A permanent pool volume is calculated through the development of a water balance within the watershed and at the sediment pond site. A permanent pool can provide significant dilution opportunities; it will also result in less available storage and detention time for the influent waters.

Spillway Design

Spillway location, type, and size directly influence the performance of a sediment pond. Spillways consist of primary spillways, which are designed to allow smaller flows out of the impoundment, and emergency spillways, which are designed to pass a peak flow and to ensure the stability of the embankment. Most treatment-type reservoirs are designed with both a primary and an emergency spillway, so that treated water can be released on a regular basis while protecting the embankment. Detention-type reservoirs may only have an emergency spillway and are either dewatered by a pump or through evaporation and seepage.

Primary spillways are often gated culverts or pipes, which can be controlled to allow periodic discharge and draining of the reservoir. Where treatment is required, a drop inlet or riser may be attached to the culvert to remove water from the upper portion of the pond. All valve assemblies should be equipped so that they can be operated from above the surface of the water. All inlets should be protected by a screening device that prevents floating objects, brush, turtles, fish, and so forth from entering and clogging the pipe (Schwab et al. 1966). The design size and capacity of the primary spillway is dictated by dewatering needs and treatment time requirements. In some instances it may be dictated by downstream water users via regulated water rights or irrigation needs.

All sediment ponds and most interceptor ponds should have an emergency spillway. In the case of the incised pond or excavated basin, the emergency spillway is generally the downstream drainage channel. Typically, an incised pond has no embankment to protect. Good design practice, as well as environmental rules and regulations, dictate the design and construction of an emergency spillway to protect an embankment from overtopping and thus leading to ultimate failure. The emergency spillway will typically be sized for a peak runoff event greater than the actual design event for the pond. For example, if the pond is designed for storage of the 10-year/24-hour storm event, the emergency spillway should be designed to pass the 25-year/24-hour storm event at a minimum. If the design life of the structure is greater than 10 years, good engineering practices may dictate a more conservative design (e.g., 50-year/24-hour). The emergency spillway should be designed to be nonerosive at the design event. This may be accomplished with a vegetated spillway, or in some cases it can be stabilized by riprap, concrete, or numerous erosion-control products. This will be discussed in the following section. The emergency spillway should be constructed such that no flow will be conveyed against the embankment, so that the affected parts of the embankment are protected.

SMALL-AREA SEDIMENT CONTROLS

Small-area sediment controls or best management practices (BMPs) allow an area of land to be subdivided such that the volume of runoff coming from a given parcel is bifurcated and reduced. If the area of land draining to a single given point can be reduced, then the volume of surface runoff traveling to

that given point will be reduced and the erosion potential will thereby be mitigated.

Best Management Practices

As is the case when designing hydrologic controls or control structures, the design life of the erosion-control structure is often the driving force when determining the type of material used for BMPs. In addition, the severity of the slope, runoff hydraulics, and potential for sediment production may also influence the determination of the type of BMP. Proper installation of erosion-control structures is critical because when overland flow has either bypassed or undermined the structure, failure is imminent. The mining professional must match the method to the site condition (i.e., straw bales may not be effective in a steep draw). In general, many of the discussed erosion-control structures (especially those designed to stabilize slopes and promote the development of vegetation) require very regular maintenance and have a short life span.

Dry Wells or French Drains

A dry well or French drain can be used to treat contaminated runoff and promote infiltration of surface water back into an aquifer. French drains are only effective at removing sediment from contaminated waters and are not designed to treat hazardous wastes. Both structures are passive, relying on gravity to pass runoff through a filter comprised of rock or gravel before it mixes with groundwater (Figure 16.4-2). These systems are often lined with filter fabric, and in general, the deeper the well or drain, the more effectively it can remove particulate pollution.

Revegetation

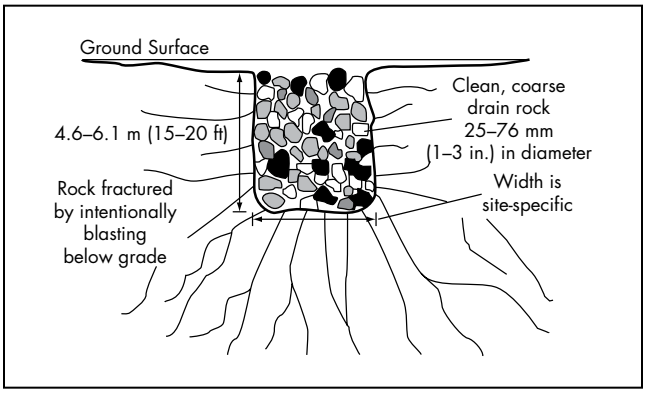
Steep, smooth, and nonvegetated slopes and spoil mounds are subject to high rates of water and wind erosion. High erosion rates decrease slope stability, result in the loss of valuable topsoil, and can be very costly to repair. Adding vegetation cover to an exposed soil surface can intercept and reduce raindrop impact and increase infiltration, which slows and redirects overland flow (Kleinman 1996). Several immediate revegetation establishment options exist, including drill seeding, hydroseeding, broadcast seeding, and transplanting entire live plants or plant cuttings. In addition, the placement of mulch can increase soil moisture, provide a temporary cover to reduce erosion risk, moderate soil temperature, and increase the likelihood of seed establishment (Kleinman 1996).

Natural and Biodegradable

Straw or hay bale check dams can be installed for temporary sediment or erosion control for one season or less. Straw or hay bale check dams are placed end to end on contour to reduce the run length of any exposed slope (Table 16.4-4). The most critical factor in proper installation of a straw bale check dam is the surface slope, which should be less than 20%. In addition to straw bale check dams, there are a variety of biodegradable matrix fabric structures that can be used both to protect steep slopes from erosion and to aid in the establishment of living vegetation (Barrett et al. 1998; North American Green 2009; RoLanka International 2009).

Artificial and Synthetic Materials

The most commonly used synthetic erosion-control product is sediment fencing. Sediment fences act as a dam, trapping sediment on the upslope side and allowing water to leave the site (Figure 16.4-3). They are effective in retaining suspended solids coarser than 0.02 mm (0.00079 in.). They are simple to construct, relatively inexpensive, and easily moved as development proceeds.



Source: Norman et al. 1998.

Figure 16.4-2 Dry well or French drain construction details

Table 16.4-4 Straw or hay bale check dam spacing

Slope, %	Maximum Slope Length Above a Single-Bale Check Dam or Between Successive Bale Check Dams, ft
≤2	250
5	100
10	50
15	35
20	25

MECHANICAL TREATMENT (SEDIMENT SEPARATORS)

The mining industry has utilized mechanical treatment for sediment removal or sediment separation for decades. Mechanical separators can be incorporated into the mineral-processing stream (placer or aggregate mines, process mills) or can be used solely for environmental purposes. For the purposes of this chapter, only environmental use of this equipment will be discussed. The principal advantage of mechanical separators is their ability to effectively remove sediment from water where space or time is limited. With the addition of flocculants, these separators can generally meet discharge effluent standards relatively quickly without requiring several acres of treatment ponds. Mechanical separators will also reduce cleanout costs because sediment traps and particulate removal is integrated within the equipment process stream.

Mechanical separators can include a swirl concentrator or inertial separator and will intercept sediment-laden flow. The swirl concentrator separates sediment from the incoming flow through the centrifugal force generated by the inherent inertia of the flow (Warner and Dysart 1983). In many ways they operate in the same way as centrifugal cyclones used in mineral processing. Effluent with a high sediment load is transmitted to a small sediment trap, while the clearer flow is discharged directly to a stream or receiving water. The inertial separator is designed to retain incoming sediment in a tank between a series of V-cut bottom-slotted troughs (Sterling and Warner 1984). The sediment-laden influent is transferred from influent troughs into the tank, and clearer flow is withdrawn by deep troughs (Warner 1992). The concept behind these mechanical separators has been transferred to stormwater

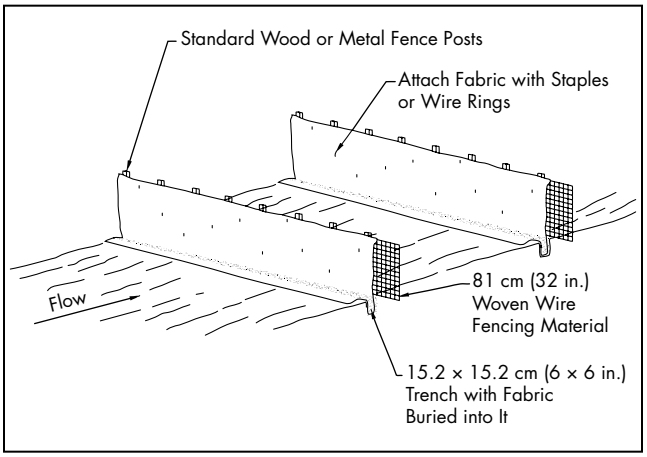


Figure 16.4-3 Sediment fence installation details

control agencies and has manifested as design guidelines for BMPs in the control of sediment-laden runoff. An example of this is presented in the Washington County (Oregon, United States) Clean Water Services design and construction standards, one of which is presented as Figure 16.4-4.

The large-scale use of mechanical separation in the mining industry may include hydrocyclone separators, which are prevalent in the gold mining industry, and water clarifiers, which are utilized extensively in the aggregate industry. Both systems allow operators to take a dirty stream of water and produce clean water immediately, concentrating the fines or solids to a thick state. Hydrocyclones are based on the centrifuge principal, where the particles are spun against the outside wall of the separator and gravitate toward the bottom and are collected in the sedimentation tank. A water clarifier is a similarly self-contained unit but uses a flocculant-driven process with a combination of settlement time and available space. It includes a dry polymer feed system, a hydraulic package, control panels, and a mud-discharge pump system. As the sediment is removed from the water, it is pumped as slurry out of the clarifier to its interim or final resting space. The clean water is pumped back into the process stream or to the receiving water. Hydrocyclones and clarifiers can typically handle greater flow volumes of up to 252 L/s (4,000 gpm) and greater sediment concentrations (up to 20% solids).

GROUNDWATER HYDROLOGY

Both surface and underground mines must develop a strategy to handle groundwater both for water quantity and quality. If the volume of intercepted groundwater exceeds an operation's ability to pump, flooding of the mine operations will result. If groundwater commingles with the mining operation, contamination of this water by the operation may ensue. Such contamination may result from interaction with equipment and haul traffic (oil, grease, sediment) or from an interaction with a by-product of mining, which could include the leaching of metals as the groundwater moves from a reduced to an oxidized state. Excess groundwater must be handled in a similar fashion as excess surface water and must be treated in advance of discharge.

The purpose of this section is to outline a strategy for designing a groundwater monitoring program that provides useful information throughout the life of the mining operation. Like surface water, if groundwater is successfully intercepted and diverted around the mining operation, the need for treatment is eliminated. Unlike surface water, groundwater interception requires extensive subsurface capital construction, which may include slurry walls, grout curtains, or a dewatering well field.

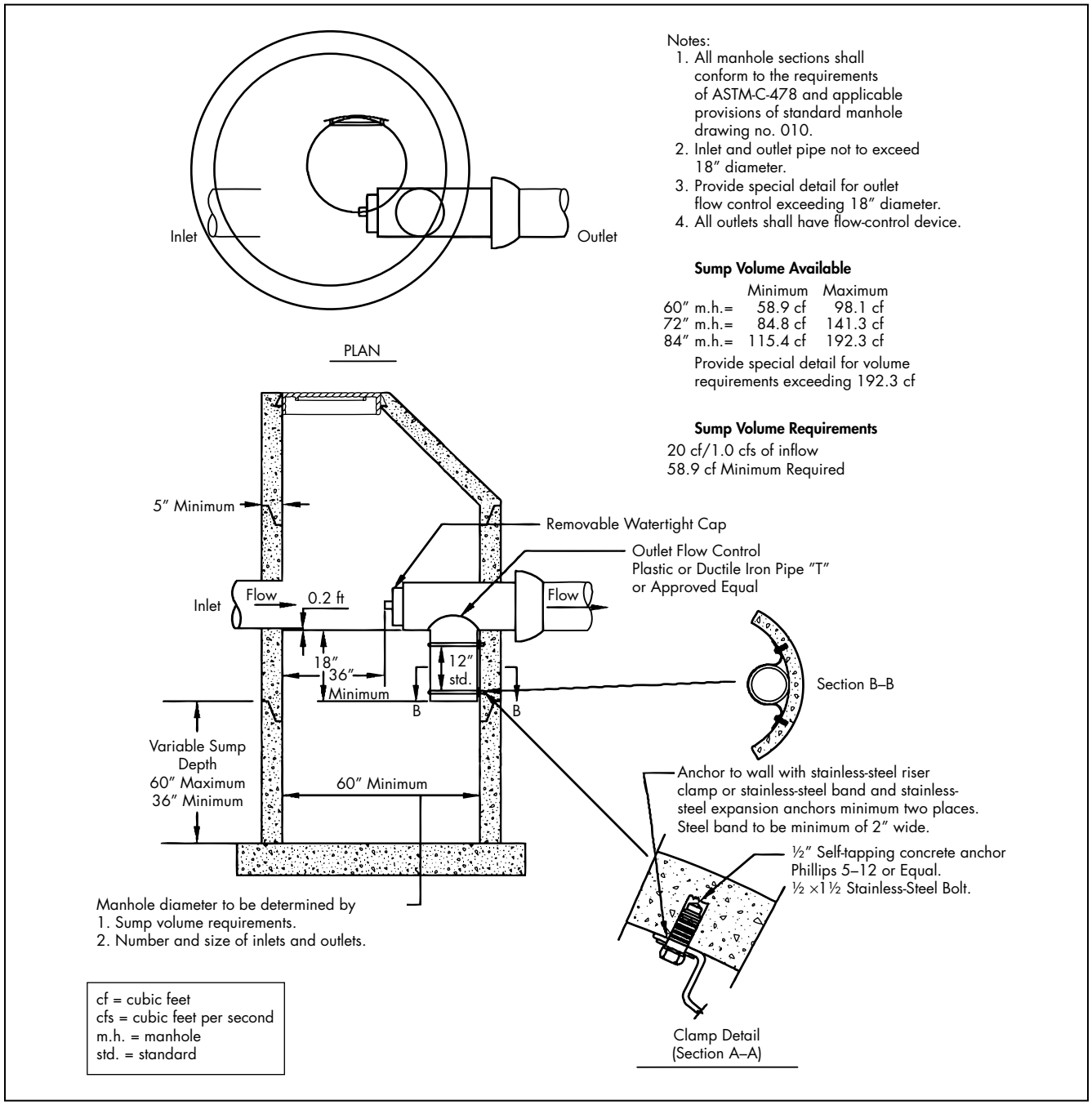
Predictions of Groundwater Inflow

Predictions of groundwater inflow require a detailed knowledge of the geologic framework, including structure, stratigraphy, and a definition of the aquifer properties. Such properties typically include an understanding of hydrologic boundaries (recharge and discharge), an aquifer's potentiometric surface, the presence of aquitards (low-permeability strata) and aquicludes (very low permeability strata), and areas of aquifer communication. Supporting data can be derived from the geologic exploration program, which may include drill-hole logs, geotechnical studies, geophysical logs, geological cross sections, monitor wells, and structure contour maps. Aquifer hydraulic characteristics must be derived from actual pump tests, which have been conducted on or near the mine property.

Aquifer Testing

Aquifer tests should be used to determine transmissivity, storage coefficient, hydrologic boundaries, leakage, aquifer homogeneity, and isotropy. For example, an aquifer pump test can be used to determine these groundwater properties in accordance with methods described by Theis (1935), Cooper and Jacob (1946), Boulton (1954), and Lohman (1979). The location and number of aquifer tests should be sufficient to characterize the different aquifers or the variability within the aquifer in the vicinity of the mine. The purpose of the data collection effort will dictate the degree of investment. For example, an in-situ mine requires an extensive network of wells and aquifer tests to not only define the hydrologic unit from which resource extraction is taking place but also to define the overlying and underlying aquifer and confining units. A coal or metal mine may require an aquifer testing program to establish baseline data and to ensure that the mine workings remain dry during and in some cases following the extraction process. Geochemical change, where groundwater passes through spent mine workings, is a long-term surface and underground mining concern.

Aquifer tests include single-well and multiple-well tests. A single-well test can allow the specialist to establish permeability or transmissivity of an aquifer on a local scale. A multiple-well test can allow for the calculation of storativity, as well as provide data for the degree of communication between a production zone and its overlying and underlying counterparts. A typical aquifer test includes the placement and testing of a pump in the pumping well and monitoring water level changes in that well and any other well in the immediate area. Well completion and aquifer testing are well presented in the literature (Sterrett 2007). The length of an aquifer test may change depending on its relative importance in the design process. A short-term test (1 to 4 hours) typically provides information in the immediate area of the well bore. An intermediate-length test (4 to 24 hours) provides good information relative to the aquifer in general. A long-term test (24 hours to 30 days) provides excellent information relative to the aquifer, aquifer sustainability, and area boundary conditions. A good guideline is that the longer the test, the greater the aquifer area tested. When designing an extensive underground mine operation, dependent on dewatering, a long-term multiple-well test is



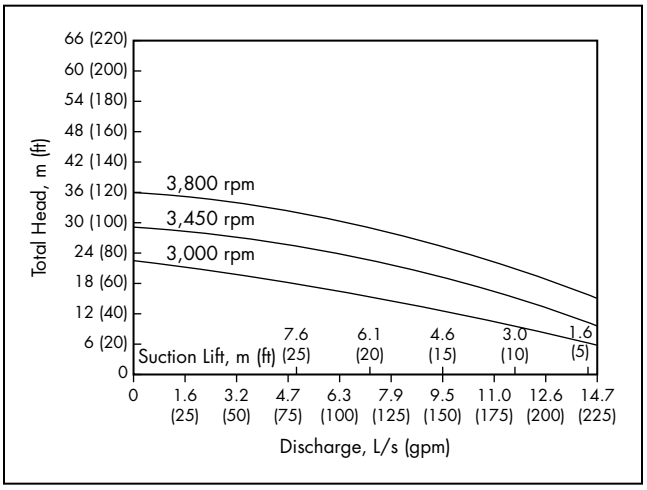
Source: Clean Water Services 2009.

Figure 16.4-4 Stormwater manhole and sediment separator

recommended. It is important to collect and maintain meteorological data throughout the course of the test.

Methods to Handle Groundwater Inflow

Groundwater can be handled before mine interaction, on interception by the mine, or after it passes through the mine. On a water quality and treatment basis, it is far better to handle groundwater before it interacts with the mine or mine workings. This can be done simply using a dewatering well field, or it can be done using slightly more complex and far more permanent slurry walls or a grout curtain. A dewatering well field requires closely spaced wells that are pumped continuously, and the respective cone of depression of each well will overlap and ensure that no groundwater passes through the well field. This is an effective means of controlling groundwater flow, but it requires nearly constant operation and maintenance. The advantages are its relatively low capital cost, its system design flexibility, and its short-term effectiveness. The disadvantages are the operational costs of keeping pumps and well screens operational, the electrical costs, and the need to move the system as the mine operation advances.



Source: Warner 1992.

Figure 16.4-5 Pump characteristic curves for engine-driven pump

A far more effective and long-term method is the construction of a grout curtain, which includes the drilling of exploration boreholes into a lower confining layer and emplacement (injection) of a concrete grout or, in limited cases, a freezing agent such as liquid nitrogen. Drill-hole spacing must be adequate to ensure the overlapping of the grout to guarantee an impermeable seal. In the case of the concrete grout curtain, its advantage is its relative permanent characteristic such that after the mine closes, groundwater will continue to be diverted around the mining operation, thereby avoiding the long-term geochemical impacts. The disadvantages are the initial capital cost and its overall ineffectiveness in some fractured aquifer systems.

A recent advancement in groundwater interception for mining is the use of slurry walls. This technique is far more effective than a grout wall in that it is constructed as a continuous trench that is backfilled with a bentonite/sand mix. Slurry walls are keyed into a lower confining layer, but may be limited by equipment reach and depth. Recent advances in slurry-wall construction allow depths up to 24 m (70 ft). The advantages of slurry walls are their relatively lower capital cost compared to grout walls and their relatively permanent behavior, even after the mine closes, so that groundwater will continue to be diverted around the mining operation. The disadvantages of such slurry walls are that they are limited to relatively shallow applications and they require a nearby source of clean sand. They are particularly effective in sand and gravel mining near or adjacent to the floodplain of a river.

When groundwater enters the mine operation, it must be collected in a sump (at a low spot within the mine workings) and removed from the mine operation by mechanical means. This is generally accomplished with a pump or a series of pumps. Pump and pipeline redundancy is absolutely critical if there is a pump or pipeline failure. Another critical concern is electrical failure and protection of the dewatering controls. Backup generators should be available at all mine sites, and the system of controls should be protected from a rising water level and/or placed at a location that is above the water table if all pumps were shut off (i.e., the recovered potentiometric surface).

Ideally, groundwater should be intercepted before it has an opportunity to be contaminated by mine equipment or oxidized waste rock. For this reason, sumps can be constructed on highwall benches or at the leading edge of an underground mine. If the groundwater is unaffected, it can be pumped directly to the surface streams. If it is affected, it must be handled in the same fashion as surface water and treated through sediment or treatment ponds. In most cases, additional treatment for mobilized metals will be required beyond pure sediment control, and this may include, but not be limited to, the addition of lime (pH control), organic matter (chelation), "bugs" (sulfate-reducing agents), or barium chloride (radium control).

Pumps

Pumps are widely used throughout the mining industry for dewatering underground mine operations and active pits, and for predraining areas adjacent to the pit prior to excavation (Warner 1992). Having the ability to control the presence of groundwater or surface water can alleviate numerous production problems and provide safer working conditions.

Centrifugal Pump

The basic components of a centrifugal pump are the inlet, the eye of the rotating impeller, the curved impeller vanes, the volute (i.e., casing), and the discharge connection. A centrifugal pump is a simple device with the only moving part being the impeller, which is attached to the shaft of a motor or engine. Assuming that the suction pipe and pump housing (volute) is filled with water to the level of the impeller eye (i.e., it is primed), then as the impellers rotate, a partial vacuum is created that allows atmospheric pressure to lift more water into the pump housing. The water, which enters the inlet opening at the eye of the impeller, is set in rotation by the impeller and creates centrifugal force, resulting in pressure at the outer perimeter of the impeller. Water moves outward from the impeller at a high velocity and pressure into an expanding volute and is then discharged.

Calculating Dynamic Head

The essential information needed to properly select a pump is the required flow rate in liters per second (gallons per minute) and the total dynamic head in meters (feet). Flow rate is dependent primarily on operational factors such as the anticipated rate of flow into a pit or underground mine. Determination of the total dynamic head requires consideration of the suction lift; the elevation difference between the pump and the discharge point; the pressure requirements, if any, at the discharge outlet; the friction along the pipes and fittings; and the velocity head.

Representative pump characteristic curves are illustrated in Figure 16.4-5. The curves show the relationship between the discharge and the total dynamic head. Also shown along the x-axis is the suction lift or net positive suction head. The discharge points along the x-axis indicate the maximum flows that the pump is capable of delivering at designated suction lifts.

Pump Troubleshooting

A properly designed and installed pump will reduce potential operational problems. A vacuum gauge, installed a minimum of one pipe diameter from the pump inlet, can solve the majority of problems. A high gauge reading indicates a partially blocked suction line, whereas a low reading often indicates air leakage through fittings on the suction side. A vacuum gauge can easily detect common problems, such as a nonprimed pump; too high a dynamic lift; excessive air in the water due

Table 16.4-5 Annotated list of computer modeling software

Use	Software	Source*
Surface water hydrology	WinTR55	USDA NRCS
	HEC-1	USACE
Surface water hydraulics	HEC-RAS	USACE
	HSPF	USGS, USEPA
	Flowmaster	Bentley
Culvert hydraulics	HY-8	USDOT, FHWA
Site grading, geomorphology	Carlson Natural Regrade with Geofluv	Carlson Software
Groundwater, aquifer characteristics	AQTESOLV	Hydrosolve
Groundwater modeling	Visual MODFLOW	Waterloo Hydrogeologic
Hydrology, sediment and structures	SEDCAD	Civil Software Design LLC
Sediment yield and transport	SIMSED99	USACE
Sediment transport	HEC-HMS	USACE
	GSTARS	USBR
	SMS	Scientific Software Group
Erosion	WEPP	USDA-ARS
Water quality	WATEQ4F	USGS
	AquaChem	Schlumberger Water Services
	QUAL2E	USEPA

*FHWA = Federal Highway Administration
NRCS = Natural Resources Conservation Service
USACE = U.S. Army Corps of Engineers
USBR = U.S. Bureau of Reclamation
USDA = U.S. Department of Agriculture
USDA-ARS = U.S. Department of Agriculture–Agricultural Research Service
USDOT = U.S. Department of Transportation
USEPA = U.S. Environmental Protection Agency
USGS = U.S. Geological Survey

to the intake being near the water surface; air leakage in the inlet pipe, fittings, or stuffing box; or the strainer, foot valve, or suction pipe that is too small or restricted by debris. If a discharge pressure gauge reads too low, the pump may not be primed, the revolutions per minute may be too low, the total dynamic head may be too high, the impeller may be rotating in the wrong direction, there may be excessive air in the water, there may be mechanical defects, and so forth. Excess pump vibration may indicate a misalignment, worn bearings, a bent shaft, or that an obstruction has lodged in one side of the impeller.

COMPUTER MODELS AND REFERENCES

Utilization of computer applications and software can offer readily available professional assistance to the mining engineer. Simulation models and software are very important tools that are used to evaluate both surface and groundwater hydrology as well as sediment and erosion processes. Much of this software is publicly available and can be downloaded for free or at a nominal charge. Table 16.4-5 identifies some of the currently available software that has been successfully used by the environmental professional, with a certain bias toward publicly available and free software. An excellent reference (including downloadable technology transfer) for design manuals and software used by the mining industry is the U.S. Department of the Interior, Office of Surface Mining Web site (DOI 2009).

A variety of software programs exist that allow the mining professional to evaluate both existing and proposed conditions. Not only are the modeling programs valuable on their own, but they can also be interfaced or joined with AutoCAD design software by Autodesk or with GIS software by ESRI.

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CHAPTER 16.5

Mitigating Acid Rock Drainage

Rens Verburg

INTRODUCTION

Sustainable mining requires the prevention, mitigation, management, and control of mining impacts on the environment. Acid rock drainage (ARD) continues to be one of the most serious and visible environmental issues facing the mining industry because it is often the transport medium for a range of pollutants, which may affect on-site and off-site water resources, and associated human and ecological receptors. The impacts of ARD on near and distant water resources and receptors can also be long term and persist after mine closure. Therefore, ARD prevention, mitigation, and treatment are important components of overall mine water management over the entire life of a mining operation.

This chapter addresses the prediction, prevention, and treatment of ARD. A comprehensive approach to ARD management reduces the environmental risks and subsequent costs for the mining industry and governments, reduces adverse environmental impacts, and promotes public support for mining. The extent and particular elements of the ARD management approach that should be implemented at a certain operation will vary based on many site-specific factors, not limited to the project's potential to generate ARD.

Acid rock drainage is formed by the natural oxidation of sulfide minerals when exposed to air and water. Activities that involve the excavation of rock with sulfide minerals, such as metal and coal mining, accelerate the process. ARD results from a series of reactions and stages that typically proceed from near-neutral to more acidic pH conditions. When sufficient base minerals are present to neutralize the ARD, neutral mine drainage or saline drainage may result from the oxidation process. Neutral mine drainage (NMD) is characterized by elevated metals in solution at approximately neutral pH, whereas saline drainage (SD) contains high levels of sulfate at neutral pH without significant dissolved metal concentrations. Figure 16.5-1 illustrates the various types of drainage schematically.

A risk-based planning and design approach forms the basis for prevention and mitigation. This approach is applied throughout the mine life cycle but primarily in the assessment and design phases. The risk-based process aims to quantify the long-term impacts of alternatives and to use this knowledge to select the option that has the most desirable combination of attributes (e.g., protectiveness, regulatory acceptance, community approval, cost). Mitigation measures implemented as part of an effective control strategy should require minimal active intervention and management.

Proper mine characterization, drainage-quality prediction, and mine waste management can prevent ARD formation in most cases and minimize ARD formation in all cases. Prevention of ARD must commence at exploration and continue throughout the mine life cycle. Ongoing ARD planning and management is critical to the successful prevention of ARD.

Stopping ARD formation, once initiated, may be challenging because it is a process that, left unimpeded, will continue (and may accelerate) until one or more of the reactants (sulfide minerals, oxygen, water) are exhausted or excluded from reaction. The ARD formation process can continue to produce impacted drainage for decades or centuries after mining has ceased, as is illustrated by the portal in Spain shown in Figure 16.5-2, which dates from the Roman era.

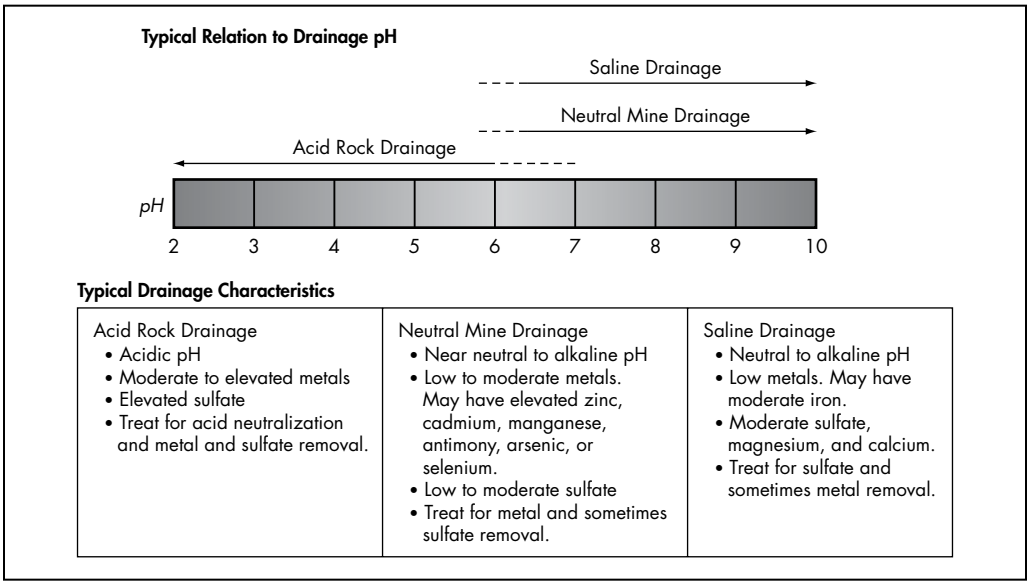
The cost of ARD remediation at orphaned mines in North America alone has been estimated in the tens of billions of U.S. dollars. Individual mines can face postclosure liabilities of tens to hundreds of million dollars for ARD remediation and treatment if the sulfide oxidation process is not properly managed during the mine's life.

This chapter draws heavily from and follows the general structure of the *Global Acid Rock Drainage Guide* (*GARD Guide*), a state-of-practice summary of the best practices and technologies. It was developed under the auspices of the International Network for Acid Prevention (INAP) to assist ARD stakeholders, such as mine operators, regulators, communities, and consultants, with addressing issues related to sulfide mineral oxidation (INAP 2009). Readers are encouraged to make use of the *GARD Guide* and its references for further detail on the subjects covered in this chapter.

FORMATION OF ACID ROCK DRAINAGE

The process of sulfide oxidation and formation of ARD is very complex and involves a multitude of chemical and biological processes that can vary significantly depending on

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Source: INAP 2009.

Figure 16.5-1 Types of drainage produced by sulfide oxidation

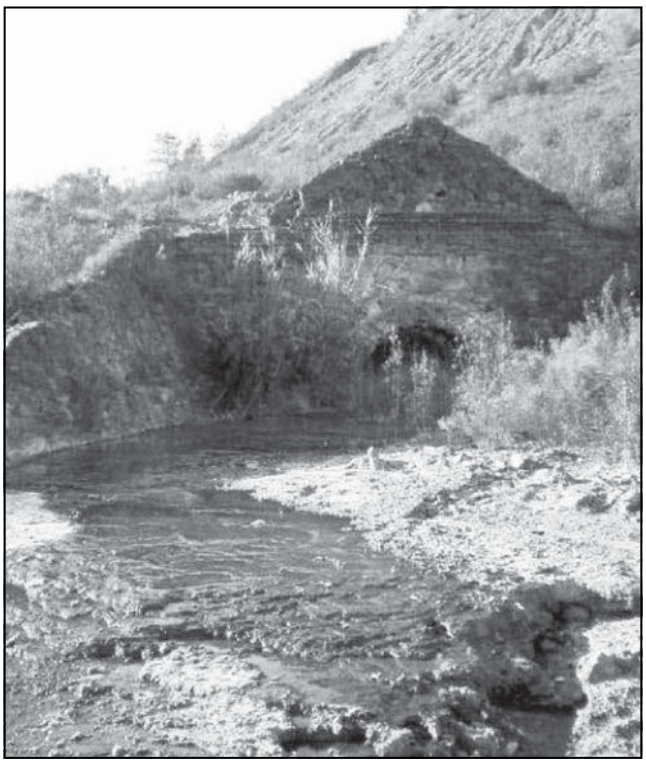
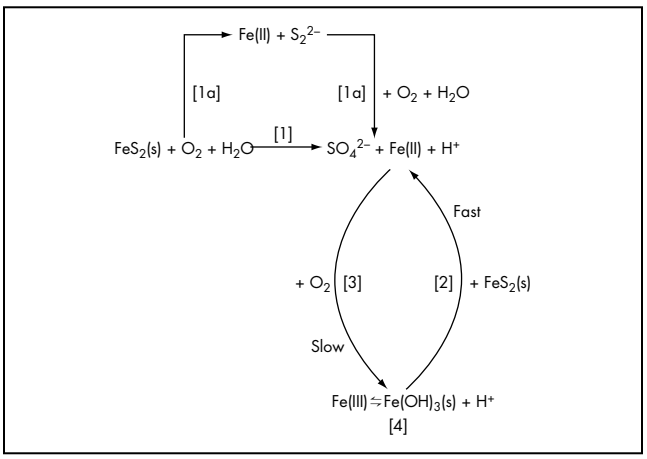


Figure 16.5-2 Roman portal with acid rock drainage—Spain



Source: Stumm and Morgan 1981.

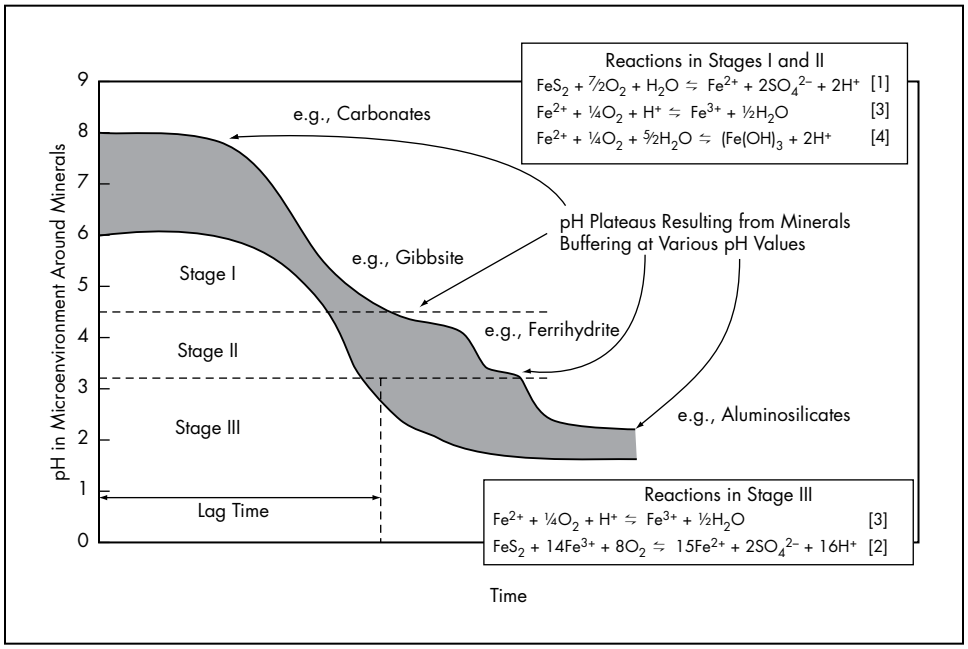
Figure 16.5-3 Model for the oxidation of pyrite

environmental, geological, and climate conditions (Nordstrom and Alpers 1999). Sulfide minerals in ore deposits are formed under reducing conditions in the absence of oxygen. When exposed to atmospheric oxygen or oxygenated waters due to mining, mineral processing, excavation, or other earth-moving processes, sulfide minerals can become unstable and oxidize.

Figure 16.5-3 presents a simplified model that describes the oxidation of pyrite, the sulfide mineral responsible for the majority of ARD (Stumm and Morgan 1981). The reactions shown are schematic and may not represent the exact mechanisms or reaction stoichiometry, but the illustration is a useful visual aid for understanding sulfide oxidation.

The chemical reaction representing pyrite oxidation (Reaction 1 in Figure 16.5-3) requires three basic ingredients: pyrite, oxygen, and water. This reaction can occur either abiotically or biotically (i.e., mediated through microorganisms). In the latter case, bacteria such as *Acidithiobacillus ferrooxidans*, which derive their metabolic energy from oxidizing ferrous to ferric iron, can accelerate the oxidation reaction rate by many orders of magnitude relative to abiotic rates (Nordstrom 2003). In addition to direct oxidation, pyrite can also be dissolved and then oxidized (Reaction 1a).

Under the majority of circumstances, atmospheric oxygen acts as the oxidant. However, aqueous ferric iron can oxidize pyrite as well according to Reaction 2. This reaction is



Source: INAP 2009.

Figure 16.5-4 Stages in the formation of ARD

considerably faster (two to three orders of magnitude) than the reaction with oxygen, and generates substantially more acidity per mole of pyrite oxidized. However, this reaction is limited to conditions in which significant amounts of dissolved ferric iron occur (i.e., acidic conditions: pH 4.5 and lower). Oxidation of ferrous iron by oxygen (Reaction 3) is required to generate and replenish ferric iron, and acidic conditions are required for the latter to remain in solution and participate in the ARD production process. As indicated by this reaction, oxygen is needed to generate ferric iron from ferrous iron. Also, the bacteria that may catalyze this reaction (primarily members of the *Acidithiobacillus* genus) demand oxygen for aerobic cellular respiration. Therefore, some nominal amount of oxygen is needed for this process to be effective, even when catalyzed by bacteria, although the oxygen requirement is considerably less than for abiotic oxidation.

A process of environmental importance related to ARD generation pertains to the fate of ferrous iron resulting from Reaction 1. Ferrous iron can be removed from solution under slightly acidic to alkaline conditions through oxidation and subsequent hydrolysis and the formation of a relatively insoluble iron (hydr)oxide (Reaction 4). When Reactions 1 and 4 are combined, as is generally the case when conditions are not acidic (i.e., pH > 4.5), oxidation of pyrite produces twice the amount of acidity relative to Reaction 1 as follows:

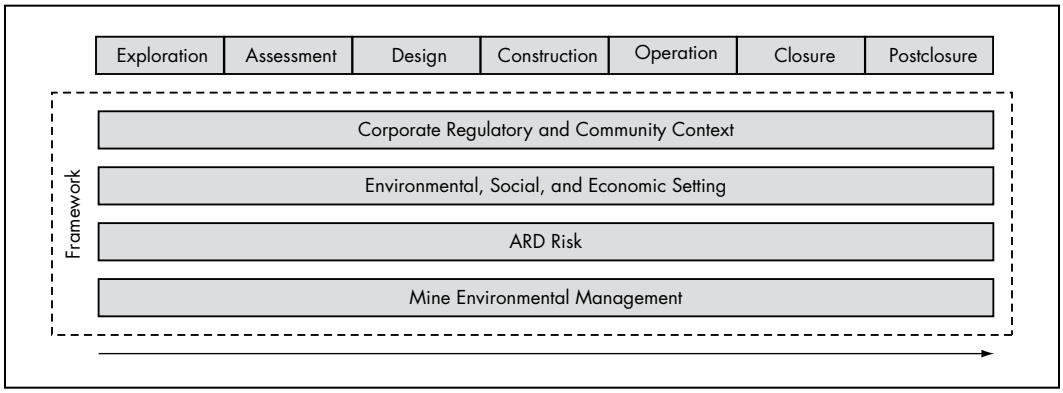
$$FeS_2 + 15/4O_2 + 7/2H_2O = Fe(OH)_3 + 2SO_4^{2-} + 4H^+$$

which is the overall reaction most commonly used to describe pyrite oxidation.

Although pyrite is by far the dominant sulfide responsible for the generation of acidity, different ore deposits contain different types of sulfide minerals. Not all of these sulfide minerals generate acidity when being oxidized. As a general rule, iron sulfides (pyrite, marcasite, pyrrhotite), sulfides with molar metal/sulfur ratios less than 1, and sulfosalts (e.g., enargite) generate acid when they react with oxygen and water. Sulfides with metal/sulfur ratios equal to 1 (e.g., sphalerite, galena, chalcopyrite) tend not to produce acidity when oxygen is the oxidant. Therefore, the acid generation potential of an ore deposit or mine waste generally depends on the amount of iron sulfide present. However, when aqueous ferric iron is the oxidant, all sulfides are capable of generating acidity.

Neutralization reactions also play a key role in determining the compositional characteristics of drainage originating from sulfide oxidation. As for sulfide minerals, the reactivity, and accordingly the effectiveness with which neutralizing minerals are able to buffer any acid being generated, can vary widely. Most carbonate minerals are capable of dissolving rapidly, making them effective acid consumers. However, hydrolysis of dissolved Fe or Mn following dissolution of their respective carbonates and subsequent precipitation of a secondary mineral may generate acidity. Although generally more common than carbonate phases, aluminosilicate minerals tend to be less reactive, and their buffering may only succeed in stabilizing the pH when rather acidic conditions have been achieved. Calcium–magnesium silicates have been known to buffer mine effluents at neutral pH when sulfide oxidation rates were very low (Jambor 2003).

The combination of acid generation and acid neutralization reactions typically leads to a stepwise development of ARD (Figure 16.5-4). Over time, pH decreases along a series of pH plateaus governed by the buffering of a range of mineral assemblages. The lag time to acid generation is a very important consideration in ARD prevention. It is far more effective (and generally far less costly in the long term) to control ARD generation during its early stages. The lag time also has significant ramifications for interpretation of test results. Because the first stage of ARD generation may last for a very long time, even for materials that will eventually be highly acid generating, it is critical to recognize the stage of oxidation when predicting ARD potential. The early results



Source: INAP 2009.

Figure 16.5-5 Conceptual ARD management framework

of geochemical testing, therefore, may not be representative of long-term environmental stability and associated discharge quality. However, early test results provide valuable data to assess future conditions such as consumption rates of available neutralizing minerals.

A common corollary of sulfide oxidation is metal leaching (ML), leading to the frequent use of the acronyms *ARD/ML* or *ML/ARD* to describe the nature of acidic mine discharges more accurately. Major and trace metals in ARD, NMD, and SD originate from the oxidizing sulfides and dissolving acid-consuming minerals. In the case of ARD, Fe and Al are usually the principal major dissolved metals, while trace metals such as Cu, Pb, Zn, Cd, Mn, Co, and Ni can also achieve elevated concentrations. In mine discharges with a more circumneutral character, trace metal concentrations tend to be lower as the result of formation of secondary mineral phases and increased sorption. However, certain parameters remain in solution as the pH increases, in particular the metalloids As, Se, and Sb, as well as other trace metals (e.g., Cd, Cr, Mn, Mo, and Zn).

FRAMEWORK FOR ACID ROCK DRAINAGE MANAGEMENT

The issues and approaches to ARD prevention and management are the same around the world. However, the specific techniques used for ARD prediction, interpretation of ARD test results, and ARD management may differ depending on the local, regional, or country context and are adapted to climate, topography, and other site conditions.

Therefore, despite the global similarities of ARD issues, there is no "one size fits all" approach to address ARD management. The setting of each mine is unique and requires a carefully considered assessment to find a management strategy within the broader corporate, regulatory, and community framework that applies to the project in question. The site-specific setting comprises the social, economic, and environmental situation within which the mine is located; the framework comprises the applicable corporate, regulatory norms and standards and community-specific requirements and expectations. This framework applies over the complete life cycle of the mine and is illustrated conceptually in Figure 16.5-5.

All mining companies, regardless of size, need to comply with the national legislation and regulations pertaining to ARD for the countries within which they operate. It is considered good corporate practice to adhere to global ARD guidance as well; in many cases, such adherence is a condition of funding.

Many mining companies have established clear corporate guidelines that represent the company's view of the priorities to be addressed and their interpretation of generally accepted best practice related to ARD. Caution is needed to ensure that all specifics of the country's regulations are met, as corporate ARD guidelines cannot be a substitute for country regulations.

Mining companies operate within the constraints of a "social license" that, ideally, is based on a broad consensus from all stakeholders. This consensus tends to cover a broad range of social, economic, environmental, and governance elements (sustainable development). ARD plays an important part in the mine's social license because it tends to be one of the more visible environmental consequences of mining.

The costs of closure and postclosure management of ARD are increasingly recognized as a fundamental component of all proposed and operating mining operations. Some form of financial assurance is now required in many jurisdictions.

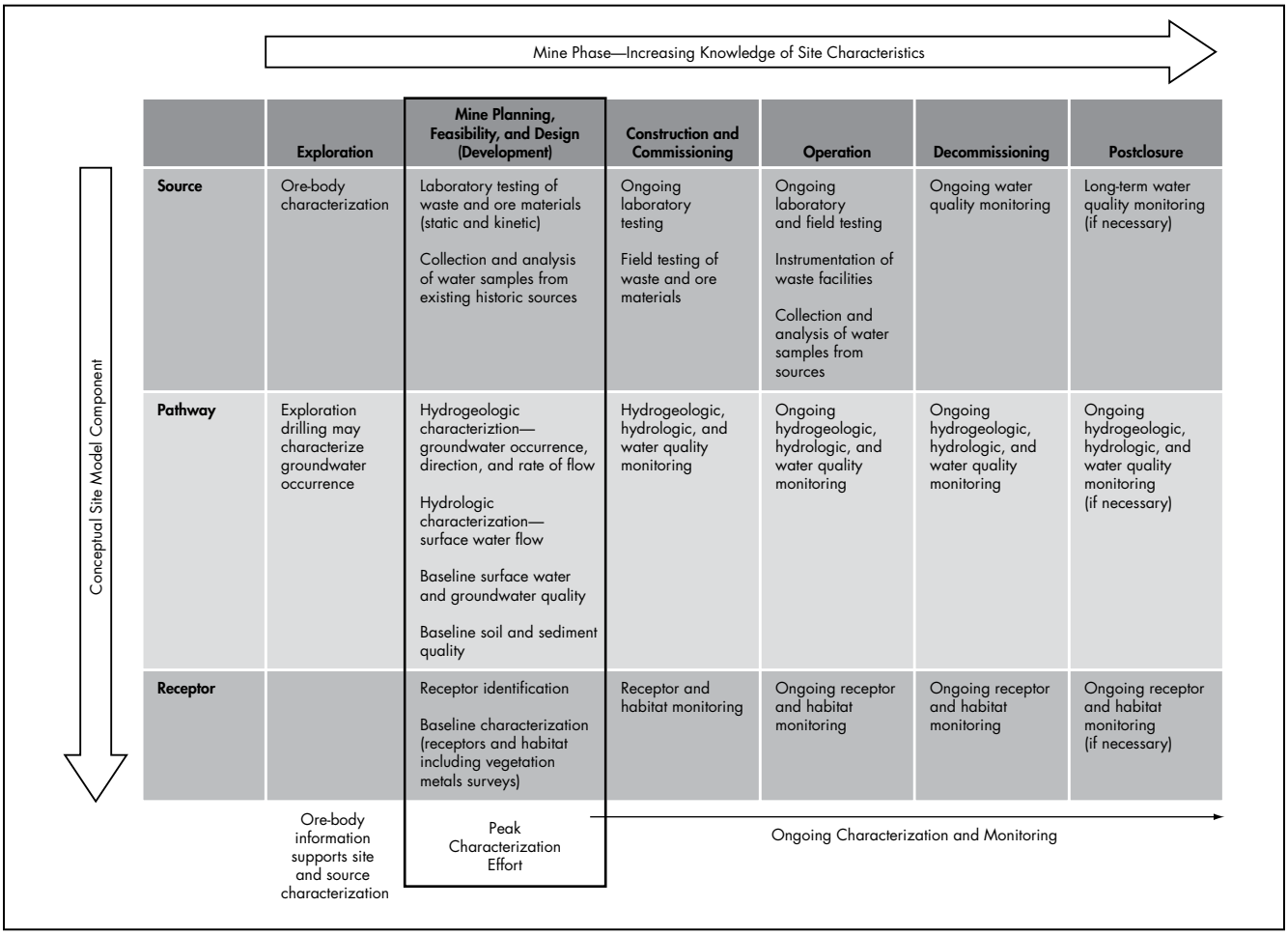
Characterization

The generation, release, transport, and attenuation of ARD are intricate processes governed by a combination of physical, chemical, and biological factors. Whether ARD becomes an environmental concern depends largely on the characteristics of the sources, pathways, and receptors involved. Characterization of these aspects is therefore crucial to the prediction, prevention, and management of ARD.

Environmental characterization programs are designed to collect sufficient data to answer the following questions:

1. Is ARD likely to occur?
2. What are the sources of ARD?
3. How much ARD will be generated and when?
4. What are the significant pathways that transport contaminants to the receiving environment?
5. What human and ecological receptors have contact with the receiving environment?
6. What level of risk would the anticipated concentrations of contaminants in the receiving environment pose to the receptors?
7. What can be done to prevent or mitigate/manage ARD?

The geologic and mineralogical characteristics of the ore body and host rock are the principal controls on the type of drainage that will be generated as a result of mining. Subsequently, the site climatic and hydrologic/hydrogeologic characteristics define how mine drainage and its constituents



Source: INAP 2009.

Figure 16.5-6 Overview of ARD characterization program by mine phase

are transported through the receiving environment to receptors. To evaluate these issues, expertise from multiple disciplines is required, including geology, mineralogy, hydrology, hydrogeology, geochemistry, (micro)biology, meteorology, and engineering.

The geologic characteristics of mineral deposits exert important and predictable controls on the environmental signature of mineralized areas (Plumlee 1999). Therefore, a preliminary assessment of the ARD potential should be made based on a review of geologic data collected during exploration. Baseline characterization of metal concentrations in various environmental media (i.e., water, soils, vegetation, animal tissue) may also provide an indication of ARD potential and serves to document the potential for naturally elevated metal concentrations.

During mine development and operation, the initial assessment of ARD potential is refined through detailed characterization data on the environmental stability of the waste and ore materials. The magnitude and location of mine discharges to the environment also are identified during mine development. Meteorological, hydrological, and hydrogeological investigations are conducted to characterize the amount and direction of water movement within the mine watershed(s) to evaluate transport pathways for constituents of interest. Identification of potential human and ecological receptors within the watershed boundary requires expertise in field biology and public health.

Over the mine life, the focus of the ARD characterization program evolves from establishing baseline conditions to predicting drainage release and transport and to monitoring of the environmental conditions, receptors, and impacts. Despite inherent differences at mine sites (e.g., based on commodity type, climate, mine phase, regulatory framework), the general approach to site characterization is similar.

Figures 16.5-6 and 16.5-7 present the chronology of an ARD characterization program and identify the data collection activities typically executed during each mine phase. The bulk of the characterization effort occurs prior to mining during the mine planning, assessment, and design (sometimes collectively referred to as the development phase). In addition, potential environmental impacts are identified and appropriate prevention and mitigation measures, intended to minimize environmental impacts and human and ecological risk, are incorporated. During the commissioning/construction and operation phases, a transition from site characterization to monitoring occurs, which is continued throughout the decommissioning/closure and postclosure phases. Ongoing monitoring helps refine the understanding of the site, which allows for adjustment of remedial measures, in turn resulting in reduced closure costs and improved risk management.

Mine Phase—Increasing Knowledge of Source Material Characterization

Waste or Facility Type—Potential ARD/NMD/SD Sources

	Exploration	Mine Planning, (Pre-) Feasibility and Design	Construction and Commissioning	Operation	Decommissioning	Postclosure (Care and Maintenance)
Waste Rock	Drill core descriptions and assay data (petrology and mineralogy) Block model (quantity of ore and waste) Review of any historical data	Laboratory testing of drill core samples—sample selection targets waste*	Ongoing laboratory testing of drill core or development rock samples* Field leach testing (barrels, test pads)	Ongoing laboratory testing* Ongoing field leach testing Collection and analysis of runoff and seepage samples from waste rock facility	Collection and analysis of runoff and seepage samples from waste rock facility	Collection and analysis of runoff and seepage samples from waste rock facility (if necessary)
Tailings		Laboratory testing of pilot plant tailings* Analysis of pilot testing supernatant	Ongoing laboratory testing of pilot-plant tailings*	Ongoing laboratory testing of tailings discharge* Collection and analysis of supernatant and seepage samples from tailings storage facility	Collection and analysis of supernatant and seepage samples from tailings storage facility	Collection and analysis of supernatant and seepage samples from tailings storage facility (if necessary)
Ore		Laboratory testing of drill core samples*		Ongoing laboratory testing*	If ore stockpiles exist, collection and analysis of runoff and seepage samples	
Pit		Laboratory testing of drill core samples—sample selection targets pit walls*		Field scale leach testing (e.g., wall washing) Collection and analysis of water samples (i.e., runoff, sumps)	Collection and analysis of pit water and pit inflow(s) water samples	Collection and analysis of pit water samples (if necessary)
Underground Workings		Laboratory testing of drill core samples—sample selection targets mine walls*		Collection and analysis of water samples (i.e., sumps, dewatering wells)	Collection and analysis of mine pool water samples	Collection and analysis of mine pool water samples (if necessary)

*Typical laboratory testing components: particle size, whole rock analysis, mineralogy, acid–base accounting, static and kinetic leach testing.

Source: INAP 2009.

Figure 16.5-7 ARD characterization program for individual source materials by mine phase

Prediction

One of the main objectives of site characterization is prediction of ARD potential and drainage chemistry. Because prediction is directly linked to mine planning, in particular with regard to water and mine waste management, the characterization effort needs to be phased in step with overall project planning. Early characterization tends to be generic and generally avoids presumptions about the future engineering/mine design, whereas later characterization and modeling must consider and be integrated with the specifics of engineering/mine design. Iteration may be required as evaluation of the ARD potential may result in the realization that a reassessment of the overall mine plan is needed. Integration of the characterization and prediction effort into the mine operation is a key element for successful ARD management.

Accurate prediction of future mine discharges requires an understanding of the sampling, testing, and analytical procedures used, consideration of the future physical and geochemical conditions, and the identity, location, and reactivity of the contributing minerals. All mine sites are unique for reasons related to geology, geochemistry, climate, commodity, processing method, regulations, and stakeholders. Prediction programs therefore need to be tailored to the mine in question. Also, the objectives of a prediction program can be variable. For instance, they can include definition of water treatment requirements, selection of mitigation methods, assessment of water quality impact, or determination of reclamation bond amounts.

Predictions of drainage quality are made in a qualitative and quantitative sense. Qualitative predictions are focused on assessing whether acidic conditions might develop in mine wastes, with the corresponding release of metals and acidity to mine drainage. Where qualitative predictions indicate a high probability of ARD generation, attention turns to review of alternatives to prevent ARD, and the prediction program is refocused to assist in the design and evaluation of these alternatives.

Significant advances in the understanding of ARD have been made over the last several decades, with parallel advances in mine water quality prediction and use of prevention techniques. However, quantitative mine water quality prediction can be challenging because of the wide array of the reactions involved and potentially very long time periods over

which these reactions take place. Despite these uncertainties, quantitative predictions that have been developed using realistic assumptions (while recognizing associated limitations) have proven to be of significant value for identification of ARD management options and assessment of potential environmental impacts.

Prediction of mine water quality generally is based on one or more of the following:

- Test leachability of waste materials in the laboratory
- Test leachability of waste materials under field conditions
- Geological, hydrological, chemical, and mineralogical characterization of waste materials
- Geochemical and other modeling

Analog operating or historic sites are also valuable in ARD prediction, especially those that have been thoroughly characterized and monitored. The development of geo-environmental models is one of the more prominent examples of the "analog" methodology. Geo-environmental models, which are constructs that interpret the environmental characteristics of an ore deposit in a geologic context, provide a very useful way to interpret and summarize the environmental signatures of mining and mineral deposits in a systematic geologic framework; they can be applied to anticipate potential environmental problems at future mines, operating mines, and orphan sites (Plumlee et al. 1999). A generic overall approach for ARD prediction is illustrated in Figure 16.5-8.

Prevention and Mitigation

The fundamental principle of ARD prevention is to apply a planning and design process to prevent, inhibit, retard, or stop the hydrological, chemical, physical, or microbiological processes that result in the impacts to water resources. Prevention should occur at, or as close to, the point where the deterioration in water quality originates (i.e., source reduction), or through implementation of measures to prevent or retard the transport of the ARD to the water resource (i.e., recycling, treatment, and/or secure disposal). This principle is universally applicable but methods of implementation are site specific.

Prevention is a proactive strategy that obviates the need for the reactive approach to mitigation. For an existing case of ARD that is adversely impacting the environment, mitigation will usually be the initial course of action. Despite this initial action, subsequent preventive measures are often considered with the objective of reducing future contaminant loadings, thus reducing the ongoing need for mitigation controls. Integration of the prevention and mitigation effort into the mine operation is a key element for successful ARD management.

Prior to identification or evaluation of prevention and mitigation measures, the strategic objectives must be identified. That process should consider assessment of the following:

- Quantifiable risks to ecological systems, human health, and other receptors
- Site-specific discharge water quality criteria
- Capital, operating, and maintenance costs of mitigation or preventive measures
- Logistics of long-term operations and maintenance
- Required longevity and anticipated failure modes

Prevention is the key to avoiding costly mitigation. The primary objective is to apply methods that minimize sulfide reaction rates, metal leaching, and the subsequent migration of weathering products that result from sulfide oxidation. Such methods involve the following:

- Minimizing oxygen supply
- Minimizing water infiltration and leaching
- Reducing, removing, or isolating sulfide minerals
- Controlling pore water solution pH
- Controlling bacteria and biogeochemical processes

Factors influencing selection of the above methods include the following:

- Geochemistry of source materials and the potential of source materials to produce ARD
- Type and physical characteristics of the source, including water flow and oxygen transport
- Mine development stage (more options are available at early stages)
- Phase of oxidation (more options are available at early stages when pH is still near neutral and oxidation products have not significantly accumulated)
- Time period for which the control measure is required to be effective
- Site conditions (i.e., location, topography and available mining voids, climate, geology, hydrology and hydrogeology, availability of materials and vegetation)
- Water quality criteria for discharge
- Risk acceptance by company and other stakeholders

More than one measure, or a combination of measures, may be required to achieve the desired objective.

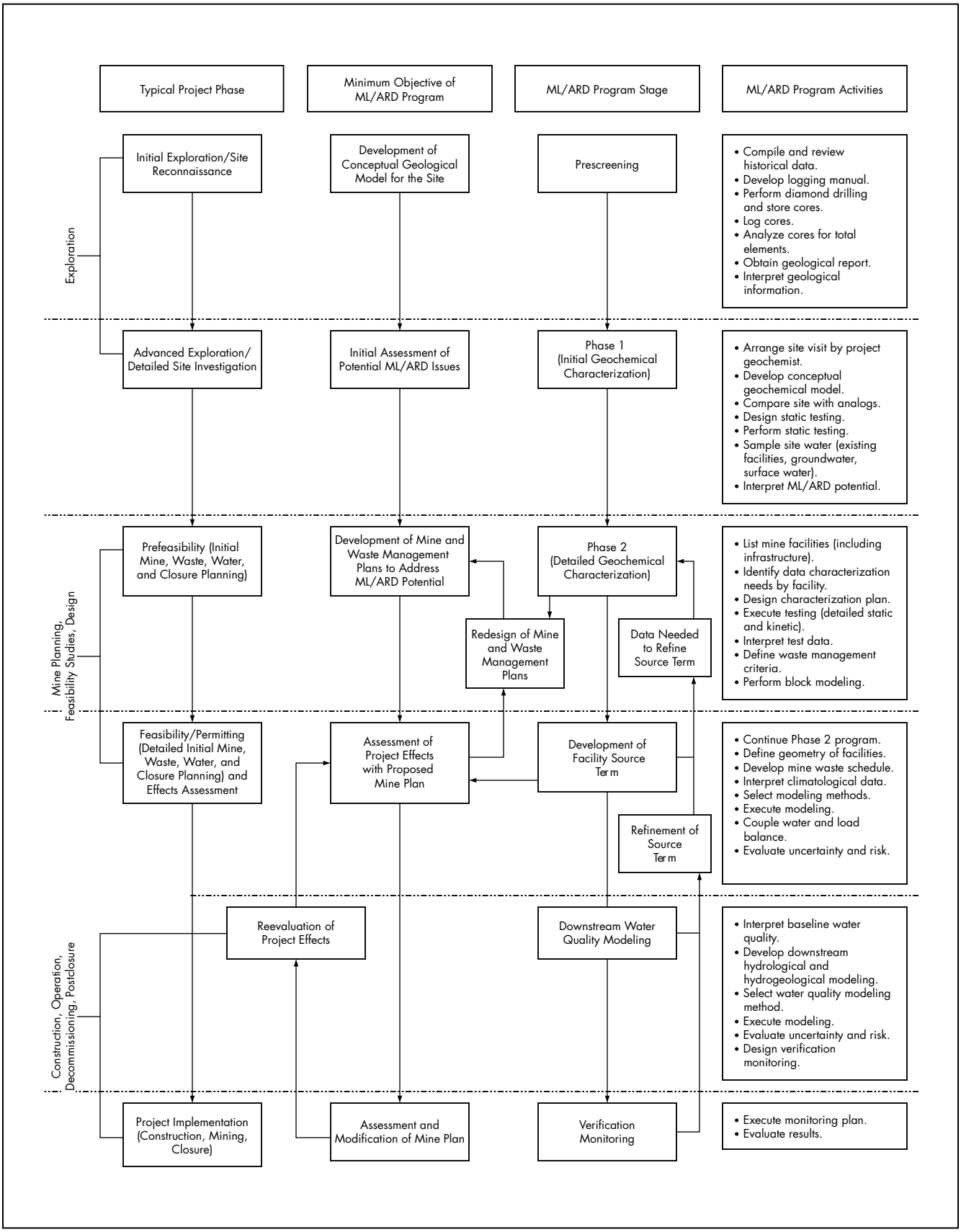
Typical objectives for ARD control are to satisfy environmental criteria using the most cost-effective technique. Technology selection should consider predictions for discharge water chemistry, advantages and disadvantages of treatment options, risk to receptors, and the regulatory context related to mine discharges. Figure 16.5-9 provides a generic overview of the most common ARD prevention and mitigation measures available during the various stages of the mine life cycle.

Treatment

The objectives of mine drainage treatment are varied. Recovery and reuse of mine water within the mining operations may be desirable or required for processing of ores and minerals, conveyance of materials, operational use (dust suppression, mine cooling, irrigation of rehabilitated land), and so forth. Mine drainage treatment, in this case, is aimed at modifying the water quality, so that it is fit for the intended use on or off the mine site.

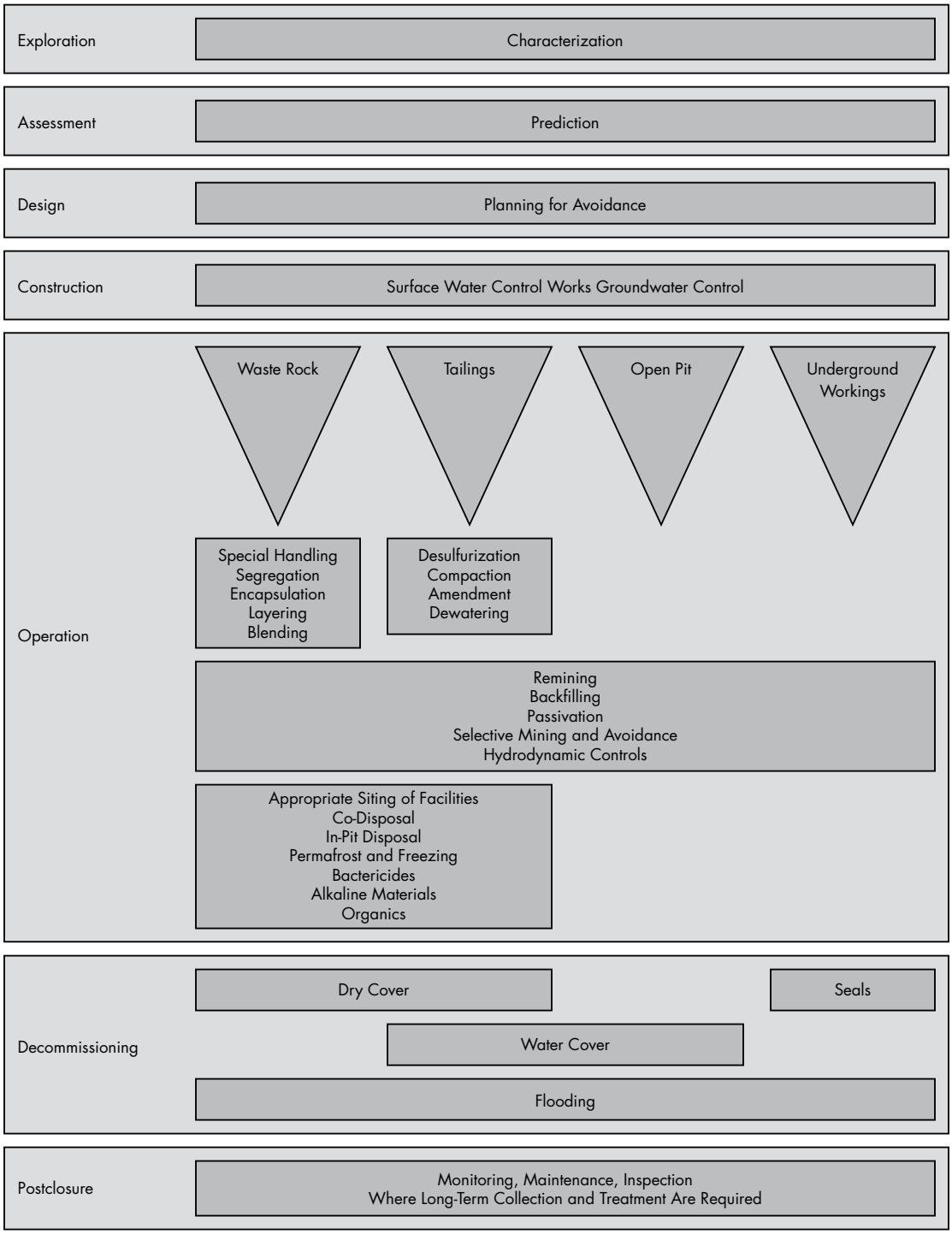
A more public objective of mine water treatment is the protection of human and ecological health in cases where people or ecological receptors may come in contact with the impacted mine water through indirect or direct use of on-site and off-site water resources. Water treatment then aims to remove the pollutants contained in mine drainage to prevent or mitigate environmental impacts.

In the majority of jurisdictions, any discharge of mine drainage into a public stream or aquifer must be approved by the relevant regulatory authorities; regulatory requirements stipulate a certain mine water discharge quality or associated discharge pollutant loads. Although discharge quality standards may not be available for many developing countries in which mining occurs, internationally acceptable environmental quality standards generally still apply as stipulated by project financiers and mining company policies.



Source: INAP 2009.

Figure 16.5-8 Generic overview of ARD prediction approach



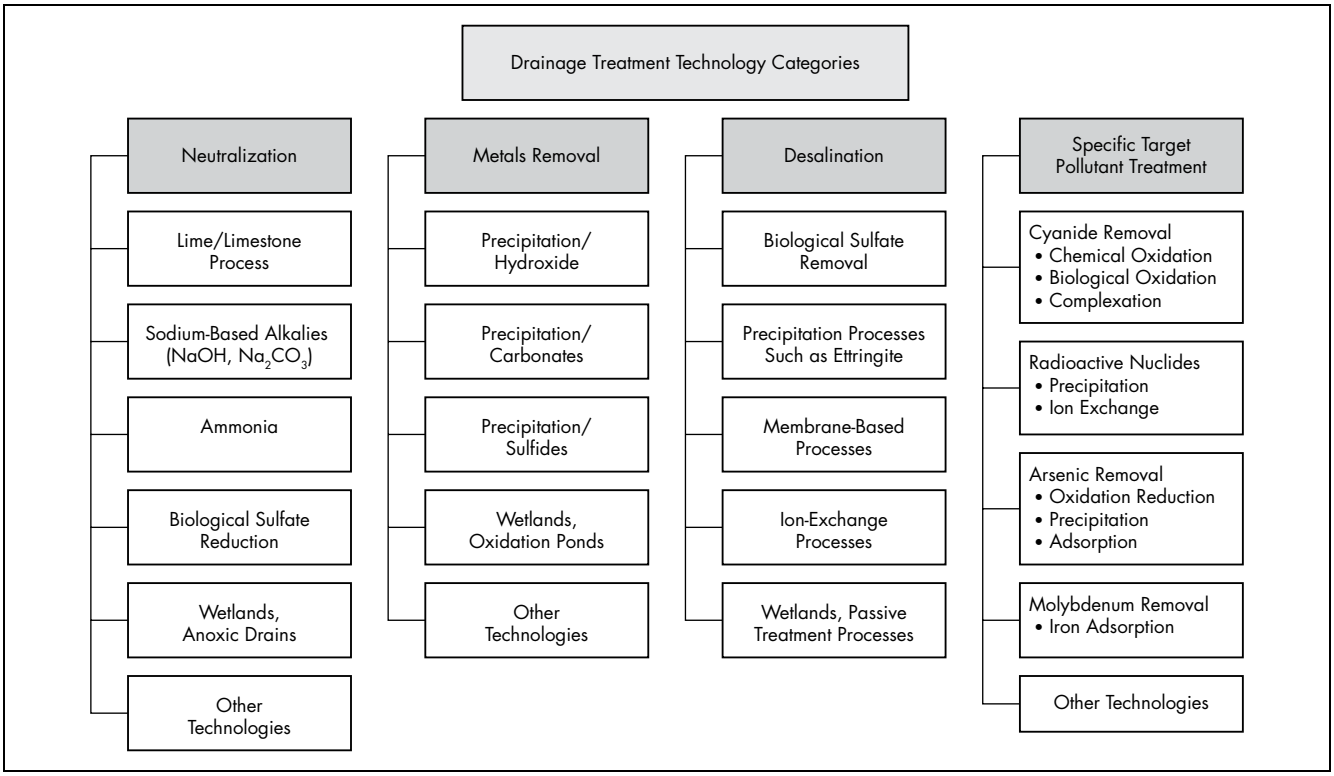
Source: INAP 2009.

Figure 16.5-9 Generic overview of ARD prevention and mitigation measures

The approach to selection of a mine drainage treatment method is premised on a thorough understanding of the integrated mine water system and circuits and the specific objective(s) to be achieved. The approach adopted for mine drainage treatment will be influenced by a number of considerations.

Prior to selecting the treatment process, a clear statement and understanding of the objectives of treatment should be prepared. Mine drainage treatment must always be evaluated and implemented within the context of the integrated mine water system. Treatment will have an impact on the flow and quality profile in the water system; hence, a treatment system is selected based on mine water flow, water quality, cost, and water use(s) and receptors.

Characterization of the mine drainage in terms of flow and chemical characteristics should include consideration



Source: INAP 2009.

Figure 16.5-10 Generic overview of ARD treatment alternatives

of temporal and seasonal changes. Flow data are especially important, as this information is required to size any treatment system properly. Of particular importance are extreme precipitation and snowmelt events that require adequate sizing of collection ponds and related piping and ditches. The key chemical properties of mine drainage relate to acidity/alkalinity, sulfate content, salinity, metal content, and the presence of specific compounds associated with specific mining operations, such as cyanide, ammonia, nitrate, arsenic, selenium, molybdenum, and radionuclides. There are also a number of mine drainage constituents (e.g., hardness, sulfate, silica), which may not be of regulatory or environmental concern in all jurisdictions but that could affect the selection of the preferred water treatment technology. Handling and disposal of treatment plant waste and residues such as sludges and brines and their chemical characteristics must also be factored into any treatment decisions.

A mine drainage treatment facility must have the flexibility to deal with increasing/decreasing water flows, changing water qualities, and regulatory requirements over the life of mine. This may dictate phased implementation and modular design and construction. Additionally, the postclosure phase may place specific constraints on the continued operation and maintenance of a treatment facility.

A variety of practical considerations related to mine site features will influence the construction, operation, and maintenance of a mine drainage treatment facility:

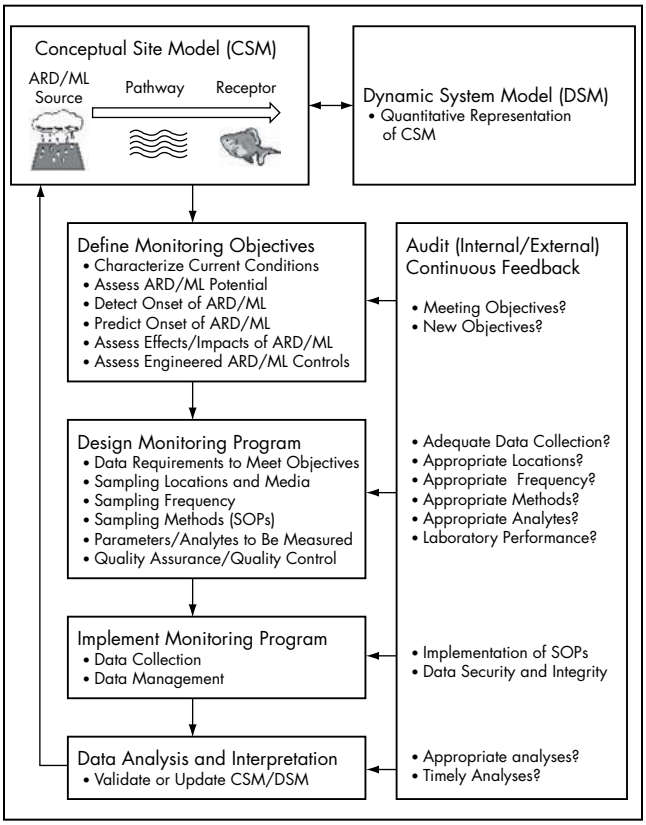
- Mine layout and topography
- Space
- Climate
- Sources of mine drainage feeding the treatment facility
- Nature and location of treated water receptors

Various ARD treatment alternatives are presented in Figure 16.5-10.

Monitoring

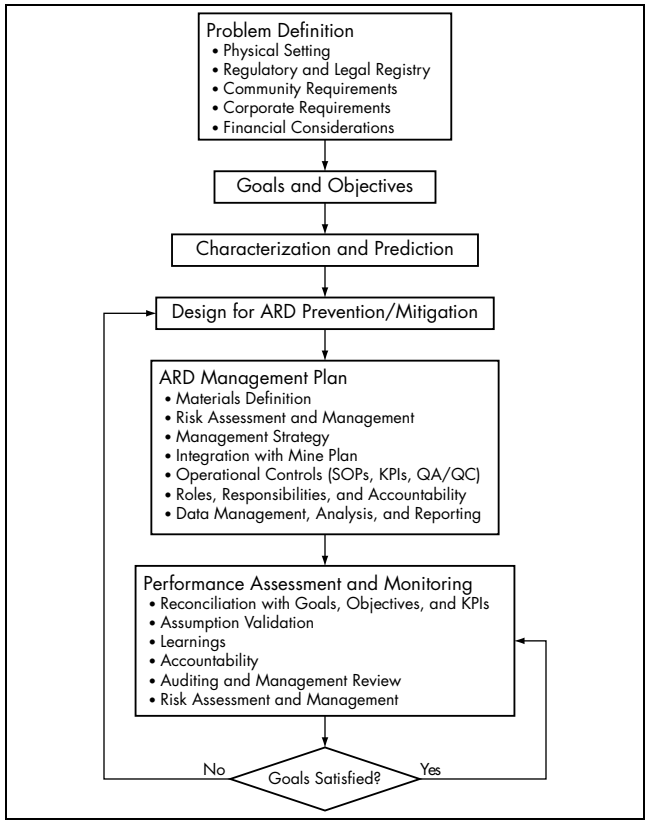
Monitoring is the process of routinely, systematically, and purposefully gathering information for use in management decision making. Mine site monitoring aims to identify and characterize any environmental changes from mining activities to assess conditions on the site and potential impacts to receptors both on- and off-site. Monitoring consists of both observation (e.g., recording information about the environment) and investigation (e.g., studies such as toxicity tests where environmental conditions are controlled). Monitoring is critical in decision making related to ARD management, for instance through assessing the effectiveness of mitigation measures and the subsequent implementation of adjustments to mitigation measures as required.

Development of an ARD monitoring program starts with a review of the mine plan, the geographical location, and the geological setting. The mine plan provides information on the location and magnitude of surface and subsurface disturbances, ore processing and milling procedures, waste disposal areas, effluent discharge locations, groundwater withdrawals, and surface water diversions. This information is used to identify potential sources of ARD, potential pathways for release of ARD to the receiving environment, receptors that may be impacted by these releases, and potential mitigation that may be required. Because the spatial extent of a monitoring



Source: INAP 2009.

Figure 16.5-11 Development of an ARD monitoring program



Source: INAP 2009.

Figure 16.5-12 Flow chart for ARD performance assessment and management review

program must include all these components, a watershed approach to ARD monitoring (including groundwater) is often required. Monitoring occurs at all stages of project development, from preoperational through postclosure. However, over the life of a mine, the objectives, components, and intensity of the monitoring activities will change. The development and components of a generic ARD monitoring program are presented in Figure 16.5-11.

Management and Performance Assessment

The management of ARD and the assessment of its performance are usually described within the site environmental management plan or in a site-specific ARD management plan. The ARD management plan represents the integration of the concepts and technologies described earlier in this chapter. It also references the engineering design processes and operational management systems employed by mining companies.

The need for a formal ARD management plan is usually triggered by the results of an ARD characterization and prediction program or the results of site monitoring. The development, implementation, assessment, and continuous improvement of an ARD management plan are ongoing processes throughout the life of a mine, which will typically follow the sequence of steps illustrated in Figure 16.5-12.

As shown in Figure 16.5-12, the development of an ARD management plan starts with establishment of clear goals and objectives, such as preventing ARD or achieving compliance with specific water quality criteria. This includes consideration of the physical setting, regulatory and legal registry, community and corporate requirements, and financial considerations. Characterization and prediction programs identify the potential magnitude of the ARD issue and provide the basis for the selection and design of appropriate ARD prevention and mitigation technologies. The design process includes an iterative series of steps in which ARD control technologies are assessed and then combined into a robust system of management and controls (i.e., the ARD management plan) for the specific site. The initial mine design may be used to develop the ARD management plan needed for an environmental assessment. The final design is usually developed in parallel with project permitting.

The ARD management plan identifies the materials and mine wastes that require special management. Risk assessment and management are included in the plan to refine strategies and implementation steps. To be effective, the ARD management plan must be fully integrated with the mine plan. Operational controls such as standard operating procedures (SOPs), key performance indicators (KPIs), and quality assurance/quality control (QA/QC) programs are established to guide its implementation. The ARD management plan identifies roles, responsibilities, and accountabilities for mine operating staff. Data management, analysis, and reporting schemes are included to track progress of the plan.

In the next step, monitoring is conducted to compare field performance against the design goals and objectives of the management plan. Assumptions made in the characterization and prediction programs and design of the prevention/

mitigation measures are tested and revised or validated. *Learnings,* or lessons learned during monitoring and assessment, are evaluated and incorporated into the plan as part of continuous improvement. Accountability for implementing the management plan is checked to ensure that those responsible are meeting the requirements stipulated in the plan. Internal and external reviews or audits should be conducted to gauge performance of personnel, management systems, and technical components to provide additional perspectives on the implementation of the ARD management plan. Review by site and corporate management of the entire plan is necessary to ensure that the plan continues to adhere to site and corporate policies. Finally, results are assessed against the goals. If the objectives are met, performance assessment and monitoring continues throughout the mine life with periodic rechecks against the goals. If the objectives are not met, then redesign and reevaluation of the management plan and performance assessment and monitoring systems for ARD prevention/mitigation are required. This additional effort might also require further characterization and ARD prediction.

The process described in Figure 16.5-12 results in continuous improvement of the ARD management plan and its implementation, and accommodates possible modifications in the mine plan. If the initial ARD management plan is robust, it can be more readily adapted to mine plan changes.

Implementing the ARD management plan relies on a hierarchy of management tools. Corporate policies help define corporate or site standards, which lead to SOPs and KPIs that are specific to the site and guide operators in implementing the ARD management plan. Where corporate policies or standards do not exist, projects and operations should rely on industry best practice.

Communication and Consultation

The level of knowledge of ARD generation and mitigation has increased dramatically over the last few decades within the mining industry, academia, and regulatory agencies. However, for this knowledge to be meaningful to the wide range of stakeholders generally involved with a mining project, it needs to be translated into a format that can be readily understood. This communication should convey the predictions of future drainage quality and the effectiveness of mitigation plans, their degree of uncertainty, and contingency measures to address that uncertainty. An open dialogue on what is known, and what can be predicted with varying levels of confidence, helps build understanding and trust, and ultimately results in a better ARD management plan.

Communicating and consulting with stakeholders about ARD issues is essential to the company's social license to operate. Because of the generally highly visible nature of ARD, skilled people are needed to communicate effectively, and the involvement of representatives from all relevant technical disciplines may be required.

SUMMARY

A thorough evaluation of ARD potential should be conducted prior to mining and continued throughout the life of the mine. Consistent with sustainability principles, strategies for dealing with ARD should focus on prevention or minimization, rather than control or treatment. These strategies are formulated within an ARD management plan, to be developed in the early phases of the project together with monitoring requirements to assess its performance. Integration of the ARD management plan with the mine operation plan is critical to the success of ARD prevention or mitigation. ARD management practices continue to evolve but tend to be site specific and require special expertise.

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CHAPTER 16.6

Waste Disposal and Contamination Management

Richard K. Borden

INTRODUCTION

Wastes generated by mining and mineral processing operations can have profound environmental impacts on land, water, and air resources if improperly managed. Mining and mineral processing generate large volumes of mineral waste, such as tailings and waste rock. At large mines the mass of mineral waste generated can commonly be measured in the tens of millions to billions of tons. Similarly, the surface area that must be disturbed for mineral waste disposal is often measured in the tens to thousands of hectares and may account for the majority of disturbance. If the mineral waste is also chemically reactive, the potential environmental impacts and the complexity of waste management increase dramatically.

Public concerns during project permitting are commonly centered on potential exposure risks and water quality impacts from chemically reactive mineral wastes and can result in project delays and costly permitting requirements. If mishandled, chemically reactive mineral waste can degrade water quality for centuries, creating large environmental and financial liabilities. Waste-related perpetual water management and treatment can account for more than half of the total closure cost at some mines. Fortunately, significant advances in mineral waste characterization and management have been made over the past several decades. Proactive mineral waste management can significantly reduce the intensity, footprint, and duration of environmental impacts. Companies that practice proactive management can reduce their financial liabilities, improve their reputations, and become miners of choice, helping ensure access to new mining opportunities.

Mining and processing operations also can generate significant volumes of nonmineral waste. Common hazardous and industrial wastes include solvents, used oil, oily debris, spent reagents, coolants, greases, batteries, and used paint. These wastes typically are sent to off-site recycling, treatment, or disposal facilities. Other nonhazardous wastes such as construction debris, packing materials, general office waste, and used tires may be recycled, sent to an off-site municipal landfill, or placed in permitted on-site landfills. Both hazardous waste and materials such as gasoline, strong acids, and some reagents must be carefully managed on-site to comply with legal requirements, protect worker health and safety, and avoid spills. Areas that have been impacted by historic hazardous material spills and releases may have contaminated soils and groundwater that can threaten human health or the environment. Active management or remediation of contaminated sites is commonly required at older mining operations.

MINERAL WASTE

The most common mineral wastes include overburden and waste rock, tailings, spent heap leach ore, block cave rubble, and slag. Waste rock, overburden, and interburden are unmineralized or weakly mineralized rock and unconsolidated sediments that must be removed to access the underlying ore. Waste rock is typically a poorly sorted mixture of clay, silt, sand, gravel, and boulder-sized material. The most common disposal method for waste rock is placement within dumps and stockpiles, although in-pit disposal is common in strip mines. Tailings are the fine-grained waste that remains after the minerals or elements of economic interest have been removed from the ore. Rejects from coal-washing facilities, fines generated by bauxite and iron ore washing, and red mud generated by aluminum refining also have many of the same physical characteristics and management requirements as tailings. Tailings are composed of the gangue minerals in the ore and residual minerals of economic interest that were not recovered, along with process water and any reagents that were added during the milling and beneficiation processes. Tailings are usually disposed of in specially engineered repositories that are capable of containing the fine-grained and often saturated tailings mass without risk of geotechnical failure.

Spent heap leach ore is the residual rock that is left after the elements of economic interest have been removed by leaching. It may have the same physical characteristics as waste rock or it may have undergone additional crushing, grinding, and/or agglomeration steps. Chemically, the spent heap leach ore has usually been altered by the addition of strongly acidic, saline, and/or basic leach solutions and the addition of reagents such as cyanide. Spent ore will also contain residual leach solution held within its pores, even after the initial drain down has been completed. In most circumstances

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spent ore is left in place on the heap leach pad after closure, but sometimes it may be actively cycled on and off the pads.

The block cave underground mining method can create a large mass of highly fractured and disturbed rubble overlying the underground workings. This rubble can have many of the same physical and chemical characteristics as waste rock, even though it has been generated in situ. Slag is the glassy waste residue generated by the smelting of metallic ores, typically at much lower volumes than other categories of mineral wastes.

Mineral waste disposal can be responsible for much of the environmental impact caused by mining. Potential impacts that must be assessed, minimized, and mitigated during mine design, operation, and closure include the following:

- **Direct land disturbance.** Construction of out-of-pit mineral waste storage facilities will typically require burial of the premining surface, its soils, and ecosystems beneath tens to hundreds of meters of waste. The surface area impacted by waste disposal is commonly larger than the disturbance required to access the ore body via open pits and underground workings.
- **Geotechnical instability.** Unless properly designed, thick waste piles may be prone to geotechnical instability and failure. Instability may range from excessive surface erosion to large deep-seated slope failures. Geotechnical risks are generally highest for tailings and other fine-grained waste materials that are saturated when deposited. Geotechnical issues and requirements related to waste management are detailed in Chapters 8.10 and 8.11.
- **Erosion and sediment release.** Erosive wastes are prone to the formation of gullies and other erosion features. The release of sediment at much higher rates than surrounding natural landforms can have negative impacts on downgradient surface water bodies and aquatic ecosystems.
- **Visual impacts.** Large-scale mineral waste transport and placement can significantly modify the landscape, creating landforms that are taller than the surrounding topography, truncating valleys and drainage lines, and creating unnatural uniform planar landforms that do not blend in with the surrounding natural topography. Mineral wastes may inhibit vegetation establishment, creating surfaces that look different from the surrounding undisturbed lands. Visual impacts are likely to be a particular concern near population centers and recreational or protected areas.
- **Direct exposure risks and phytotoxicity.** As detailed in the next section, chemically reactive mineral waste may pose direct chemical exposure risks to people, plants, and animals that live on or near the waste. The pH, salinity, or metals content of the waste can also inhibit vegetation establishment and prevent successful rehabilitation of waste surfaces.
- **Water quality degradation.** Chemically reactive mineral wastes can degrade the quality of water that runs off of or seeps through the waste material. Unless properly managed, this can cause degradation of surface and groundwater quality, impacts to aquatic ecosystems, and loss of the beneficial use of water resources far from the point of initial waste placement.
- **Dust release.** Wind erosion and dust release can degrade air quality because of increases in suspended particulate matter. If the dust is derived from chemically reactive waste, wind transport may disperse potential contaminants over a broad area.

Chemically Reactive Mineral Waste

Chemically reactive mineral waste has chemical or mineral constituents that could now or in the future create an environmental hazard above and beyond the risks posed by inert dust in air or inert total suspended solids in water. Several types of chemical hazards are associated with mineral waste and are discussed in the following paragraphs.

Acid Rock Drainage

The oxidation of naturally occurring sulfide minerals such as pyrite and chalcopyrite releases acidity, metals, and sulfate. Sulfide-bearing mineral waste can generate drainage with low pH, high salinity, and high metals concentrations, and can prevent the establishment of vegetation. Acid rock drainage (ARD) is most commonly associated with ore deposits that formed as a result of sulfide mineralization such as precious (gold, silver) and base metal (copper, nickel, lead, zinc) ore bodies. However, ARD is possible at any ore body or associated country rock that formed or exists under reducing conditions such as those typically found below the premining water table. For example, ARD has also been observed at coal, diamond, iron ore, and uranium mines. Operations that use acid for processing (such as uranium or copper heap leach operations) may also create acidic solutions and wastes with properties similar to ARD. ARD represents the most significant and most common chemical hazard posed by mineral waste and is discussed in more detail in Chapter 16.5 Good overviews of ARD chemistry, prediction, and management are provided in the *Global Acid Rock Drainage Guide* (INAP 2009) and the Mine Environment Neutral Drainage Program manuals (MEND 2001).

Metals Soluble or Bioavailable at Neutral pH

Many elements are soluble at neutral pH and can leach from mineral waste at concentrations that may be of environmental concern. Once in solution, these elements may be toxic to aquatic life or may make the water unfit for use as drinking, irrigation, or stock water. Many of these elements are also bioavailable at neutral pH and can inhibit vegetation establishment or pose direct exposure risks to animals and people via uptake by vegetation growing on the waste or incidental ingestion. These elements may include boron, cadmium, cobalt, chromium, fluoride, mercury, manganese, molybdenum, nickel, selenium, thallium, and zinc (Smith and Huyck 1999). Neutral drainage problems may be associated with many types of ore bodies and processes but are particularly common at sulfide-bearing ore bodies, which also contain significant neutralization potential. In this case the acidity released by sulfide oxidation is neutralized in situ, but the metals released from the sulfide minerals are soluble at neutral pH and can be mobilized out of the waste material.

Salinity and Sodicity

Some mineral wastes contain readily soluble salts, which can impact water quality and inhibit vegetation establishment. Major ions typically associated with salinity impacts include sodium, magnesium, potassium, calcium, chloride, sulfate, carbonate, and bicarbonate (Richards 1954). Elevated salinity can cause environmental impacts through general osmotic effects or specific ion effects. Saline soils can inhibit vegetation establishment, and solute loading to surface water can degrade water quality, harming aquatic life and ecosystems. Saline soils with a high proportion of exchangeable sodium

relative to calcium, potassium, and magnesium are defined as sodic soils. Sodic soils and wastes can have poor soil structure, are prone to erosion, and have poor drainage. Salt loading can make water unfit for use as drinking, stock, or irrigation water. Salinity guidelines for soils and water are usually expressed in terms of total dissolved solids concentration or electrical conductivity. However, water quality guidelines for specific major ions such as sulfate have also been promulgated in many jurisdictions. Many sectors of the mining industry may have salinity issues, including coal, borax, salt, potash, and any sulfide-bearing ore bodies. Operations that use seawater in the process can create tailings salinity issues. The evapoconcentration of zero-discharge or low-discharge water bodies and process circuits such as pit lakes and tailings impoundments may also result in high salinity.

Radionuclides

Radionuclides such as uranium, thorium, radium, and radon can pose exposure risks because of toxicity and/or radiological hazards. The release of uranium and its daughter products are an issue at uranium mines. However, radionuclides may also pose hazards at heavy mineral, coal, and rare earth ore bodies and at ore bodies associated with granitic rocks. Radon exposure may be a particular concern at some underground mining operations in granitic terrains.

Cyanide and Other Reagents

Cyanide is most commonly used in the mining industry to extract gold from ore and can also be used at low concentrations as a flotation reagent. As such it may be present in spent heap leach ore and tailings. Because of the toxicity of cyanide, airborne and aqueous releases to the environment are highly regulated in most jurisdictions. A good overview of cyanide geochemistry and environmental behavior is provided by Smith and Mudder (1999). Best practice for management of cyanide and cyanide-bearing wastes is provided by the International Cyanide Management Institute (ICMI 2009a, 2009b). Many other organic and inorganic reagents may be used in the flotation process or other ore-processing techniques. These reagents are commonly highly diluted within the resultant processing mineral waste but under some circumstances could pose risks to the environment and human health.

Nitrogen Compounds

Nitrogen compounds (ammonium, nitrate, and nitrite) are most commonly associated with incomplete combustion of blasting agents at hard-rock mines and can be present in both waste rock and tailings. Nitrate is also a common cyanide degradation product. Ammonium is also commonly used in uranium processing and is likely to be present in some uranium tailings. Nitrogen compounds have the potential to be directly toxic to aquatic life and can degrade water quality so that it cannot be used as drinking or stock water. The release of nitrogen and/or other nutrients such as phosphorus can cause algal blooms and eutrophication of surface water bodies.

Naturally Occurring Asbestos and Other Mineral Dust Health Issues

Although little commercial asbestos mining is still performed, asbestiform minerals may be present at low concentrations (as naturally occurring accessory or gangue minerals) in other types of ore bodies. These may include vermiculite, talc, and enriched banded iron formation ore bodies, and ore bodies associated with skarns, carbonatite intrusions, ultramafic igneous rocks, mafic alkaline intrusions, serpentinite, and metamorphosed carbonates (Virta 2002). Gangue asbestos in waste rock and tailings can pose airborne exposure risks to workers and nearby communities. Worker exposure to silica minerals, amorphous silica, and other fine mineral dusts can pose health risks. A good summary of the potential health effects of mineral dusts is provided by Ross (1999).

High pH

Many jurisdictions specify a maximum discharge limit of 8.5 to 9.0 for pH to protect aquatic ecosystems and drinking water resources. Aluminum refining generates bauxite residues (red mud), which typically have pH values of greater than 12. Alkaline leaching operations for the recovery of uranium, gold, and other commodities may also generate mineral waste with a high pH.

Spontaneous Combustion

Spontaneous combustion may pose a risk at operations that disturb or expose carbonaceous and/or highly reactive sulfide-bearing waste. Oxidation of the material may release heat faster than can be dissipated, until temperatures become high enough to trigger combustion of the waste. This is most likely to occur at coal mines but may also be a risk at operations that disturb carbonaceous and/or reactive pyritic shale.

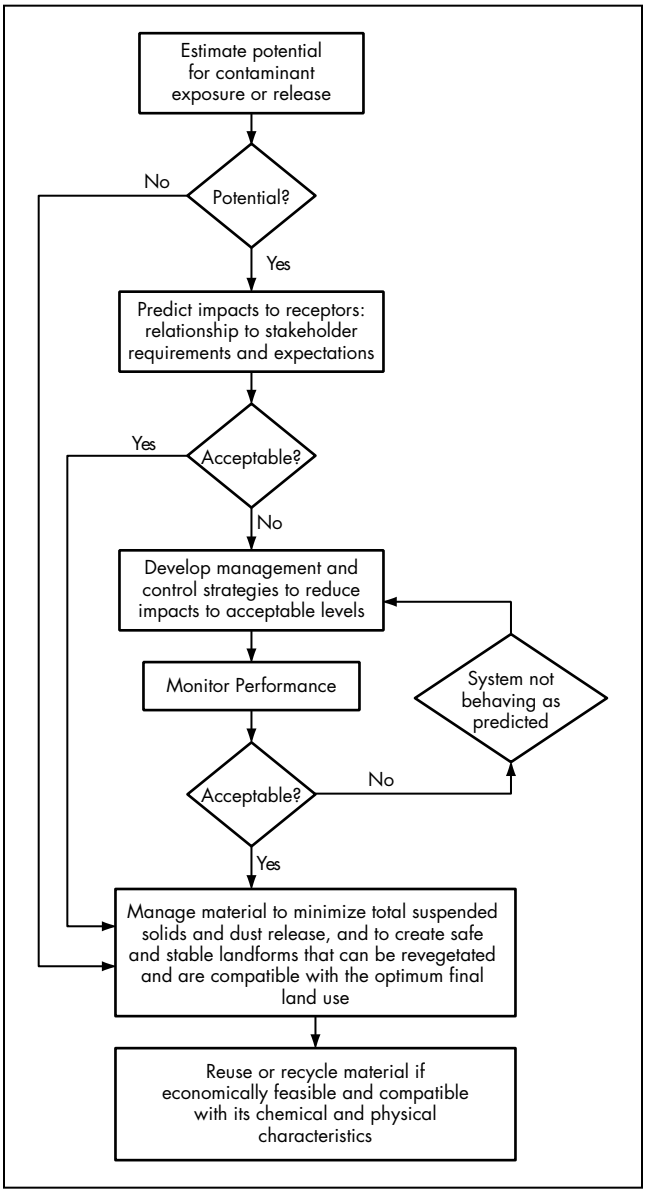
Compared to inert waste, the presence of chemically reactive mineral waste can significantly increase the complexity and cost of waste management. Successful management of chemically reactive mineral waste requires a thorough understanding of pertinent regulations, well-designed characterization programs, careful site selection, good facility design, and rigorous ongoing management and monitoring (Figure 16.6-1).

Waste Characterization and Impact Prediction

When developing a waste characterization program, operations must identify and understand the physical and chemical characteristics and hazards of all mineral wastes that will be disturbed, exposed, produced, or imported over the life of the operation. The characterization program must be rigorous enough to provide reliable predictions of the long-term physical and chemical behavior of the waste. Ultimately, the program will be used to select appropriate management strategies that comply with pertinent regulations for each waste type; ensure that all repositories are physically and chemically safe and stable; and allow for successful rehabilitation and closure.

Physical hazards include materials that are highly erodable and may be prone to shallow or deep-seated geotechnical failure under local conditions. Very blocky or poorly graded coarse-grained materials may also inhibit vegetation establishment. The potential existence of each of the chemical hazards described in the previous section must be assessed. The characterization work should be implemented in a phased approach, beginning with a broad-based sampling program combined with simple laboratory tests, followed by more complex testing if required. Initial testing should be designed to determine if a potential problem exists, while later testing may be focused on defining geologic distribution, release rates, and ecological or health effects.

For most geochemical hazards, laboratory analysis typically progresses from



Courtesy of Rio Tinto.

Figure 16.6-1 Overarching management strategy for chemically reactive mineral waste

- Total element analyses, paste pH and conductivity, and short-term static leach tests; through
- Longer-term kinetic or sequential leach tests, mineralogical analysis, and ecotoxicity tests; to
- Field scale tests and trials.

For example, a typical phased testing program for ARD may begin with simple total sulfur analyses performed on a suite of well-distributed samples that represent all major lithologies, alteration types, ore grades, and waste types. If all samples contain very low sulfur values, then the risks posed by ARD are likely negligible, and more complex testing is unlikely to be required. At most sites, ARD risks are likely to be negligible if all total sulfur values are less than 0.1%, but under some geologic and environmental conditions, material with total sulfur values as low as 0.05% may require special management (e.g., if the waste contains negligible neutralization potential and is located in a sensitive environment with little buffering capacity).

If materials that pose an ARD risk are identified by the preliminary testing, increased sampling density coupled with more detailed acid–base accounting, net acid generation testing, and mineralogical analysis should be implemented to further define the geochemical behavior of the material under surface weathering conditions. Acid–base accounting compares the acid-generating potential of the rock sample (as measured by sulfide sulfur) against its acid-neutralization potential (most commonly related to the calcium and magnesium carbonate content of the sample). Rock that contains excess neutralization potential will theoretically maintain a neutral pH as it weathers, even if it contains abundant sulfide minerals. Because of the uncertainties created by differential reaction kinetics, leaching rates, and mineral distribution in the rock, a commonly used screening criterion assumes that rocks with excess neutralization potential may still be acid generating unless the neutralization potential is at least twice the acid potential (Price et al. 1997).

Long-term kinetic leach tests should be carried out on selected samples if contact water quality and contaminant release rates need to be predicted. Kinetic leach testing should be performed in a manner consistent with the conditions to which the material will be exposed (e.g., will it be deposited above the water table or under water?). The assessment must also consider the effects of processing and deposition on the materials (e.g., will sulfides become concentrated during processing?). More detail on ARD characterization and prediction is presented in Chapter 16.5.

Preliminary physical analysis commonly consists of standard laboratory classification tests such as

- Grain size distribution and unified soils classification,
- Plasticity (Atterberg limits),
- Estimated placement density, and
- Moisture content.

More sophisticated tests may be required to assess weathering, strength, permeability, and erosion potential. Similarly, revegetation tests for mineral wastes may progress from

- Nutrient analyses and grain size distribution, to
- Greenhouse trials, to
- Field revegetation test plots and plant tissue sampling (to determine metals uptake).

Materials characterization data should be integrated with mine plans, waste inventories, and environmental baseline data such as climatic setting, hydrology, hydrogeology, and receiving environment sensitivity to develop an integrated conceptual understanding of how contaminants may be released, migrate, and impact receiving environments, biota, and people. Transport and migration pathways to be considered include direct runoff, wind erosion and dispersion, infiltration and water discharge from the facility's toe, and infiltration and migration to groundwater. Receiving environments and resources that may be impacted include surrounding terrestrial ecosystems; nearby communities; downstream aquatic ecosystems; groundwater-dependent ecosystems; and surface and groundwater resources used for drinking, stock, and/or irrigation water.

It is usually useful and often important to develop a conceptual model that provides a unified overview of how the

waste behaves in the environment and how it may impact key receptors. Although a model may be as simple as a schematic diagram, flow chart, or summary text, in some cases, it will need to be supplemented with numerical computer models to predict geochemical, hydrogeologic, and geotechnical behavior. Conceptual and numeric models should be validated and periodically updated in response to changes in operations and new monitoring and characterization data.

Site Selection and Disposal Area Footprint

Waste disposal facilities should be located in areas that minimize environmental impacts and long-term environmental liabilities. The selection process should include a review of site regulatory requirements, baseline conditions, and environmental considerations (see Chapter 16.1), environmental consequences, and the direct surface impacts caused by disposal. In general, the following factors should be considered when selecting locations for waste disposal facilities (modified from Rio Tinto 2008a):

- Only place waste within legally permitted areas.
- When practicable, preferentially place waste within inactive open pits, underground workings, or existing disturbed areas. This not only minimizes direct land disturbance but may allow rehabilitation of open pits that otherwise could not be returned to productive postmining use and can provide significant benefits for the management of some chemically reactive mineral waste.
- Avoid permanent disruption of drainage systems.
- Tie waste repositories into the surrounding topography to maintain natural, free-draining landforms and to reduce visual impacts.
- Avoid placement on land with high biodiversity or ecosystem services values.
- Avoid placement in areas with significant archaeological or social value.
- Avoid placement in close proximity to local communities.
- Preferentially place chemically reactive wastes in drainage basins that already contain reactive waste (thereby avoiding placement in undisturbed drainages).
- Limit the footprint of chemically reactive mineral waste to the maximum extent practicable. This will limit the volume of precipitation that may ultimately contact the waste, mobilizing contaminants and degrading water quality. It will also limit the surface area that may ultimately need to be covered or capped, often at significant cost.
- Avoid placement in areas with poor foundation conditions due to topography, underlying geology, or hydrology.
- Avoid placement of chemically reactive mineral waste over significant aquifers or groundwater recharge zones. Preferentially place waste over aquitards and aquicludes and in areas where hydrogeologic containment can be demonstrated.
- When the choice is available, such as in some mountainous terrains, preferentially place chemically reactive waste in areas with significantly dryer climates.
- Balance economic considerations such as haul profiles, potential resource sterilization, and pumping costs with environmental, social, and closure considerations.
- Avoid placement in or near perennial surface water bodies or in large ephemeral drainage lines when practicable, unless this represents the preferred environmental alternative.

Uncontrolled river or shallow marine disposal, which could allow rapid transport of the mineral waste away from the point of initial placement, should almost always be avoided. This does not necessarily apply to the return of mining-related inert dredge spoils into the river or shallow marine environments from which they were derived. River flow, wave action, and long-shore currents can transport wastes far from the point of initial discharge, making it difficult to control where the wastes are ultimately deposited. Aquatic environments can also be severely damaged by the smothering of benthic organisms, degradation of water quality, metals uptake by organisms contacting the waste, and increases in turbidity, which inhibits photosynthesis. However, controlled deposition into lakes and the deep marine environment may represent a preferred environmental alternative under some circumstances. For example, lake disposal may be preferred if

- ARD or geotechnical stability concerns need to be managed;
- It is restricted to well-defined, permanent deposition areas;
- Disturbance of sensitive terrestrial habitats needs to be avoided; and
- Placement can be controlled so that significant fisheries, high-biodiversity freshwater habitats, and down-gradient water resources are protected.

Deep marine mineral waste disposal should generally be considered only if the waste can be discharged and deposited below the maximum euphotic, upwelling, and mixed-layer depths in the surrounding ocean. The actual maximum depths of these three zones must be determined though detailed field studies. Coral reefs, kelp forests, and almost all other high-biodiversity marine environments are restricted to the euphotic zone (where light penetration allows photosynthesis), which typically extends to maximum depths of anywhere from 30 to 200 m in the ocean. Within this marine zone, productivity, fishery resources, ecosystem services, and biodiversity are highest, and waste can be routinely transported by wave action and currents. In most cases, deep marine disposal involves pretreatment, discharge from a pipe more than 100 m deep, and final tailings deposition in water greater than 1,000 m deep.

Waste Management Strategy and Facility Environmental Design

The design and engineering of waste disposal facilities should take into account local legal requirements; the physical, chemical, and mineralogical properties of the waste; and the environmental setting of the disposal area, including considerations of climate, water balance, underlying geology, hydrogeology and foundation conditions, potential seismic events, and nearby receiving environments and communities. Priority should be given to the proactive management of chemical hazards so that exposure risks are controlled and long-term liabilities such as perpetual water treatment are minimized. The selection of appropriate management strategies typically begins by comparing the impacts as predicted by conceptual or numeric models to environmental compliance and performance objectives. If needed, a strategy is selected to reduce the potential impacts and ensure that all compliance and performance objectives will be met during start-up, operation, and closure.

The selected strategy does not have to completely prevent any solute or contaminant release but must ensure that release rates meet regulatory requirements and are low enough

to be assimilated by the receiving environment without causing harm to people, ecosystems, organisms, or resources. Numerous geochemical prevention and control strategies are available for consideration (Table 16.6-1). Based on the chemical hazard, waste characteristics, and environmental setting, many of the control strategies listed in Table 16.6-1 will be impractical or impossible to implement at a specific site. Final selection among the remaining viable control strategies should be based on a rigorous comparison of the potential options (or combination of options) and giving preference to strategies that

- Will consistently meet or exceed local regulatory or permit requirements;
- Pose the lowest risk of short- or long-term failure;
- Can be realistically implemented under actual field conditions (the simpler the better);
- Are compatible with life-of-mine and short-term mine plans including short-, intermediate, and long-term material balances;
- Are proactive preventive measures rather than reactive remedial measures;
- Minimize postclosure liabilities and most closely approach a "walk away" solution at closure; and
- Meet the surface discharge limits, groundwater quality requirements, air quality standards, and/or receiving environment objectives in the most cost-effective manner.

Not all strategies listed in Table 16.6-1 are applicable to each chemical hazard. Strategies that depend on subaqueous disposal to limit oxygen flux into the waste are extremely effective when attempting to control ARD, sulfate-dominated salinity, and metals release related to sulfide oxidation. The maximum concentration of dissolved oxygen in surface water is approximately 30 times less than in the atmosphere. More importantly, the diffusive transfer of oxygen in water is on the order of 10,000 times slower than in air, whether the water is fully oxygenated or not (MEND 2001). Subaqueous disposal in a nonerosive landform, such as a flooded inactive open pit or underground workings, usually represents the most effective and secure disposal option for sulfide-bearing mineral waste. However, subaqueous disposal is not appropriate for waste-containing metals and salts that are already in a soluble form, because the constituents will go into solution where they can be easily transported out of the waste. Subaqueous disposal may also not be appropriate for oxidized wastes, which may contain metals that are stable under oxidizing conditions but which would be mobilized under reducing conditions. Almost any kind of nonerosive inert cover will control dust release for hazards associated with naturally occurring asbestos and other contaminants that may be entrained in the dust. However, to minimize contaminant seepage from a waste repository, a cover must be designed to limit infiltration of precipitation or surface water into the waste.

Figure 16.6-2 provides an example of an integrated mineral waste management strategy at the Rio Tinto Barneys Canyon gold mine in the western United States. About 5% of the 150 Mt of waste rock produced at the mine was unoxidized, enriched in some metals, contained abundant pyrite, and was prone to ARD generation. Proactive control of this potential ARD source required implementation of several of the strategies from Table 16.6-1.

- **Selective handling:** Based on field inspection of color and periodic confirmatory sampling, the unoxidized sulfide-bearing rock was segregated from the benign oxide rock so that it could be specially handled.
- **In-pit disposal and selective waste placement:** The sulfide-bearing rock was placed in two small repositories: one an inactive open pit and another an area already used for oxide waste rock disposal. The repositories both had a small footprint (limiting incident precipitation), were located on previously disturbed land, were not in contact with significant drainage lines or springs, and were located away from property boundaries and groundwater resources.
- **Waste removal and consolidation:** Some sulfide-bearing, low-grade ore stockpiles were also moved into the repositories after it was decided that the material would not be processed.
- **Recontouring to promote runoff:** The repositories were constructed to tie into surrounding native hillsides and to have a final slope of at least 10°. This configuration minimizes run-on and promotes runoff during heavy precipitation events.
- **Infiltration limiting covers:** The mine is located in a semiarid climate, so a store-and-release infiltration limiting cover was constructed on the repositories. Evapotranspiration was maximized by construction of a 4-m-thick cover with good moisture retention properties and planting a mixture of deep-rooting native perennial grasses, forbs, and shrubs to maximize plant water usage. Both repository surfaces were also sloped to the south to maximize sun exposure, growing season, and evapotranspiration.

Store-and-release cover systems, which seek to limit water contact with the underlying mineral waste, are seldom viable in regions with high rainfall, particularly with distinct wet and dry seasons. When managing sulfide-bearing wastes in humid climates, it is commonly better to try and minimize oxidation of the waste by ensuring it remains saturated in perpetuity or is capped with a fine-grained, compacted cover that remains near saturation. Figure 16.6-3, the Rio Tinto Ridgeway gold mine in the eastern United States, is an example of a typical flooded, chemically reactive mineral waste repository designed to limit oxygen ingress into the waste.

Waste storage facilities should be designed and constructed so that postclosure environmental impacts and financial liabilities are minimized. For example, landforms can be constructed to minimize costly recontouring and earthmoving requirements at closure. In many jurisdictions, maximum allowable final slopes for landforms constructed of mineral waste can be no steeper than 2:1 or 3:1 (horizontal/vertical). Rehabilitation costs can be significantly decreased by benching final outer waste rock dump faces in short stair-stepped lifts to match the required final slope angle. Constructing final lifts out of benign waste material—whose physical and chemical properties can support vegetation, will not pose direct exposure risks, will not degrade water quality, and is not prone to erosion—can prevent the need to import costly cover material to meet these objectives at closure. Constructing waste disposal facilities in a manner that allows progressive rehabilitation can also allow operations to minimize the footprint of unreclaimed land at closure. Progressive rehabilitation and

Table 16.6-1 Common management strategies for chemically reactive mineral waste

Strategy	Features
	Proactive Preventive Measures
Selective mining	Early identification of rock masses that pose the greatest geochemical risk allows the mine plan to be designed to avoid exposing or disturbing these materials. The additional handling, closure, and water treatment costs associated with acid-generating or other chemically reactive materials should be included in any economic analysis when determining ore reserves and cutoff grades, open-pit geometry, and the layout of underground workings. Early characterization of relatively benign rock masses is also needed so that materials that could be used for clean fill, construction activities, encapsulation, mixing, or covering are identified and can be incorporated into the mine plan as appropriate.
Improved recoveries	Improving recovery rates during the beneficiation process means that less sulfide ore minerals (such as chalcopyrite and bornite) remain in the resultant tailings or spent ore, where they produce no economic benefit, can contribute to acid potential, and may increase metals concentrations in the waste. Similarly, improving leaching and recovery rates of elements with economic value can lower their concentration in leached tailings and spent heap leach ore where they may be potential contaminants.
Lowering cutoff grade	Anytime cutoff grade can be lowered, the volume of waste rock produced will be reduced. Because the highest-grade waste material is converted to ore when cutoff grades are lowered, this process commonly removes rock with the highest potential contaminant concentrations from waste rock production.
Reagent selection and management	Potential environmental impacts caused by reagents added during ore processing can be minimized by selecting less toxic but equally effective reagents or ensuring reagents such as cyanide are added at, but not above, rates that are optimum for processing. Ensuring that blasting techniques are optimized so that blasting agents are efficiently used can lessen the risks posed by residual nitrogen compounds in waste rock and tailings.
Chemical treatment of tailings	Tailings can be treated before they are discharged to waste repositories. For example, acid-leached tailings can be neutralized to raise the pH and lower metals solubility; rinsing and cyanide destruction can be used to reduce the hazard posed by cyanide leached materials; and red mud generated from aluminum refining can be neutralized with seawater addition.
Selective waste placement	Waste repositories should be designed so that transport pathways that may allow contaminants to migrate out of the waste are minimized. Issues that should be considered include avoiding placement of reactive wastes in areas where they may contact surface water flows or in areas where groundwater discharges to the surface, over highly permeable near-surface aquifers, over or immediately up-gradient of sensitive environments (such as lands with high biodiversity value) or facility boundaries, in unstable or highly erosive landforms, and in watersheds or in areas that have not already been impacted by previous chemically reactive mine waste disposal; and minimizing the ultimate footprint of the chemically reactive waste disposal areas.
Up-gradient water diversion	In locations where surface or groundwater comes into contact with chemically reactive waste materials or wall rock, it is beneficial to capture the clean up-gradient water before it contacts the reactive material. This will reduce the likelihood of contaminants being mobilized and will protect water quality from being degraded. A surface water control plan should be developed as part of the mine plan for all disturbed areas.
Selective handling/encapsulation	Chemically reactive mineral waste may be selectively handled and surrounded with benign waste to limit the contact of runoff water, infiltration, and groundwater; reduce direct plant uptake and animal exposure; reduce the risk posed by wind erosion; provide a growth media over phytotoxic waste; provide alkalinity to underlying acidic waste; or limit the access of oxygen to sulfidic waste. Benign materials can also be used for foundations in areas where water may perch at the base of the waste or in areas where groundwater may discharge into the base of the waste pile via seeps and springs.

(continues)

Table 16.6-1 Common management strategies for chemically reactive mineral waste (continued)

Strategy	Features
In-pit or underground disposal	This method is viable where mined-out pits or underground workings of sufficient size are available. With effective mine planning, the early closure of one of a series of pits or workings may allow for the effective disposal of chemically reactive wastes. Depending on the wall-rock permeability and the location of the water table, waste placed in open pits may need to be encapsulated. For pits and underground workings located below the postmining water table, this disposal method may be used in conjunction with subaqueous disposal to prevent sulfide oxidation but may not be appropriate for wastes containing soluble contaminants. Backfilling open pits above the postmining water table can also be used to prevent evapo-concentration and the creation of hypersaline conditions in arid or semiarid climates and the formation of pit lakes that may provide a direct exposure pathway to terrestrial animals.
Subaqueous disposal	The placement of acid- generating wastes below the water table or in a permanent surface water body and the permanent flooding of underground workings or pits will greatly limit the flux of oxygen into sulfide-bearing materials. This can lower the rate of acid generation and metals release by several orders of magnitude. This technique is most effective when flooding or covering is accomplished rapidly—before the material has time to oxidize. Subaqueous disposal is generally not a viable control option for wastes containing contaminants that are already in a soluble form and whose rate of release is not dependent on the supply of oxygen.
Blending/mixing	The blending and mixing of acid-generating wastes with net neutralizing materials may be used to ensure that bulk neutral pH conditions are maintained within the waste mass as a whole, either in perpetuity or for a sufficiently long period to allow another long-term management technique to be implemented. This control technique may be difficult to implement because of preferential flow paths established within the waste, armoring of neutralizing rock surfaces, and acidified pockets that are likely to form within the waste column. Proper implementation of the technique requires good ore and waste characterization and control, coupled with sequential mining and good dispatch control to ensure that acid-generating and acid-neutralizing wastes are intimately mixed at the appropriate ratios.
Enhanced tailings consolidation	Techniques that enhance tailings consolidation such as the construction of underdrains, ensuring subaerial placement, controlling the rate of rise, active removal of pore water with extraction bores, or initial placement as thickened, dry stacked, or paste tailings will create a more stable material with lower permeability. If solutes within the pore water pose an environmental risk, enhanced consolidation will also reduce the total contaminant mass and lower the potential flux of contaminants that could discharge from the tailings.
Codisposal	Materials with different physical characteristics may be mixed for codisposal to enhance stability and resistance to erosion or to reduce permeability. This technique may be applied to the codisposal of tailings with waste rock. The mixing of tailings and waste rock may create a well-graded material that typically has lower hydraulic conductivity and higher water storage capacity, and is relatively resistant to erosion. The geochemical compatibility and the physical behavior of the mixtures require careful evaluation. Techniques to deposit whole tailings (such as dry stacking, paste, or thickened tailings) that are not strongly segregated into coarse and fine fractions, or segregated into sulfide-depleted and sulfide-enriched fractions, may also provide lower hydraulic conductivity, maintain saturation and, if the parent material is net neutralizing, may prevent the creation of net acid-generating zones within the tailings.
Sulfide flotation	Gangue sulfides such as pyrite may be separated from the bulk of the tailings in order to reduce the tailings' acid potential and ensure they are net neutralizing. The relatively small volume of concentrated waste sulfide can then be specially handled to minimize its environmental risks. Metals that are associated with the sulfides may also be preferentially removed from the tailings by this technique.

(continues)

Table 16.6-1 Common management strategies for chemically reactive mineral waste (continued)

Strategy	Features
Compaction during placement	Chemically reactive waste rock or overlying benign material may be compacted during placement to lower the hydraulic conductivity of the material and limit the potential flux of water through the compacted layer. Compaction may also aid in the establishment of capillary breaks and/or perched water tables associated with the compacted layers, which will inhibit the flux of oxygen. Although engineered compacted layers should usually be specially constructed, some compaction of the waste can also be accomplished by dumping in shorter lifts to enhance the compaction that occurs naturally as mine vehicles travel over the surface.
Liners	Liners can be constructed under reactive mineral wastes to allow capture of contact water that would normally discharge to the surface or to groundwater. Liners can be simple and relatively inexpensive or complex and expensive, depending on the required performance criteria (generally the allowable design leakage rate). Complex liners are generally required for heap leach pads containing very hazardous leach solutions. For mineral wastes, complex liners are usually only practicable for use with small volumes of very reactive mineral waste. If hydrogeologic containment can be demonstrated because of underlying aquicludes, aquitards, and/or strong upward vertical gradients, then liner systems may not be needed, or less complex systems may be adequate.
Temporary synthetic covers	Synthetic covers such as high-density polyethylene (HDPE) sheeting may be used to temporarily cover reactive mineral wastes or ore stockpiles. This isolates the material from precipitation and greatly limits the volume of contact water that may discharge from the material. This is generally not a viable long-term strategy to limit water contact because the synthetic cover will degrade with time, but it can be used on a short-term basis until the ore is processed or the final closure strategy for the mineral waste is implemented. However, when temporary covers are removed, sulfide oxidation products and other soluble constituents may dissolve and produce a large spike in contaminant release.
Micro-encapsulation (passivation)	This process involves the coating of mine wastes or pit walls with products that encapsulate sulfide surfaces and inhibit oxidation reactions. The mechanism involves mixing the reagents with waste or spraying pit walls with treatment solutions. Treatments that are being investigated include phosphate, amorphous silica, polymer, and potassium permanganate application. Encouraging bioshrouding of sulfide minerals by promoting the growth of microbe biofilms with nutrient addition is also under investigation. However, these techniques are likely to be costly on a per-ton basis; the potential exists for environmental problems associated with the use of some reagents; retroactive treatment of large waste storage facilities is unlikely to be successful; and the long-term viability of the techniques has not been proven at an operational scale.
Bactericide application	Abiotic sulfide oxidation rates are typically several times slower than biologically controlled oxidation rates. Bactericides such as thiocyanate that inhibit the growth of sulfide-oxidizing microorganisms can substantially slow ARD generation rates. However, these techniques are likely to be costly on a per-ton basis; the potential exists for environmental problems associated with the use of some reagents; retroactive treatment of large waste storage facilities is unlikely to be successful; and the long-term viability of the techniques has not been proven at an operational scale.
Incorporation of organic matter and/or nutrients	Incorporation of organic matter or nutrients into wastes can promote the establishment of reducing conditions within the waste material (inhibiting sulfide oxidation and promoting sulfide formation) and may reduce metals solubility by providing sorption sites. Organic matter and nutrients can also promote the establishment of sulfate-reducing bacteria, potentially removing dissolved sulfate and metals from pore waters. Such systems may not be appropriate for use with some oxidized wastes because reductive dissolution of iron or manganese oxides may release adsorbed trace metals such as arsenic.
Accelerated sulfide oxidation	In specially designed heap leach pads, it may be possible to accelerate sulfide oxidation processes, depleting all the reactive gangue sulfide minerals before closure. This would help reduce the risk of long-term contaminant release from the mineral waste. This technique would likely only be viable for copper heap leach pads and other operations that are already designed to maximize the oxidation of sulfide ore minerals. However, costs are likely to be high and the viability of this option has not been proven at an operational scale.

(continues)

Table 16.6-1 Common management strategies for chemically reactive mineral waste (continued)

Strategy	Features
	Reactive Remedial Measures
Recontouring to promote runoff and inhibit infiltration	Waste surfaces can be recontoured to prevent precipitation from pooling and infiltrating at low points on the surface and/or to allow precipitation to be more efficiently shed from the surface. The surface can be contoured to naturally shed precipitation via runoff, or low points can be established from which runoff is collected and piped off the surface.
Direct revegetation	Some waste surfaces may be directly revegetated after minor physical or chemical modification, such as ripping to reduce compaction, addition of alkaline materials to increase the pH to near neutral, or the addition of organic matter. The establishment of vegetation can reduce erosion, significantly increase evapotranspiration, and reduce the amount of water that infiltrates the underlying waste material. Direct revegetation may allow many of the benefits of a store-and-release cover to be realized without the need to import large volumes of cover material.
General purpose covers	Covers can be constructed over reactive wastes using inert mineral waste, imported fill, or soil to reduce infiltration, preserve runoff water quality, allow vegetation to become established on phytotoxic wastes, limit plant uptake of bioavailable metals in the underlying waste, limit wind transport of reactive wastes, limit direct exposure pathways for animals and humans to metals and radionuclides in the underlying wastes, limit oxygen ingress into the underlying mineral waste, and/or provide alkalinity to the underlying waste.
Alkaline covers	Under certain climatic and geochemical conditions it may be beneficial to construct covers with relatively reactive carbonate material. The carbonate can provide alkalinity to the underlying waste via infiltrating precipitation to control ARD from weakly acid-generating wastes or to increase the thickness of the underlying neutralized waste layer, allowing plants to root more deeply.
Infiltration or oxygen-limiting covers	Two general types of covers to limit oxygen and/or water ingress may be constructed on top of existing mineral waste: (1) low-permeability covers to limit the flux of oxygen and water into the underlying waste or (2) store-and-release covers that limit the amount of water that may infiltrate the underlying waste. In general, low-permeability covers are best suited to wet climates, and store-and-release covers are best suited to dry climates (where evaporation is at least twice precipitation). Low-permeability covers that inhibit the entry of oxygen, using natural materials such as compacted clay, generally require that nearly saturated conditions be maintained within the cover at all times. Low-permeability covers are often subjected to natural processes that may cause degradation in performance over time such as biointrusion by plants and animals and increasing permeability because of natural freeze/thaw and wetting/drying cycles. It is important for compacted layers to be protected from these processes by a thick overlying cover layer. The performance of most store-and-release cover systems is strongly dependent on the thickness and texture of the cover material (i.e., high-moisture storage capacity and low susceptibility to cracking), as well as the density, rooting characteristics, and species composition of vegetation that is established on the cover to maximize evapotranspiration rates.
Infiltration or oxygen-limiting covers (continues)	Covers can be simple, thin, and relatively cheap or complex, thick, and relatively expensive depending on the required performance criteria (generally the design allowable flux of water or oxygen through the cover and into the underlying reactive mineral waste). Covers designed to limit infiltration may also include capillary breaks to inhibit vertical water movement.
Rinsing	Rinsing can be used to flush soluble constituents from mineral wastes such as spent heap leach ore. Generally, several pore volumes must be exchanged to reduce concentrations to acceptable levels. If all the potential contaminants are not present in a readily soluble form, then rinsing is unlikely to be viable. For example, rinsing would not be a viable alternative for sulfide-bearing wastes until almost all of the sulfides have been oxidized.

(continues)

Table 16.6-1 Common management strategies for chemically reactive mineral waste (continued)

Strategy	Features
Flooding	The permanent flooding of existing wastes, open pits, and workings can limit the access of oxygen to the waste material and greatly restrict the rate of sulfide oxidation. Ideally, flooding should occur as soon as possible after the waste material is exposed. Waste materials that have undergone long periods of subaerial storage prior to flooding may contain abundant soluble sulfide oxidation products that may be released when the material is initially contacted by water. It may be necessary to add neutralizing agents to weathered materials that have acidified before they are flooded or to add alkalinity to the flooding waters. Flooding is generally not a viable control strategy unless contamination release rates are dependent on the flux of oxygen into the waste. Large fluctuations in water level so that wastes are periodically exposed and resubmerged must be avoided.
Down-gradient collection and treatment of contact water	This is required for sites where contaminated water is already being released or where it cannot be avoided and involves the capture of impacted surface and groundwater for treatment and disposal. Potential collection and recovery systems may include gravity flow into catchment ponds, drains, and trenches, as well as pumping from pits, underground workings, and extraction wells. Management options include active water treatment such as the addition of alkaline materials or the use of reverse osmosis, internal mixing, and dilution with controlled release to receiving water bodies, evaporative disposal, reuse in the process, reactive barriers, settling ponds, and wetlands treatment for polishing.
In-pit water treatment	The water chemistry of pit lakes may be manipulated in situ to promote physical or chemical conditions and biological processes that improve or maintain water quality. Some examples include controlling the depth of flooding to completely cover acid-generating wall-rock zones, to prevent water from contacting chemically reactive zones, or to create lakes that are unlikely to turn over; the direct addition of alkaline materials to maintain a neutral pH and inhibit sulfide oxidation below the water surface; the addition of nutrients to promote biological activity that removes metals from the water column and helps establish anaerobic conditions at depth; and the addition of sorbing agents such as iron to remove dissolved metals from solution.
Demonstrating hydrogeologic containment	For open pits that intersect the water table in arid and semiarid climates it may be possible to demonstrate hydrogeologic containment of solutes and contaminated water within the pit. Evaporative losses from the pit lake may be sufficient to maintain radial groundwater flow toward the pit in perpetuity. Containment can be enhanced by reducing the volume of runoff water that reaches the pit floor or by pumping and treating enough pit water to keep the lake level below water levels in the surrounding bedrock. Contaminated groundwater underlying waste rock dumps and tailings impoundments located within the zone of groundwater capture of the open pit may also be permanently contained. In some arid climates it may be possible to preferentially direct contaminated surface water into the pit for disposal by evaporation.
Waste removal and consolidation	In some cases it may be beneficial to move tailings and waste rock from one location to another if they are broadly dispersed or located in a particularly sensitive location. The selective removal and consolidation of highly reactive wastes into a smaller footprint may allow for better mitigation, containment, and control of the waste.

Source: Adapted from Rio Tinto 2008b.

revegetation of tailings impoundments can be particularly important to prevent dust generation from inactive tailings surfaces as they dry out. Other closure requirements and considerations that are pertinent to mineral waste disposal facilities design and construction are detailed in Chapter 16.7.

Ongoing Management and Monitoring

After a mineral waste management strategy is selected and waste storage facilities have been designed, they must be constructed and successfully managed on an ongoing, long-term basis. If benign and chemically reactive wastes are to be segregated and specially handled, the complexity of mine planning and management is increased. Waste management strategies need to be fully integrated into mine plans and truck control and dispatch systems. Strategies and costs should also be fully integrated into any economic models used by mine planners. Standard operating procedures need to be developed to ensure that different types of waste are identified, hauled to the correct location, and then placed and managed appropriately. Successful field implementation of strategies and designs will typically require cooperation between environmental personnel, senior management, geologists, short- and long-term mine planners, field supervisors, equipment operators, and surveyors. At many operations, development of a formal mineral waste management plan may be justified. These plans typically include a brief description of management objectives and strategies, waste storage facility design criteria, detailed field procedures, clear assignment of responsibility for different tasks and actions, and ongoing sampling and monitoring requirements.

Monitoring is required to ensure successful implementation of the mineral waste management plan and to ensure that the strategy is leading to the intended results. Physical monitoring programs for waste disposal facilities typically will include at a minimum

- Regular visual inspections of surface structures and facilities such as spillways, piping, dykes, ditches, and other water management systems;
- Regular visual inspections for signs of excessive surface erosion and shallow or deep-seated failure on the outer slopes of waste repositories; and
- Monitoring of water levels and pore pressure within embankments and the waste.

Programs to monitor the geochemical behavior of waste disposal facilities will typically include

- Periodic sampling of runoff water and water discharging from the facility's toe in order to monitor flow volumes, solute concentrations, and the solute mass that is being released from the waste;
- Periodic sampling of down-gradient monitoring wells and surface water bodies to ensure that seepage from the waste is not adversely impacting receiving environment water quality; and
- Periodic assessment of revegetation success such as total cover, species composition, and plant health.

If needed, more specialized monitoring data may be collected, including

- Paste pH, conductivity, and leach testing of soils forming on the surface of the waste or on covers constructed over the waste;

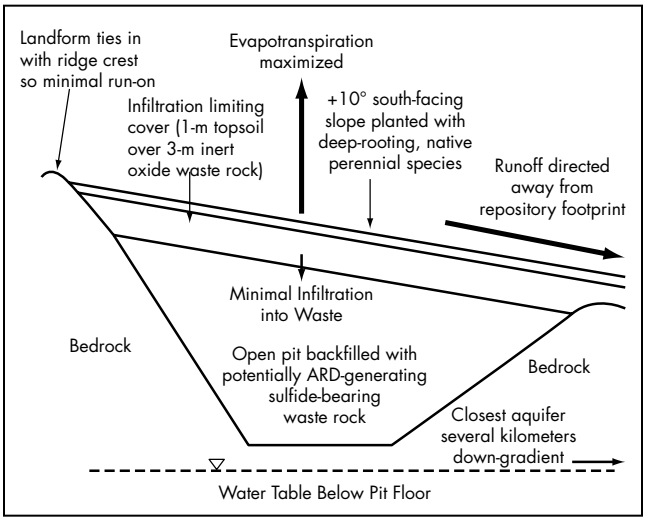


Figure 16.6-2 Example of a chemically reactive mineral waste repository in a semiarid climate

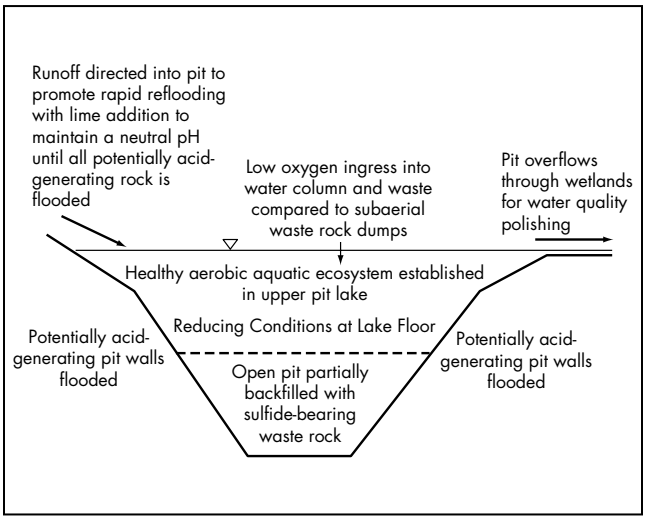


Figure 16.6-3 Example of a chemically reactive mineral waste repository in a humid climate

- Tissue sampling and metals analysis for plants growing on the waste;
- Monitoring of infiltration rates into the waste;
- Internal monitoring of the waste disposal facility for pertinent parameters such as temperature, oxygen content, moisture content, and pore water chemistry;
- Radiological measurements at the surface of the facility; and
- Monitoring of ecosystem health.

Monitoring data should be reviewed regularly, and historical trends should be examined so the longer-term chemical behavior of the mineral waste can be assessed. Time series of monitoring data should be maintained so that long-term changes in water quality, flow rate, or other key parameters can be tracked and significant changes can be identified. Significant unexpected changes should trigger a review of predictions, performance objectives, management strategy, and/or field implementation procedures. Appropriate feedback mechanisms

should be in place to ensure timely management response to out-of-compliance, unexpected, or upset conditions. It is usually beneficial to maintain internal monitoring points in addition to the monitoring required at mandated external compliance points. Internal monitoring can allow early warning of potential problems before out-of-compliance conditions develop.

Closure

Waste disposal facility closure should create safe and stable landforms and allow land and water resources to be returned to productive postmining uses. Geochemical exposure hazards should be controlled, and long-term water management liabilities should be minimized. General reclamation and rehabilitation requirements are detailed in Chapter 16.7. Typical closure activities that are specific to mineral waste disposal facilities include the following:

- Removal of any surface debris or infrastructure.
- Large-scale recontouring to create natural-looking, safe, and stable landforms; to reestablish drainage lines; to reduce angle of repose slopes; and to comply with any other mandated closure criteria.
- Construction of long-term erosion and runoff control structures (Chapter 16.4).
- Placement of any cover material needed to prevent wind erosion, allow revegetation, preserve runoff water quality, limit infiltration, or control oxygen ingress. Cover material may be composed of benign mineral waste or imported borrow material with the appropriate physical and chemical characteristics. If erosion is a concern, a well-graded material should generally be used to support vegetation while also being resistant to erosion. If mineral waste is used as a growth medium, it should generally have the following characteristics:
  - Circumneutral pH (ideally between about 5.5 and 8.0) (Buckman and Brady 1969; Tucker et al. 1987),
  - Low salinity (typically an electrical conductivity of less than 2 dS/m [deciSiemens/meter] for moderately salt-sensitive species) (Maas 1990; ANZECC/ARMCANZ 2000),
  - Excess neutralization potential and low sulfide sulfur content (ideally a neutralization potential ratio of greater than 2:1 [Price et al. 1997] and total sulfur less than about 0.5%);
  - Metals concentrations that will not pose an exposure risk to plants or animals, and
  - Good moisture retention characteristics (adequate sand, silt, and clay in coarse wastes).
- Ripping or scarifying compacted surfaces to remove physical barriers to plant growth.
- Placement of topsoil and/or any soil amendments needed to promote plant growth. Topsoil that is free of weed seed will almost always enhance desired plant growth, and good topsoil management is an important part of most reclamation programs. Soil amendments may be needed when direct planting into mineral waste in order to modify the pH, increase water-holding capacity, improve soil structure, increase organic matter content, and add nutrients. Care should be taken not to overapply biosolids or other nutrients which may promote long-term weed dominance.
- Seed application and/or the planting of seedlings. The species selected for planting are likely to be based on the planned final land use, regulatory requirements, and stakeholder expectations.

Postclosure water management is commonly required for disposal facilities that contain chemically reactive mineral waste. Contaminated water that discharges from the waste material will typically need to be collected and treated before it can be discharged to the environment. Depending on the rates at which sulfide minerals oxidize and at which soluble constituents migrate out of the waste, water management may be required for decades or even centuries after closure. Spent heap leach pads containing hazardous process fluids may need to be flushed with many pore volumes of clean water to meet water quality discharge requirements. Long-term water management costs can be high, in many cases exceeding the cost of the initial rehabilitation. Postclosure monitoring programs are also usually required to ensure that surfaces are not eroding, the mandated vegetation is permanently established, and chemical exposure risks and impacts remain within acceptable ranges. If only inert waste is present, postclosure monitoring may be completed within a few years of closure, but if chemically reactive waste is present, monitoring may be required for many decades.

NONMINERAL WASTE

Common hazardous and industrial wastes generated at mining and mineral processing operations include used oil, oily debris, antifreeze, greases, batteries, solvents, coolants, spent reagents, used paint, and contaminated soils. These wastes are typically sent to off-site recycling, treatment, or disposal facilities. Other nonhazardous wastes, such as construction debris, packing materials, general office waste, and used tires, may be recycled, sent to an off-site municipal landfill, or placed in permitted on-site landfills. Remote operations that are not connected to municipal sewage systems must also construct and manage on-site sewage treatment systems.

Each waste stream needs to be characterized so that its hazards are understood, legal requirements are defined, and appropriate recycling, treatment, or disposal options are identified. Wastes need to be safely accumulated on-site until they are shipped off-site or placed into permanent on-site storage facilities. The volume of waste generated may be reduced by improving process efficiency, substituting hazardous materials with less hazardous but equally effective products, and by identifying on-site recycling and reuse opportunities.

The degree of effort required to characterize each waste stream will generally depend on its potential risks and management requirements. Some waste streams may only require detailed assessment during initial characterization and in response to significant changes in the generation process. Other wastes may require much more frequent analytical testing if the characteristics of the waste are highly variable or the final treatment or disposal method has strict physical or chemical requirements. For example, used oil burned for energy recovery typically has restrictions on metals and total halogens content and flash point. Municipal landfills and many industrial waste landfills are only permitted to receive nonhazardous wastes, so waste characteristics must be defined in accordance with local regulatory requirements to identify appropriate disposal facilities. Depending on the waste stream, key sources of information are likely to be

- Material safety data sheets;
- Other manufacturer and/or supplier information;

- Process and engineering knowledge and materials balances;
- Local regulations, which may list wastes generated by certain processes and/or industries and mandate how they must be handled;
- Analytical testing such as
  - Primary chemical constituents;
  - Trace chemical constituents and impurities such as metals, organic compounds, asbestos, and carcinogens;
  - Physiochemical characteristics such as flash point (typically considered hazardous if <60°C), corrosiveness (typically considered hazardous if pH <2 or >12.5), and reactivity (unstable when heated, compressed, or in contact with water or other wastes);
  - Toxicity leach tests such as the toxicity characteristic leaching procedure (Method 1311, EPA 2008) or the synthetic precipitation leaching procedure (Method 1312, EPA 2008); and
  - Radiological analysis.

Temporary on-site waste accumulation and storage areas should be clearly defined, utilizing fencing and appropriate signage as required by local regulations. Dispersed short-term accumulation areas typically will be located at or near the point of waste generation. Wherever possible, wastes should be segregated at source, which allows for improved recycling or recovery of materials, may increase the number of recycling or disposal options available for the waste, and may reduce the cost of waste disposal. In particular, disposal costs for mixed wastes may also be significantly higher if hazardous wastes are mixed with nonhazardous wastes.

Waste storage and containment practices must meet local, national, or international standards as applicable and must be commensurate with the risk posed by the waste. In many jurisdictions mines are required to have a central waste accumulation site where hazardous and industrial wastes are consolidated and stored before being shipped off-site. Central accumulation areas should typically be located in a secure location, on an impermeable surface to contain spills, and with a roof to minimize rainwater contact. New or used storage containers can be used, but they should be in good condition and must be compatible with the contained waste. Chemically incompatible wastes, which could react with each other, should not be stored together. Waste containers should be appropriately labeled so the contents and their primary hazards can be identified. In some jurisdictions, hazardous wastes have strict limits on the time they can be stored on-site before being shipped to a permitted disposal facility.

All waste must be permanently disposed in accordance with applicable laws, regulations, and permit conditions. Where local or national laws are weak or do not exist, internationally recognized good practices should generally be followed. The design and ongoing management of off-site waste disposal facilities must also be consistent with the regulatory classification and the hazards posed by the waste. Waste shipped off-site should be tracked and/or manifested in order to document its ultimate disposal location. It is often prudent to periodically audit off-site treatment, storage, and disposal facilities that are handling large volumes of hazardous waste to ensure that it is being appropriately managed. There are many examples of off-site contractors who have not correctly handled wastes, creating significant financial liabilities for the original waste generators.

Permanent on-site waste disposal should only occur within designed facilities with engineering and management controls commensurate with the health, safety, and environmental risks posed by the waste. In most cases the on-site disposal facilities will need to be permitted with the appropriate regulatory agencies. To ensure adequate levels of engineering design and management, active on-site landfills or waste storage facilities will generally require

- Clear restrictions on what types of waste materials can and can't be placed in the facility;
- Clear boundaries and limited access to the facility to ensure waste delivery is controlled and to minimize exposure risks to animals and people; and
- Operating procedures describing waste delivery and placement practices, inspection frequency, and covering requirements.

For active storage facilities and landfills that accept chemically reactive wastes capable of posing direct exposure risks or degrading surface and groundwater quality, additional controls are likely to be required, including

- Leachate and stormwater management systems consistent with the chemical hazards posed by the waste, for example:
  - Composite liners with leachate collection systems for chemically reactive wastes that may degrade groundwater quality,
  - Covered and enclosed structures for long-term storage of wastes that are unstable in contact with water, and
  - Stormwater collection systems for wastes that may degrade runoff water quality;
- Closure plans detailing how the facility will be closed—in particular, final cover designs needed to limit direct exposure risks to animals and people, to preserve runoff water quality, to limit infiltration, and to control wind dispersion of the waste; and
- Monitoring programs for groundwater, surface water, air quality, landfill gas, and windblown debris to ensure that potential impacts associated with the waste are being adequately controlled.

Remote sites may find it economic to operate dedicated recycling and/or incineration facilities for waste materials such as antifreeze, used oil, solvents, and medical waste. Again, these facilities must be designed and operated in accordance with all application waste management regulations and air quality standards.

HAZARDOUS MATERIALS AND CONTAMINATION CONTROL

Hazardous material is broadly defined as any substance (solid, liquid, or gas) that could pose a significant threat to human health or the environment if improperly handled or accidentally released. Typically, hazardous materials are toxic, corrosive, flammable, explosive, strongly oxidizing, or radioactive. Hazardous materials that are commonly managed at mining operations include cyanide, strong acids, caustic soda, lime, chlorine, pesticides, herbicides, explosives, solvents, gasoline, and hazardous waste. Other substances that can be considered hazardous under some circumstances include paints, diesel fuel, coolants, and lubricants. The actual risks posed by the handling of these materials depend on their innate hazards, the volumes that are present, potential receiving environments, and transport

pathways that could connect the point of release with potential receptors.

The mishandling or unintentional release of hazardous materials can create acute or chronic health problems for workers and nearby communities, damage ecosystems, degrade land productivity and water quality, and create costly cleanup liabilities. Hazardous materials should be managed to minimize the frequency and severity of any releases. Pipes and tanks should be designed to safely transport and store hazardous materials with minimal risk of failure. Pipelines and tanks should be designed for the pressures used and the material being transported, be well maintained, and be regularly inspected. In general, the use of underground bulk storage tanks should be avoided. If the risks posed by contaminant release are high, additional safeguards such as secondary containments, leak detection systems, and double-walled pipelines may need to be considered.

Secondary containment systems such as bunds, berms, and liners for bulk storage tanks need to be built and maintained in compliance with all local and national requirements and permit conditions, or, in their absence, internationally recognized standards. Secondary containment systems may be designed for individual tanks or for entire tank farms and drum storage areas. Containment and capture systems (such as concrete pads sloped to collection sumps) may also be constructed in areas prone to recurring spills such as haul truck refueling areas.

Secondary containment facilities should be sized to hold a volume greater than that of the largest tank in the containment, along with any piping that drains back into the tank or containment and additional capacity for the design storm event (typically 110% of the largest foreseeable leak into the containment is sufficient). Stand alone double-walled tanks can provide valid secondary containment if designed to contain all leakage from the internal tank. The impermeable material used to construct the secondary containment must have a design leakage rate that is consistent with the environmental hazards posed by the material. In most cases, the secondary containment should be constructed with highly impermeable materials such as concrete, asphalt, or synthetic liners like HDPE. However, if risks are relatively low, more permeable materials such as earth berms and compacted fine-grained soils may be acceptable.

Liner design for the storage of hazardous materials should also always meet or exceed local legal and regulatory requirements. Leach pads and process water ponds that contain hazardous fluids, such as cyanide-bearing solutions or extremely acidic, basic, and/or saline solutions, will generally need to be underlain by a composite liner. Tailings impoundments that contain hazardous pore fluids may also need to be underlain by a liner system unless sufficient hydrogeologic containment can be demonstrated along with adequate seepage collection and monitoring systems. Liner systems may also sometimes be required for chemically reactive waste rock dumps in wetter climates unless sufficient hydrogeologic containment can be demonstrated along with adequate seepage collection and monitoring systems. Many types of nonmineral waste landfills will also require liners to allow capture and collection of leachate and landfill gases.

Composite liners are typically constructed with an underlying low-permeability, compacted clay layer and an overlying synthetic liner. Again, the actual design leakage rate of the liner system must be based on a risk assessment that takes into account the hazards posed by the liquid and its potential environmental impacts. If a higher leakage or failure rate is determined to be acceptable, then a composite liner may not be needed. Conversely, if the hazards posed by the fluid and the environmental setting are high, then a double or triple liner system with intervening leak detection and pressure relief layers (to minimize head on the lower liners) is likely to be needed.

Leach pads and other process circuits that contain large volumes of hazardous fluids should be designed for gravity flow into engineered process ponds and containment structures during power outages and other upset conditions. The capacity of these structures should be large enough to hold the maximum credible volume of the fluid that would be released by drain down before the upset condition could be corrected. In many cases, backup emergency power systems for pumps may be required.

If there is an uncontained spill, rapid spill response can minimize the environmental damage by limiting the volume of material released, the surface area impacted, and the exposure duration. Emergency response procedures for spill containment and cleanup should be developed and appropriate spill response equipment stored on-site. Typical spill response kits may include sorbent materials, neutralizing agents, drums, shovels, and appropriate personal protective equipment. At port facilities where spills onto open water are possible, spill response boats and booms may be maintained. Most operations should also maintain a trained spill-response team.

CONTAMINATED WATER MANAGEMENT

Mining operations can impact surrounding water resources and water-dependent ecosystems by (1) water withdrawal and dewatering impacts, and (2) the discharge of contaminated water. Surface and groundwater withdrawals to dewater the ore body or to supply operations can lower surrounding groundwater water tables, causing seeps, springs, and wells to dry up; harming groundwater-dependent vegetation and ecosystems; and reducing in-stream flow. If necessary, dewatering impacts must be predicted, monitored, and mitigated. Mitigation strategies may include

- Improving water efficiency through process and management improvements so that less water needs to be withdrawn,
- Intentional surface water discharge at key locations to maintain in-stream flow,
- Providing alternative water resources for impacted communities,
- Intentional recharge of groundwater to minimize drawdown impacts, and
- Construction of slurry walls and other subsurface flow barriers to minimize hydrogeologic connections.

Water may be discharged to the environment through centralized, engineered, and permitted discharge points; more dispersed, uncontrolled stormwater discharge locations; and/or uncontrolled subsurface flow. If the released water is contaminated, it may degrade down-gradient water quality, harming ecosystems, human health, and water resources. Water that contacts chemically reactive mineral waste is commonly contaminated, but poor-quality water can also be generated by ore-body dewatering, mineral processing, sewage discharge, recurring hazardous material releases, and outflows from pit lakes and underground workings.

As discussed in the earlier section on chemically reactive mineral waste, mine-impacted water typically poses risks

because of ARD, salinity, metals soluble at neutral pH, radionuclides, nitrogen compounds, and added reagents such as cyanide. In many jurisdictions, discharge limits are established by law, regulation, or permit for potential contaminants of concern. In such cases, water may usually only be released from designated outfall points and must consistently meet the mandated discharge limits. These limits are typically based on maximum concentration, average concentration weighted by flow and/or time, and/or mass loading. In other cases, discharge limits must be selected and justified based on site-specific data and conditions. In these circumstances it is important to understand not only the potential discharge water quality but also migration pathways and the assimilative capacity and sensitivity of the receiving environment. It is important to differentiate between impacts to receiving environment water quality and environmental harm. Discharged mine water may result in measurable changes in solute concentrations in receiving water bodies without causing environmental harm. For example, metals concentration may measurably increase, while the ambient concentration remains well below threshold values for protection of aquatic ecosystems and quality of drinking, stock, and irrigation water. It is also important to consider cumulative impacts to the receiving environment. If multiple mines exist in a single drainage basin, the cumulative impact of all mine discharges must sometimes be considered.

Discharge water quality and volumes partly depend on the innate geology and climatic setting of the ore body being mined. However, mine and process design and management can also have a profound influence on the quality and volume of water that must be managed. The water and solute balance should be predicted and monitored during mine design, operation, and closure. These data can be used to select appropriate water management strategies to meet discharge limits and protect down-gradient water quality.

Water treatment before discharge can be costly, sometimes exceeding several dollars per 1,000 L. At large mines with significant ARD flows, cumulative treatment costs can be measured in the tens to hundreds of millions of dollars. Implementation of internal proactive management strategies that reduce the volume of water that must be treated and/or reduce the solute load in the water can be cost-effective, as well as ultimately more protective of the environment. The strategies discussed previously to minimize water contact with, and solute loading from, mineral wastes are a key component of overall site water quality management. Other broad internal water management strategies include the following.

- **Water diversion:** Capture and diversion of clean surface and groundwater flows up-gradient of the operation can limit the volume of water that may be contaminated by contact with the operational footprint.
- **Water segregation:** Within the footprint of the operation, careful segregation and separate handling of waters with different qualities can limit the volume of water that ultimately requires treatment. Clean stormwater can often be discharged without treatment if its quality is preserved. Unless needed as makeup water, precipitation and stormwater should generally be isolated from process water circuits, reactive mineral waste, hazardous materials, and contaminated areas.
- **Recycling and reuse:** Internal recycling and reuse of poor-quality water can reduce the volume of makeup water that must be withdrawn from the environment and allow beneficial use of poor-quality water, thereby reducing the volume of water that may need to be treated before discharge.
- **Improved water use efficiency:** Improvements in water use efficiency can also reduce the volume of water that must be imported into the operation and that ultimately must be discharged.
- **Reagent management:** Process water quality can be improved by the efficient use of reagents and/or replacement of hazardous reagents with less hazardous but equally effective substitutes.
- **Safe on-site storage and controlled release:** If sufficient safe water storage capacity exists on-site, mine water can sometimes be discharged without treatment at rates and times that can be assimilated by the receiving environment. For example, saline waters can be stored on-site during the dry season and discharged during peak storm-related runoff events, when natural salinity is low and in-stream flow is highest.
- **On-site evaporation:** Evaporative losses within the footprint of the operation will reduce the volume of water that must be discharged. Natural evaporation from storage ponds and tailings dams can be a significant source of water loss in some climates. Evaporation can be enhanced with simple sprinkler systems, and in some extreme situations energy intensive crystallizers may be used. Although evaporation can be effective for reducing the volume of water that must be managed, it does not remove soluble salts from the system and will ultimately produce saline solutions or soluble solids that must be managed.

In humid environments it is likely these proactive water management strategies will reduce but not eliminate the need to discharge water. However, mines in arid and semiarid climates may maintain a negative water balance because of evaporative losses, entrainment losses to shipped product, and long-term storage within tailings. The water balance implications of any significant operational changes such as new ore-body development, new processes, or footprint expansion must be carefully assessed. It is also important to distinguish between operational water balances when mining and processing activities may consume water, and postclosure conditions when these water sinks are no longer available.

In many cases, water treatment before discharge will be required. At a minimum, most mining operations will need to control sediment discharge via on-site erosion control and the use of settling ponds. Sewage treatment facilities will usually be required unless operations are connected to a municipal treatment system. Oil water separators are also likely to be needed to capture fuel, oil, and grease from refueling stations and maintenance shops. Mine waters that have been impacted by ARD, salinity, metals soluble at neutral pH, reagent addition, or other contaminants may require additional treatment before they can be discharged. Numerous mine-impacted water treatment technologies exist for different contaminants of concern. These technologies are too numerous to discuss in detail here, but many good summaries are available in the literature (Brown et al. 2002; Younger et al. 2002; INAP 2009). The various treatment technologies are most commonly designed to neutralize acidity and raise the pH, remove metals from solution via precipitation or adsorption, or remove salts from solution by membrane-based or ion exchange technologies.

Neutralization and metals precipitation is most commonly accomplished by the addition of alkaline materials such as lime,

limestone, and soda ash, but many other viable technologies are available. In some situations, multiple technologies must be used in sequence such as neutralization to raise the pH and remove metals from solution, followed by flocculation and filtering to remove fine total suspended solids and then reverse osmosis to remove dissolved salts. Treatment systems may be either active or passive. Active systems usually require extensive infrastructure, electrical power, and continuous staff involvement for operation and maintenance. They are usually required for large flows of poor-quality water that must be discharged from active mines. Passive systems generally do not require continuous reagent and energy inputs or operator involvement and so are appropriate at some closed operations and remote sites. However, passive systems are commonly only viable for smaller flows (typically less than 10 L/s), for water that is only weakly contaminated, or as a polishing step for water pretreated with an active system. Common passive treatment systems include wetlands, aeration and settling ponds, and anoxic limestone drains.

The water treatment systems that are selected and used should reliably meet the discharge water quality criteria in the most cost-effective manner. Keys variables that should be considered during the selection process include

- Initial water quality and its variability;
- Required discharge water quality;
- Volume and variability of flow that must be treated;
- Operational versus closure requirements;
- Capital, operations, and maintenance costs;
- Regulatory requirements;
- Reliability of the technology;
- Power, labor, and footprint constraints; and
- Solid waste (generally sludge) management requirements.

Contaminated water with high solute concentrations can generate large volumes of sludge. Although commonly overlooked during initial analysis, the costs and technical challenges to manage water treatment sludges can be significant.

CONTAMINATED SITES MANAGEMENT

Unplanned releases of hazardous materials will typically lead to the contamination of soils, sediment, groundwater, and/or surface water. Historic waste and hazardous materials management was generally less rigorous than current practices, so contaminated sites are most likely to exist at older mines and at new mines in historic mining districts. Common historic contaminated sites at mining operations include leaking underground storage tanks; locations of recurring spills such as refueling areas; historic landfills and waste storage areas; old mineral processing, smelting, and refining sites; and chemically reactive mineral waste repositories. Common contaminants of concern include hydrocarbons, chlorinated solvents, PCBs (polychlorinated biphenyls), asbestos, acids, and metals such as arsenic, cadmium, chromium, copper, lead, mercury, nickel, selenium, and zinc.

Many jurisdictions have clear guidance on acceptable levels for these contaminants in soils, sediment, and water. When available, the appropriate local screening or investigation levels should be used. These criteria are generally derived from simple exposure models with conservative assumptions to protect human health, plants, and animals. Concentrations above these screening levels will generally trigger further remedial, management, or administrative actions to control the risks to human health and the environment. The appropriate criteria must be selected based on the media, the environmental setting, and the potential receptors requiring protection. Examples include standards for drinking water and the protection of aquatic life; and guidelines for stock and irrigation water, animal feed, and soil and sediment cleanup. Because the screening levels tend to be conservative, in some cases the cost and time required to derive site-specific criteria through a formal ecological or human health risk assessment process may be justified.

If current or potential future exposure risks are determined to be unacceptable, then remediation or management strategies (corrective actions) will need to be developed and implemented. Consideration should be given to stakeholder expectations, legal requirements, and the land's planned final use when defining remedial objectives. Corrective actions should then be selected to meet the environmental objectives in the most reliable and cost-effective manner. Only strategies that will consistently meet or exceed local regulatory or permit requirements should be considered. In some cases, more aggressive but more costly remediation strategies may be justified if they will raise the final value of the land or provide significant additional environmental or reputational benefit. Corrective action selection and refinement should be an iterative process with a range of potentially viable strategies assessed before the optimum strategy or combination of strategies is selected. Common general remediation and management strategies include those discussed in the following paragraphs (modified from Rio Tinto 2008c).

Removal Actions

Contaminated soils and sediments can be removed and transported to an engineered repository or treatment facility where the contaminants are neutralized or destroyed. Removal actions typically have a cleanup target in which all contaminated soils or sediments over a certain concentration are removed. The cleanup goals are typically based on the final assumed land use, such as industrial/commercial uses, which typically allow the highest residual concentrations to be left in place, and residential land uses, which usually require the lowest. Open space, wildlife habitat, or grazing and agricultural lands will typically (but not always) have intermediate values.

In-Situ Treatment

Some exposed or near-surface contaminated soils may be remediated in place without excavation and transport. Examples of such in-situ treatment techniques include

- Lime application to neutralize acidity and limit metals bioavailability and mobility,
- Application of nutrients or microbiota and mixing to accelerate the biological degradation of organic contaminants such as hydrocarbons,
- Air sparging and soil vapor extraction for removal of volatile compounds,
- Application of oxidizing agents to treat cyanide spills, and
- Bioremediation with plants that extract metals from the soil and can then be harvested.

Capping in Place

Contaminated soils can be capped with clean fill, soils, and impermeable layers (such as compacted soils, synthetic liners, asphalt, and concrete). Capping can serve several purposes, including

- Preserving runoff water quality;
- Controlling the dispersion of contaminated soils by wind, runoff, or tracking;

- Eliminating direct exposure risks to animals and people from contaminants exposed at the surface;
- Limiting plant contaminant uptake and subsequent exposure risks posed to browsing and grazing animals; and
- Limiting infiltration and subsequent contaminant migration to groundwater.

The cap must usually be designed to meet the environmental objectives for the long term because the contamination is left in place.

Groundwater Remediation

Contaminated groundwater may be remediated through a variety of pump and treatment systems, by in-situ treatment technologies, or by other subsurface control systems such as reactive barriers to control contaminant migration. Pump and treat systems typically involve either removal of the most contaminated portion of the plume (to remove the contaminant source) or capture of the leading edge of a migrating plume (to control its movement). Pump and treat systems for hydrocarbon contamination are usually focused on the removal of free phase product that is floating on the surface of the water table. Examples of in-situ treatment technologies for groundwater include injection of alkalinity to neutralize acidity; air sparging and vapor extraction to remove volatile organic compounds; and the addition of oxygen, microorganisms, and/or nutrients to promote biological degradation. Groundwater remediation will usually continue until water quality meets applicable criteria for drinking water, aquatic life, irrigation water, or stock water.

Groundwater Flow Control

The flux of contaminated groundwater may be reduced so that contaminant loading into and concentrations within down-gradient aquifers or surface water bodies remain within acceptable levels. Controls may include

- Impermeable covers that limit infiltration to groundwater through contamination in the vadose zone,
- Impermeable subsurface barriers that divert up-gradient flow around the contamination, or
- Impermeable subsurface barriers that divert down-gradient flows away from sensitive receptors.

Land-Use Restrictions

Exposure risks to people and animals may be controlled by establishing long-term land-use restrictions such as prohibitions against residential development and farming or fencing to keep even incidental visitors out. Areas where contaminated soils are capped in place will typically require restrictions against development or uses that could breach the cap and re-expose the underlying contamination. In some countries, long-term land-use restrictions may be difficult to enforce.

Natural Attenuation

Many organic contaminants such as solvents, hydrocarbons, and cyanide will degrade with time after they are released to soils, sediment, or water due to oxidation and biological activity. Acidic waters may be neutralized through mixing with more-alkaline waters or reaction with aquifer materials, and dissolved metals may be removed from solution by sorption and precipitation. Down-gradient mixing and dilution will also limit contaminant concentrations below the point of release. If these natural attenuation processes are identified and quantified, it may be possible to justify no further action other than long-term monitoring, if allowed in the local jurisdiction. However, generally, a rigorous evaluation will be needed to demonstrate that impacts to important receiving environments and receptors will remain within acceptable limits throughout the natural attenuation process.

In all cases, some form of post-remediation sampling or monitoring is generally required to ensure that environmental objectives are being met. For removal actions, this typically involves short-term sampling immediately after the excavation has been completed to confirm residual contaminant concentrations are below the target values. If natural attenuation is selected, then long-term monitoring will generally be required to ensure that contaminant concentrations in the affected media are declining at the rates predicted and that receiving environment criteria are being met.

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CHAPTER 16.7

Closure Planning

Evelyn L. Jessup Bingham

INTRODUCTION

In recent years the minerals industry has recognized the importance of sustainable, integrated closure planning. Because of the finite nature of ore bodies, all mine and related facilities will eventually close, and the reputation of the industry is very much dependent on the legacy it leaves. Whether a facility has 10, 20, or 50 years of operational life remaining, integrating closure planning into the mining and metal processing business can result in value for both the company and the wider community.

Activities associated with mineral resource development have a significant impact on the social and environmental aspects of a site. Any site disturbance, even in the early stages of a project, creates a legal and financial closure obligation for the mining company. Communities, governments, and lending institutions are more and more aware of the benefit of good closure planning and demand it as part of the permitting process for new or expanded operations.

Closure of an asset, be it a mine or mineral processing facility, is generally not straightforward and requires dealing with the uncertainty of future conditions and events concerning commodity prices, stakeholder demands, regulatory requirements, environmental conditions, and ore grade. Ideally, closure is carefully planned and occurs with the exhaustion of the mineral resource, but it is not uncommon for closure announcements to come suddenly with the fall of commodity prices or some other unplanned event. Risk-based closure planning is inclusive of such uncertainty.

Closure of an asset is a process that works toward an end point that can be referred to as completion. Planning for a successful closure is a complex, multi-disciplinary task that is essential to minimizing long-term risk for the mining company, the environment, and affected stakeholders.

CHAPTER ORGANIZATION AND APPLICABILITY

This chapter is intended to provide basic practical guidance on closure planning. The information presented should be considered just a first glimpse of the wide-ranging discipline of closure planning.

In recent years a number of comprehensive and specialty documents have been created by governments, industry associations, and nongovernmental organizations (NGOs) on the subject of asset closure planning and financial assurance, such as those available through the International Council on Mining and Metals at www.icmm.com.

The core information provided in this chapter includes a number of practical tables and lists of information that have been found to be useful for closure planning practitioners who are creating or reviewing best-practice integrated closure plans.

After a brief overview of closure planning issues and a description of what integrated closure planning means, a set of principles for creating a framework for closure planning is presented. Following this, the steps involved in closure planning are outlined, and detailed components of an effective closure plan are described according to the closure planning steps. Also presented are a number of common risks to be considered in closure planning. Lists of this type are often useful in preparing for closure risk assessment workshops with a multi-disciplinary team to assist in the completeness of the review. Finally, a checklist is offered, showing how the detail and contents of a closure plan can change through the life cycle of a mineral resource development project.

The chapter content is applicable to a wide range of facilities and activities including mines, processing facilities, and greenfields and brownfields projects. The concepts and tools are appropriate for creating a closure plan for the first time or reviewing an existing closure plan.

COMPLETING THE ASSET LIFE CYCLE—CLOSURE PLAN OBJECTIVES

A successfully completed closure process is one in which the land affected by the asset has been rehabilitated to a sustainable condition. Ideally, in this completed state, the mining lease ownership has been relinquished by the mining company to another land user with minimized continued liability. This sustainable completed state can only be achieved through careful and informed planning over the long term. Closure planning is the responsibility of the mining company, but it cannot be done in a vacuum. Mining companies must involve governments, the communities of which they are part, and

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other stakeholders in closure planning to achieve a successful closure outcome.

Increasingly, financial institutions lending capital to mineral resource development projects set a high standard for asset closure planning as they seek to maximize value from the project and minimize risk. These institutions have learned hard lessons from the past where poor operational closure practices have resulted in ongoing legacies of water pollution, safety and health hazards, and the resulting public outrage and continuing liabilities. Many countries have specific regulations concerning closure planning and completion. Currently, however, there is no overarching world governance body covering mine and mineral processing facility closure, and there are still areas in the world where closure regulation is minimal or nonexistent.

External mining stakeholders such as local communities, conservation groups, and biodiversity advocates are becoming more and more sophisticated about the outcomes of good and bad closure planning practices. Given the opportunity, external stakeholders can become partners in closure planning, ultimately to the benefit of all.

Good closure planning toward the outcome of sustainable mine completion requires communication and engagement with stakeholders to develop agreed-on closure plan objectives, which should include the following:

- Comply with regulatory requirements or industry-leading practice where local regulatory requirements are insufficient to manage risk.
- Protect public health, safety, and welfare over the long term.
- Maximize environmental sustainability.
- Mitigate socioeconomic impacts during and after closure.
- Provide a reasonable basis for estimating and managing closure costs.
- Minimize costs for the mining company, the government, and the public.
- Achieve sustainable land-use conditions as agreed with stakeholders.
- Enhance the reputation of the mining company as a responsible corporate citizen, and ensure that shareholder value is preserved.

Timing and how these objectives are achieved through the life of the asset life cycle is very site specific. Certain types of mines (such as porphyry copper deposits and other hard-rock mines) and mineral processing centers may not allow for any concurrent or progressive rehabilitation or reclamation (for the purposes of this chapter, *reclamation* and *rehabilitation* are used interchangeably) during the operating phase because the disturbed areas (such as the open pit, tailings impoundment, waste dumps, and concentrator buildings) are in constant use during the operations. Assets of this type may spend many decades planning for closure activities that will occur at the end of the asset life.

Coal mines, on the other hand, generally have the opportunity to execute and obtain sign-off on rehabilitation through the life of the asset. However, even projects conducive to progressive closure will face challenges associated with life-of-asset (LOA) plans changing as commodity prices go up and down. It is not unheard of for coal mines to have to dig up or cover over well-established rehabilitation because coal seams once thought uneconomic are now scheduled to be mined.

Economic conditions going in the other direction more commonly present a serious obstacle to sustainable closure planning and ultimate completion of the closure process. A downturn in commodity prices usually results in a lack of resources for progressive rehabilitation (if the mine continues to operate) and for closure execution if sudden closure of the asset is announced.

To minimize the ups and downs of economic cycles and resource availability, effort on the part of the mining company must be made to understand and forecast changing conditions that might affect closure planning and execution as well as possible changes in stakeholder needs. Closure plans must be adjusted through time and adequate resources allocated for difficult economic conditions. Postmining land use and other objectives may need to change according to changes in the LOA plan and external factors.

INTEGRATING CLOSURE PLANNING FOR EFFECTIVENESS

A mining company's commitment to good closure planning practices will pay off in reduction of liabilities for the company and improved reputation with stakeholders. In the past, closure planning has often been a stand-alone activity in the business, usually done as a one-time effort to meet a regulatory obligation.

It is not uncommon for the asset closure plan to be the responsibility of the environmental function at a site while the LOA plan is the responsibility of the mine planners and the updates to the closure financial provision are handled by the accounting department, whose focus is on meeting accounting guidelines. A lack of communication between these various departments can cause the closure plan scope and details to fall well behind the inevitable updates to the LOA plan and the closure provision carried on the books to be out of date with both.

To address closure planning issues and meet business objectives to manage risk, it is necessary to ensure that closure planning is fully integrated in the core business of the asset. As business conditions change, LOA plans change, and so must the closure plan change. Additionally, the detail and accuracy of closure plans must change through the life cycle of the asset—starting out as conceptual and progressively becoming more detailed over time. Finally, as regulations and stakeholder expectations change, the closure plan needs to change. Effective closure plans must be integrated in a number of levels including the following:

- Integration with the LOA and long-term business plan. The closure plan necessarily links past disturbance, current operations, progressive rehabilitation, and future disturbance planned until the end of asset life.
- Integration with external stakeholder affairs. A focus of closure planning should be engagement with the external stakeholders affected by closure. This will include working with the government, local community, and NGOs to understand their needs and concerns. Interaction with external stakeholders during closure planning not only results in compliance with regulations but also affords the development of robust closure criteria to support an acceptable end land-use and communities that are better prepared to support the company when it comes time for completion and lease relinquishment.

- Integration with all the relevant functional disciplines on-site. Closure planning and ultimately the execution of the closure project toward completion will involve senior management, human resources, project engineering, business improvement specialists, environmental managers, operations, planning engineers, community and external affairs, legal advisors, risk management, and finance; all are necessary for effectiveness. Commonly, the following key roles and responsibilities will be essential to the multidisciplinary team:
  - Technical services provide the engineering design, LOA plan, and rehabilitation plan.
  - Project engineers and cost specialists are responsible for cost estimates, design of necessary pilot tests, and execution by project management.
  - Environmental specialists are responsible for developing closure criteria and performance requirements working with government representatives to meet regulations.
  - Community specialists are responsible for stakeholder engagement and closure scenario building.
  - Finance and accounting personnel are responsible for provisioning and maintaining closure risk registers.
  - Management is responsible for setting key performance indicators and milestones for closure.

Integrated closure plans are developed, reviewed, and updated starting in the LOA concept phase and moving through prefeasibility, feasibility, execution, operation, and closure phases of a mineral development project. Throughout this lengthy period, integration in the key areas will increase the probability of a closure plan meeting the business objectives upon execution.

CREATING A MANAGEMENT FRAMEWORK FOR INTEGRATED CLOSURE PLANNING

In order to fully integrate closure planning in the business, a long-term commitment to a set of basic principles is necessary from senior management. Closure planning is not a one-time endeavor but one that takes place over many years or decades. The people involved in closure planning will come and go; however, if the company has adopted a framework of principles, closure planning can proceed and improve through the life cycle of the mineral development project.

The following basic list of principles should be considered and adjusted as required for adoption as a company "closure policy" to form a solid framework from which the closure planning team can work over time.

- **Framework Principle 1: Regular Review and Updates**
  - Closure plans shall be maintained throughout the life cycle of all mineral development assets.
  - Closure plans shall be reviewed annually in conjunction with the LOA planning process. Closure plans and associated cost estimates shall be revised when there is a significant change to the LOA plan, and when there are regulations or factors significantly affecting the closure plan and associated cost estimates.
  - Closure planning processes and closure plans and cost estimates shall be subject to reporting, audit, and governance procedures, particularly where there is a risk of material misstatement on financial reports or a significant effect on business decisions.

Table 16.7-1 Closure plan upgrades across life-cycle phases

Life-Cycle Phase	Closure Plan Class
Projects in study phase	Conceptual-level closure plan and cost estimate ±50% to 30%
Assets with >10 years life remaining	Conceptual-level closure plan and cost estimates, e.g., ±30%. Expectation of progressive improvement in uncertainty and cost estimates annual review.
Assets 10–3 years life remaining	Prefeasibility-level closure plan and cost estimates, e.g., ±20%. Expectation of progressive improvement in uncertainty and cost estimates annual review.
Assets with 3–1 years life remaining	Feasibility-level closure plan and cost estimates, e.g., ±15%.
Assets in closure execution or postclosure	Execution-level closure plan and cost estimate, e.g., ±10%

- **Framework Principle 2: Management of Business Risk**
  - Closure planning shall include an assessment of all risks associated with closure including, but not limited to, risks associated with
    - Postclosure liabilities;
    - External stakeholder needs, aspirations, concerns;
    - Deferral or delay in progressive rehabilitation during operations;
    - Uncertainty in environmental, regulatory, or community aspects;
    - Uncertainty in cost estimations;
    - Uncertainty in technical aspects of closure; and
    - Delay in closure execution.
  - The management of closure risk shall be developed in accordance with formal risk management standards and associated guidance approved by company management.
- **Framework Principle 3: Accurate Estimates of Cost**
  - Closure plans shall be prepared in accordance with project management principles and contain sufficient detail to determine the cost of closure to the extent necessary to manage closure risk.
  - Closure plans will be progressively upgraded according to the life-cycle stage of the mineral development project as shown in Table 16.7-1.
  - Environmental and engineering studies will be performed to increase the certainty of closure cost estimates as an investment comes closer to the need to execute the required closure activity.
  - All closure plans shall include an estimate of the provision for closure, in accordance with international accounting guidelines.

THE CLOSURE PLANNING PROCESS

Whether creating a closure plan from scratch for a greenfield project, exploration project, or simply reviewing an existing closure plan for a large, mature mining property, the steps in the closure planning process are essentially the same. The level of detail and effort required to develop the closure plan through this process will vary as a function of site-specific issues and the life-cycle phase of the project.

Sudden or unplanned closure due to a downturn in commodity price or other adverse business decision may call for an acceleration of this process and require such things as interim care and maintenance plans to be implemented while the closure execution plan is being finalized.

Workshop Approach

The process of developing and updating an integrated closure plan should take as much thought and effort as that dedicated to design and development of any significant project. The result of the closure planning, even in the early phases, has the potential to significantly affect the business and stakeholders. Closure planning sets the course for years of activities, and it involves the expenditure of material funds for studies and, ultimately, closure execution.

Because of the multi-disciplinary nature of closure planning, facilitated workshop sessions are useful at intervals in the closure planning process. An internal closure risk assessment workshop is useful in the beginning to set the context, agree on strategy, and explore risks.

Workshop sessions with external stakeholders can be effective in building relationships and obtaining input about expectations. If available, an expert in workshop facilitation can add significant value, keeping the participants on point, while enabling constructive brainstorming, debate, discussion, and challenge.

A single point of accountability, a team organizational chart, and a work breakdown structure and schedule should be developed for the closure planning team to refer to. This should be updated annually during the regular closure plan review.

Basic Steps in Closure Planning

Recommended steps for closure planning are as follows:

1. Set the context by reviewing background data regulations, LOA plans, environmental baselines, and previous closure plans. Review stakeholders and engage as needed to understand their issues. Review the current status of the progressive rehabilitation plan and implementation schedule to determine whether the LOA plan, progressive rehabilitation plan, and closure plan align.
2. Establish closure plan strategy and criteria including scope, objectives, assumptions, design criteria, and completion criteria. This sets the basis for all subsequent steps. It is essential that the closure team reaches agreement on the standards and strategy to efficiently complete the planning.
3. Assess risks and evaluate closure options. The closure risk assessment process forms the backbone of integrated closure planning, and therefore a standardized risk assessment process, such as that developed by Australia and New Zealand (AS.NZS 4360:2004), should be used. Involving the proper set of people with the detailed knowledge of the site is essential to identifying all the risk issues. Standard options analysis techniques that would be used on any large project involving significant costs should be applied to ensure sufficient rigor in providing the basis of the preferred closure plan.
4. Select a preferred closure plan from the option analyses to control closure risks at an acceptable level. This step is usually an iterative process with Steps 3 and 5, as cost estimate and the risk profile necessarily enter into the decision-making process. The preferred closure plan will be adjusted with changes to the LOA plan and other changes to the context and risk profile of the site as the annual reviews proceed over time.
5. Generate robust cost estimates for the preferred closure plan in accordance with sound engineering and project management principles. The engineering cost estimate serves as the basis for a number of subset estimates, including internal business investment valuation, accounting provision for financial statements and government financial assurance or bonding. Both deterministic and probabilistic estimates may be generated depending on the level of uncertainty. Probabilistic techniques such as Monte Carlo simulations may require specialist effort but can very effectively incorporate the uncertainty of such items as undefined contamination and cost fluctuation. As the site nears closure, specialist demolition, salvage value, and remedial experts will be required to establish the cost of an executable closure plan.
6. Establish an action plan for improvement. Effective integrated closure planning requires improvement in the knowledge base in preparation for the next review cycle and beyond. This will include further stakeholder consultation, engineering and environmental studies, site characterization studies, definition of existing contamination, and rehabilitation pilot testing. These actions may require allocation of significant resources, and therefore a formal action plan for improvement establishing schedule and budget is necessary.
7. Update the closure document and record management system. The closure plan process represents a significant investment in human and financial resources over many years. A readily retrievable archive of the process, the information used, and the decisions made is essential for annual reviews and audits.

COMPONENTS OF THE INTEGRATED CLOSURE PLAN

Table 16.7-2 provides a guide to the components of a typical closure plan organized by the high-level steps taken by the closure planning team. Because of the site-specific nature of closure planning, components will vary according to many factors. The list in Table 16.7-2 can be used in a number of ways: as a closure planning checklist, a table of contents for a closure plan, an update list during the annual closure plan review, or a closure plan audit checklist.

CLOSURE RISK

Assessing closure risk requires a systematic, structured evaluation. As mentioned, the risk assessment forms the basis of the closure plan and cost estimates. The focus of the closure risk assessment will be to establish an acceptable risk profile for the company and all other stakeholders upon completion of the closure project.

Materiality of risks will be an important consideration for the closure team to prevent the closure planning process from becoming bogged down in unnecessary detail. It is useful for the team to establish a materiality threshold at the beginning of the closure planning process as an assumption. This will depend a great deal on where the asset is in the life cycle, the class of closure plan (concept, prefeasibility, feasibility, etc.), and the overall magnitude of the closure plan.

For example, a closure team may decide that risk issues identified which have a worst-case cost of addressing less than $200,000 are not material to the process of updating a concept-level closure plan that will total more than $100,000,000. The closure planning team may record these lesser risks or review them in subsequent updates but not spend valuable time putting them through the full risk assessment and option evaluation process.

Table 16.7-2 Key components of an integrated closure plan

	Plan Component	Description and Comment
▶	**Closure Planning Step 1**	**Set the context.**
1a.	**Life-of-asset (LOA) plan summary and map**	The LOA plan is the basis of the closure plan. Additionally, the closure plan can affect the LOA plan when the cost and risk profile of the closure plan is considered. The LOA plan establishes the impacted area and scope of works to be decommissioned, rehabilitated, closed, and released. Summaries of the final configuration of pits, dumps, and plant areas are included in the closure plan. LOA plans necessarily change with changing economic conditions and changing technology. Closure plans should be fully integrated and updated with the LOA plan.
1b.	**Progressive rehabilitation plan summary and map**	Plans to rehabilitate areas of disturbance back to final configuration during the operating phase (prior to closure execution) are integrated in the closure plan. Progressive rehabilitation plans are integrated with the LOA plan and closure plan, and are reviewed annually. Expenditures for progressive rehabilitation are part of the operations budget and commonly are excluded from the closure plan and cost estimate. A significant closure planning risk is that progressive rehabilitation during operations falls behind schedule and unexpectedly needs to be completed as part of closure execution. Similarly, sudden closure (e.g., due to commodity price downturn) can result in a breakdown in the progressive rehabilitation schedule and an underfunded closure plan.
1c.	**Studies register—environmental, cultural, heritage, and biodiversity conditions**	A reference list of studies documenting conditions for the LOA plan footprint, pollutant management area, and regional area impacted must be established. Baseline studies and recent reviews of conditions are important in establishing closure criteria. In particular, incorporating the ongoing results of groundwater monitoring, surface water monitoring, and vegetation studies into closure planning is critical. Basis for remedial projects (current and forecasted) and their expected outcome should be reviewed regularly and factored into closure planning.
1d.	**Legal and corporate requirement register**	A register of regulatory/legislative mandatory government requirements that cover the environmental, social, financial, and legal aspects of the closure plan must be kept up-to-date and reviewed annually. These requirements vary significantly from country to country and site to site, and must be well understood as they become the basis for the absolute minimum criteria for the closure plan to meet. Site-specific permit requirements are specified. Legal commitments to joint-venture partners, financial institutions, or other entities affecting closure planning must be met. A register of company requirements, such as internal, environmental, or community standards and industry-leading closure criteria and practices, must be kept. Distinctions must be made as to which are considered mandatory (as in the case where a constructive obligation has been created by publicly indicating a standard higher than legislative requirements) and which are discretionary.
1e.	**Land and property ownership register**	A register summarizing the site land information impacted by the site disturbance should be created. Property boundaries, geographic information system data, lease arrangements, deeds, and related obligations must be clearly recorded and reviewed for their impact on the closure plan.
1f.	**Stakeholder analysis register**	External and internal closure stakeholders must be identified; known stakeholders' needs, concerns, and expectations must be assessed and considered. Requirements and community expectations in developing overall closure objectives and criteria for postclosure land use and cultural and heritage values must be evaluated. This register is updated as closure planning proceeds through the years to reflect current demographics and changing views as the stakeholder consultation and external affairs management plan (discussed further in step 6b) is implemented.
▶	**Closure Planning Step 2**	**Establish closure plan strategy and criteria.**
2a.	**Closure boundary and scope summary**	This may be a simple list and diagram of the key elements of the site, established by the multifunctional closure team after review of existing information. Establishing the boundaries of the closure plan may not be completely straightforward. The closure plan may or may not include such elements as a joint venture or contractor-run coal processing plant. A rail spur on-site may be included in the closure plan at one site, while another site's third-party-owned railway passing through the site is not included. Intangibles such as the human resources severance plans or a company-sponsored community support trust are to be key elements recorded and clearly included, or not included, in the closure plan to avoid confusion and errors in closure provisions as time goes on.
2b.	**Closure strategy, objectives, and assumption documentation**	Site-specific high-level vision, strategy, and objectives, followed by a more detailed listing of supporting assumptions (e.g., target closure date, costing basis in-house or contractor), should be established. These are updated as needed during the annual review to reflect any changes in stakeholder and company requirements. The materiality of risks should be established for consideration in the risk assessment for the rest of the closure process.
2c.	**Closure criteria register—design criteria and performance indicators**	Determining closure design criteria and performance indicators is an interactive process that takes form as the closure team proceeds through the risk assessment and closure options analysis. Examples of specific criteria include depth of topsoil, minimum requirement for slopes to control erosion, and storm criteria for sizing water management structures. The specifics of these items will vary depending on where in the life cycle the asset is, starting out as conceptual when closure is many years away and becoming very specific during the closure execution and postclosure monitoring phase. The closure criteria register should provide a high-level summary of rationale and references for how the criteria were determined (e.g., reference to international engineering standards, site-specific risk control basis, local regulatory mandate).

(continues)

Table 16.7-2 Key components of an integrated closure plan (continued)

	Plan Component	Description and Comment
▶	**Closure Planning Step 3**	**Assess risks and evaluate closure options.**
3a.	**Closure risk register**	The closure plan can effectively be thought of as a set of risk control measures. Documenting the output of the formal detailed assessment of all business, environmental, and social sustainability aspects and risks of closure is critical to the closure planning process. It is essential that risk assessments are reviewed and updated annually or whenever there are major changes in the LOA plan or regulatory requirements. The risk register should document the full range of closure risk, risk rating, and enough background information to enable efficient review and communication for audit. The risk register also serves as the basis of the annual closure plan review. Table 16.7-3 provides a list of examples of common closure risks and the associated closure plan controls.
3b.	**Closure opportunities register**	While risks associated with closure are usually the primary focus of the closure planning team, it is important to identify and develop any closure opportunities. Postclosure land uses for the large expanses of land typically involved in a mining property should be explored. Industrial parks, emerging carbon sequestration techniques, alternative energy (such as solar and wind power generation stations) may present economic development opportunities. Timber, agriculture, and grazing may be appropriate. Biodiversity preserves and wildlife conservation areas are postclosure land uses that may present long-term opportunities. The community or regional government may have an interest in using infrastructure such as roads, bridges, power plants, and warehouses after operations have ceased. The capacity of a third party to manage these facilities needs to be carefully gauged prior to the release of the infrastructure. It is not uncommon for the mining company to find the liability coming back unexpectedly if the recipient of the infrastructure finds itself unable to bear the maintenance or operating costs. Other common opportunities to evaluate include third-party reprocessing of low-grade ore stockpiles prior to rehabilitation, recovery of metals from postclosure water treatment, and revenue created by salvage value during demolition. Closure opportunities assessment should be evaluated and documented in the same manner as downside risk.
3c.	**Closure option analysis summary**	As the closure team works through the evaluation of risks and opportunities, various closure plan options will be generated. Formal and informal options analysis will be required to choose the preferred set of activities that become the updated closure plan. As options are considered, objectives and criteria may need to be adjusted, data gaps may be identified, and the interactive nature of closure planning becomes apparent. Concisely documenting options considered will make subsequent reviews and updates more efficient.
▶	**Closure Planning Step 4**	**Select preferred closure plan.**
4a.	**Postclosure land use and final landform plan**	Postclosure land use will be developed and agreed upon with relevant stakeholders (both internal and external) and as the result of risk assessment and options analysis. The postclosure land use will have a significant effect on the final landform plan, and both are tied to developing an acceptable postclosure risk profile. Examples of this interaction include reducing erosion risk, where final landform slope angles are minimized; establishing self-sustaining vegetation, where landforms are designed to promote or discourage access for timber or grazing activities; or achieving flora and fauna biodiversity goals. More than one postclosure use may be compatible at the same site, such as a solar panel installation on a rehabilitated waste dump, which may work well with managed grazing. Conversely, postclosure land uses may be at odds with each other or pose unacceptable risk, such as managed grazing vs. biodiversity uses or access to water for grazing animals vs. public safety issues. Landform plans should reflect the postclosure public access situation. Re-contouring of land to deter public access where necessary may be possible in lieu of fencing that requires maintenance. Public safety and long-term environmental sustainability should always be an overriding consideration and must be worked through with applicable stakeholders. The final landform plan should include maps and other documentation that clearly show the final landforms and how they support the postclosure land use.
4b.	**Closure execution plan**	An adequate description and schedule of closure actions and activities are required. The level of detail included in the closure execution plan will depend on the class of closure plan (conceptual, feasibility, etc.), regulatory requirements, and the company's internal project management requirements. This plan is sometimes referred to as the annual closure plan update or preferred closure plan and commonly addresses all activities required for closure including • Infrastructure decommissioning; • Remediation of contaminated soils; • Water remediation; • Earthmoving, topsoiling, revegetation; • Social management—community and work force (training, economic development); and • Project management and administration. Attention to completeness is required, as this will provide the basis for the cost estimates. Supporting documentation may include organizational charts, work breakdown structures, and contingency plans. It is not uncommon for an asset to develop and mange multiple versions of a closure execution plan that contains varying levels of detail depending on the whether the audience is community stakeholders who want a high-level summary or the engineering functions needing the detailed plan.

(continues)

Table 16.7-2 Key components of an integrated closure plan (continued)

	Plan Component	Description and Comment
4c.	**Postclosure plan**	It is critical that a schedule of postclosure actions and activities be developed and updated. Following the initial phases of decommissioning and rehabilitation, the postclosure period continues until the mining lease is relinquished or the company is otherwise released of liabilities. This postclosure phase needs to be considered in the risk assessment process. Indeed, it can be a surprisingly lengthy and expensive phase, perhaps continuing into perpetuity at some older more problematic operation sites. Postclosure activities include long-term water monitoring, data analysis, and reporting to company and government. They also include maintenance of remaining facilities, such as fences, water treatment plants, and water conveyance structures; actions required to repair erosion; and other rehabilitation repairs. The project management and administrative activities required for the postclosure period must be included in the plan.
4d.	**Completion plan/mining lease relinquishment plan**	In most cases, one of the prime objectives of a closure plan is to meet the performance indicators set by government and other stakeholders and to obtain formal relinquishment of the mining lease and sign off that all company obligations are complete. Unfortunately, there are very few examples where this has been successfully accomplished. Developing a mining lease relinquishment plan in the early phases of mine life is intended to give clarity to the process and mechanisms required for completion of closure. Ideally, the plan is developed in conjunction with the formal approval body such as the state or federal government. Attention should be given to the fact that most governments around the world (in both developed and undeveloped countries) do not have adequate procedures to sign off on the relinquishment plan or officially release the company, even when it is clearly demonstrated that the closure criteria have been met.
▶	**Closure Planning Step 5**	**Generate cost estimates with sound principles.**
5a.	**Deterministic cost estimate and range analysis**	Cost estimates are made for the activities that constitute the selected closure execution plan. Typically, a deterministic range analysis estimating the range of costs of each activity is captured as line items in a spreadsheet. Cost estimates should follow the standard project management and engineering estimate procedures that would be undertaken for any large-scale, multimillion-dollar projects. It is useful to forecast expenditures for a 30-year period following the targeted closure date to see the larger, long-term picture and to ensure that all costs are captured to avoid underestimating the cost of closure. The closure team will need to review applicable government, accounting, and corporate requirements to decide if a deterministic cost estimate is a sufficient end point or if probabilistic modeling is required.
5b.	**Probabilistic modeling for expected cost of closure**	Probabilistic cost modeling, using various techniques including the Monte Carlo simulation method, can be very effective in managing the inherent uncertainty of closure cost estimates. Probabilities can be assigned to the cost range analysis, event risk occurrence, and timing of key aspects affecting present value calculations and the like. The results of a probabilistic cost model for closure may satisfy a number of government, accounting, and corporate requirements to report the "expected value" of closure that demonstrates an incorporation of risk management into to the cost estimate. The closure team will want to include the expert advice of probabilistic financial modelers as would be done for any large-project costing effort.
5c.	**Closure cost summary report**	The results of the base cost estimate (whether deterministic or probabilistic) will be manipulated for various purposes according to specific guidelines. The three most common types of closure cost estimates required are the following: • Business valuation. This all-inclusive cost estimate is used primarily for internal business decisions and follows company-specific investment standards. The closure valuation looks at the LOA disturbance (rather than existing disturbance), integrates progressive rehabilitation, and is usually adjusted to account for salvage value and tax benefits. • Accounting provision. International accounting standards are very specific for determining an acceptable provision to be declared in the mining company's financial statements. Review of current accounting standards by an accounting professional is necessary to ensure compliance with corporate government requirements such as the Sarbanes–Oxley Act of 2002 in the United States. Generally, provision calculations do not include salvage value, do not allow tax benefits to be recognized, and only address current reporting year current disturbance (rather than LOA disturbance). • Government financial assurance. Guidelines vary greatly country to country and state to state. It is critical that the closure team adequately reviews the site-specific regulatory requirements for calculating financial assurance. Progressive rehabilitation is most often treated as an operating cost and is not included in any of these closure cost estimates. Progressive rehabilitation is budgeted and tracked as any other operational expenditure according to local accounting guidelines. Discount rates are generally applied to all of the above cost-of-closure calculations. Assumptions made to determine present value may vary and should be carefully documented to avoid errors and confusion. Documenting assumptions made and guidelines followed to determine the reported costs of closure is important to avoid confusion in subsequent reviews and to provide a basis for internal and external auditing.
5d.	**Closure plan review and audit plan**	Regular, formal review of the closure plan and associated cost estimates are critical to keeping the closure plan accurate and effective for managing risk and liabilities and for making sound business decisions. An annual review by the closure planning team, integrated with the LOA plan update, should be scheduled and results documented. Review by an entity outside the closure team can be very helpful due to the complex nature of closure planning and the potential for errors in the risk-assessment process. This review can take many forms, including expert peer review, internal audit, audit by joint-venture partners, or consultant review. The scope and details of the internal and external peer reviews will vary with the level of site-specific business risk and where the mining venture is in the life cycle. As the site nears closure execution, detail reviews and audit of the closure plan and cost estimates become critical to managing business risk.

(continues)

Table 16.7-2 Key components of an integrated closure plan (continued)

	Plan Component	Description and Comment
▶	**Closure Planning Step 6**	**Establish an action plan for improving the closure plan.**
6a.	**Closure study plan**	A schedule of environmental and engineering studies and projects is required to systematically reduce the uncertainty in the closure plan. Examples include piloting wast dump covers, vegetation trials, groundwater studies, waste characterization, and community surveys. The closure study plan should contain sufficient detail on goals and objectives to be effective. The study plan should include estimates of human and cost resources required and should identify where the resources are budgeted.
6b.	**Stakeholder consultation and external affairs management plan**	A plan for engaging and informing stakeholders around closure issues should be prepared. Economic development for the community postclosure must take place over many years to be effective and should be explicitly addressed in the external affairs management plan. Additionally, planning is necessary to receive sufficient input on postclosure land use and sustainable development. Frequency and range of activity will change over time depending on the life cycle of the asset. Human resources and budgets required should be identified and scheduled.
▶	**Closure Planning Step 7**	**Update closure document and record management system.**
7a.	**Closure document and record retention management system**	Closure planning updates will take place over many years or decades and will be subject to audit and regular review. A formal system with accountability will need to be established for the short and long term. Information gathered during the closure planning process should be documented in a manner that allows easy access and change management. Particular attention to effective document retention is required in the postclosure period and following completion and lease relinquishment.

Table 16.7-3 Common closure planning risks

Type	Closure Risk Issue	Possible Causes	Potential Impacts	Example Controls
Safety and health	Worker injury during closure execution	Asbestos, exposure to hazardous materials, inexperienced demolition crew	Injury, loss of life, company reputation, financial loss, regulatory penalties	Effective health and safety, safety and change management program, use of qualified people
	Public injury postclosure	Subsidence of underground workings, open pits, high-wall failure, unsecured open voids, standing water	Injury, loss of life, company reputation, financial loss, regulatory penalties	Engineering design eliminating hazard, controlled access, public education, monitoring program, maintenance plan
Environment	Water contamination	Acid rock drainage, metal and salt seepage from waste rock/ tailings/pit/underground workings, hydrocarbon soil/water contamination from poor operations practices, water catchment overflow, erosion, pit lake water quality deterioration	Third-party legal action, extensive remediation postclosure financial loss, relinquishment delay, loss of company reputation	Long-term water monitoring programs, baseline data collection, preventive actions during operations, adequate closure design criteria
	Water balance, water management	Inadequate quantitative water balance, poor storm design criteria, climate change	Uncontrolled discharge, legal action, penalties, contamination flow off site, overflow or failure of dams, catchments, injury or loss of life, financial loss, loss of reputation	Baseline climate and water data studies, quantitative water balance inclusive of water quality issues, understanding of surface/groundwater interaction, proper storm design criteria, climate change risk analysis
	Air	Improper air pollution control during operation (dust/stack emissions), decommissioning activities, inadequate cover/vegetation, improper void management	Soil contamination off property, financial loss, third-party lawsuits, loss of company reputation, community complaints	Effective air control programs during operations and closure execution, monitoring cover design criteria

(continues)

Table 16.7-3 Common closure planning risks (continued)

Type	Closure Risk Issue	Possible Causes	Potential Impacts	Example Controls
Environment (continued)	Tailings and landform failure	Slope failure, visual impairment, soil loss, storm erosion, vegetation failure, fauna habitat impaired, unsustainable ecosystem, seismic stability	Water contamination, financial loss, nongovernmental organization and community complaints, loss of company reputation, delay in relinquishment	Adequate baseline studies, rigorous engineering criteria development inclusive of ecosystem considerations and stakeholder desires
	Tailings and waste disposal	Geochemistry of rock dumps, tailings. On-site disposal of hazardous waste, long-term stability compromised by seismic or erosion events, radiation isolation, unknown hazards in landfill	Long-term water treatment, financial loss	Adequate environmental impact studies, qualified expert consultation, engineering criteria development
	Failure of remedial plan or closure activity	Poor execution by third-party contractor, soil or water contamination more serious than anticipated, vegetation failure	Financial loss, regulatory penalty, delay in relinquishment	Due diligence of third-party contractors, appropriate indemnities and insurance, independent review of high-risk plans, adequate environmental studies, and pilot testing
Community	Employee	Union strife, management retention during decommissioning, rehabilitation, and postclosure	Company reputation, access to qualified employees for closure project and future projects	Redundancy/severance planning, training programs, incentives for retention/hiring qualified personnel
	Local community outrage	Loss of income, real estate devaluation, population decline, loss of tax base, unrealistic expectations of mining company viewed as "deep pockets," failure of third party to maintain infrastructure handed over postclosure, safety hazard postclosure	Socioeconomic disruption, poverty; relocation of population; loss of emergency, medical, and educational services; company reputation; access to new projects; financial loss; long-term liability returns to company	Proactive community planning over entire life cycle, economic development partnerships, education and regular information updates to external stakeholders on LOA plan
Financial and legal	Government requirements change	Current regulations not clear, changes over time, sudden changes with change in government party. Common considerations: pit backfilling, sulfate limits for water discharge, return to original contour, unacceptable postclosure land use	Closure plan inadequate, financial loss, inability to achieve completion	Engagement of government and nongovernmental organizations, monitoring of regulations, obtaining government sign-off incrementally, participating in industry forum, adjusting LOA plan as required (including consideration of early closure)
	Loss of reputation/ third-party actions	Contamination of surface water, groundwater, air; relinquishment of closed property to third party unable to provide proper care and maintenance; exposure to workers and public during operations and closure	Financial loss through payment of compensation, return of property and liabilities after relinquishment, loss of reputation, environmental penalties, and other statutory penalties	Thorough risk-based remediation, external review of third parties and appropriate sign-off, monitoring programs during operations in closure phase
	Inadequate provisioning	Failure to consider all possible risks/contingencies including tax consequences. Common shortfalls: long-term water treatment costs, soil remediation costs, lack of topsoil and cover availability, inadequate estimate of postclosure monitoring time period, inadequate project management, overestimation of salvage value	Financial loss, misstatement of financial report, poor business decisions	Use of standardized risk assessment processes throughout the asset life cycle, audit and other assurance activities, review by legal and tax experts, closure project planned and executed by project management professionals
	Sterilization of resource	Untimely execution of closure	Resource not available when commodity price rises	Use of standardized risk assessment to evaluate closure decision

Table 16.7-4 Life-cycle closure plan comparison checklist

Life-Cycle Phase	Concept Phase and Prefeasibility Study	Feasibility Planning and Execution/Project Construction	Operating Phase	Closure Execution Phase—Decommissioning and Postclosure
General considerations	Closure options are considered in these early study phases of an asset. If a study phase involves pilot testing, drilling, or other actual land disturbance, a closure plan is prepared to address any liabilities created.	A preferred conceptual closure plan for the asset is generally selected during the feasibility phase. As soon as construction of the mining asset begins, land disturbance is created and a closure plan and closure cost estimate should be in place and regularly updated to accurately reflect the closure liability as it is created.	Operation is usually the longest phase in the asset life cycle. A life-of-asset (LOA) plan is updated annually and addresses all closure liabilities by integrating the asset business plan, progressive rehabilitation plans, and closure activities to be executed at cessation of operations.	Following the announcement by management that operations have ceased, the closure plan becomes a project to be managed according to standard project management principles. The closure project does not end until relinquishment of the property has occurred.
Closure plan class	Preparation of a conceptual class of closure plan and cost estimate ±50% to 30% to support project evaluation. Expectation of progressive improvement in uncertainty and annual review of cost estimates. When asset has less than 10 years of life remaining, upgrade closure plan and cost estimate class to prefeasibility/feasibility level.		Transition to prefeasibility/feasibility-level closure plan and cost estimates, e.g., ±20% to 15%	Execution level closure plan and cost estimate, e.g., ±10%
1. Basic closure planning actions	The following applies to closure planning in all phases of the life cycle: ☐ Identify fatal flaws in operation plan that would prevent compliance as a result of closure. ☐ Identify information gaps, update plan, and schedule to rectify. ☐ Establish current government and other stakeholder requirements.			
2. Completion criteria and exit strategy	☐ Identify likely completion criteria. ☐ Outline completion strategy.		☐ Establish completion criteria and preliminary relinquishment sign-off process through studies and stakeholder engagement.	☐ Refine and implement relinquishment sign-off process.
3. Stakeholder engagement	☐ Establish stakeholder identification register. ☐ Identify stakeholder issues. ☐ Outline concept of how stakeholders are to be engaged.		☐ Prepare, maintain, and implement community relations plan that includes community consultation with respect to closure issues.	☐ Implement community relations plan focused on decommissioning, postclosure, and completion issues. ☐ Confirm community acceptance of closure actions for completion.
4. Closure risk management	☐ List closure risks identified in order of priority. Formally assess top closure risks to avoid fatal flaw issues.		☐ Conduct formal qualitative closure risk assessments. ☐ Establish and update a documented closure risk register.	
5. Closure plan cost estimate	☐ Document basis of closure cost estimates. ☐ Document use of deterministic and probabilistic methods used. ☐ Confirm that closure cost estimates are risk based and clearly tied to closure plan, i.e., expected value at the appropriate accuracy level for risk management.			
6. Government	☐ Prepare an overview of requirements.	☐ Generate a register listing requirements and high-level plan for compliance.	☐ Generate a register listing completion criteria requirements with plan, schedule, and cost for compliance.	☐ Obtain formal acceptance for and implement the relinquishment sign-off process.
7. Environmental baseline for issues affecting closure	☐ Identify baseline issues including preexisting contamination, sensitive ecosystems, and biodiversity.	☐ Implement baseline study plans fit for purpose. Compile information and use to plan for additional work as needed.		
8. Postclosure land use	☐ Describe current land use. ☐ Conceptualize postclosure land use for project and alternatives. ☐ Identify land tenure risks and issues for closure. ☐ Plan for additional information needed. ☐ Plan for stakeholder engagement on postclosure land use.	☐ Determine detailed land capability/postmining/postclosure land-use alternatives. ☐ Discuss postclosure land-use alternatives with stakeholder engagement. ☐ Prepare a preliminary LOA plan that includes postclosure land-use issues.	☐ Include postclosure land-use goals when LOA and closure plan are integrated into business. ☐ Progressively evaluate rehabilitation against final land-use status (species regrowth, fauna assemblages, etc.). ☐ Validate land-use status against closure completion criteria.	☐ Document how postclosure land-use goals are met.

(continues)

Table 16.7-4. Life-cycle closure plan comparison checklist (continued)

Life-Cycle Phase	Concept Phase and Prefeasibility Study	Feasibility Planning and Execution/Project Construction	Operating Phase	Closure Execution Phase—Decommissioning and Postclosure
9. Water management	☐ Describe current surface and groundwater status. ☐ List broad closure water management goals. ☐ Identify on-site and off-site water-related risks. ☐ Establish storm design criteria.	☐ Prepare a conceptual water management plan to address on- and off-site risks including • Surface and groundwater, • Impact on quantity, and • Impact on quality. ☐ Finalize landform drainage plan. ☐ Generate a preliminary monitoring plan. ☐ Assess on- and off-site risks. ☐ Determine postclosure goals including final use and completion criteria for dams and voids.	☐ Clearly define and document water completion criteria. ☐ Progressively update current water status report including • Surface runoff patterns, • Storm sizing criteria, • Aquifer delineation, • Surface and groundwater quality and quantity. ☐ Evaluate and validate status against closure completion criteria. ☐ Schedule and cost pollution prevention and remediation plans. ☐ Schedule and cost monitoring plans. ☐ Quantify final landform and drainage patterns (including creek diversions, dams, future flooding impacts).	☐ Implement closure water management, remediation plans. ☐ Decommission and/or rehabilitate any water facilities (wells, boreholes, dams, canals, diversions, etc.) that are not part of postclosure land use. ☐ Monitor and evaluate progress toward final hydrology and water quality completion criteria.
10. Plant/ infrastructure	☐ Identify major infrastructure and define footprint. ☐ Prepare a conceptual decommissioning plan including what infrastructure will remain in place postclosure.	☐ Summarize demolition and decommissioning quantities. ☐ Identify off- and on-site disposal needs. ☐ Establish rehabilitation plan for footprint and adjacent areas.	☐ Progressively update decommissioning plans and cost estimates. ☐ Monitor salvage value and adjust closure cost estimate as required.	☐ Perform decommissioning. ☐ Perform remediation if necessary. ☐ Monitor and evaluate progress toward completion.
11. Overburden/ rock piles, tailings dams, open-pit features	☐ Describe quantity and quality of material. ☐ Conceptualize alternative locations and forms identified. ☐ Conceptualize disposal and closure methods described. ☐ Quantify volumes. ☐ Quantify types including acid generation potential. ☐ Prepare topsoil stripping and management plan. ☐ Prepare preliminary LOA plan for dumps, tailings, pit features.	☐ Generate LOA plan to include • Final slope angles; • Structural safety defined for tailings and rock/ overburden piles; • Water and sediment control features; • Structural stability including settlement, subsidence program; • Facility closure and rehabilitation methodology (impoundment capping, revegetation, final void); • Topsoil and other cover material plan; • Rehabilitation schedule; and • Configuration.	☐ Revise and update LOA plan in conjunction with development of closure plan annually. ☐ Progressively monitor final voids (including borrow pits and quarries), and report against closure goals. ☐ Regularly address gaps found in integrated LOA plan, progressive rehabilitation plan, and closure plan.	☐ Perform rehabilitation. ☐ Perform remediation if necessary. ☐ Monitor and evaluate progress toward completion.
12. Waste and chemical management	☐ Identify types of chemicals used. ☐ Identify types of waste to be generated. ☐ Describe general management and disposal plan. ☐ Quantify chemicals. ☐ Quantify waste generated.	☐ Generate chemical spill prevention and management plan. ☐ Develop waste management plan. ☐ Develop spill prevention and response plan. ☐ Develop rehabilitation plan for on-site waste disposal facilities.	☐ Track waste generation and annual disposal capacity against plan. ☐ Quantify accumulation of waste on-site. ☐ Develop soil decontamination plans. ☐ Document disposal plans for cumulative waste over operating life and at closure.	☐ Implement soil decontamination plans. ☐ Implement closure waste disposal plan for project. ☐ Monitor and evaluate progress toward completion.

(continues)

Table 16.7-4 Life-cycle closure plan comparison checklist (continued)

Life-Cycle Phase	Concept Phase and Prefeasibility Study	Feasibility Planning and Execution/Project Construction	Operating Phase	Closure Execution Phase—Decommissioning and Postclosure
12. Waste and chemical management (continued)	☐ Identify gaps between generation and disposal capacity. ☐ Identify off-site disposal facilities.		☐ Within 3 and 5 years of projected management decision to close, finalize detailed closure waste disposal plan.	
13. Haul roads and access roads	☐ Identify road system. ☐ Identify roads to remain at closure and likely completion criteria. ☐ Conceptualize rehabilitation plan for roads.		☐ Formalize completion criteria for roads to remain at completion of closure. ☐ Develop progressively more detailed rehabilitation plan for roads. ☐ Perform progressive decommissioning of roads where feasible.	☐ Perform decommissioning and rehabilitation of roads that will not be retained. ☐ Configure roads to be left upon completion of closure for long-term use. ☐ Perform remediation if necessary. ☐ Monitor and evaluate progress toward completion.
14. Air quality management	☐ Identify baseline air quality status. ☐ Describe project risks including air contaminant impact off- and on-site at closure.	☐ Conduct baseline air monitoring to establish air quality key performance indicators. ☐ Prepare air-quality management plan for minimizing on- and off-site impact.	☐ Develop a mitigation plan for surface treatments and revegetation to reduce dust and odor in accordance with key performance indicators at completion.	☐ Complete surface treatments and revegetation to meet air quality key performance indicators at completion.
15. Underground workings	☐ Describe subsidence-control issues. ☐ Describe water management issues.	☐ Identify underground workings areas. ☐ Prepare LOA subsidence management and mitigation plans. ☐ Prepare LOA underground water management plans.	☐ Develop underground decommissioning plan. ☐ Develop long-term subsidence management plan.	☐ Implement project plan for decommissioning underground workings.

Closure risks are very site specific and must be assessed by a cross-functional team familiar with the site and issues. However, there are a number of risks issues that are commonly encountered. Table 16.7-3 may serve as a helpful guide to brainstorming site-specific closure risks.

LIFE-CYCLE CLOSURE PLAN COMPARISON

Table 16.7-4 provides a comparison of how the closure plan should change as the mineral development project moves through the life cycle from a study project and feasibility plan into the lengthy operating phase and ultimately to the closure phase. The table is not meant to be exhaustive but can be used to prompt discussion and develop a high-level checklist to ensure that closure plans contain the correct level of detail as the mineral development project progresses.

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REFERENCE

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PART 17

Community and Social Issues

CHAPTER 17.1

Community Issues

Robin Evans and Deanna Kemp

INTRODUCTION

> The excavation, refining and shipping of this ore to the smelters of Japan could bring great profit over the next 20 years to the shareholders of Rio Tinto–Zinc—at the cost of damage to the physical, social and spiritual well-being of Bougainville, which, until the mine came, was a peaceful and prosperous island. Moreover there is a danger that arguments over the ownership of the mine could cause political strife, even civil war, in this part of the South Pacific (West 1972).

In 1989, following a series of increasingly violent protests that included sabotage of power supplies and attacks on mine workers, Bougainville Copper Ltd. evacuated its work force from the Panguna mine located on Bougainville Island in Papua New Guinea. The operation was shut down at relatively short notice, with most equipment left in place, and has not operated since. In the year prior to the shutdown, Bougainville Copper was capitalized at US$1.5 billion (Humphreys 2000), and the operation represented one of the world's largest open-pit mines. In the ensuing years, the civil unrest developed to a full-scale conflict between the Bougainville Revolutionary Army and the Papua New Guinea Defence Force that devastated the island, with several thousand deaths and approximately 50,000 people (a third of the island's population) displaced from their homes (Regan 2001). A peace process that commenced in 1998 between the government and local communities has returned some calm to the island, but the events that surround the abandoned Panguna mine remain the most vivid and tragic example of community conflict surrounding a major mining operation.

The underlying causes of the conflict that erupted on the island were many and complex: they included ethnic differences and the emergence of a secessionist movement prior to the transition of Papua New Guinea from Australian administration to full independence in 1975. The mine became a catalyst, with community concerns about the distribution of economic benefits and the environmental impacts of mine waste on the local river system featuring prominently. According to Denoon (2000), for many of the landowners "Panguna was a social, economic and environmental disaster, and a spur to militant protest."

Mines and the communities they are associated with have always been inextricably linked via a complex network of relationships and issues such as these. Local community members usually form part of the work force at a mining operation, while others in the area supply goods and services. At the same time, individuals, families, and sometimes whole communities can be displaced by the development of a mining lease, while some may be affected by environmental impacts associated with an operation. Community livelihoods can be impacted both directly by land-use changes, and also by changes within local social structures. While safety standards in the industry have improved significantly in most areas, there is a history of workplace accidents and health issues that remains an important factor in the relationship between mining companies, work force, and communities. In some locations, operations can be in competition with parts of the community for scarce resources such as productive land and/or water. In others, mining companies are welcomed as an agent of development that can bring infrastructure, essential services, and economic opportunities to a region. In most cases there is a continuum of views held about proposed or existing mining operations.

The balancing of the benefits and costs of mining operations for local and regional communities has attracted debate for centuries: "Thus it is said, it is clear to all that there is greater detriment from mining than the value of the metals which the mining produces" (Agricola 1556). While Agricola himself went on to staunchly argue the case for the 16th-century mining industry, others have taken more critical positions when considering modern operations. Notwithstanding the resilience and intrinsic strengths of some mine-affected communities for dealing with changes and transformations brought about by mining, there are often power disparities between mines and many remote, rural, and/or indigenous communities (e.g., Banerjee 2001). Compared to companies,

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communities usually have more limited access to information, knowledge, technology, and capital that can be leveraged to shape the nature and pace of mining development. Such disparities have been the primary driver for the increasing involvement of rights-based nongovernmental organizations (NGOs) that have seen international campaigns launched against companies or particular projects when grievances escalate. The boom/bust nature of the industry adds another dimension to the debate about whether mining brings positive or negative change, with mining's contribution to development challenged by "resource curse" theories.

In short, the issues associated with interactions between mining companies and communities have become more prominent, and have increasingly required more attention from those associated with managing resource companies and mineral operations. Those working in the industry are increasingly required to respond to "community issues" in ways that their predecessors were never expected to. It is also likely that many technical staff who get involved in social aspects of mining have little training or prior experience in these issues. The purpose of this chapter is to provide an introduction to the types of social and community issues involved with mining, and the contexts in which they could emerge. It explores what is meant by the term *community*, and also reviews how community issues have increasingly been investigated in the last 20 years, particularly through projects such as the Global Mining Initiative (GMI). Subsequent sections explore more systematically the contextual factors that influence community relationships, and also the types of impacts that attract most attention. More detailed chapters follow this overview and address specific themes in more detail, including indigenous peoples and mining projects, and specific processes to assess and manage social impacts.

WHAT IS "COMMUNITY"?

The concept of "community" is usually used in the minerals industry to describe those who live in the geographic region of an operation, either in defined settlements or dispersed settings. However, there are other equally valid ways to consider the term, especially as modern industry practice has moved to include a greater incidence of fly-in/fly-out arrangements whereby workers and their families live in a distant location. Another relevant example is where traditional owners of the land associated with the mining development have maintained their links to the land but reside elsewhere. In such cases, community impacts can occur many hundreds of miles from an operation. Mining projects often include transport infrastructure and supply chains that span large distances, connecting networks of mines, processing centers, and ports, significantly increasing the range and types of other potential community impacts. There are also many different definitions of community that are not geographically based, such as communities of practice and spiritual communities. However, the mining industry tends to emphasize physicality and proximity to an operation, either spatially or by issue. Leading industry practice acknowledges that communities are complex, evolving, political, and heterogeneous entities (DITR 2006). In 2005, the Australian Ministerial Council on Mineral and Petroleum Resources (MCMPR) defined community as

> a group of people living in a particular area or region. In mining industry terms, community is generally applied to the inhabitants of immediate and surrounding areas who are affected by a company's activities.
>
> The term *local* or *host* community is usually applied to those living in the immediate vicinity of an operation, being indigenous or nonindigenous people, who may have cultural affinity, claim, or direct ownership of an area in which a company has an interest.
>
> *Affected community* refers to the members of the community affected by a company's activities. The effects are most commonly social (resettlement, changed services such as education and health), economic (compensation, job prospects, creation of local wealth), environmental, and political.

A community is usually a diverse group of people with some common bonds. Diversity can come in the form of gender, ethnicity, religion, race, age, economic or social status, wealth, education, language, class, or caste. As a result, members of any community are likely to hold diverse opinions about a mining operation and its activities, as well as most other subjects. As mentioned earlier, individuals within a community will have different and sometimes overlapping associations with the mine as neighbors, employees, suppliers, and so on. It is not uncommon for disagreement and sometimes conflict to develop between different sections of a community in relation to mining operations. To different degrees, conflict will also exist within a community prior to the start of mining, from low-level tension to violent conflict. Some companies choose to operate in conflict or postconflict zones, which will involve yet another layer of complexity to the process of understanding the local community.

More recently, the term *stakeholder* has become a common term that is related to but distinct from *community*. The idea that business has responsibilities broader than its traditional role of generating a return for shareholders is reflected in stakeholder theory, popularized by Freeman (1984) in his seminal work, *Strategic Management: A Stakeholder Approach*. This theory holds that successful companies recognize that they have responsibilities to *stakeholders*, a term referring to any individual or groups who can affect or are affected by a corporation's activities (and where corporate responsibility extends beyond maximizing a financial return to shareholders). Stakeholder theory has not only been a powerful force in academic circles in terms of developing its own research tradition, but also in encouraging the corporate sector to see community concerns and aspirations as key considerations.

A common definition of stakeholders is "those who have an interest in a particular decision, either as individuals or representatives of a group. This includes people who influence a decision, or can influence it, as well as those affected by it" (MCMPR 2005).

Stakeholders might therefore include local community members, NGOs, governments, shareholders, and employees. The use of the term *stakeholder* has been contested on the basis that some communities are in fact *rights holders*—a stronger term than stakeholder—due to rights defined by relevant national or state law. For example, indigenous peoples in Australia with a recognized claim to an area have the right to negotiate under the Native Title Act (Commonwealth of Australia 1993). While local communities are usually viewed as key stakeholders, the potential range of all stakeholders is considerably broader. Both terms are relevant to the discussion that will occur in the next few chapters, although the

main focus will be on the narrower group of "community" as outlined in its definition.

MINING AND SUSTAINABLE DEVELOPMENT

The increased focus on the mineral industry's impact on the environment and society, including the local community, can be attributed to both global trends associated with attitudes to private enterprise and large multinational corporations, and also to the mineral industry's poor track record with high-profile cases such as the Bougainville crisis and various other environmental incidents (e.g., Ok Tedi in Papua New Guinea, Marcopper in the Philippines, and Baia Mare in Romania). During the 1990s the minerals industry came under increasing challenge from various quarters, with the result that it found itself losing its "social license to operate." This term has now become a popular way of describing the influence that society in general has over the ability of an organization to carry out its activities, above and beyond the legal license issued by governments, which govern the extraction of mineral resources. Social license is variously described but commonly considered an ongoing process of approval from the community that is given at a point in time, and not necessarily for the future (AccountAbility 2004).

In response to this increased societal pressure, there were several early initiatives in different countries as local industry bodies sought to engage with both governments and their critics. One example is the Whitehorse Mining Initiative in Canada (Cooney 2008), a multistakeholder initiative developed in 1994 involving industry, government, and NGOs, which aimed to develop general principles for responsible mining. In Australia the Minerals Council of Australia developed the Australian Minerals Industry Code for Environmental Management, modified in the late 1990s to include additional requirements focusing on social and community issues. These types of initiatives were often linked to the concept of sustainable development, a term that has emerged over the last 20 years as a key organizing framework for the global community to consider the links between the development and environmental protection agendas. Dresner (2008) outlines both the history and the politics of this process, including key milestones such as the Brundtland report (WCED 1987) and the United Nations Rio Earth Summit in 1992. Sustainable development remains a contested concept, with many competing definitions and sets of principles, but its popularity has meant that many organizations have chosen to use it to frame their own activities in social and environmental areas.

In 1998, a dialogue between a small group of senior mining industry CEOs including Hugh Morgan of Western Mining Corporation and Sir Robert Wilson from Rio Tinto led to the formation of the GMI, an industry-led process that expanded to involve many of the world's largest mining companies. The initiative had three elements: a 3-year research project to investigate the activities of the industry through the lens of sustainable development; a major conference held in Toronto in 2002 to review the outcomes of the project; and the creation of a new global industry body, the International Council on Mining and Metals (ICMM), charged with implementation of the industry response to the outcomes of the project. The Mining, Minerals and Sustainable Development (MMSD) research project was managed by an independent research group, the International Institute for Environment and Development, and resulted in the *Breaking New Ground* report produced in 2002 in time for the Toronto conference (IIED 2002). It is beyond the scope of this chapter to explore the full range of issues covered by the MMSD report (many are touched on in other chapters in this handbook), but it is noteworthy that one of the nine key challenges identified for the industry to address was the area of mines and communities. A number of supporting research reports addressed specific issues under this theme such as indigenous peoples' rights, social impact analysis, and socioeconomic development. Danielson (2006) provides a comprehensive account of the origins and progress of the MMSD project, including a range of different stakeholder perspectives on various elements of the process. There is little doubt that the GMI and its associated research has been the most influential process to date of all those designed to examine the social and environmental aspects of mining industry activities.

In reviewing the progress and outcomes of both the Whitehorse Mining Initiative and the GMI, Cooney (2008) suggested that the companies who launched the two initiatives were primarily concerned about the public image of mining. Public criticism was being driven by misinformation about the actual impacts of mining, and improved communication would help address this. However, he suggested that through the course of both initiatives and subsequent processes such as the World Bank's *Extractive Industries Review*,

> the mining industry learned matters both of process and of substance: from the engagement process, mining companies have learned different models of comprehensive dialogue and consensus building with critics; by listening to their critics, the companies have learned different approaches to analyzing and managing critical issues. Self-education was not the mining industry's initial purpose in either the Whitehorse Mining Initiative or the Global Mining Initiative, but it was to be the outcome.

In addition to these industry-driven initiatives, many other stakeholder groups have initiated reviews and developed frameworks for reviewing environmental and social aspects of projects, some specific to the resources sector but others more generally focused. Such nonregulatory drivers are pushing minerals companies to focus on local-level social and community issues. Particularly for publicly listed companies, the screening process for socially responsible investment or ethical investment funds, other indexes such as the Dow Jones Sustainability Index, and public ratings are also influencing behavior. Most of these indexes and rating agencies require that organizations establish a systematic approach to managing the social dimensions of their projects, as they would environmental and economic aspects. Many funding agencies, such as the International Finance Corporation (IFC) also require this as a condition of finance. In the absence of mandatory legislation, pressure from third parties has helped to sustain attention on community relations in the mining industry. This growth of "soft" regulation has been significant in the past 10 years—examples that are relevant and referenced by the minerals industry include the following:

- **ICMM Sustainable Development Framework**—Ten principles and various elements that provide guidance on applying sustainable development principles to mineral

operations. Reporting against this framework is obligatory for members of the ICMM.

- **Kimberley Process Certification Scheme**—A multistakeholder initiative that aims to provide product certification for diamonds and reduce the trade in "blood diamonds."
- **Extractive Industries Transparency Initiative**—A voluntary initiative focused on ensuring transparency of payments associated with resource projects.
- **Equator Principles**—A set of environmental and social benchmarks developed by a group of major international banks for addressing environmental and social issues in development project finance.
- **IFC Environmental and Social Standards**—A standards framework developed by the IFC to apply to projects in which they invest World Bank funds.

As well as pressure from this type of "soft" regulation to improve social and environmental performance, many governments are becoming increasingly involved in regulating the community aspects of the development, operation, and closure of mines (Brereton 2002). The focus of governments was initially on environmental issues, but social dimensions are increasingly being regulated in developed and developing countries alike. In Australia, for example, most states have made basic community consultation mandatory for major new development projects, including mining, often as part of environmental and social impact assessments. The Australian Native Title Act (AustLII 1993) has also become a central part of the Australian regulatory regime, providing indigenous groups with the right to negotiate and a potential vehicle to deliver both social and economic benefits for indigenous communities. South Africa has a regulatory framework in place to progress black economic empowerment in the mining and petroleum industries. The Philippine Mining Act of 1995 requires that proposed projects undergo a comprehensive environmental assessment, which must consider socioeconomic as well as environmental impacts and provide evidence of broad social acceptability. In Canada, the requirement to incorporate sustainability considerations in environmental assessment processes has seen community issues come to the fore in several high profile and controversial mining cases such as the Voisey's Bay project, as described by Gibson (2006), and the more recent Kemess North review.

CONTEXT IS CRUCIAL

Before considering some examples of the types of community issues that emerge, it is important to emphasize the influence and importance of context. Many factors can have a significant impact on the interactions and relationships between mining operations and communities, including various social and political aspects, as well as the stage of the mining life cycle involved. Mining is a truly global activity, involving many different types of organizations and communities in settings that range from arid mountains in parts of the Andes, to remote areas within the Arctic Circle, to established agricultural regions in developed countries, and to tropical rainforest settings in developing economies in Asia. In addition to the obvious geographical differences, other contextual factors can be very important.

The history of mining in a country and region can influence community attitudes to mining projects. In some cases mining may be a relatively well-accepted activity, as, for example, in certain well-developed coal-mining areas of the United States and Australia. A number of well-known mining towns around the world have been created close to major mining deposits whose economies continue to be based around the exploitation of these resources. Examples include Kalgoorlie in Australia, Sudbury in Canada, and Cerro de Pasco in Peru. However, in some of these areas, the cumulative and historical social and environmental impacts associated with the mining operations can still be the subject of community debate. As the industry increases its activities in less-developed countries with little or no experience of large-scale mining, community attitudes can vary widely. A good example is Mongolia, which has only recently opened its industry to direct foreign investment. Large-scale mining by global companies is a new phenomenon, and the management of mineral revenues was the principal issue in the 2009 election held in that country.

Existing community land uses can include broad-scale farming, intensive agriculture, open grazing of livestock, and hunting and fishing (among many others). Although the physical footprint of mining operations is usually relatively low, its interaction with other land uses in terms of impacts on other resources such as water, labor, and infrastructure can be significant. In some locations, small-scale or artisanal mining (particularly of gemstones and precious metals) may already be a significant community activity—some estimates put the number of people involved in this type of mining worldwide as high as 20 million, including large numbers of women and children. In this sense, the displacement of artisanal activities by large-scale mining projects can have significant negative ramifications for certain groups of people. By the same token, mining companies have worked with artisanal mining communities to address health and environmental issues and to find opportunities for artisanal miners to find alternative sources of income.

In many parts of the world, elements of the physical landscape play an important role in the culture of local communities. A desert clay pan may represent a physical feature to be managed by a mining company, but it is a significant site to a local indigenous community. In some parts of Australia, red ochre is used by male members of indigenous groups in cultural ceremonies, and therefore its presence in mine overburden becomes a major logistical issue that requires an appropriate response. There are cases where the development of mining projects has resulted in the destruction of sites of major cultural significance, such as at the Argyle mine in Western Australia, for example. There are also cases where the presence of such sites have been the main reason for mining projects not going ahead, and some sites where mining projects have been designed to accommodate cultural heritage considerations.

Political and legal frameworks within a country will have a significant impact on the scale and nature of the mining industry and can also often be the subject of intense community focus. Government capacity to regulate the minerals industry and manage the benefits of mining for the local communities has been identified as a crucial aspect by recent studies (e.g., ICMM 2006) and has been the subject of recent World Bank projects in several developing countries, such as through the provision of technical assistance to the government in Laos and other countries in Asia to strengthen their ability to manage the burgeoning mining sector. Also of interest are legal and customary rights concerning land management, especially in cases where mining occurs on lands

claimed by indigenous peoples. In Papua New Guinea, there is a well-developed system of customary land ownership that is linked to family descent, whereas in some parts of Africa communal tribal ownership and shared land-use arrangements make negotiations for land access for mining purposes considerably more complex. All around the world, many rural or traditional communities have operated on a system of land use and ownership that has not required formal land ownership. If a company compensates only those people with formal land ownership, there is a high potential that it will negatively affect livelihoods, culture, and social traditions that have been in place for generations. Where land titles are in place, it is also men who often hold title and receive compensation for land, highlighting the potential for women to be negatively impacted unless strategies are in place to ensure equitable distribution of benefits.

Other important contextual factors include the nature and scale of the mining operation itself. Large-scale open-pit and strip mines can result in more visible manifestations of mining activity in the form of spoil piles and waste dumps and can be more disruptive to other land uses such as agriculture. Underground mines generally employ more selective mining methods and produce less waste, but subsidence effects in longwall mines can result in impacts on surface environments and water resources. In some countries, the safety record of underground mining is significantly worse than surface mines, as, for example, in the underground coal industry in China (which includes many smaller, informal operations), where the reported fatalities in 2007 from a series of methane and dust explosions and cave-ins numbered nearly 3,800. It is believed that many fatalities and injuries go unreported in China and other countries.

The nature, size, and reputation of the company involved can also be influential. The larger global multinational companies tend to be engaged in the sustainable development debate and signatories to many of the frameworks and conventions that deal with community issues, whereas many of the smaller mining and exploration companies are less active in these areas. Size does not, however, always correspond to enhanced social performance. Not all operations within a large company perform to the same standard. Small or junior companies with a single asset may find creative ways to work with the local community to address social concerns, although they usually have significantly less capacity to adequately resource a community relations function and therefore do not always give social aspects the attention that is warranted. Smaller companies can also have a short-term outlook, as their focus is on discovering and developing the asset to the extent they can sell it, rather than operating the mine themselves. This tends to lead to an avoidance of social investment. Whether a company is from the country of origin can also determine the extent of community opposition. In several parts of the world, there has been community opposition to the involvement of foreign companies in mining and exporting valuable minerals when much of the profit is perceived to go offshore.

Finally, the stage of the mining life cycle can also have a significant bearing on the development of community issues. Some issues are specific to certain phases. For example, the impacts associated with accessing hitherto virgin jungle for the purposes of exploration or with a community hosting a large, temporary construction work force are very different from those that occur during actual mining operations. Closure is often associated with community concerns over the withdrawal of the mining work force and related economic activity. Some closures are planned, where the mine comes to a natural end because the resource has been exhausted, but other mines close suddenly because of changes in commodity prices, which means the mine becomes unprofitable. Mines can also be disrupted or closed because of community protest or conflict, as in the case of Bougainville discussed earlier. Community issues are not fixed; they evolve throughout the life of the operation.

Consider the example of Newmont Waihi Gold's mine located in Waihi, New Zealand, shown in Figure 17.1-1. At this historic mining town, the conversion of old underground workings into an open-pit development during the 1980s has brought the upper benches of the mine to within feet of residences and the town's main street. Although mining has been an activity in the area for more than 100 years, the impacts associated with the mine have affected many residents living close to the edge of the pit, and also the community more broadly. The community relations landscape has been dominated by these issues of amenity, including the impacts of noise, dust, and blast vibrations. However, there are many other contextual issues that also influence how the community perceives and interacts with the operation. The mine and its work force make up about 25% of the town's economy, with many local people employed at the operation. The development of the open pit removed a small hill that was of cultural significance to some Maori tribes of the area, who continue to oppose the presence of the mine on the basis of their traditional beliefs and values but remain in discussion with the company on the management of cultural issues. The mine operates in a developed country with strong environmental protection legislation, and is located in a farming area close to a major tourist region. Recently, the company worked with the community to develop a vision for the town after mine closure, which has been imminent for the past few years, and it is the potential closure of the operation and its impacts that are the current focus for many in the community. A good understanding of the overall context—economic, cultural, political, social, and environmental—as well as an understanding of the issues particular to each community and each group within the community, is therefore crucial in identifying and addressing the many issues that emerge from the closure process.

What Are Community Issues?

The introduction of the term *triple bottom line* (Elkington 1997) and its application in the context of sustainability reporting has seen social or community issues identified as a separate category to environmental or economic issues within the sustainable development framework. In reality, however, most communities are extremely focused on all aspects of mining development and do not necessarily separate out these issues into neat categories. Consider a remote traditional community that relies on the local waterway for catching fish as well as for spiritual worship and ceremony. They may explain the water as important to their survival, their traditions, their family, and their future, seeing these aspects as interconnected rather than separating them. Although there are different ways of understanding these dynamics, it is often environmental or economic aspects that are the main focus of attention. Figure 17.1-2, while not intended to be comprehensive, illustrates both the breadth of issues under consideration and their interrelated nature.



Courtesy of Newmont Waihi Gold.
Figure 17.1-1 Newmont Waihi Gold's operation in Waihi, New Zealand

Some themes have become particularly prominent in the last decade, partly as a result of such initiatives as the MMSD project but also due to a range of other drivers. Some of these have been mentioned previously, such as nascent "social regulation" in the form of legislation, voluntary initiatives, and pressure from NGOs and civil society. In recent years some investment funds have been deliberately disinvested from major companies in the minerals industry because of concerns about social risk. Several of these issues, which have become the focus of various groups and organizations, are explored briefly in the following sections.

Economic Development

The positive influence of mining projects on local, regional, and national economies has always been an argument used by proponents to support new developments. In contrast, the resource curse hypothesis suggests that in fact countries with high levels of natural resources suffer lower rates of economic growth than those with more diversified economies, in effect suffering a paradox of plenty. Much research supports arguments both for and against this proposition, one example being the Resource Endowment Initiative involving the ICMM, the United Nations Conference on Trade and Development, and the World Bank (ICMM 2006). This study concluded that mining investment does provide opportunities for economic growth, poverty reduction, and engagement in the global economy, pointing to specific examples such as Chile and Botswana where increased mining investment has coincided with an upturn in national economic growth. However, it also emphasized the need for effective and transparent governance regimes for the management of mineral wealth. Transfer of some of the benefits from taxes and royalty streams back to the regions where mines are located has been an issue of some contention in several countries such as Peru, for example, where several problems have developed as a result of changes to legislation (Arellano-Yanguas 2008).

In recent years more research has been focused on economic impacts at the local and regional levels. Companies are now starting to report on how much of the added value associated with wages and purchase of goods and services for mining operations stays at the local and regional levels. For example, see the economic indicators in the *Sustainability Reporting Guidelines and Mining and Metals Sector Supplement* (Global Reporting Initiative 2009). In addition, a company's contribution to community development activities are directed at growing local economic activity, often with a focus on nonmining-related businesses, in order to provide for a postclosure future. However, the additional cash flows injected into local economies can have negative impacts as well, as in the form of disproportionate inflation, for example. This can apply equally in less-developed contexts where market economies may be significantly changed by the introduction of mining industry wages, as well as developed economies where industry expansions can result in distortions in real estate and labor markets, with consequential impacts for other sections of the community.

Water and Mining

Access to fresh water represents an essential human need. Water is also fundamental to other ecosystem services required to sustain human life and is high on the political agendas of all levels of government, including the United Nations. It was recognized as a key theme in the MMSD project, with mines operating in the driest and the wettest regions on the planet. While not extracting as large a quantity of water as the agricultural sector in most countries, individual mines are often large consumers in their local context, and their impacts can be significant. Mining companies compete for water use within a range of market and nonmarket jurisdictions, and can often afford to pay considerably more than others, thereby running the risk of reducing the viability of other industries. Companies can also affect access to water if mining development does not consider the usage patterns of the local community. For example, by building roads or operations, some mining developments can inadvertently make access to water sources in developing countries more difficult. At the same time, mining companies are often responsible for developing water infrastructure used by other industries and communities (e.g., Brereton and Parmenter 2006).

Water use in many processing activities results in bodies of contaminated water in tailings storages and flooded pits that, if incorrectly managed, pose risks for downstream users. Many environmental legacies of mining have involved pollution of water systems, including the frequent incidence of acid mine drainage at closed or abandoned mines on many continents. Potential impacts of mining operations on both the quantity and quality of surface and groundwater resources are increasingly being raised as concerns by local communities, and in several cases have been the principal reason why some projects have not gone ahead.

Community Health

There are many ways in which mining operations can impact the area of community health. Population changes including in-migration in developing countries such as Papua New Guinea can be responsible for the spread of diseases such as human immunodeficiency virus (HIV) and tuberculosis. The ICMM recently released a report titled *Good Practice Guidance on HIV/AIDS, TB and Malaria* (ICMM 2008) for its members, reflecting the incidence of HIV in mining work forces and communities in different parts of the world. Other direct health impacts can come from emissions from processing operations, such as high blood lead levels found in communities located near older lead smelters on several continents. Local controversies have developed over the potential health impacts of riverine and marine tailings disposal processes



Figure 17.1-2 Examples of community issues in mining

such as those at Marinduque in the Philippines and Minahasa in Indonesia, for example. Both of these cases attracted significant international attention.

On the other hand, mining developments are often responsible for the establishment of health infrastructure and services in remote areas. Despite the environmental impacts associated with riverine waste and tailings disposal at the Ok Tedi mine in Papua New Guinea, the antimalarial and health clinic campaign introduced by the company to isolated communities near the mine resulted in extremely significant improvements in infant mortality and life expectancy statistics in those areas. A mining company–initiated campaign to eliminate filariasis from the island of Misima in the same country realized similarly impressive results, and there are many examples in Africa of similar involvement in regional antimalarial campaigns. While such initiatives at one level represent a risk reduction measure for the company's own work force, they often extend well beyond the level required for pure mitigation.

Resettlement and In-Migration

Mining often requires or results in movements of people, either out of or into a mining area. When minerals are located where people are living, resettlement is often required in order for the resource to be exploited. Resettlement may involve one family or a whole community. The IFC's resettlement standard has, by default, set the industry standard for resettlement. Aside from physical resettlement, mining can result in social, economic, and cultural dislocation, often stemming from physical resettlement, but not necessarily. For example, a haul road may inadvertently divide an otherwise connected group of people, or may impact access to resources such as productive land or water. But while it may serve to dislocate in some instances, a new road can also increase people's mobility and access to new markets for expanded economic opportunity, although these opportunities may not be experienced across the board.

There is a vigorous debate around whether mining companies should ever undertake resettlement involuntarily. There are some sites where communities have been involuntarily settled in the past, and as a result harbor resentment, particularly when relocation was facilitated by force or violence. In some cases, new owners have been faced with legacy issues from prior resettlement that cannot be ignored. While some communities are moved to enable resource extraction, there are other circumstances that result in migration into an area as a result of people seeking economic benefit through direct or indirect employment or other business opportunities, sometimes known as the honey pot effect. In-migration can be just as challenging for a company as resettlement, particularly if it is uncontrolled. Often companies expect governments to take responsibility for community planning around mine settlements or squatter camps, which, if not done adequately, can often cause great resentment.

Security and Human Rights

Human rights discourse has recently grown in strength, as have calls for business, and in particular multinational companies, to ensure that their activities do not harm the rights of others. The key international reference for human rights is the Bill of Rights, which includes the United Nations' Universal Declaration of Human Rights, proclaimed in 1948 by the UN General Assembly, and the two covenants—the International

Covenant on Civil and Political Rights, and the International Covenant on Economic, Social and Cultural Rights. As elements of the bill are ratified by governments, not companies, it has not been clear how human rights responsibilities confer to companies, particularly those operating on a transnational basis, including whether and how companies should be held accountable for their actions should they abuse human rights. The nature of global businesses has meant that it is often difficult to apply home country laws to companies operating abroad, although there have been attempts in both Canada and Australia to introduce legislation that specifically focuses on mining companies in this area. Further, the laws of some host countries may not have enshrined human rights into law, or may have weak or ineffective legislative frameworks that fail to hold companies to account for human rights violations. While the debate about the human rights responsibilities of companies continues, a recent report from the UN special representative of the secretary-general clarified the different yet complementary roles of governments and companies with respect to human rights (Ruggie 2008). The three-part framework outlines three key principles: the state duty to protect, the corporate responsibility to respect, and access to remedies. Under the recently extended mandate, "operationalizing" the framework will gain particular attention.

In the mining industry, human rights are most often raised in the context of the use of security forces to protect mining operations, which has been particularly controversial in militarized regions. Safety and working conditions have also been the subject of intense scrutiny. Attention is also being called to the human rights issues associated with environmental impacts, particularly if mines impact on the ability of local communities to establish sustainable livelihoods, as well as a number of other rights-related impacts, such as on culture (e.g., if sacred sites are impacted by mining activities), discrimination in the workplace, and the right to an adequate standard of living. While attention is often called to the negative impacts that mines have on local communities, it also needs to be recognized that companies help uphold a variety of human rights in their day-to-day business. Their contribution to economic growth, for example, can provide the support necessary for fulfillments of various economic, social, and cultural rights. Responsible mitigation of environmental impacts, close consultation with local communities, and employment procedures also help uphold different sorts of rights (ICMM 2009).

Mining, like many other economic activities, has often been conducted with little regard to the rights and interests of indigenous peoples on whose lands resources were located. Over the last few decades, indigenous peoples have been recognized as a distinct category of human societies under international law and, to varying degrees, in national law as well. The adoption in 2007 of the United Nations Declaration on the Rights of Indigenous Peoples has brought a sharper focus to this area. Harms that indigenous peoples suffer as a consequence of poor practices have included dispossession and forced relocation, destruction of culturally significant sites, loss of livelihoods, exposure to disease and "social vices" such as alcohol and prostitution, and in extreme cases, total cultural and social breakdown. It is vitally important that the rights of indigenous peoples are taken into account in any mining project. However, as human rights assessments are not yet common in the industry, particularly those that are publicly disclosed, the degree to which companies respect or otherwise impact human rights—indigenous or otherwise—is not always clear.

COMMUNITY ENGAGEMENT AND DEVELOPMENT

The common terminology for the processes undertaken by mining companies to understand and address community issues include both *community engagement* and *community development*. Community engagement is usually undertaken so that the company can better understand community perspectives. Community engagement is increasingly required by legislation, most commonly as part of project approval processes. It can also be undertaken voluntarily, as part of developing good relationships with local or host communities. Community engagement is not new in the sense that mining companies have always interacted with a range of groups around mining operations. However, in recent times, the focal point and rationale for community engagement has increasingly fallen under the banner of sustainable development and been linked to a broader range of issues than previously considered.

Community development is focused on the needs and aspirations of the community. Community development is concerned with issues of social justice, human rights, and empowerment of all groups in a community, including the most vulnerable. Community development can contribute to the management of the social impacts of a mining project, but this is not the core focus. Effective engagement is essential for community development, but community development does not automatically flow from engagement. The mining industry tends to present community development as a "mature" form of stakeholder engagement practice. A common reference in this area is the spectrum of public participation developed by the International Association for Public Participation (IAP2 2009), which shows a progression from informing and consulting to collaboration and empowerment of local communities. Under this model, community development is "high end" participation compared with public relations, which focuses more on corporate reputation through formal communication and information dissemination.

There are several stages of mine life, from exploration and project development to construction, extraction operations, and closure. In different ways, depending on the context, community engagement and development is important and relevant. Exploration is important in terms of establishing early relationships. This stage sets the scene for the future. By the time project development commences, there should have been sustained dialogue between the company and the community, various studies and assessments undertaken to determine potential impacts and benefits that might occur, and strategies for either impact mitigation or maximization discussed and agreed. The construction phase can result in significant changes and impacts to local communities. Regular dialogue remains important in these phases. It is often only after operations commence that profits are reaped and significant money starts to flow. Community development often becomes a central focus at this point, but the foundations for good development are ideally laid from the very beginning of the project life cycle. Closure and postclosure considerations should be discussed from the very outset. If community development has succeeded, a community will be well placed to deal with the changes that closure may bring.

The mining industry tends to use community development to describe activities undertaken directly or indirectly with communities in the geographic proximity of operations

that aim to achieve positive economic, environmental, and/or social outcomes for those communities. Some companies also use terms such as *community programs*, *community support*, and *social investment*. The sphere of activity varies from site to site, depending on the context, size of operation, commodity, impact of the operation, and various social and political expectations. Activities typically included in sustainability reports under community development tend to include local employment (direct or indirect through the supply of goods and services), training and skills development, provision of infrastructure (such as roads, water, and sanitation facilities), service delivery (such as health and education), employee volunteerism, donations, and nonmining-related opportunities (such as capacity building and empowerment programs). However, it is often not the activities themselves that denote community development, but the processes used. An employment program can serve company goals, but if community-directed and empowering, it can also meet the needs and aspirations of the local community. Process is key to successful community development.

Of particular interest to this chapter is the increasing alignment of various mining companies with the international development and poverty agenda as expressed, for example, by the CEO of Anglo American Corporation at a mining industry conference in South Africa: "Mining companies are not development agencies, but we are important development actors…. I believe that the mining sector will—and must—play an increasingly important role in development and poverty alleviation in the continent" (Carroll 2008). Several multinational mining companies have publicly aligned themselves to the Millenium Development Goals, a set of ambitious targets released by the United Nations to address pressing global development needs. This move toward community development is unsurprising, given that mining companies increasingly operate in some of the poorest and most marginalized communities of the world. Many of the world's leading companies, particularly those operating within a sustainable development framework, are becoming involved in the provision of infrastructure and services, such as for health and education, and economic opportunities through compensation, royalties, direct or indirect employment, and small business enterprise, as well as initiating capacity building and other community development programs. However, while the core business of mining can have adverse impacts, so too can development projects initiated to secure social license to operate, and companies should always analyze the potential for perverse consequences of development projects, including the issue of community dependency.

ADDRESSING COMMUNITY ISSUES

As a testament to the increasing voice that society has over company affairs, and the continued emphasis on the social aspects of mining, in 2009 Newmont Mining Corporation released a major community relationships review. This was undertaken in response to a shareholder resolution at its 2007 annual general meeting put forward by Christian Brothers Investment Services Inc., endorsed by the board, and supported by almost 92% of shareholders. The aims of the review were defined as follows:

- To better understand Newmont's current community relationships and their contexts;
- To assess future risks and opportunities to Newmont with regard to these relationships;
- To analyze the relevance and effectiveness of Newmont's policies, systems, and controls as they relate to community relationships; and
- To identify the impact of resources, capacity and governance on the implementation of these policies and controls (Newmont Mining Corporation 2009).

The project was undertaken by an independent working group and reviewed by an external advisory panel whose members included some who had been publicly critical of the industry's approach to dealing with communities. It was designed around case studies of five Newmont operating sites located in different continents and with vastly different contexts. In reviewing the final report, the Environment and Social Responsibility Board Committee commented on the project outcomes:

> The Company has learned much about the need to foster and maintain good relationships with governments, communities and other stakeholders, not just the ones who support the Company in its mining ventures, but also those who object to mining in general or the Company in particular. We firmly believe that the future viability and sustainability of the Company's business requires that the Company manage our community relationships more effectively and with consistency. The Company must ensure that community engagement, community relations and conflict management become a more integral component of the Company's business (Newmont Mining Corporation 2009).

In responding to the emerging agenda around mines and communities, the industry has made some significant changes at a number of levels, including both policy and planning activities to address community issues. Typically, corporate policies of leading companies in the minerals industry now explicitly address a range of broader social justice objectives, which include such aspects as local and indigenous employment, security and human rights, sustainable livelihoods, culture and heritage, ethical procurement, and stakeholder and/or community consultation. Increasingly, these policies are focused not only on mitigating the negative impacts of mining on the environment and people, but also on delivering sustainable benefits for local and regional communities. However, "good deeds" undertaken in one area in order to maximize benefits does not compensate for social harms. At a minimum, companies must focus on avoiding or mitigating social harm that occurs as a result of their activities.

At the level of process, there is a growing emphasis on the need to adopt a more participatory and inclusive approach to interacting with stakeholders, including local communities. In order to achieve this, there are a series of essential processes that should be undertaken from the outset and repeated throughout the mine life, including inclusive engagement and relationship building, social mapping and baseline studies, social impact assessment, and social risk assessment and community planning. These processes are explored in more detail in a subsequent chapter. In an effort to improve their performance in community relations, some minerals companies

are now investing considerable resources in developing and implementing management systems for this area. This entails adaptation and extension of the approach being taken in other dimensions of sustainable development, such as occupational health and safety and environmental management. The elements of these systems typically include an annual planning process, detailed documentation of procedures, regular reviews and audits against defined corporate standards, and a strong focus on information management. Examples that illustrate the "systematization" of community issues include cultural awareness protocols and training, and procedures for managing community complaints or grievances.

Structural arrangements for community relations vary according to contextual and organizational factors. Some operations include community relations departments within the communications, public relations, or external affairs functions, whereas others position community relations as part of the environmental or sustainable development departments. Many large mine sites, particularly those in developing countries or where there is a large indigenous population, have dedicated community development units or departments. Some companies detach or semidetach community development through a dedicated foundation or trust. At the corporate level, there is often a senior executive or separate department that has responsibility for social policy, as well as a board subcommittee in the area of sustainable development or social and environmental policy.

There are no known studies, either regionally or globally, that provide a comprehensive profile of organizational arrangements for community work within the global minerals industry. A notable contribution, however, was a study undertaken by Reichardt and Moshoeshoe (2003) that considered the structural arrangements and capacity for sustainable and community development undertaken in the context of the African mining industry. In the Australian context, a practitioner survey undertaken by Kemp (2004) found that people involved in community relations work were generally well educated and had considerable industry experience, but the majority did not hold qualifications in the social sciences. In addition, the group had a low level of prior experience in community relations–type work, either within or outside the industry. Subsequent anecdotal evidence suggests that the industry has recently targeted people with social science backgrounds from sectors such as international development, aid, social services, and community development while also building the social science and relational capacities of those from technical or operational backgrounds (Kemp 2009). The policy, structural, recruitment, and professional development arrangements outlined above speak to an ever-increasing focus on the institutionalization and professionalization of community relations in mining.

SUMMARY

Although many of the issues explored in this chapter are not new to the industry, and indeed have been the subject of public debate for decades, there is no doubt that the focus on community issues in mining has increased substantially in recent years. There are many reasons for this, some of which have been identified earlier, but in general the change does reflect an underlying shift in societal attitudes toward business and society, and the extractives industry in particular. For those working in the industry, it means that significant attention needs to be paid to this area, and not just by those working in dedicated community roles. In particular, engineers should recognize the diversity and complexity of communities, rather than viewing them as a collective entity; understand the direct and indirect ways in which mining operations impact communities; and appreciate the significance of contextual and cultural factors that might affect community responses to mining activities. They should also understand that the nature of community issues will change over the life cycle of a project, and they should appreciate the need for early and continuous engagement with the communities associated with their operations.

The mining industry has, to a great extent, accepted aspects of the moral arguments being made in this area. The industry also appears to be persuaded that investing in community relations activities to address these types of issues makes good business sense. A former chief economist for Rio Tinto identified two main sets of business case arguments, which he termed "show-stopping pressures" and "competitive pressures" (Humphreys 2000). The former includes disruptions and delays to projects that threaten their existence, while the latter focuses on the desire to be the "developer of choice" and the ability to move faster in the project establishment process. However, in the same paper the author adopted a less-instrumental argument: "How companies behave reflects underlying currents in the value systems of the world in which they operate. They and their employees are not something apart from civil society. They are an integral part of it and, as such, need to be sensitive to changes in its priorities and perceptions." The maintenance of the industry's "social license to operate" has become an imperative in an increasingly connected and challenging global environment.

The Panguna mine on Bougainville referred to at the beginning of this chapter has not operated since 1989. There have been suggestions in recent years that some in the local communities would like to investigate the possibility of reopening the mine. Other landowners remain bitterly opposed because of the environmental and social legacies left by the operation and subsequent conflict. As in many other locations, the possibility of using the economic benefits of resource development to provide opportunities for local communities remains attractive. Amidst the complexity, one thing is clear—in negotiating the future of any renewed mining activity on the island, the approach taken by mining companies, governments, and local stakeholders in addressing "community issues" will need to be very different from the past. To be successful, mining companies must reflect a better understanding of the needs, aspirations, strengths, and perspectives of local communities and focus on these social aspects as much as on the engineering design of any new mining proposal.

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CHAPTER 17.2

Social License to Operate

Ian Thomson and Robert G. Boutilier

ORIGIN OF THE TERM

Mining is accepted by the public at large because of the role it plays in society as a provider of minerals and metals for the public's needs and general well-being. There can be no doubt as to the historic role that mineral exploitation has played in the advance of societies, and in more recent times in the economic growth and industrialization of specific countries such as Australia, Canada, Chile, South Africa, and the United States. At the level of individual mining projects, however, this acceptance is neither automatic nor unconditional, and since 1990 has become increasingly tenuous.

During the 1990s, the mining industry found itself under close public scrutiny following a series of well-publicized chemical spills, tailings dam failures, and increasing conflict with local communities around exploration and development projects (Thomson and Joyce 2006 provide a review of this time period). In 1996, a Roper opinion poll showed mining to rate last among 24 U.S. industries in terms of public popularity, behind the tobacco industry (Prager 1997). Internationally, mining became a pejorative term in many circles and widely regarded as a problem industry that was the cause of unwanted pollution and undesirable social impacts. This pervasively negative reputation constituted a liability to the industry.

At a meeting with World Bank personnel in Washington in early 1997, Jim Cooney, then director of international and public affairs with Placer Dome, proposed that the industry had to act positively to recover its reputation and gain a "social license to operate" in a process that, beginning at the level of individual mines and projects, would, over time, create a new culture and public profile for the mining industry. The concept and terminology surfaced in May 1997 in discussions within a conference on mining and the community in Quito, Ecuador, sponsored by the World Bank, and soon entered the vocabulary of the industry, civil society, and the communities that host mines and mining projects.

CHARACTER AND DEFINITION

At the level of an individual mine or mining project (including exploration projects), the social license is rooted in the beliefs, perceptions, and opinions held by the local population and other stakeholders about the mine or project. It is therefore "granted" by the community. It is also intangible, unless effort is made to measure these beliefs, opinions, and perceptions. Finally, it is dynamic and nonpermanent because beliefs, opinions, and perceptions are subject to change as new information is acquired. Hence the social license has to be earned and then maintained.

The social license has been defined as existing when a mine or project has the ongoing approval within the local community and other stakeholders (Business for Social Responsibility 2003b; AccountAbility and Business for Social Responsibility 2004), ongoing approval or broad social acceptance (Joyce and Thomson 2000), and most frequently as ongoing acceptance.

The differentiation of approval (having favorable regard, agreeing to, or being pleased with) and acceptance (disposition to tolerate, agree, or consent to) can be shown to be real and indicative of two levels of the social license—a lower level of acceptance and a higher level of approval. Although the lower level is sufficient to allow a project to proceed and a mine to enjoy a quiet relationship with its neighbors, the higher level is more beneficial for all concerned, including the industry as a whole.

On occasion, the social license can transcend approval when a substantial portion of the community and other stakeholders incorporate the mine or project into their collective identity. At this level of relationship, it is not uncommon for the community to become advocates or defenders of the mine or project since they consider themselves to be co-owners and emotionally vested in the future of the mine or project.

EXAMPLES FROM REAL COMMUNITIES

In the spring of 2008, a tourist traveling across southern British Columbia, Canada, checked into a motel in the community of Trail and chatted with the receptionist. As part of the exchange, the receptionist mentioned how excited she was about "our" new project. Thinking this to be some improvement to the motel, the visitor asked for more information and was interested to learn that the new project had captured the imagination of the whole community. It was, in fact, a process

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to recover recyclable scrap metals at the nearby Teck Cominco smelter. Clearly, the receptionist and her neighbors had very positive feelings toward the project, the company, and the smelter; they not only approved of it, they identified with it, and in their collective minds the project was fully "licensed" to go ahead.

The same type of community identification with a company's operations can be found in small cities and towns throughout the world. In the information economy, San José, California, United States, and the surrounding vicinity identifies itself with the high-tech industry, dubbing itself "Silicon Valley." With this level of support, community and private sector organizations see their interests as highly aligned, both politically and economically.

PROBLEMS OF INTERPRETATION

The concept of an informal social license is comfortably compatible with legal norms in countries that operate under the principles of common law. However, the concept runs into difficulties in countries such as those in Latin America that operate under the principles of civil law, whereby only an official authority can grant a license. As a consequence, while communities and civil society are eager to see the social license in terms of a dynamic, ongoing relationship between the company and its stakeholders, regulators (and in turn many companies) see the license in terms of a formal permission linked to specific tasks and events in which the regulator plays the central role in granting the license.

In parallel with the above, there have been attempts in Canada and the United States to link the social license to specific tasks, and also to reformulate the concept as something that is established by the company and, once gained, becomes permanent. For example, Shepard (2008), suggests that the social license be defined as "a comprehensive and thoroughly documented process to have local stakeholders and other vested interests identify their values and beliefs as they participate in scoping the environmental impact assessment and in identifying alternative plans of operation for the project." (Notably this does not stipulate that the community, stakeholders, and other groups accept, approve, or support the project).

Neither of these capture the vision of the social license as being dynamic, granted by the community (or at the higher level, society in general), descriptive of the quality of the relationship between company and stakeholders, or involving the reputation of the company and hence industry. Moreover, the words *to operate* are sometimes confused with the strictly operational phase of a mine's life cycle when ore is extracted for processing. A better sense of the term *to operate* is to continue the project, no matter where in the mine life cycle, from the initiation of exploration through to closure.

COMPARISON WITH OTHER FORMS OF CONSULTATION AND CONSENT

FPIC is the acronym used to describe free, prior, and informed consultation and free, prior, and informed consent. Both involve close interactions with communities around resource development projects and have become critical issues for the mining industry. Superficially, there appears to be much in common between a social license and FPIC. From an operational perspective, however, while there is convergence, there are also significant differences between them, with FPIC limited in scope and time to gaining permission to enter land and/or initiate a mining project. More specific definitions can clarify the distinctions.

The free, prior, and informed consultation version of FPIC is perhaps best expressed in the Performance Standards on Social and Environmental Sustainability of the International Finance Corporation (IFC), although it is widely incorporated into national regulations for environmental impact assessment and the permitting of mining projects. In the IFC performance standards, free, prior, and informed consultation is defined as "an obligation of private sector project proponents to engage with project affected populations in a process of consultation that is 'Free' (free of intimidation or coercion), 'Prior' (timely disclosure of information—in effect before any decision is made), and 'Informed' (relevant, understandable, and accessible information)" (IFC 2006). Guidelines accompanying the IFC standards indicate that the process of FPIC should lead to "broad community support," an aspect that is substantially similar to gaining a social license. However, the FPIC process is a specific action or series of actions limited to the period before a one-time decision is taken about a project and focuses on application to the provision of information about a project and its potential consequences.

The free, prior, and informed consent version of FPIC is rooted in two international instruments: the International Labour Organization (ILO) Convention 169, also known as ILO 169, and the United Nations Declaration on the Rights of Indigenous Peoples, or UNDRIP (ILO 1989; UNGA 2007). It is important to note that these instruments are directed to governments, not the private sector, and are limited to interactions with indigenous and tribal peoples. Nevertheless, they are of concern to the private sector because companies are typically the advocates for development projects and frequently introduce the idea of a mining project before any such discussion has taken place between government and the indigenous people. Therefore, as the initiators, companies should take care that discussions with indigenous communities do not proceed ahead of consultation between government and the indigenous people, neither in the timing nor direction of the content.

ILO 169, which has been adopted by 20 nations (the majority in Latin America), requires governments to "consult the peoples concerned…whenever consideration is being given to legislative or administrative measures which may affect them directly;…establish means by which these peoples can freely participate…at all levels of decision-making"; and stipulates that "the consultations…shall be undertaken, in good faith…with the objective of achieving agreement or consent to the proposed measures" (Article 6). With respect to mineral resources and mining projects, ILO 169 is quite specific in requiring governments to consult indigenous and tribal people "with a view to ascertaining whether and to what degree their interests would be prejudiced, before undertaking or permitting any programs for the exploration or exploitation of such resources pertaining to their lands" (Article 15).

UNDRIP includes a much stronger statement on free, prior, and informed consent in requiring governments to "consult and cooperate in good faith with the indigenous peoples concerned through their own representative institutions in order to obtain their free and informed consent prior to the approval of any project affecting their lands or territories and other resources, particularly in connection with the development, utilization or exploitation of mineral, water or other resources" (Article 32.2).

In summary, FPIC is limited in scope and timing to a period before exploration or development can take place, is linked to a one-time decision-making process, and, in the case of ILO 169 and UNDRIP, is an obligation of governments. In contrast, the social license to operate is an expression of the quality of the relationship between a private sector project/company and its neighbors. In terms of mining, this begins with first contact at the initiation of exploration and continues through the entire life of a project, which, if successful, includes mine construction, mine operation, closure, and increasingly into postclosure.

WHO GETS SOCIAL LICENSE AND WHO GRANTS IT

A social license is usually granted on a site-specific basis. Hence a company may have a social license for one mine or operation, but not for another. Usually the company has to have an operation that affects other groups, organizations, or aggregates of individuals. The bigger the effects, the more difficult it becomes to get the social license. For example, an independent Web-site designer working from a computer in his bedroom will normally get an automatic social license in most communities. A mining company wanting to relocate an entire village faces a much bigger challenge.

The license is granted by the *community*, a term used generically throughout this chapter to describe the network of stakeholders that share a common interest in a mining or exploration project and make up the granting entity. Identifying that the entity is a *network* makes salient the participation of individuals, groups, or organizations that might not necessarily be part of a geographic community, and recognizes the reality that communities are inherently heterogeneous. Calling them *stakeholders* means the network includes individuals, groups, and organizations that are either affected by the operation or that can affect the operation. For example, ranchers who would have to accept a land swap involving part of their pastureland would be affected by a proposed mining operation, without having much effect on it, provided they accepted the deal. By contrast, a paramilitary group of insurgents or an international environmental group that might attack the mine site, each in their own way, would *affect* the operation, without being affected much by it. They would be stakeholders too.

Use of the terms *community* and *stakeholder network* implies that the license is not granted by a single group or organization. It is a collective approval granted by a network of groups and individuals. Therefore, the existence of a handful of supporters amid a larger network of opponents would mean that the license has not been granted.

The requirement that the license be a sentiment shared across a network of groups and individuals introduces considerable complexity into the process. It invites the question about whether a coherent community or stakeholder network even exists. If one exists, how capable is it of reaching a consensus? What are the prerequisites a community must have before it becomes politically capable of granting a social license? These complexities make it more difficult to know when a social license has truly been earned. The complexities of the granting process are discussed later in this chapter.

COMPONENTS OF THE SOCIAL LICENSE

Accumulated experience supports the proposition made by Thomson and Joyce (2008) that the normative components of the social license comprise the community/stakeholder perceptions of the legitimacy and credibility of the mine or project and the presence or absence of true trust. These elements are acquired sequentially and are cumulative in building toward the social license. The mine or project must be seen as legitimate before credibility is of value in the relationship, and both must be in place before meaningful trust can develop.

THE BUSINESS CASE FOR INVESTING IN IT

Earning a social license to operate plays havoc with financial and engineering schedules. It involves a fuzzy relationship-building process that does not fit comfortably into technical planning frameworks that are built around concepts like material objectives, deadlines, and deliverables. So, why is this necessary?

The same question could be asked about safety. Doing safety training does not seem to directly contribute to the bottom line, so why bother? Several decades ago, health and safety was viewed in the same way that many people view social responsibility today. It was the responsibility of a specialist in the company. Accidents were accepted as the aspect of the job that justified higher pay. Today, safety is everyone's responsibility. The leading companies have systems in place to identify dangerous situations so that accidents can be prevented before they happen. The same will eventually be true for the social license. The leading companies will establish positive stakeholder relations before complaints and controversies erupt. Seeing the earning of a social license as an essential part of the project requires a higher-level perspective that is usually only the concern of executives and members of the board. However, project planners need to understand this bigger picture too, in order to allocate sufficient time and resources to earning a social license. The next two sections deal with this bigger picture.

Dependence on Resources Controlled by Stakeholders

It is an axiom of the mining industry that mines are made where mineral deposits are found. This truism reflects the fact that economically viable mineral deposits occur in relatively rare situations. As a result, the mining company has to adapt to the location of the deposit and engage with the people who live there and who consider the location part of their "backyard." By way of contrast, manufacturing industries can select the location for a new plant or facility based on a variety of factors, including the willingness of the local community to accept them. Indeed, during the first part of the decade, there was open competition between communities in North America to host new manufacturing and assembly plants and wholesale distribution depots.

Mineral deposits always have neighbors. Sometimes they have a village right on top of them. Roads always go through someone's land or affect someone's traffic or hunting ground or watershed. Even the most remote mines have to rely on the closest towns as supply centers and transportation links. The many ways that mines affect their neighbors make those neighbors stakeholders. However, the effects go in the opposite direction too. The mine can be affected by road closures, road use regulations, water rights negotiations, protests, blockades, and many other tactics that restrict access to essential resources. Even with the legal right to mine from a government, a company can still be stopped by a community determined to withhold the social license to operate. For example, in Peru alone, at the time of this writing, there were more than 80 sites where communities had prevented mining companies from exploring or exploiting mineral deposits.

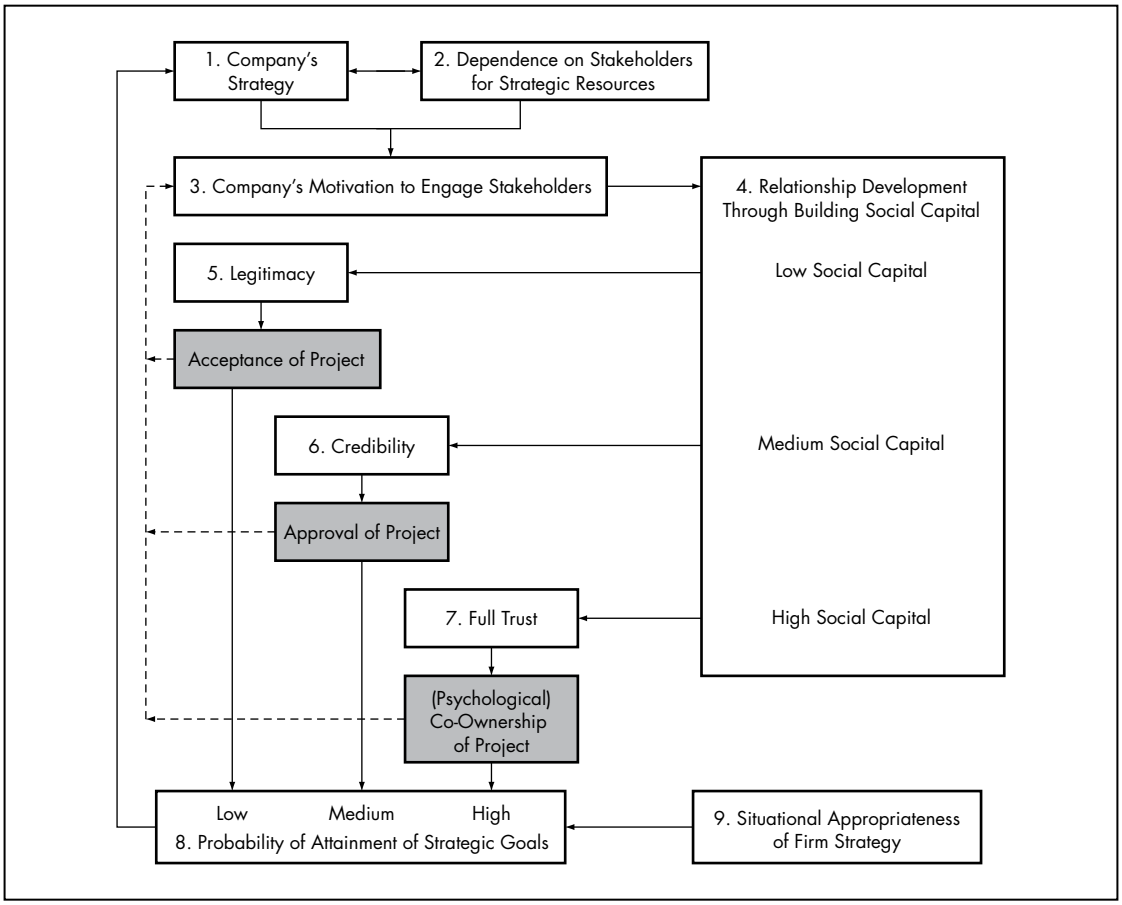


Figure 17.2-1 Adaptation of resource-based view to include process of gaining a social license to operate

The management theory that covers this type of challenge is called the *resource dependence* view of competitive advantage and the firm (Barney et al. 2001 provide a review). It explains why some companies thrived while others struggled. The theory emphasizes that companies need good relationships with those who control essential resources that are scarce, non-substitutable, or imperfectly imitable. That can include things like labor skills, some materials, risk financing, access to some markets, special legal permits, and, for mining, access to land and mineral deposits.

The Role of Strategy

Figure 17.2-1, shows the resource-dependence view augmented with acknowledgment of the role of strategy in setting priorities among resources. The chosen strategy (box 1) determines which resource-controlling stakeholder (box 2) must be dealt with. At the same time, because strategy is often based on existing strengths, the stakeholders the company can deal with best or worst influence what strategy will be chosen. Together, these factors identify the stakeholders with whom the company must engage (box 3).

Different Strategies at Different Stages of the Mine Life Cycle

Companies have different strategies for financial success at different stages in the life cycle of a mine. In the earlier stages of exploration and feasibility, the strategy is to buy the rights inexpensively, prove the richness of the deposit, and sell the property to a mine operating company. The financial gain depends on demonstrating the quality of the ore deposit (grade, tons, mineralogy, feasibility, etc.) and finding a motivated buyer who has the capital and knowledge needed to develop the property into an operating mine.

In the later stages, operating companies pursue a strategy of getting the property into production as quickly as possible in order to reap the financial rewards of selling the minerals from the site. Financial gain depends much more directly on the prices that mineral processors and industrial manufacturers are willing to pay for metals.

Mine builders want to buy properties that are fully permitted. They avoid properties at which stakeholders threaten to block construction or access to the deposits or other vital resources (e.g., water, power). Unfortunately, too many exploration companies think they can ignore or hide these kinds of stakeholder problems.

For example, from late 2003 to late 2004, the stock price for Monterrico Metals, a mineral exploration company, rose from around 150 to 600 pence per share on the London Stock Exchange. Drilling results had shown that the company had a world-class copper deposit at its Rio Blanco property in northern Peru, and the company was actively seeking a buyer to develop a mine. During 2005 and 2006, however, local peasants demonstrated against the development. A group of academics and British parliamentarians known as the Peru Support Group went to Rio Blanco to investigate. Their findings noted a permitting dispute between two Peruvian government bodies and, more particularly, widespread community

opposition (Bebbington et al. 2007). The news of the apparent absence of a social license was followed by fall in share prices with their value ultimately dropping almost 70%. In early 2007, a Chinese mining company bought control of the company amid complaints from some Monterrico investors about the low price being paid. At the time of this writing, the project continues to be mired in conflict and there is no indication that a mine can be developed in the near future. In retrospect, the Monterrico investors should be satisfied with the price paid by the Chinese, but they have also learned the hard way that a social license to operate has a financial value when a buyer is sought to develop the property.

Identifying Stakeholders Based on Strategy

Boxes 1 and 2 in Figure 17.2-1 cover the criteria for identifying stakeholders. Those who will be *affected by the company* are implicated by the company's strategy (box 1). Those who have the potential to *affect the company* are covered by the company's dependence on others for resources (box 2). The combined set represents the stakeholders with whom the company should establish relationships.

When an exploration team determines that there is an interesting deposit, the activities in the first three boxes transpire. The team must identify by name local stakeholders who make up the community and then determine who speaks for each stakeholder group as an opinion leader in relation to the project. When the exploration team first introduces itself to those stakeholders, the social capital in the firm-stakeholder relationship is very low (box 4). The social capital dimension of stakeholder engagement and relationship is explored later in this chapter. For now it is sufficient to think of social capital as the collaborative capacity in the relationship.

The pivotal role of relationship building (box 4) implies that a policy aimed at keeping a low profile raises the sociopolitical risks associated with the project. This is especially true during the earlier stages of the project. If the company does not engage with all elements of the community, there is no way the community can collectively grant a social license. If relationship development is delayed, the social license will also be delayed, usually at a high financial cost to the company. Thus, anyone who argues that there is no time for a lengthy relationship-building process with the community is unwittingly advocating for lengthy delays to be imposed by the community.

Building Social Capital and Earning Higher Levels of Social License

Engagement entails the building of a relationship with the stakeholders, who are often already organized into a geographically based network (the community). If the relationship proceeds well, the community may view the company as legitimate (box 5 in Figure 17.2-1) and may grant a provisional/conditional social license (gray box, "acceptance of project").

Acceptance of the project can heighten the company's motivation to engage with the stakeholders, provided the company realizes that this is an investment in raising the probability that corporate strategic goals will be attained. Because not all companies realize this, the line feeding back to a company's motivation (box 3) may be dashed. Assuming the company does choose to build more social capital in the relationship, the community will come to see the company as credible (box 6). Consequently, it will grant full approval of the project (gray box, "approval of project"). Ideally, the relationships should evolve to this point before the company embarks on the feasibility and permitting stages of a project.

Probabilities of Success

If the company continues to put effort into the relationship (dashed line to the left in Figure 17.2-1), it can raise the probability of realizing its strategy and lower the inverse probability of sociopolitical risk. Taking the relationship to the level of full trust (box 7) yields exponentially more benefits for the project and the parent company. At that point, the social license reaches the level of "psychological co-ownership." This will result in the success of the project, provided that the company chose a winning strategy in the first place. Outside factors, however, like the price of metals, can always stop a project (box 9).

Figure 17.2-2 shows how the process of earning a social license to operate impacts a company's bottom line. *No access* means "red ink." Good access makes "black ink" possible, provided all other aspects of the project are positive. Although the relationship-building process might sound at first like a nonessential nicety, it is in fact a make-or-break factor for the successful development of a project. Every continent has its "off limits" mineral deposits that represent projects which failed at the relationship-building stage.

Dynamics of Stakeholder Influence in Networks

Some stakeholders have legal powers to affect the company. Governments, for example, can often act unilaterally to impose conditions or restrictions on mining activities. Most stakeholders, however, have their effects through political maneuvering. Some have direct control over the resources the company needs. They can exert influence by restricting access to those resources through tactics like prohibitive pricing, blockades, and boycotts.

Still other stakeholders have no legal power and no direct control over essential resources. They can, nonetheless, form alliances that pressure those with such direct power to act in solidarity with them. Environmental and human rights groups, for example, usually fall into this category. They often use tactics such as putting pressure on senior governments to change mining regulations or on municipal governments to withhold resource access (e.g., water) or on consumers to boycott the product (e.g., the blood diamonds campaign).

Because political alliances among stakeholders often determine their level of influence, it is important to know about those networks in order to identify the stakeholders who deserve more attention. Methods for acquiring this kind of information are discussed later in this chapter. For the moment, it is simply important to recognize that the sociopolitical dynamics in the stakeholder network do affect the project's support among stakeholders. Thus, acquiring a social license to operate is partially a sociopolitical task.

Resource Access Restriction at the Corporate Level

Stakeholders can also have influence over a project in less direct ways. For example, they can exert pressure through control of access to local resources; they can also affect access to the resources that the parent company needs. The local project may or may not be constituted as a subsidiary company of a larger company. In any case, the "parent(s)" sponsoring the project also needs resources. These include equity or debt

financing, new highly trained employees, access to minerals at other sites in the future, and willing customers.

Stakeholders sometimes restrict the parent company's access to those resources by launching national or international campaigns that damage the company's reputation. The effect on share value, for instance, can restrict access to equity financing, which is particularly damaging to junior exploration companies, but can also affect operating companies. A company with a negative reputation will also have more trouble attracting talented employees. If the international campaign includes a consumer boycott (e.g., the dirty gold campaign), the pool of willing customers will shrink and income could be depressed. A bad reputation in community relations can also limit the number of sites a company can have access to in the future, no matter whether they are greenfield sites or acquisitions.

At the international level, the absence of a world government to regulate mining has led to the creation of a world governance regime instead. This network of international agreements, standards, and principles is supported by organizations that frequently finance mining projects, including the IFC and the banks that have adopted the Equator Principles (EPFIs 2006) as criteria for project financing. They stand ready to legitimize the complaints of local stakeholders and impose a variety of sanctions against the offending company. Again, these affect the parent company's reputation and access to money, talent, willing customers, and even further national legal permits.

In summary, earning a social license to operate is necessary for protecting the financial viability of the project and the company. It is as deserving of time, money, and career prestige as drilling to estimate ore reserves or preparing quarterly financial statements.

PHASES OF EARNING A SOCIAL LICENSE

A social license has distinguishable levels. At the same time, the process of moving from one level to another can be thought of as a smooth gradient of continuous relationship improvement through increasing social capital.

Figure 17.2-2 shows the four levels of social license and the three boundary criteria that separate them. The levels represent how the community treats the company. The boundary criteria represent how the community views the company, mostly based on the company's behavior. These transition criteria were derived from language repeatedly heard from communities themselves. Although the terms *legitimacy*, *credibility*, and *trust* emerged out of years of conversations while consulting with mining communities, social science and management literature has been shown to support these common-sense views.

The levels and boundary criteria are arranged in a hierarchy. It is possible to go both up and down the hierarchy. For example, if a company loses credibility, approval will be withdrawn and the project will hobble along on acceptance only. If a company loses legitimacy, the project will be shut down. If full trust is gained, the community will support and protect the project as its own.

Withholding/Withdrawal Level

Starting from the base, the rejection level of a social license is the worst-case scenario. This is when the community stops progress on the project. Many mineral deposits cannot be exploited because the community does not grant any level of social license to proceed.

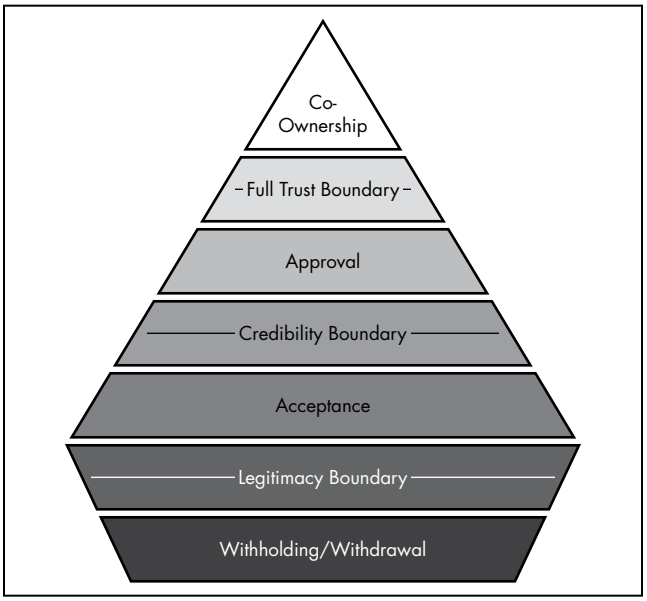


Figure 17.2-2 Levels of social license with boundary criteria between them

The withholding/withdrawal level is shown as narrower than the acceptance level above it in order to symbolize the possibility that, globally, more projects are accepted than rejected. However, at present, this is a supposition. The empirical studies to determine the correct width of each level in Figure 17.2-2 have yet to be done.

Legitimacy Boundary Criterion

The academic meaning of the term *legitimacy* is evolving as interest grows in how organizations gain or lose it, and what they can do with it once they have it (Deephouse and Suchman 2008 provide a review). Knoke (1985) defines legitimacy in the context of stakeholders and politics as "the acceptance by the general public and by relevant elite organizations of an association's right to exist and to pursue its affairs in its chosen manner." This adequately summarizes the bare minimum of legitimacy even when the company has no social license. Suchman (1995) offered a typology in his definition, which has become a touchstone for scholarship on legitimacy. He proposed a three-category typology of "legitimacies." They were (1) pragmatic, based on audience self-interest; (2) moral, based on normative approval; and (3) cognitive, based on comprehensibility and taken-for-grantedness. Each of these has aspects that continue to be important through higher levels of social license. For each type, the community asks itself questions such as the following:

- What do they want, and what is in it for us? How will the consequences of their actions affect us? How will the project affect the environmental resources we absolutely depend on for survival? If we cannot know that for sure, then can we at least discern whether they will be responsive to our concerns, or even share decision making with us? (Pragmatic legitimacy)
- Does anyone in authority recognize/respect us? Are they conforming to our social, cultural, or political norms? Have they followed the specific norms for approaching us with their proposal? Will the consequences of their

activity promote the general welfare of the community, according to our (i.e., the community's) own values? (Moral legitimacy)
- Does what they say make sense, or is it confusing or strange? Has this been done anywhere else? Are their proposals routine practice, or is this uncharted territory? Does that company have the capacity to do what they say they could do? In the case of an expansion of an existing operation, it is precisely the absence of questioning that indicates cognitive legitimacy. This occurs when the presence of the company and its activities are already taken for granted. They are seen as an inevitable part of the community economy. (Cognitive legitimacy)

In addition, particularly for mining exploration projects, the community may also ask questions about legal legitimacy. Do they have the legal permits and permission to do what they want to do?

The behaviors that lead to a company's gaining legitimacy are associated with spreading awareness about the company and what it does, listening to community concerns, and following the official and unofficial local norms, customs, and practices. The company should also have legal status nationally, inform the general public about how the proposed approach has worked for the benefit of other communities elsewhere, and solicit participation in planning and decision making in order to allay fears about the company implementing an arbitrary, uninformed, or high-handed development process.

Failure to engage all segments of the community (e.g., young, old, men, women), to inform them and to solicit their opinions, is often seen as evidence of illegitimacy by those excluded. It is normally important to communicate directly with the bases and not rely solely on leaders. At the same time, following local norms and protocols requires an understanding of, respect for, and use of local social structures and decision-making processes.

Acceptance Level

When legitimacy is established, the community response is that they will listen to the company and consider its proposals. If, by their own standards, they have no reason to doubt the company's credibility, they may allow the project to tentatively proceed. This constitutes the acceptance level of social license. It is a minimal objective for any company.

As can be seen in Figure 17.2-2, the acceptance level is bounded by the legitimacy criterion and the credibility criterion. This represents how acceptance requires that the company's legitimacy must be firmly established and its credibility should at least not be damaged.

Credibility Boundary Criterion

Credibility is the foundation of trust. When a company is regarded as credible, it is seen as following through on promises and dealing honestly with everyone. There is little danger of the company saying one thing one year and a different thing the next. Moreover, the company's policies are the same for everyone that it deals with.

The promise-keeping aspect of credibility is related to the company's responsiveness to community concerns and requests. Legitimacy can be earned by just listening; credibility requires doing something about what has been heard. Community members need to see action following their discussions.

An essential component of credibility comes from openness and transparency in the provision of information and decision making that demonstrates the company to be consistent in the way it treats different groups. Alternatively, transparency reveals the principles that determine why one group might be treated differently than another, reducing the risk of feelings of discrimination or marginalization.

The community asks itself questions such as these:

- Will they deliver on their promises? Are they living up to their responsibilities? Are they acting yet on what we have already said concerns us? Do their promises sound unrealistic?
- Do we understand why they treat some groups differently and how that is consistent behavior?
- Are they keeping any secrets? Do they avoid contact or avoid answering certain questions? Do they acknowledge difficulties or do they often sound glib?

Establishing credibility involves the cycle of listening to the community, responding with information and proposals, and implementing approved proposals. Demonstrations of responsiveness and principled action on a number of quick, short-cycle initiatives help to build a reputation for credibility in the early stages. There is pressure to show immediate benefits, as opposed to long-term promises. The planning horizons in rural communities seldom reach beyond one agricultural cycle. By contrast, exploration teams may make promises to be fulfilled by the future buyers of their properties.

In communities, credibility rests in a person's willingness to keep one's word. This is a significant challenge for the credibility of an exploration company that does not expect to be around when its promises are to be fulfilled. Many communities are suspicious of anyone who intends to transfer responsibility for stated promises to some unspecified, hypothetical person who may or may not even appear in the intangible marketplace for mineral deposits.

To earn credibility, the company should make and keep short-term promises. This is best done by using participatory processes to identify community priorities that the company can help make real. It helps to bring in third parties to verify the truth of company statements and to empower the community to be the watchdog on the company's activities (e.g., community-based environmental committees). Formal agreements and contracts give structure; manage expectations; avoid misunderstandings; and define rules, roles, and responsibilities. Providing false or incomplete information or failing to deliver on promises will quickly destroy credibility and lead to questions as to whether the company is even legitimate. Acceptance can be rapidly withdrawn.

Because companies usually carry so much economic clout, they are often expected to provide the kinds of services that are provided by the government in the developed world (e.g., education and health care). In contexts where governments have not provided these, a company's reluctance to talk about them may be taken as a sign that the company cannot be trusted to make an effort around basic community needs. The best way to address these expectations is with programs to help governments develop the capacity to shoulder such responsibilities. This shows a concern for the problems and a respect for local institutions. It avoids the trap that occurs

when governments decide they do not need to do anything because the company is taking care of the community.

Even when relations with the community are good, there may still be attacks on the credibility of exploration companies from civil society sources that warn about the broken promises or social or environmental problems experienced by other communities that accepted mines. Even if some of these stories are false or exaggerated, the sources are credible because they do not appear on the surface to have anything to gain from misrepresenting the facts. When it is their word against a company's word, the company appears to have the greater motive to distort the truth.

Approval Level

When a company has established both legitimacy and credibility, a community is likely to grant approval of the project. This means the company has secure access to the resources it needs. The community regards the project favorably and is pleased with it. The community is now resistant to the "us-versus-them" rhetoric of antimining groups.

This level of social license represents the absence of sociopolitical risk. However, it is really only at the threshold of opportunity. Moving toward the top of this level, as represented in Figure 17.2-2, a company can begin to reap some positive benefit from their relationship building, such as strong community support for expansion of operations.

Full-Trust Boundary Criterion

In management research, trust has been shown to be important in relationships between and within organizations. It is just as important in relationships between organizations and stakeholders. Trust is especially important when bridging the boundary between businesses and civic sector organizations, which include many community groups.

Many typologies of trust have been proposed, some of which range from weak or superficial trust to deeper, more comprehensive trust. As used here, the term *full trust* means a broader and deeper trust.

Credibility is a basic level of trust related to honesty and reliability. A full trust relationship is one where there is a willingness to be vulnerable to the actions of others. Communities that have a full level of trust in a company believe that the company will always act in the community's best interests. Before moving to that level of trust, a community asks itself questions such as these:

- Have they fulfilled their promises repeatedly and consistently?
- Did they handle the unexpected problems in a way that showed they had our best interests at heart?
- Did they share power in a partnership approach?

In order to gain full trust, a company has to go beyond fulfilling its promises to jointly envisioning new development goals with the community. The company will have initiated activities to strengthen the community's ability to plan and achieve its goals for the future. For example, this can involve training in project management for local nongovernmental organizations (NGOs) or technical training for government employees. The company has managed to explain its decision-making principles to the degree that people accept rejection of proposals calmly because they trust the decision makers.

When fulfillment of a promise will be delayed, the company explains why and asks the community for ideas about how to overcome the obstacles to fulfillment. Whenever possible, the company involves community groups in decision making to ensure that no opportunity for capacity building and economic development is missed. The company hands over responsibility for specific aspects of the project to the community so that it takes ownership of both the opportunities and the risks.

Trust is the much desired quality in a relationship that is only earned over time and typically emerges as the product of shared experiences that have positive outcomes for all parties. Trust is hard to earn, easy to lose, and very difficult to recover once lost.

Co-Ownership Level

When the community sees the company as having full trust in it, the community takes responsibility for the project's success. Psychologically, both parties come to view it as a co-ownership arrangement. The limits of the responsibilities of each party are clear, as are ultimate decision criteria.

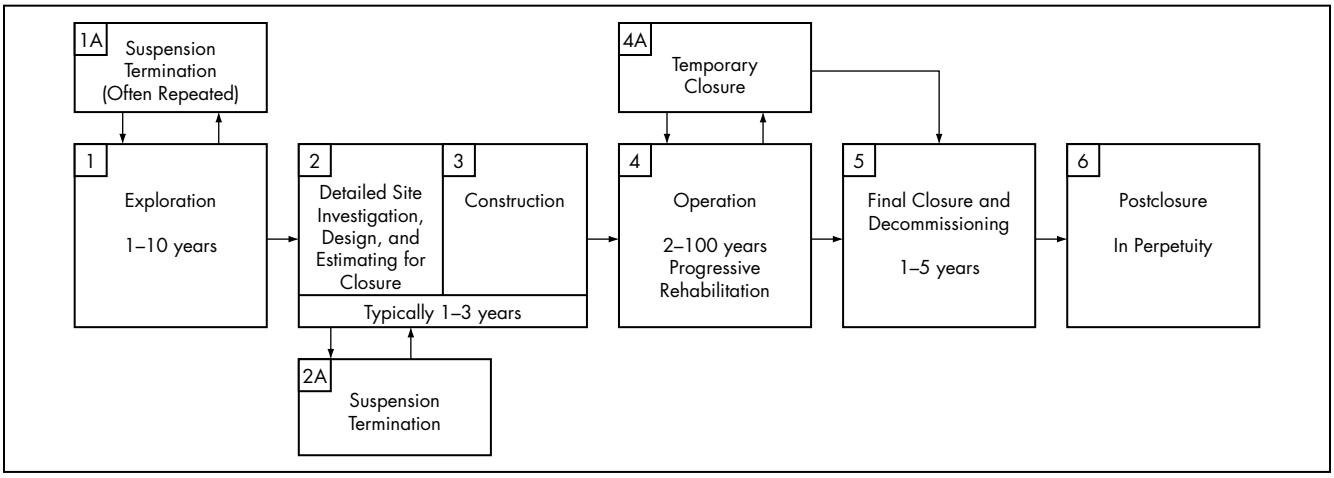
At this co-ownership level of social license, the company becomes an insider in the community social network. In social sciences, social identity theory describes this level of relationship as the dissolution of the us–them boundary (Williams 2001). Working closely together, the company and community often develop creative solutions to all types of challenges. If outside stakeholders, like the national government or an international NGO, move against the interests of the company, the community will mount a campaign in defense of the company. There have been cases where community members have traveled to foreign countries to challenge false information being promoted by NGO critics. In another case, community members marched on the national capital to make a point to politicians who were proposing a new tax that would affect the mine.

Few companies have taken their community relations to the co-ownership level. Many have difficulty seeing beyond the immediate transactions to the much greater benefits of establishing strong collaborative relationships. Nonetheless, as awareness of the potential benefits grows, more companies are attempting to win a higher level of social license.

MINE LIFE CYCLE AND FIRST IMPRESSIONS

The level of social license that a company enjoys can fluctuate throughout the life cycle of a project. Figure 17.2-3 shows the six stages of the mine life cycle: (1) exploration, (2) feasibility, (3) construction, (4) operation, (5) closure, and (6) postclosure. Obviously, the earning of an initial social license to operate has to be complete before construction begins (stage 3). However, in the past many companies have treated it as an afterthought sometime during the feasibility stage (2). Experience shows that ignoring the social license matter during exploration only creates problems in later stages.

In the first two stages, the key issue affecting the social license is access to land for exploration and feasibility studies. The exploration stage is especially important because that is when first impressions are made. It is a challenging period that can affect community relations during the whole mine life cycle. A positive relationship can lead to the early acquisition of a social license. If that is maintained, it can create the tolerance and mutual understanding needed to deal with conflicts and different interests during the whole life of the mine. Conversely, bad relationships during exploration can lead to social tensions, conflict, and premature shutdown of the project.



Source: Adapted from Gadsby and Hodge 2003.
Figure 17.2-3 Stages of the mine life cycle

A DIFFERENT STARTING POINT IN EVERY COMMUNITY

First impressions are important, but the preexisting disposition to trust also matters. Some societies are less trusting than others. Figure 17.2-4 shows percentages of generalized trust by population in selected countries from a survey by Delhey and Newton (2005). Obviously, earning a social license in Brazil would be much more difficult than in Norway. The reasons for these differences are not entirely clear, but there are suggestions that lower trust of people in general might be associated with fractionalization in terms of income inequality and political diversity and with higher levels of commitment to family, as opposed to the collective society.

In addition to these cultural factors, each community has its own history. Some have had negative experiences with previous operators of a mine or exploration project. Some have had bad experiences of a more general nature, such as exploitation by foreign companies or governments during colonial times. In many developing countries, there is still suspicion of corporations from developed countries as a result of the Third World debt crisis that followed World Bank policies aimed at encouraging developing countries to imitate rich countries.

All of these factors affect how long it will take to earn a social license in any particular community. Just because it took 18 months to earn enough of a social license to buy land in one community, it cannot be assumed that the next community will require the same amount of time.

DIFFERENT CULTURES, DIFFERENT EXPECTATIONS

There are profound cultural differences between corporations and communities, especially communities that are rural and operate at subsistence level. Companies tend to expect communities to be coherent, cohesive, rational economic actors with a keen sense of judicial impartiality regarding the issuing or revoking of a social license. However, cultural differences can radically redefine the meaning of these qualities. The community's perceptions are filtered through their world view. For example, the Dongria Kondh tribe, in the state of Orissa, India, consider their mountain sacred and view the bauxite mining proposal by Vedanta Resources as an assault on all that is good and holy (Survival International 2008). In these cases, the company must decide if it should withdraw, and avoid the risk of conflict, or invest in working with the community and other stakeholders over a longer period with the hope that a creative compromise can be found.

Many rural communities work on the tribal principles of personal authority and obligations to relatives. Such communities may see profit making as proper for the benefit of a family or community but improper for the benefit of any other body, from individual to company to state. In these societies, all companies, except family and community-owned businesses, start out as morally illegitimate because of the way they treat money and resources.

In many rural communities, nepotism is seen as commendable, not condemnable. Being seen as legitimate entails a willingness to treat community members like family. The company is expected to hire on the basis of local residence, rather than qualifications. Becoming qualified is seen as something that comes after being hired, rather than before.

Time is a common point of difference. Companies want things to happen according to a firm schedule, usually related to a budget. Communities, on the other hand, react organically, take time to reach decisions, and typically have no concern for the schedule set by the company.

THE LOW-PROFILE FALLACY

Earning a social license is complicated work. Occasionally companies are tempted to opt for a strategy that seems easier. They decide to keep a low profile or "stick to their knitting," especially in the exploration stage. This, of course, makes establishing legitimacy, credibility, and full trust quite impossible. The following anecdote describes the consequences.

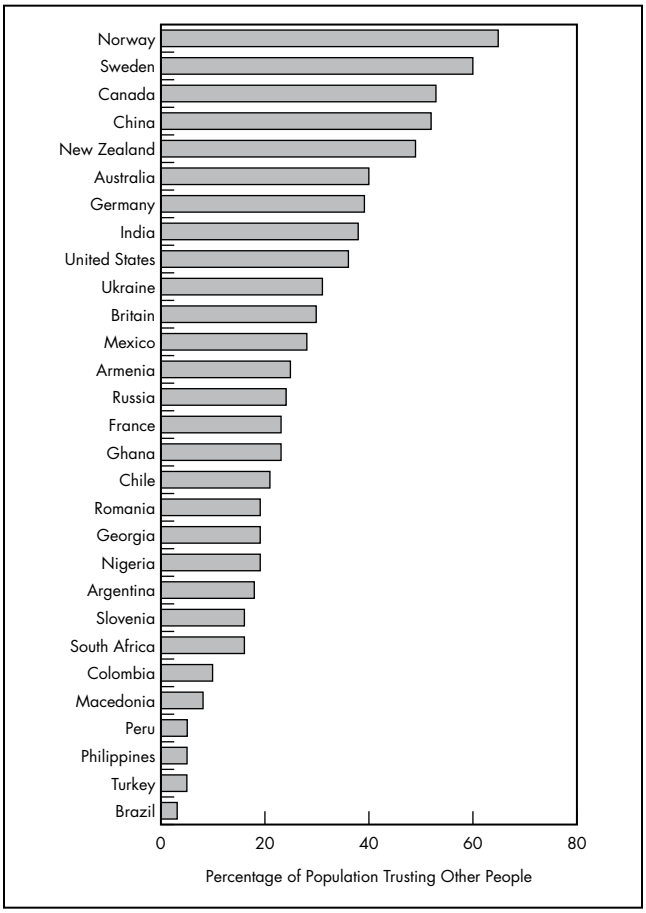
> In July 2002, Meridian Gold, Inc., completed its acquisition of Brancote Holdings and took possession of the Esquel gold deposit, an exploration project located in the Cordón de Esquel in northwestern Chubut, Argentina. At the time, the concept of developing a mine at the Esquel gold deposit enjoyed general acceptance in the nearby town of Esquel as a result of successful efforts by Brancote to develop a positive relationship with the community. However,

popular sentiment changed very quickly as Meridian failed to respond to requests for information and assistance, and as local permitting authorities expressed a lack of confidence in the company and its actions. Over the following months, the situation became more polarized with the formation of community-based groups organized to seek information and clarification from the company on various issues. In the absence of replies that satisfied these community groups, an organized opposition to the idea of a mine formed in the town. Various protests, marches, and meetings took place with a common theme of "No to the Mine." Attempts to reach a solution through dialogue led by the Church failed to gain any momentum and, under increasing pressure from the local population and growing national and international opposition to the project, the mayor of Esquel authorized a plebiscite on the future of the proposed mine development. On March 23, 2003, the population of Esquel voted 81% (of a 75% turnout) against the mine. In a subsequent independent investigation by Business for Social Responsibility (2003a), it was noted that the Esquel gold mine project was characterized by a lack of engagement by Meridian with the community. This lack of engagement and the inability to have a meaningful dialogue about potential risks (and benefits) was the dominant factor influencing the community not to support the mine. In practice, the company did not provide timely and useful information, and on occasion made it difficult to obtain information, for example when the company refused to provide copies of the environmental impact study, a public document, available in CD format. What community members wanted was to feel informed and listened to, to participate in a real dialogue, and for the company to be responsive to their concerns. In short, the community wanted a partnership, something that the company never attempted to deliver (Business for Social Responsibility 2003a). The collapse of the Esquel project was financially disastrous for Meridian, with the company obliged to take a write-down of US$542.8 million. As of June 2009, the project remained frozen with a majority in the community firmly opposed to any form of mineral exploration or mine development.

Such scenarios are becoming more frequent. They are not always caused by a failure to talk to the community stakeholders. In most of these types of cases, the exploration teams have held community meetings. They have explained their plans. The community representatives showed approval for the projects. So what went wrong? Why do things so often fall apart?

Practical experience points to a number of problems created by the exploration team themselves. The most common problems are as follows:

- A failure to invest fully in relationship building, an approach generally justified by the knowledge that most exploration projects fail. Explorers often choose to delay full engagement with stakeholders until the "right moment" or when there is certainty that the project will go ahead.
- Selective engagement. Talking only to those who are already friendly toward the project creates a group marginalized from information and participation that may well respond by rejecting the project.
- Undermining the company's credibility by failing to deliver on promises already made or by providing false or incomplete information.
- Failing to understand the internal structure of the community and existing relationships between stakeholders and thereby unwittingly creating or amplifying rivalries or divisions that lead to conflict.
- Failing to respect and listen to the community and relying on legal positions and permits to operate.
- Failing to maintain close contact with the community when there is a change in project management or ownership, thereby weakening the continuity needed to sustain the social license.
- Misunderstanding, usually overestimating, the quality of the relationship with the community by confusing acceptance with approval, and cooperation with trust.



Source: Data from Delhey and Newton 2005.

Figure 17.2-4 Percentages of generalized trust for selected countries

The last pitfall is particularly pernicious. Individuals and groups will cooperate with the company for many reasons, including courtesy, a desire for gain, a perception of having no alternative, or, as is often the case with authorities, a sense of obligation. Cooperation for these motives does not necessarily require a trusting relationship.

MAINTAINING A SOCIAL LICENSE

The four-level-phase view of earning a social license is simplistic. It helps mark progress and breaks the process into a practical, phase-like approach. It also highlights important pitfalls. However, every community and stakeholder network is unique in some ways. There is no one-size-fits-all process for earning a social license. There is always a need to adapt and adjust to the specifics of each case. For that reason, it is also important to understand the theoretical process underlying the phases. Any necessary adjustments and adaptations can then be derived from first principles.

Continuous Building of Social Capital

The movement from legitimacy through credibility to full trust is a process of building and balancing the social capital in the relationships between the company and the members of the network.

Definition of Social Capital

Adler and Kwon's (2002) definition of *social capital* says, "Social capital is the goodwill available to individuals or groups. Its source lies in the structure and content of the actor's social relations. Its effects flow from the information, influence, and solidarity it makes available to the actor." In other words, the effects of social capital flow from what it is, namely, information, influence, and solidarity.

Examined more closely, social capital can be seen as a set of benefits that network members can enjoy because of their unique patterns of connections in a network. (In social network analysis, the technical terms for connections in a network is *ties.*) The benefits fall into three general categories. First, social capital is access to the *information* that flows through the network. People who act as bridges between otherwise unconnected clusters of network members are especially likely to enjoy this type of benefit. They have "bridging" social capital. Second, social capital is *solidarity* in support of group norms and shared identity. For example, members of immigrant communities often stick together and police one another's behavior for mutual protection and benefit. This is an aspect of "bonding" social capital. Third, social capital is *influence* in the network. Those who have a good balance of bridging and bonding often enjoy a greater capacity to solicit assistance and resources from others, whether it is for a sociopolitical purpose or a more personal goal.

Sources of Social Capital

Adler and Kwon (2002) say the source of social capital is in the structure and content of the social relations of network members. The structure refers to the network pattern—a bridging or bonding pattern, for example. The content refers to qualities of the relationships between pairs of network actors. Adler and Kwon drew upon a set of distinctions proposed earlier by Nahapiet and Ghoshal (1998), who also identified a *structural* source of social capital, but divided the content into two distinct sources, the *relational* and the *cognitive*. The relational source encompasses qualities like reciprocity, mutual trust, and shared identity. The cognitive source covers shared language, shared problem-solving paradigms, common strategies, and even shared goals and values.

By cultivating these three sources of social capital simultaneously, a company can raise social capital in the community. Then it can "cash in" the social capital for a social license to proceed.

Tapping Sources of Social Capital Across the Phases

Developing the Structural Source of Social Capital

The structural source of social capital deals mostly with the patterns of connections in a network. For example, many networks have a core-periphery structure. The members at the core are well connected and influential. The peripheral members are more isolated with less access to resources. The structural dimension also deals with the strength of each tie. In a community network, for example, the groups or individuals with many weak ties will have access to a greater variety of information than others who have only a few strong ties. However, those with strong ties will be more capable of collaborating with one another to get things done.

When company representatives first arrive in a remote community, they begin forming ties with the network of community stakeholders. In the process of everyday transactions, they form very weak network ties with the hotel desk clerk, the restaurant owner, and the taxi driver. Since these people are all well embedded in the community, these ties represent bridges between the community network and the company's internal network. The leftmost panel of Figure 17.2-5 shows one thin line going from the company (i.e., the solid circle) to one rather central member of the community network. This is meant to represent a bridging relationship between the company and community. With nothing more than a weak bridging relationship in the community, the company is a peripheral member of the community network.

As the project proceeds, the company's presence grows in terms of numbers of representatives and in numbers of contacts with community members. Some of these relationships deepen. The company goes from having bridge relationships, to actually being a semiperipheral member of the community network. At this point, the company may also have created enough trust (relational source of social capital) and understanding (cognitive source of social capital) to be able to use its social capital to get an acceptance level of social license. The network structure would look something like the middle panel of Figure 17.2-5, where the original line representing the relationship formed in the leftmost panel becomes thicker in order to represent a stronger relationship.

The rightmost panel of Figure 17.2-5 shows the company as a member of the core of the community network. In this position it has considerable influence in the community and much more ready access to community-controlled resources. This would be a typical pattern for a company with an approval level of social license.

Developing the Relational Source of Social Capital

The relational source of social capital includes related qualities of social interactions like reciprocity, shared identity, and trust.

Reciprocity. Reciprocity begins with the simplest of courtesies in polite interaction. In many cultures this quickly involves mutual gift giving. As relationships develop, it is most prominent as various kinds of mutuality, including mutual respect and mutual self-disclosure. Reciprocity shades into generalized trust when the parties forgo immediate advantages to themselves in favor of actions that support the long-term interests of the group or of the bilateral relationship. This shows reciprocity because it requires a faith that such sacrifices will be reciprocated by others. In this way, reciprocity contributes to the solidarity benefit of social capital.

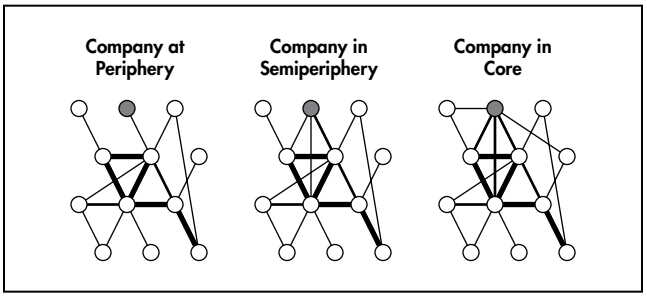


Figure 17.2-5 Structural dimension of social capital at each stage of the social license

Shared identity. Shared identity is important in preventing us–them dynamics from taking root. Members of stakeholder groups all share an identity by virtue of their membership in that group. Likewise, company representatives share an identity, especially in their roles as representatives. Two strategies can be used for avoiding intergroup conflict in such situations.

First, ties that cut across groups insulate members of each group from viewing members of the other group as "them." Crosscutting links either bilateral relationships or group memberships (that include people from both the stakeholder group and the company). For example, some company employees and stakeholder group members might both attend the same church and participate cooperatively in events there.

The second strategy is to heighten the salience of group memberships that completely encompass both groups. For example, both stakeholder group members and company employees might be citizens of the same country. Appeals to nationalistic sentiments can dissolve intergroup rancor and focus people on their shared interests.

At the highest level of social license, psychological coownership, the community, and the company share an identity centered around the industry itself. This happens, for example, when a community starts describing itself as a "mining town."

Trust. Trust is a very important aspect of the relational source of social capital. As Figure 17.2-5 shows, a company will start at different levels of trust even before any relationship begins. Early in relationships, trust is based more on calculations of risks and benefits and is limited to specific short-term transactions. Later on, trust comes to be based more on knowledge of the other party gained from experience over a period of time. Trust grows as the other party consistently keeps promises and demonstrably acts in the interests of the perceiver. This knowledge-based trust is important for gaining the acceptance level of social license. Companies can foster this kind of trust early by taking every opportunity to listen to the community and respond quickly (Svendsen et al. 2003).

Developing the Cognitive Source of Social Capital

The cognitive source of social capital comes from mutual understanding and agreement between the parties in a relationship. At the most basic level, speaking the same language creates a modicum of social capital. The level of language, however, is just as important. Companies must always use nontechnical language in their communications with stakeholders, even when it means using more words or losing some nonessential precision in meaning.

At a more abstract level, parties might use the same cognitive tools for approaching problems. For example, if both community stakeholders and the company habitually use strategic planning frameworks like cost–benefit analysis, strength-weakness-opportunity-threat analysis, critical-path analysis, and decentralized decision making, then they will enjoy more social capital together. More often, the stakeholders do not share these cognitive frameworks with companies. In those cases, both sides need to learn each other's ways of approaching collective action tasks. As the community learns the company's analytic techniques, it gains capabilities that it can use to achieve whatever goals it sets for itself, whether those goals are poverty reduction, ecological footprint reduction, or general community development.

Quite often the issues on which firms and stakeholders must develop a mutual understanding concern the general direction of future development in the community. Community stakeholders want to know what life will be like after the mine opens or after the mine closes. They need to know about downsides such as population influx and the possibility of more crime. For its part, the company wants to know what the community's priorities are for future development. Balancing short- and long-term interests here means not promising too much just to get an acceptance level of social license. In these cases, a future visioning process is in order. The process helps set planning priorities and taps the cognitive source of social capital.

How the Sources Work Together

Although the sources of social capital can be described as distinct, in practice they are intertwined in every action and activity. Likewise, the stakeholders' perceptions of legitimacy, credibility, and full trustworthiness develop organically along with the structural, relational, and cognitive aspects of the relationships.

Structural with Cognitive

Speaking the same language allows more communication to occur in the same amount of time. Inversely, communicating more frequently tends to create a shared set of understandings and frameworks. Shared understanding contributes to credibility because the stakeholders understand more thoroughly how mining works. Understanding, of course, also contributes to the cognitive aspect of legitimacy.

Structural with Relational

The gradual integration of the company into the stakeholder network (e.g., Figure 17.2-5) begins to create a shared identity. The community gradually sees the company as a fixture in the network. As crosscutting ties increase in number, a shared identity can emerge, thus tapping into the relational source of social capital. Likewise, any preexisting shared identity can facilitate the company's integration into the local network. For example, a local or domestic company might be able to create and strengthen ties more easily within the network.

Integration into the network builds legitimacy. Early in the process it creates the awareness and helps spread understanding needed for cognitive legitimacy. Later it contributes to the taken-for-grantedness of the company's presence, a more advanced form of cognitive legitimacy. Similarly, building trust, reciprocity, and shared identity contributes to moral legitimacy. Cognitive understanding contributes to pragmatic legitimacy, and vice versa.

Relational with Cognitive

A future visioning process affords an opportunity for the company to demonstrate honesty, transparency, concern for the community's priorities, respect for the community's leaders, responsiveness, and a willingness to share decision making on matters of mutual concern. All of these, in various proportions, contribute to stakeholder perceptions of legitimacy, credibility, and trustworthiness.

Often the process of coming to hold shared goals and visions for the future entails putting oneself in the other party's shoes. This can be difficult, but social capital from the relational source can make it much easier and often overlaps with the mutual perspective-taking feat (e.g., shared identity).

Dynamism in the Process

The levels of social capital in the firm–stakeholder relationships can fluctuate through the life cycle of the mine. Some of the determining factors may be beyond the control of either or both parties. More often, the actions of one or both parties raise or lower the level of social capital, with predictable effects on the level of social license. An example from an actual mine illustrates the point.

Figure 17.2-6 provides a graphical representation of the condition of the social license for the San Cristobal project over a 14-year time period from 1994 to 2008. Today, San Cristobal is a large zinc, lead, and silver mine in the southern Altiplano of Bolivia, operated by Minera San Cristobal (MSC), which is 65% owned by Apex Silver Mines Limited and 35% by Sumitomo Corporation. The quality of the social license was estimated initially from historical documents and the experience of persons who had been present throughout the life of the project using a basket of indicators and then verified through interviews with community members.

As can be seen in Figure 17.2-6 the quality of the relationship changed over time in response to various factors, which may be summarized as follows with reference to the numbers beside the graphical trace in the figure.

1. Rights to the mining concession are acquired by Mintec, a Bolivian mining consulting firm, which, together with permission from the community to access the surface, establishes formal legal status and permission to start work on the ground. The company rapidly builds social legitimacy with the local community by providing information and employment.
2. Geological indicators of widespread mineralization are identified. Mintec, and later ASC Bolivia LDC (ASC), an Apex Silver Mines Limited subsidiary that had acquired Mintec, steps up a dialogue with the community and provides additional benefits. Social and environmental baseline studies for an environmental impact assessment are initiated.
3. Clear indications of a very large mineral system are demonstrated through drilling. ASC believes that a mine can be developed in the near future. Accordingly, the company begins a process of consultation with the community of San Cristobal to relocate the population away from the mineral deposit. ASC empowers the community to manage essential aspects of the relocation such as selection of the new town site, design of houses and infrastructure, eligibility, and a benefit package. The community comes to feel that they are partners (co-owners) in the project—ASC has the mineral deposit, but the community has given its land to make the mine possible. Negotiations begin with the sister community of Culpina K for land for the tailings facility.
4. A comprehensive resettlement agreement is signed with the San Cristobal community. A land purchase agreement is reached with Culpina K. Trust reaches an all-time high.
5. The community of San Cristobal is relocated to the new town site together with the colonial church and cemetery.
6. The women, who were largely excluded from the negotiations and planning for the relocation, voice complaints about the houses: "They are not what we wanted." Culpina K residents start to think that they have made a bad deal.
7. The project is then transferred by ASC to MSC. Trust is eroded because the company has failed to deliver on commitments made in the resettlement agreement. Further, although the company has obtained necessary permits to construct and operate a mine, there are now doubts as to the feasibility of the project, and the company has drastically reduced the number of employees.
8. The project fails feasibility because of low metal prices. MSC closes all field operations. The communities are frustrated and trust is lost. Contacts between the company and the communities become infrequent. The company remains noncompliant with the terms of the resettlement agreement. Credibility is lost. The project remains legitimate in the minds of community members because they want the employment and the better future they hope it will bring.
9. Realizing the need to stabilize and strengthen the relationship with the community, the company initiates a program of assistance centered on employment for local people, agriculture, and tourism. Credibility is restored with delivery of the program.
10. A highly innovative program to encourage tourism is launched. Culpina K becomes heavily involved; San Cristobal, less so because it would rather have the mine.
11. An upturn in market conditions renders the project positive, and MSC announces the start of construction. Credibility peaks as the communities welcome the construction of "their" mine and the potential for employment during construction.
12. New management is installed to supervise construction that has no knowledge of the social history of the commitments to the communities. Contact between the company and the communities breaks down as the company drops meetings with the communities that involve top management. The communities feel deserted and disenfranchised because of a perception that they are not being respected and commitments for employment and training are not being honored. Credibility is lost almost immediately. Despite employment for almost all local people, social legitimacy drains away as the communities feel they have been overlooked in the construction phase and commitments dating back to 1999 remain unfulfilled. They still believe in "their" mine and mourn the loss of the partnership that existed at the time of the relocation of San Cristobal.
13. A contractor opens a road though community gardens outside of the agreed area of construction operations. For the community of Culpina K, this is an illegal act. Social legitimacy is compromised, and community relations collapse for a period with demonstrations and confrontation

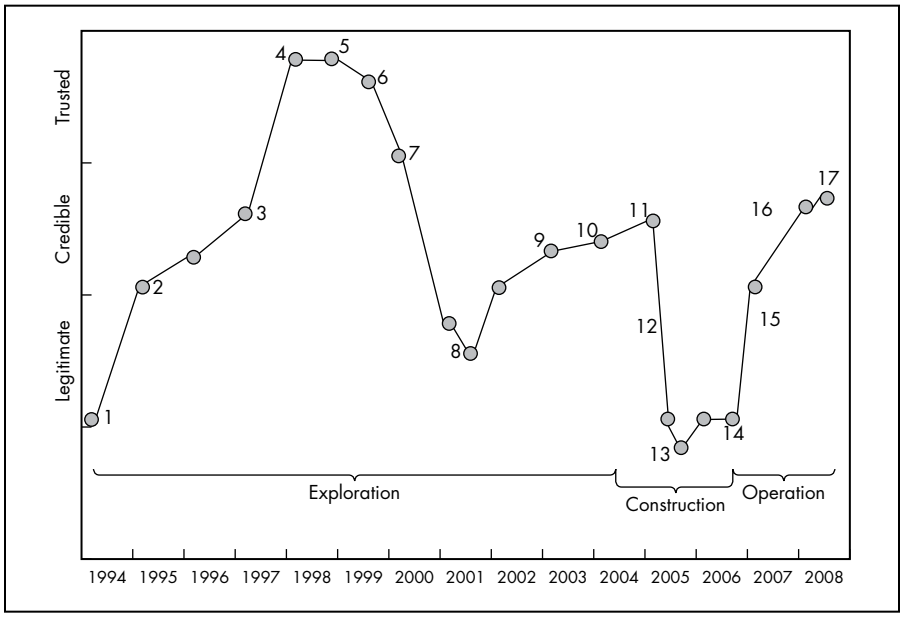


Figure 17.2-6 The ups and downs of social capital at San Cristobal mine

with the company. MSC is able to negotiate a letter of agreement that allows work to continue and begins a protracted process of negotiation for the acquisition of additional land for mine infrastructure facilities. Culpina K remains at a distance because of rising concerns that they will be adversely affected by the tailings facility.

14. Management comes to recognize the problems and risks created by the damaged relationship and takes action to improve the situation. The company proposes the formation of a community-based process for the design and management of community-development programs. The communities see this as an opportunity to take control of their own future and feel empowered. At the same time, MSC begins an accelerated program to comply with all prior commitments. The communities see progress and feel reassured. Construction ends; training and full-time employment is available to all. Dialogue is established with Culpina K around the issue of management of the tailings. Legitimacy is regained.
15. A management team is appointed to run the mine, replacing the construction team, which brings stability and quickly establishes a positive dialogue with the communities. All prior commitments have been met or are in a visible process of being met. An accelerated local employment program is initiated. Community-based planning for social and economic development is under way. Credibility is restored.
16. MSC responds to community concerns regarding the management of tailings by forming a joint monitoring committee with the community. Essential infrastructure improvements are made in San Cristobal and Culpina K. Credibility is strengthened. As "co-owners" of the mine, the communities make representations to the national government in support of the company in response to statements of increased taxes and threats of nationalization. Trust, however, remains elusive.
17. With the mine in operation and the communities fully involved in the management of their own development, in collaboration with MSC and local and regional government, there are early indications that trust could return to the relationship.

RELATIONSHIP BUILDING CHANGES BOTH SIDES

The Fiction of "Community" as an Entity

Up to this point, the term *community* has been used in a reified way, that is, in a way that grants it an identity and singleness of purpose that does not always exist. Many so-called communities are really aggregations of communities and/or kinship and interest groups. However, the concept of the social license to operate presupposes that all the families, clans, interest groups, and institutions in a geographic area have arrived at a shared vision for their future, or at least an agreed-upon set of priorities for today. This kind of cohesion is frequently absent. In such cases, it has to be built. That is why earning a social license to operate often involves the company in building social capital, also known as community building, capacity building, and institution strengthening.

Industries, and the companies in them, have to learn how to participate in the kind of community partnerships that earn a social license. Reciprocally, communities that want the benefits of participating in the mainstream economy have to develop the social structure that makes them capable of issuing a legitimate, credible, and trustworthy social license.

Communities Not Qualified to Issue a Social License to Operate

The key to a community's capacity to issue a meaningful social license is the pattern of social capital it has in its network structure. Without the right patterns of social capital, a community cannot stick to its commitment through the inevitable changes in local politics. Companies want a license that reduces the risk of investing in mining infrastructure. Figure 17.2-7A, B, and C illustrate three examples of how communities can have a social capital network structure that makes them unlikely to be capable of issuing the social license to operate that companies want.

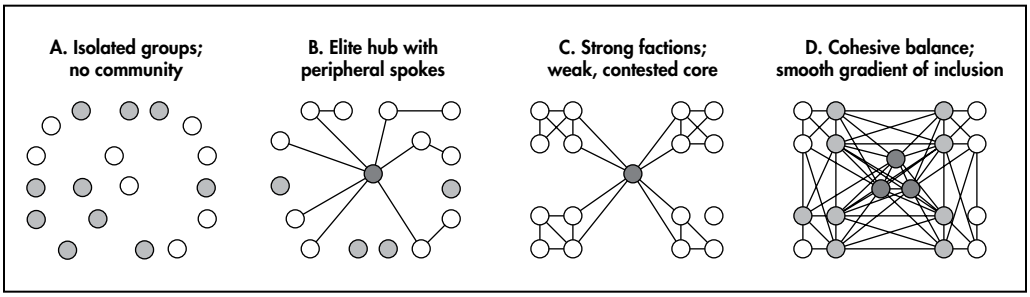


Figure 17.2-7 Four prototypical patterns of stakeholder network structure

Figure 17.2-7A shows a community with a severe lack of social capital. It has no core. It is an aggregate of groups that do not talk to each other. This is the pattern one would expect in a region after a natural disaster or war. There may be pockets of groups that deal with each other, but over the whole geographic region, or across all the stakeholders in the issue, there is little communication. No one can speak for the whole community, and therefore the community can neither issue nor revoke a social license to proceed. In this circumstance, companies usually fall back on the legal licensing process at the national level. Unfortunately, if the community subsequently comes together and finds a voice, that legal license can become worthless.

Another common hindrance to being capable of issuing a social license is the existence of an elite that purports to speak for everyone (see Figure 17.2-7B). The core has numerous relations of bridging social capital to peripheral groups. The peripheral groups have little social capital among themselves. Although dealing with this kind of structure is administratively convenient, there are dangers related to the elite's own legitimacy. Such elites are often tempted to put their own interests ahead of the interests of the whole community. Sometimes they stay in power by keeping everyone else divided, ignorant, or intimidated. Multinationals often fall prey to such groups when only the elites know the native language of the company representatives. Often they have been educated in the developed world and know the norms for doing business in the company's home country. The elites will say they can issue a social license, but once development gets underway, the objections of the excluded groups come to the surface. The company can then be lumped together with the elite as part of an illegitimate and corrupt establishment. This, of course, calls the social license into question.

Figure 17.2-7C shows another dysfunctional community structure. There is plenty of bonding social capital, but it is confined to tight-knit factions. To the extent that a core exists to speak for everyone, it is beholden to the leaders of the factions, be they mayors, war lords, clerics, or members of parliament. Because the factions are continually creating and destroying alliances in attempts to dominate the core, there is never a community with enough cohesion to ensure that any social license will be durable. The company can easily be drawn into the political machinations, either as a benevolent ally of one faction or an evil villain against which another faction rallies support.

Diagnosing Community Readiness to Issue

Companies that want a social license clearly need to know the patterns of the social capital in the network from which they expect to get the social license. Figure 17.2-7D is the ideal social capital structure for a stakeholder network. This type of community can make decisions that include everyone and that will work for everyone's benefit. It can stand behind the decisions for the long term but also has the flexibility to adapt its goals to changing circumstances.

Fortunately, social capital has been studied quite thoroughly in the social sciences for more than 15 years. Ways have been developed to quickly assess its patterns in a network, whether that is a geographic community or a wider set of stakeholders that includes national government departments and international NGOs. The basic approach is to interview the stakeholders about four areas of concern: (1) what their concerns are, (2) which other groups they consider to be stakeholders, (3) the quality of their relationship with the company, and (4) the quality of their relationships with other stakeholders. Data from interviews of this type provide a well-rounded basis for a stakeholder engagement strategy leading to a social license.

Ways to Approach Identified Problems

After a map of the social capital structure in the community is at hand, it is possible to determine where the effort should be focused. If the community looks like Figure 17.2-7A, it is necessary to start with basic communications among groups. If the community looks like Figure 17.2-7B, the first step is to make connections with the periphery groups and connect them with each other. If it looks like Figure 17.2-7C, efforts to strengthen the core are needed. A group of leaders from each faction must work together on a project long enough to develop a sense of shared purpose that is community-wide.

Each community has its own specific issues and problems that provide plenty of opportunities for bringing people together. Often the initial focus has to be setting community priorities and then identifying those that can be achieved in collaboration with the company. Wilson and Wilson (2006) offer numerous examples of initiatives that build social capital. These span diverse areas of operations from security to catering. Activities that build social capital rarely cost very much money. Usually they simply involve redirecting funds from an approach that ignores or damages the company's social license to an approach that improves and secures the social license. Sometimes they actually reduce costs, as occurs when the work to improve the capacities of local suppliers creates a stable of lower-cost local sources for goods and services previously imported from elsewhere.

Social License, Poverty Reduction, and Sustainable Development

The process of helping a community acquire the balance of social capital needed to issue a social license amounts

to strengthening the community to the point where it can be a strong, reliable partner in achieving whatever goals it sets for itself. This happens to be the same process that community development specialists follow in initiatives to reduce poverty. Social capital building and balancing initiatives have been shown to reduce poverty, but as Figure 17.2-7 indicates, the pattern of social capital has to be balanced (e.g., Figure 17.2-7D). In addition to a strong core, the overall pattern should integrate all subgroups to avoid isolation of the periphery. When these conditions are met, communities can benefit from the presence of a national or international corporation. The company can create or broker links to the larger economy outside the community. Such links are called *linking* social capital (Woolcock 1998). Through linking social capital, a community can gain access to more distant markets, for both its goods and its services (i.e., remittance income). Therefore, gaining and maintaining a social license employs the same set of activities that promote sustainable community development. Both goals depend on strengthening the community to the point where it can be a stable, committed partner in development initiatives.

TOOLS AND GUIDELINES

Companies use numerous tools and guidelines to help them do the things that earn a social license. These tools often bear labels related to "corporate social responsibility," "sustainability," or various more specific issues like corruption, transparency, or the environment. Leipziger (2003) reviewed hundreds of such codes, lists of principles, and guidelines. She reviewed more than two dozen of them in detail. Some of them reiterate many of the principles discussed above. Those that contain recommendations for specific steps and activities will not always work. Communities and circumstances are too varied to be amenable to any one-size-fits-all program. That is why the more general principles must take precedence over concrete recommendations. Although some guidelines sound too vague to be practical, it is actually the more specific elements that are less practical because they help only in some circumstances but not in others. Ultimately the social, cultural, and political dynamics of the individual community or communities that are neighbors to mining projects have to be understood, and tools and guidelines must be adapted to meet the characteristics encountered on the ground.

Exploration Practices

For explorers, *The Principles and Guidance for Responsible Exploration*, produced by the Prospectors and Developers Association of Canada (PDAC 2009a) together with the *e3Plus* on-line reference manual for good practice (PDAC 2009b) are invaluable aides to gaining legitimacy and credibility with local communities and hence acceptance and a conditional social license.

For projects that advance to feasibility and construction, the key sources of information for the management of social, socio-environmental, socioeconomic, and certain cultural matters are the Performance Standards on Social and Environmental Sustainability of the International Finance Corporation (IFC) of the World Bank (IFC 2006) and the Equator Principles (EPFIs 2006), a parallel set of guidelines created by the private-sector banks. While not created specifically for the mining sector, the IFC standards and a series of accompanying manuals are based on extensive practical experience that is of immediate relevance to mining projects. The IFC standards emphasize disclosure of information, consultation, participatory processes of investigation, and decision making with the community and other stakeholders; inclusion and protection of vulnerable and marginalized groups; respect for rights; participation in the benefits of mining; and so forth, all of which are actions that can create social legitimacy and, properly implemented, credibility for the company and hence a base for building trust.

Mine Operation from Construction Through Postclosure

Business for Social Responsibility (2003b) and AccountAbility and Business for Social Responsibility (2004) provide case histories and discussion of the social license together with practical advice on how a social license may be gained.

The principles and guidelines of the International Council on Mining and Metals (ICMM) provide a point of reference at the global level for good practice that aids acquisition of the social license (ICMM 2003). In addition, the ICMM has prepared a series of good-practice handbooks that offer practical advice on the management of various aspects of mining, including social and community development. At the same time, mining associations in a number of countries have developed guidelines optimized to national legal, cultural, and social norms; both Canada and Australia, for example, have developed such instruments.

The Mining, Minerals and Sustainable Development (MMSD) project identified a portfolio of initiatives that support gaining and maintaining a social license and are critical if mining projects are to make a contribution to sustainable development (MMSD 2002a). They include

- Supporting people (social capital) and investing in people (human capital),
- Supporting and strengthening existing governance structures (social capital),
- Facilitating continuous learning and improvement (human capital), and
- Helping local populations anticipate and take advantage of mine development to improve their quality of life (as defined by them) beyond the life of the project or mine (balancing human and social capital).

The MMSD project also created a framework for assessing a mining project's contribution to sustainable development, which includes material that can be used to design projects in a way that helps in gaining the social license to operate (MMSD 2002b).

As with the examples above, full implementation of the recommendations contained in these documents in ways that are sensitive to local social and cultural norms can significantly assist in gaining and maintaining the social license.

General Guidelines for All Types of Businesses

In the developed world, various existing political forces came together to create an ad hoc governance regime to regulate global corporations. The result of the ad hoc, piecemeal collaboration has been the emergence of some mechanisms for influencing corporate behavior. The influence strategies are each centered on a different set of stakeholders.

One of the most important stakeholder groups has been investors. The socially responsible investment movement applies investment screening criteria that have been promoted by the aforementioned loose coalition. In 2007, ethically screened investments accounted for US$4.93 trillion in assets

in United States and Europe alone (Social Investment Forum 2009). Prominent rating systems in the ethical investment field include the Calvert Social Index, the Dow Jones Sustainability Index, the FTSE4Good Index, and KLD's Domini 400 Social Index.

The same general influence mechanism has been called into play with lenders. The IFC has developed Performance Standards on Social and Environmental Sustainability, which are incorporated into the Equator Principles that have been adopted by most of the major commercial banks that finance mines. Companies seeking financing must pass independent assessments of their compliance with these standards for social and environmental performance.

In terms of employees and suppliers, there have been campaigns against child labor and sweatshops. The ILO has been very active in the area of mine safety. Its standards apply to governments, which in turn are supposed to enforce them on companies to whom it grants licenses.

The ILO 169 gives explicit instructions to governments to ensure that indigenous people are consulted before exploration and development of natural resources and that they share in the benefit that comes from resource development. This convention has a strong influence on corporate behavior even where governments are weak or absent, as is the case in many rural areas. The UNDRIP gives an even stronger instruction to governments to obtain the consent of indigenous people before any exploration or development of mineral resources.

Another approach is centered on consumers as stakeholders. Green and socially responsible consumer movements usually focus on specific product classes. The "dirty gold" and "blood diamonds" campaigns are aimed directly at the minerals industry.

The development of these influence channels has included the emergence of general codes and standards that spell out what is expected of companies. These expectations provide general guidance regarding how to earn a social license. Popular codes and standards that apply to all industries include the AA1000, the Global Reporting Initiative (GRI), and the venerable United Nations Universal Declaration of Human Rights (UNGA 1948).

The AA1000 was developed by the Institute for Social and Ethical AccountAbility (AccountAbility 2009). It consists of checklist-style framework for implementing changes in a company and a separate assurance process that verifies the company's adherence.

The GRI deals with standards for transparent reporting on nonfinancial matters. The most recent version was developed with input from thousands of companies from all parts of the world. It is a comprehensive checklist that was intended to apply to all industries. (GRI 2006). To help mining companies reduce the burdensome paperwork, the ICMM has published a guide for focusing on the parts of the GRI that are relevant to its sustainable development principles (ICMM 2005).

SUMMARY

The fundamental tools for building relationships between mining projects and communities are the values that permeate the actions of the company. These include

- Respect and inclusiveness;
- Transparency and honesty;
- Listening and empathy;
- Responsiveness and promise keeping;
- Goodwill, care, and protecting the interests of the other; and
- Clear rules and principled actions.

These are "soft" attributes that many mining company personnel find difficult to work with since they are not amenable to quantification in the same way as engineering and accounting. Nevertheless, they are keys to the creation of good, stable relationships and in many ways parallel the attributes of lasting personal relationships, friendships, and, indeed, a marriage.

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CHAPTER 17.3

Cultural Considerations for Mining and Indigenous Communities

Ginger Gibson, Alistair MacDonald, and Ciaran O'Faircheallaigh

THE GROWING IMPORTANCE OF INDIGENOUS CULTURAL CONSIDERATIONS

Article 1 of the International Labour Organization's (ILO's) Convention 169 defines indigenous peoples as

> (a) Tribal peoples in independent countries whose social, cultural and economic conditions distinguish them from other sections of the national community, and whose status is regulated wholly or partially by their own customs or traditions or by special laws or regulations; (b) peoples in independent countries who are regarded as indigenous on account of their descent from the populations which inhabited the country, or a geographical region to which the country belongs, at the time of conquest or colonisation or the establishment of present state boundaries and who, irrespective of their legal status, retain some or all of their own social, economic, cultural and political institutions (ILO 1989).

A cornerstone definition for the United Nations has been the "Cobo definition" of indigeneity, which highlights sacred attachment to and traditional ownership of land that has been appropriated by outsiders. This stands in contrast to people who self-identify as indigenous, which is the basis of the ILO definition.

The mining industry has not historically been required to closely analyze its potential impacts on indigenous cultures. This is no longer the case. The industry cannot afford the risks associated with the damage its activities can cause to indigenous culture. Nor can mining companies, in a highly competitive environment, ignore the potential of indigenous culture to add value to the enterprise.

More and more exploration and mining projects are located near indigenous communities whose culture is already under threat from dominant settler societies. The culture of indigenous communities, traditionally reliant on the land and its renewable resources, can be further threatened as they become exposed to and engaged with the mining industry (IFC 2007). Mining often becomes the new field of contestation between settler and indigenous cultures, as well as creating new relationships between the indigenous group and the distinct corporate culture of the modern mining firm.

Indigenous groups are not without levers to both shape these relationships and to impose sanctions on mining companies that refuse to acknowledge and respect their cultures. Many indigenous communities have gained enough power in recent decades to vigorously assert their rights to land, and have had these rights recognized through national law and international declarations and conventions such as the ILO's *Convention 169 Concerning Indigenous and Tribal Peoples* (1989) and the *United Nations Declaration on the Rights of Indigenous Peoples* (2007). This recognition has partly been driven by successful litigation and direct political action by indigenous peoples, often with the support of civil society groups. As part of this recognition of indigenous rights, cultural impact assessment of individual projects is now required in many regions. The Mackenzie Valley Resource Management Act in Canada's Northwest Territories, for example, requires environmental impact assessments (EIAs) to have regard to "the protection of the social, cultural and economic well-being of residents and communities of the Mackenzie Valley" (Government of Canada 1998).

Many of these indigenous cultures are resilient, vibrant, and resurgent. For example, in recent years, thousands of Aboriginal people from throughout Australia's Pilbara iron ore region have participated in annual ceremonial activities. Innu and Dene people of Canada are recapturing their cultural heritage through bushwalks and mass canoe trips (Zoe 2007). Indigenous communities have united to protect special cultural places and spaces. Another indicator of cultural resilience is that indigenous groups have increasingly challenged companies to make space for traditional cultural activities and values to be expressed at mine sites, to assist in protecting sensitive valued cultural components, and to adapt the corporate work culture of mining to the needs of indigenous people

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(e.g., through alternative work rotations or respect for communal grieving) (Argyle Diamonds Ltd., Traditional Owners, and Kimberley Land Council Aboriginal Corporation 2004).

When cultural issues are not given appropriate consideration, the sanctions that indigenous groups impose on companies have become increasingly onerous. Companies who cannot show that they will not cause irrevocable damage may be denied the permits and licenses required to establish new projects. They may also be unable to secure mine financing if potential lenders perceive they are unable to manage cultural and social risks. Here are a few examples of downside risks resulting from inadequate community engagement or the presence of significant cultural impacts:

- Innu and Inuit landowners in Labrador, Canada, delayed the Voisey's Bay nickel project through court cases and activism because of the company's failure to prepare an environmental and cultural protection plan before starting exploration on indigenous lands (R. Gibson 2006).
- At Esquel in Argentina and Tambogrande in Peru (MacDonald and Gibson 2006), there was a fundamental lack of early and effective community engagement, and, as a result, insufficient effort to deal with indigenous concerns about adverse environmental, social, and cultural impacts. This led to a refusal by indigenous groups to grant companies a "social license to operate" (Joyce and Thompson 2000).
- Canada's federal government ultimately rejected a proposal for a modest drilling program in the remote Upper Thelon River basin when it became clear to the co-management board in charge of environmental assessment that the area held great cultural significance to the indigenous residents (MVEIRB 2007a).

(Note: The bulk of examples in this text are taken from the authors' experiences in Australasia and North America, but the tone is one focusing on expressing general concepts that emerge between indigenous communities and mining projects around the globe, and many of the general concepts, tips, and tools that can be applied in a variety of cultural contexts. However, there is no substitute for learning the site-specific cultural context and engaging culture holders in the cultural impact assessment process.)

Examples like these have done little to alleviate industry fears that dealing with indigenous cultures will consume valuable time, create new project costs, and open a Pandora's box of new risks for mining projects. Geologists, mining engineers, and other project managers have historically lacked training in cross-cultural engagement. This lack of experience can heighten a perception that cultural engagement is the first step in either a total rejection of their proposed project, a drastic reduction in its operating scope, or the imposition of restrictions that cost a great deal of money and reduce the life of any potential project.

Such a perception is both inaccurate and damaging to mining companies and their shareholders. In reality, the majority of potential cultural impacts are manageable at modest cost if they are clearly identified and appropriately addressed, and particularly if this occurs early in project life. Further, cross-cultural engagement can generate substantial value for the enterprise. For example, Canadian diamond mines have worked with indigenous groups to devise a 2-week-in/2-week-out work rotation schedule that maximizes the amount of time hunters can be out on the land. Any rotation greater than 2 weeks was considered too long, given that a herd of caribou can move through an area in weeks and be too far from communities to harvest by the time workers returned. In this case, the cultural practices and communal desires of the potentially affected culture groups were integrated into project planning. The result has been a win–win situation for the companies and the communities, with a more-committed work force for the companies, and an indigenous work force able to balance traditional and wage economies (G. Gibson 2008).

In areas largely populated by indigenous people, cross-culturally sensitive workplaces can add value by increasing indigenous employee retention rates. Unlike many nonindigenous mining employees who migrate or "commute" to remote mine sites, local indigenous people have a long-term commitment to live in the regions where mines are located. If they can be recruited and retained by mining companies, this can significantly reduce work-force turnover, which in turn can reduce company expenditure on staff recruitment, training, relocation, and severance, and enhance work-force cohesion and morale. In addition, resources committed to upgrading the skills of indigenous workers can generate higher returns for the corporation, as their ties to their "home place" make them less likely to move to other operations elsewhere, taking their new skills with them. Use of local labor can also reduce the costs involved in transporting workers into remote sites and providing them with camp accommodations. Companies may incur additional up-front costs in recruiting and training indigenous workers but, especially if a longer time frame is adopted in assessing returns, this investment can generate significant net gains (Barker and Brereton 2005; Barker 2008; Harvey and Gawler 2003; NWT and Nunavut Chamber of Mines 2005; SIWGMI 1996; G. Gibson 2008). Cross-culturally sensitive work environments can also minimize downtime and costs associated with turnover among indigenous and nonindigenous employees, reducing management–employee conflict and minimizing costly disruptions to the work environment.

Respectful cross-cultural engagement is essential if companies are to gain access to traditional indigenous knowledge, an outcome that can generate considerable value for the firm. Traditional knowledge is sometimes envisioned by developers as an input useful only during initial project planning, and as a legal requirement of the formal EIA process. Experience shows that these resilient cultures and their knowledge constitute a valuable resource that companies can draw from throughout the life of the mine. Cultural knowledge can be a valuable environmental planning and early warning tool for mining companies. Indigenous peoples are often in a unique position to assist mining companies in minimizing negative environmental impacts and in returning mine sites to acceptable environmental standards (O'Faircheallaigh 2010). One critical resource possessed by indigenous people is time depth in relation to information on the existing environment ("baseline data"). Typically, mining companies only start collecting baseline data some 2 to 3 years before project construction is planned to commence. Because environmental conditions can be highly variable over time, a few years' data may not provide a sufficient basis on which to develop an understanding of environmental dynamics. Indigenous peoples draw not only on decades of experience in observing environmental conditions on their traditional lands, an understanding of which is vital to their subsistence activities, but also on generations of knowledge handed down to them. Thus, their participation is essential in achieving a full and accurate understanding of

existing environmental dynamics, which in turn is the foundation for effective environmental protection (Nadasdy 2003; Usher 2000).

The intimate and ongoing contact that many indigenous people have with their ancestral lands and waters also allows them to quickly detect ecological changes that may signal problems with a mine's environmental management system, helping to avoid potential damage and quickly take remedial actions. A range of more specific indigenous knowledge can also greatly assist effective environmental management. This can, for example, include understanding of animal behavior as essential in designing effective wildlife monitoring and management regimes.

The knowledge and skill of indigenous people can also be mobilized to make major technological shifts that increase productivity. As an example, using elders' knowledge of sea ice re-freeze conditions and time requirements to build a floating bridge, Inuit workers decreased wait times for shipping of ore from the Voisey's Bay mine in Labrador (I. Paine, personal communication).

In all of these cases, the requisite traditional knowledge may only be readily accessible if a respectful and trusting relationship, including strong ground rules for the protection of cultural knowledge, is established between the developer and knowledge holders.

Positive cross-cultural engagement can also allow companies to

- Demonstrate their corporate social responsibility, a major benefit in an industry with an often problematic public relations image;
- Develop the effective communication and understanding of the indigenous party (and vice versa) essential for negotiating, interpreting, and implementing agreements between indigenous communities and mining companies; and
- Obtain certainty of land tenure and permits in a timely manner.

Against this background, mining companies, small and large, have to learn how to deal with indigenous expectations that impacts on culture will be managed, and with the increased leverage exercised by indigenous people. Avoidance of cultural impact identification will only delay risks until a time in the future when greater financial losses will accrue because increased project investment has occurred. Industry associations are certainly taking heed that cultural issues merit closer consideration to both take advantage of upside benefits and avoid downside risks. As an example, the International Council on Mining and Metals has published ten principles of sustainable development, including the requirement that operations should "uphold fundamental human rights and respect cultures, customs and values in dealings with employees and others who are affected by our activities" (ICMM 2003). Also, the Prospectors and Developers Association of Canada has developed e3 Plus: A Framework for Responsible Exploration that includes expectations of "respecting and protecting local culture and traditions" (PDAC 2009).

Cultural impact assessment and the expectation that companies will achieve effective cross-cultural engagement with indigenous people are thus both "on the table." Despite the importance of cultural factors—their growing legal recognition, the demands of indigenous peoples around the world that they be considered before developments proceed, and the business case for positive engagement—culture quite often remains the "elephant in the living room" of mining project development. Everyone knows it is there and it is awfully hard to ignore, but dealing with it effectively is another matter. While some companies are making serious efforts to respectfully engage with indigenous cultures, common barriers to effective cross-cultural engagement and doing good cultural impact assessment still include

- Narrow interpretations of "culture," usually to the detriment of "intangible" dimensions of culture;
- Weak initial cross-cultural engagement often associated with lack of preparation and lack of in-house expertise;
- Power differentials and incompatible cultural values between settler society, indigenous groups, and mining companies;
- Emphasis on the technical/scientific/quantitative over the oral/traditional/qualitative in formal EIAs;
- The complex variety of contributing factors to cultural change;
- Lack of agreement on what constitute appropriate indicators of cultural impact;
- Lack of preexisting thresholds of acceptable change for cultural indicators;
- Companies not wanting to take on pseudo-governmental sociocultural protection roles; and
- Lack of formal legal requirements for implementation of cultural protections.

These are not insurmountable challenges; they can be overcome by changing attitudes with additional guidance and more effective approaches to cultural engagement and impact assessment, both at the corporate head-office level and by company managers "on the ground."

The purpose of this chapter is to identify some of the key issues that must be addressed, and principles and methods that can be used, during engagement between indigenous communities and mining companies. It seeks to help ensure that the current gaps in cultural impact assessment and community engagement practice can be overcome, relationships are respectful and appropriate, the benefits of cross-cultural reciprocity can be fully realized, and adverse impacts on indigenous culture can be avoided throughout the life cycle of an operation and at the various sites of cross-cultural interaction.

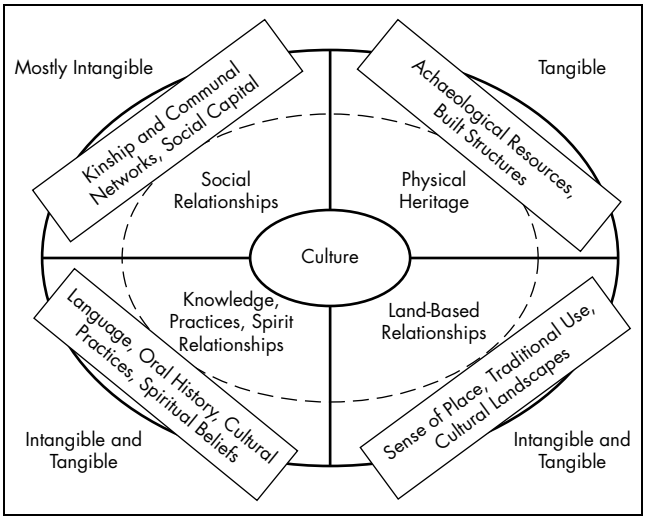
The sections that follow define culture, discussing intangible and tangible cultural assets; scoping, which requires engaging culture holders and understanding the potential impacts a project may have on culture; the collection and analysis of appropriate data and prediction of likely impacts and their significance; and the identification and implementation of appropriate mitigation, the means by which potential adverse impacts can be avoided or minimized and, equally importantly, benefits to (and of) culture can be maximized.

CONCEPTS OF CULTURE

Before exploring how mining development can impact indigenous culture and vice versa, it is necessary to know what culture is, who holds it and where, and how the values and priorities of indigenous cultures and mining's corporate culture tend to differ.

Tangible and Intangible Cultural Assets

The study of indigenous cultures as they relate to mining has often focused on the protection of historic artifacts that may be



Source: Adapted from Candler 2008.
Figure 17.3-1 Tangible and intangible cultural assets

physically damaged by development. This "stones and bones" approach to cultural impact assessment ("heritage resources impact assessment") is practiced largely by archaeologists and other physical anthropologists, and has helped identify and protect the material manifestations of past occupation—features such as historic sites, gravesites, middens (refuse caches), rock and cave paintings, and scatterings of stone tools as well as dwellings or ceremonial sites and trails (King 2002; O'Faircheallaigh 2008). Each of these features leaves a physical clue as to how the earlier generations lived and traveled. They focus attention on the *tangible* signs of a culture's past.

Culture is, of course, much more than stones and bones; it is a living, continually adaptive system, not a remnant of the past. It is also highly complex, which makes precise or exhaustive definition of the concept impossible. A simple, general definition of culture is "a way of life; a system of knowledge, values, belief, and behavior, passed down between generations." As early as 1871, Sir Edward Tylor defined culture as "that complex whole which includes knowledge, belief, art, morals, law, custom, and any other capabilities and habits acquired by man as a member of society" (Tylor 1920).

Although it may be difficult to define it precisely, it is possible to identify some key elements of culture. Aside from the physical manifestations most often analyzed and protected during a heritage resources impact assessment, culture also includes values and worldviews, and the tools through which they are communicated—symbols, language, and behavior. It includes communally held knowledge, spiritual practices and beliefs, traditions and traditional pursuits, social relationships, and a common understanding of humankind's connection to the natural environment. These are among the *intangible* assets of a culture group. Manifestations of intangible cultural heritage include oral traditions, language, performing arts, ritual and festive events, knowledge concerning nature and the universe, traditional craftsmanship skills, traditional social practices, spirituality, and sense of place (UNESCO 2003; Andrews and Buggey 2008).

Common elements of culture, and particularly of indigenous cultures, are identified in Figure 17.3-1.

Common languages, stories, myths, songs, conversation, traditional activities, and art are used to communicate and transmit culture from generation to generation. Often in indigenous communities, the stories, knowledge, and practices that are critical to living well are transmitted as people are out on the land, engaged in the traditional economy as they pick berries and plants, hunt, trap, and fish. In these primarily oral societies, historical and mythological events are often recorded in landscape features. Literally for them, "the land is like a book" (Dene Elder Harry Simpson, quoted in G. Gibson 2008). As young people travel on the land to significant places, they engage with their elders, leaders, and families, strengthening their bonds and knowledge, while they come to know their history through storytelling at the significant sites of their people. The shared stories ensure that the younger generation acquires appropriate cultural information about traditions, way of life, and values.

The key thing to remember for those involved in cross-cultural engagement is that impacts on culture can come not only from changes to "things you can touch" or from physical effects on tangible and visible heritage resources, but they can also come from changes to "things that touch you"—intangible cultural assets that while perhaps not visible from without, are felt from within and are vital to cultural retention and maintenance.

Domains of Culture: Who Holds It and Where?

It is essential to understand the "domains" of culture—the places, spaces, and other planes of interaction where culture is transmitted, regenerated, and altered. Culture is acted out, transmitted, and received between the self, the family, and social networks; within and between communities; between different cultural groups; and within the context of an increasingly global popular culture. In many indigenous cultures there is an ongoing struggle between the norms and values realized through the practice of traditional ways such as subsistence harvesting and the values of a dominant settler society.

Culture has a spatial component as well. All cultures have a "home place" and members have a shared sense of place. For the mining company's corporate culture, the home place may be the corporate head office or the mine site. Indigenous cultures have ties to the physical habitations they reside in, just like settler communities do, but indigenous cultures may hold a much stronger, even symbiotic, relationship with their wider traditional lands.

For indigenous people the land itself is often a key repository of knowledge, an interpretive tool, and a key locator of meaning and history. There is growing international recognition (MacKay 2004; UNDESA 2004) that the ability to live on, care for, and use resources from their ancestral lands is central not only to the economic and social well-being of indigenous people but also to their very survival. Land is critical

- To physical sustenance;
- For social relationships that are bound up with relations to land;
- For law and culture, which are interwoven with use of the land and its resources; and
- For spirituality and religion, which have as their basis beliefs about the creation of the land, the way in which creation spirits continue to occupy the land and influence contemporary life, and the way in which ancestors and future generations are tied to the current generation through the land.

Indigenous knowledge of the land is tied to knowledge of past events related to people first by their ancestors and

then passed by them to their own descendants. Quite literally for indigenous people, "wisdom sits in *places*" (Basso 1996, emphasis added).

Wisdom resides in spaces too. Places are sites. Spaces are routes, relational lines between points on the map. They are larger than a single place, and they can also represent a larger meaning as *cultural landscapes*. Cultural landscapes have been defined as "landscapes that are lived in" and that bring attention "to the way people within the landscape live, their traditions and everyday life" (NWT Cultural Places Program 2007). Even where there are no visible signs of human usage, there may be historic events, culture hero stories, or significant usages of an area.

> These kinds of activities can give cultural meaning to natural features in the landscape, such as hills, rocks, caves, water, trees, animals and plants. Natural features remind people of stories about events in their history and of how important animals, plants and spirits have always been for the people of their community (NWT Cultural Places Program 2007).

In a cultural values assessment for the proposed Sahyoue-Edacho National Historic Site in Canada's Northwest Territories, the importance of this cultural landscape was described as follows:

> These areas represent the relationship between the Sahtugotine and their land; the landscapes of Sahyoue and Edacho blend the natural and spiritual worlds of the Sahtugotine, creating a living web of myth and memory…[they] remain important places not only for subsistence, but also for spiritual renewal and a sense of cultural identity (Janes and Hanks 2003).

Cultural landscapes are the embodiment of cultural heritage at a grand scale, incorporating individual sites of heritage resources into a broader network of stories and collective memory tied to the land, essential to defining the people similarly tied to that land (Rose 1996). Buggey (2004) argues that a cultural landscape expresses an indigenous group's "unity with the natural and spiritual environment. It embodies their traditional knowledge of spirits, places, land uses, and ecology. Material remains of the association may be prominent, but will often be minimal or absent."

It is this lack of tangible presence that can be a problem for the outsider, both in coming to understand the values and priorities of an indigenous group and in determining the significance of impacts on cultural assets during impact assessment. The value of different cultural assets cannot be discerned from without; it must be determined from within, at least with the assistance of the indigenous peoples themselves.

In general, three types of "values" are associated with cultural assets:

1. *Use value* is associated with continuing utilization of an area. For example, an area may be important because it is used currently by culture holders on a day-to-day, seasonal, or annual basis for practice of the traditional economy.
2. *Bequest value* is associated with any cultural assets the holder desires to pass on unchanged to future generations and the satisfaction and peace of mind associated with this knowledge.
3. *Existence value* (sometimes known as wilderness value) is the benefit attributable to knowledge that an environmental resource exists, even in areas that are not used and likely will never be used by the individual. These values can be important in situations where indigenous groups are unable to access parts of their traditional lands for extended periods of time because, for instance, they are distant from settlements.

Often places carry multiple layers of meaning, deepening their value to culture holders. Cultural landscapes such as the Upper Thelon River basin in the Northwest Territories may be all the more important and sensitive because they hold use, bequest, and existence/wilderness values. For the Denesoline people, the area is "the place where God began." It is critical habitat for wildlife including the vitally important barren ground caribou herds. It is viewed as a place with which the Denesoline desire continued connection in its current, relatively pristine status to sustain their cultural history, traditional activities, and spiritual base into future generations (Lutsel K'e Dene First Nation 2008). The Upper Thelon River basin is also a highly valued conservation and ecotourism area (one of the largest forested areas north of the tree line), valued for its existence/wilderness value both among Canadians and internationally. This serves as a reminder that while impacts from mining may be primarily felt by those closest to the development (indigenous peoples), there may be impacts on other cultures associated with special places.

The existence and bequest values associated with special places and spaces belie using quantitative indicators like the existence of known physical heritage resources or intensity of current use for traditional activities as the only measuring sticks. Some places take much of their value from the peace of mind people enjoy from knowing they are not being disturbed. Other places are important not (or not only) because they house valuable renewable resources, but because they hold power and resources that should be avoided except in times of great need or in very specific circumstances, as, for instance, in a particular ceremonial context (NWT Cultural Places Program 2007). Some places may require avoidance by all people at all times, or should be visited only by particular people who hold specific knowledge, because they can be dangerous to the ignorant or the uninitiated. In many parts of Australia, specific places are variously considered dangerous to men or to women, or to any individual who has not completed specific initiation ceremonies, or to anyone who is not accompanied by custodians who can perform rituals to render the visitors safe. Aboriginal attempts to restrict access to such places are designed not only to protect them from damage, but also to protect non-Aboriginal people who would otherwise suffer serious injury by visiting them inappropriately (O'Faircheallaigh 2008).

Indigenous culture holders take their guardianship of spiritual and cultural places and spaces very seriously, and there may be severe repercussions for their failure to protect them. For instance, the destruction of an important women's cultural site by the Argyle diamond mine in Western Australia in the mid-1990s had serious social consequences for its female custodians, although they fought strenuously to stop the development (Dixon and Dillon 1990).

Culture is not only practiced on the land, it is practiced in the home and in the community. Cultural change is not limited to alterations on the land, but also occurs when social relations change, or when practice of language and other tools of cultural maintenance or one's sense of self, place, and community are altered (Basso 1996; G. Gibson 2008).

Understanding Cultural Differences

The work sites of operating mines, although they are often envisioned as "culture free" areas (Trigger 2005), are themselves one of many cross-cultural "mixing zones" associated with mining in indigenous communities (see Figure 17.3-2). Cultural differences are also felt during initial community engagement, around the negotiating table when agreements are being developed, and during the mine planning and formal cultural impact assessment processes.

Indigenous culture interacts with the institutional culture that prevails in mining camps and companies (Heyman 2004). The mining company can be distinguished by its different sites of operation: the operating mine site is quite distinct from the parent company in managerial style and strategic priorities. A parent company may watch over many operating mines and shepherd old mines out of existence. The management and corporate culture of the individual mine is concerned with the ore bodies on site, the understanding of which changes frequently and often expands as the ore body is newly delineated and made real through development and extraction.

The operating mine is characterized by key values, hierarchical relationships, strict lines of authority and communication, and prescribed policies for managerial and human resource systems. Key values of an operating mine are maximizing productivity and site safety (G. Gibson 2008). A head office's primary and overall responsibility is to maximize economic returns to company shareholders. The means to these culturally prioritized ends may conflict with the primary driver of indigenous culture: to protect the values imbued in the land under which the mineral resource lies.

"People" priorities may differ as well. Cross-cultural leadership qualities, social science skills, and human relations acumen are not necessarily valued prerequisites in mine management, as long as the individual has a track record of productivity. This can and has led to conflicts between indigenous workers and their settler culture managers (G. Gibson 2007, 2008). Indigenous workers are expected to assimilate into the occupational site, discarding, at least temporarily, their own practices and values in favor of those of, primarily, the institutional culture, and secondly, the settler society. As one senior manager put it, workers have to "take off their kimonos" and participate in the "industrial bubble" (Trigger 2005). Participation in the mine site can powerfully affect how individuals are able to reproduce and express their own values and practices (G. Gibson 2008). Exposure to a new workplace in an isolated fly-in/fly-out work camp where the dominant culture is defined by the employer not only exposes the indigenous person to new values and world views of people from settler cultures, but also to the institutional culture of the modern mining workplace. Common differences between the corporate culture of the mining work site and indigenous groups are outlined in Figure 17.3-3.

The difficulties for indigenous workers in adapting back and forth between the industrial production culture and the home community can create major challenges

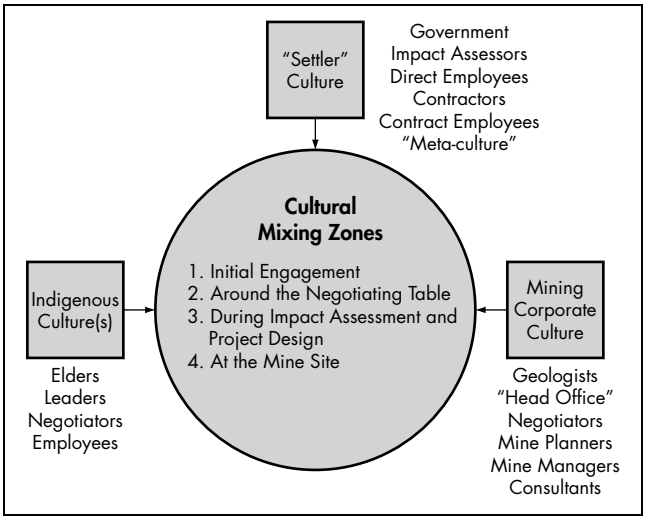


Figure 17.3-2 Cultural "mixing zones" associated with mining activities

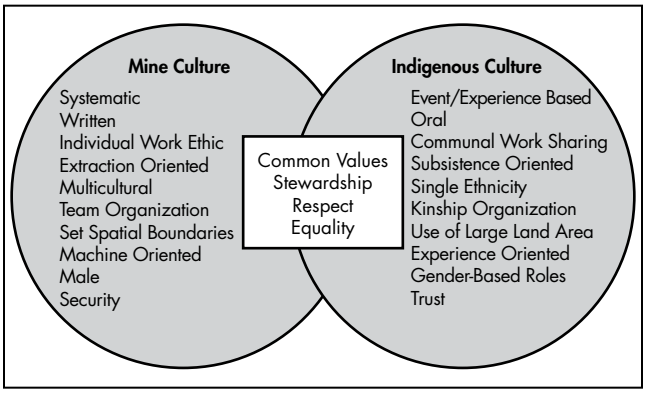


Figure 17.3-3 Comparing values

(O'Faircheallaigh 1995; Barker and Brereton 2005). Attempts to divorce the "workplace" culture from the "home place" culture lead to problems such as cross-cultural disputes and high turnover rates, often associated with increased familial tensions as well as workplace conflict. G. Gibson (2007, 2008) identified the top 10 cross-cultural conflicts encountered between indigenous people and mine managers at the diamond mines in northern Canada, a few of which are discussed here:

- In an area such as the open pit, it takes constant shifting, training, and machine jumping to gain the skills necessary to understand the full range of machines in the area. People who move up in the mines have been afforded these opportunities by their pit managers, given the time to train, and often have a few characteristics the manager sees as necessary. They may be aggressive and self-promoting. These characteristics and values may run contrary to the nature of an indigenous person.
- When a non-kin member of a family dies, corporate-leave policy may constrain managers from allowing a worker to attend the funerals of other than immediate relatives. In a society where kinship is defined by different rules, an indigenous person may need to attend funerals as part of a communal grieving custom.

Table 17.3-1 Elements of cultural difference that may be exposed during a cultural impact assessment

Factor	Mining Culture	Indigenous Culture
"On-the-ground" evidence of culture in area proposed for mining	Found archaeological sites	Stories, trails, use patterns (current and historic), density of place names, archaeological sites and potential for unfound sites, cultural landscapes
Prioritized cultural impact consideration	"Bones and stones"	Tools/images/objects/ stories in combined place memory; innate values of the land
Value focus in cultural impact assessment	Current use value and bequest value associated with physical heritage resources	Historic and current use; bequest value and sometimes intrinsic existence values
Geographic scope of cultural impact assessment	Places and sites that will be physically altered or otherwise directly affected by the proposed development	Places, sites, spaces, cultural landscapes and relationships between them and people, home community, lived experience
What constitutes "heritage resources"?	Heritage resources as physical artifacts only	Heritage resources also include culture holders' shared history and experiences, knowledge of the land and resources, and ways of knowing and transmitting knowledge
Role of intangible cultural resources in cultural impact assessment	Minimal; view that vulnerability of things like language and time spent on the land is linked to cultural change beyond scope of project-specific cultural assessment	Project-specific cultural impact assessment should consider contribution of proposed development to potential erosion of cultural maintenance tools in context of cumulative change

Cultural differences must first be identified before there can be any successful coexistence and cross-cultural engagement of the many cultures on site. Only then can human resources policy be adapted to fit these unique cultural needs (work-site cross-cultural conflict mitigation strategies are discussed later in this chapter).

Obviously, the work site is not the only place where mining companies and indigenous communities interact. Common sources of tension between corporate/settler culture and indigenous cultures in other cultural mixing zones include

- Prioritization by companies of the written word and technical expertise over oral discussions in plain language, often characterized by company use of overly wordy and complex documents, presentations, and other materials about the development;
- Different canons of proof, with Western, "scientific" knowledge being privileged, and characterization of traditional knowledge and oral history, often provided by elders, as "hearsay" or "anecdotal" information without scientific support;
- Access to and control over information, including sensitive traditional knowledge;
- Misinterpretation of cultural signals (e.g., silence in settler society may indicate agreement, while for an indigenous society it may mean nonsupport);
- Conflict between settler laws and traditional indigenous laws, with corporations seeking to rely exclusively on the former;
- Lack of incorporation of ceremonies required prior to, during, and after meetings, breaking the land and agreements; and
- Different understandings of formal arrangements like impact benefit agreements as either final "set in stone" legal settlements (from the mining companies' perspective) or as the starting point for an ongoing, open, and adaptable relationship (from the viewpoint of the indigenous people) (G. Gibson 2008).

Many of these cultural differences are exposed during early community engagement and subsequent cultural impact assessment. Cultural differences not only govern how people interact but also how they perceive the environment and potential for impacts from mining projects. Table 17.3-1 provides a comparison of values in relation to key factors involved in a cultural impact assessment.

Some understanding of the cultural attributes of each other is essential prior to entering a formal relationship. Gaining a contextual knowledge of the community is essential so that relationships are based on mutual understandings of priorities and reasons behind actions. Without this in-depth knowledge, misinterpretation and a lack of trust can emerge. For example, concerns are sometimes raised about whether indigenous culture holders invent, fictionalize, or otherwise augment their histories to make potential cultural impacts appear more severe. This concern surfaces when an impact emerges in the last moment in an environmental assessment. This misperception is linked to the fact that the nature of information sought in order to understand culture is often highly sensitive. Depending on the cultural values of the potentially affected group, it may be hurtful or improper to custodians and to the wider indigenous community for cultural information in the form of traditional or local knowledge to be divulged, and so information may only be released at the very last moment when it becomes clear that to do so is the only way to protect a place. A conflict between the desire of formal impact assessment to quantify cultural assets and their value, and the desire for indigenous groups to retain control over the sensitive cultural knowledge and expression of value of their cultural resources often emerges. Sometimes cultural resource issues are simply too sensitive for public disclosure (King 2004; O'Faircheallaigh 2008). This is often the case with indigenous people who are concerned with the impact of mining activities on spiritual or religious sites. To divulge information about a site may actually put knowledge holders or the place itself, as well as recipients of the knowledge, at risk. This explains the reluctance to divulge it unless knowledge holders become convinced that protecting sites requires releasing information about them, even though revealing their location or significance may reduce their future safety.

EFFECTIVE EARLY COMMUNITY ENGAGEMENT

Early and appropriate community engagement can create a respectful relationship between parties and assist in proper identification of cultural issues that may arise from a project in a specific location, a process referred to as *scoping*. To scope

properly, the developer needs to understand the unique cultural context in which the project is proposed and how mining activities have tended to impact indigenous cultures in this or other similar environments.

There is no broad-brush approach that can be used to understand indigenous peoples' lives. Each indigenous community will be unique, with issues, concerns, and an underlying context that will emerge only through engagement. At the outset of a project, developers need to develop a basic understanding of the cultural context in which they are operating and the unique vulnerabilities and resiliencies of the people involved. Developers need to be aware of

- The historic background of the community;
- The relationship between community members and the environment;
- Demographic characteristics of the community;
- Demographic subgroups within the community who are particularly vulnerable to adverse cultural impacts and are underrepresented;
- Internal (traditional laws and governance) and external (state) political and legal structures and customs;
- The community's relationship with regional, territorial, and federal governments;
- Existing community goals and aspirations for social and cultural well-being, as expressed, for instance, in development plans or proposals for protected areas;
- The cultural values that shape the perspectives of community members; and
- Vulnerabilities and strengths of the community, (e.g., strong language and other cultural programs) and trends in levels of subsistence activity.

Developing this contextual knowledge requires a substantial and early commitment of resources but has important benefits. It assists in accurate issue identification, which allows identification of the proper scope and scale for the required impact assessment. It also helps to identify locally and regionally relevant mechanisms and processes that are already in play to support and protect valued cultural components, such as the regulatory and legislative context for identifying and managing cultural impacts. Regulations on burial sites, cultural items, sacred sites, religious freedoms, and archaeological sites will also need to be understood (King 2004; Parker and King 1990).

Sources of contextual cultural information include

- Land-use plans and supporting documentation;
- Literature on protected areas in the vicinity, which often include some description of cultural values and specific spaces identified as requiring special protection;
- Documentation in relation to prior EIAs in the region in question;
- Published anthropological or sociological literature about the people and places involved;
- State and federal government departments responsible for indigenous populations' health and well-being, and cultural protection;
- Community-specific engagement protocols;
- Museums or cultural institutes dedicated to the study of the indigenous culture, which among other resources often house archaeological databases identifying physical heritage resource locations and areas of high heritage resource potential;
- Ethical guidelines for traditional knowledge collection and dissemination (e.g., MVEIRB 2005a); and
- Existing traditional knowledge reports, which may have certain confidentiality restrictions.

It is prudent to identify and use all available secondary (desktop) sources of information before committing to any dedicated primary (fieldwork) cultural impact assessment. Besides creating a useful contextual understanding to help avoid misunderstandings during initial engagement, this can also help avoid the "consultation fatigue" often cited by indigenous communities. Despite the obvious differences among indigenous communities, there are also remarkable consistencies in the issues they raise in relation to proposed developments. Therefore, it may be useful when determining the "cultural risk profile" of a development to consider case studies of similar developments in other indigenous environments. Table 17.3-2 identifies some of the activities a developer may want to undertake prior to formal engagement with a community.

Table 17.3-2 Checklist of recommended activities prior to and during early community engagement

☐ Identify what land access agreements will be required in the area of the proposed development, and identify any overlapping land claims by different culture groups.
☐ Identify relevant community plans, regional land-use plans, and other planning documents. Identify whether the proposed development conforms to these plans.
☐ Use community engagement handbooks and talk to community liaison specialists before developing a consultation strategy. Consider whether a consultant or local expert from the cultural region is required.
☐ Identify which potentially affected communities and culture groups should be consulted initially.
☐ Identify whether each identified community, or the region as a whole, has specific policies or protocols that dictate how developers should conduct early community engagement, or for the collection and protection of traditional knowledge.
☐ Conduct desktop research on the social, economic, and cultural environment of communities that may be affected by the proposed development.
☐ Identify appropriate community contacts.
☐ Produce a preliminary in-house list of potential impacts and public concerns that may be raised (see the "Impacts of Mining on Indigenous Cultures" section later in this chapter). Use information about the proposed development, research on past developments in the potentially affected region, case studies of similar developments, and existing developments in the region as source material for this list.
☐ Distribute a plain-language description of the proposed development to potentially affected communities.
☐ Be familiar with cultural impact issues commonly brought up by indigenous community members, be ready to listen to whether and how these concerns relate to the proposed development, and be prepared to respectfully consider issues that may not seem to you to be directly relevant to the proposed development.
☐ Take responsibility for keeping detailed meeting minutes, be ready to provide them to the community consulted for verification, and establish a flexible and appropriate mechanism for follow-up communications with communities.

Source: Adapted from MVEIRB 2007b.

Early community engagement can be essential in minimizing the type of misunderstandings that can damage, even irrevocably, a project in traditional indigenous territory. Important contextual information about the cultural group in question may in some cases only be unearthed during discussions with key contacts in the communities. The key here is to make sure that contact is "appropriate," meaning it is made through the proper channels according to relevant community engagement protocols. To bypass these protocols, which are set up by the culture holders to protect their interests, can get the engagement process off to a bad start.

Rio Tinto's (Rio's) Marandoo iron ore mine in the Pilbara region of Western Australia offers an example of the costs associated with failure to address cultural heritage protection early and in cooperation with indigenous custodians. The developer complied with the provisions of Western Australia's Aboriginal Heritage Act, but traditional owners believed this legislation offered little by way of effective protection and wished to negotiate stronger protective provisions. After Rio failed to establish a productive engagement with them during exploration for the project, the traditional owners undertook a number of legal actions designed to frustrate Rio's plans and slow development. While Rio eventually overcame these hurdles and secured project approvals, the action of traditional owners resulted in a delay of more than 2 years. This not only led to higher capital costs but meant that Marandoo came onstream too late to take advantage of an expected spike in iron ore prices, and Rio incurred significant losses as a result. Another result was, according to a Rio executive, "a low point in Aboriginal relationships for the company…[and] a legacy of deep distrust and bitterness in the [Aboriginal] communities" (Eggleston 2002). The experience contributed to a fundamental policy shift at Rio Tinto, with the company undertaking a major initiative to establish productive relationships with all indigenous communities located adjacent to its mine sites (Eggleston 2002; O'Faircheallaigh 2006).

IMPACTS OF MINING ON INDIGENOUS CULTURES

A second key task for managers is to understand how a project might impact culture. Many developers enter an area without fully understanding the likely impact of their activities on indigenous culture. This requires both a basic understanding of how mining projects have been known to impact indigenous culture, and specific knowledge of both the proposed development and the cultural context it is proposed to be situated in.

Common Impacts of Mining on Indigenous Cultures

At a workshop in Perth, Australia, in May 2008, EIA practitioners from around the world identified the cultural impacts that most concerned them in relation to mining. The results are indicated, along with the authors' own experiences, in Table 17.3-3 (updated from Gibson et al. 2008). The list is by no means is definitive.

Cultural impacts from mining activities occur to places and people. For example, the first layer of impact may be direct and tangible damage to heritage resource sites through construction, blasting, and earthmoving. As a result, the physical record of a culture group's past occupation may no longer be marked into the landscape (Angelbeck 2008). This destruction can in turn indirectly reduce the ability of people to practice rituals, ceremonies, and traditions, as well as add to disconnection from nature (Pilgrim et al. 2009). The space might also be a fundamental source for food, shelter, clothing, medicine power, or a wellspring for much less tangible but equally important cultural resources (Angelbeck 2008). These changes can have cascading, potentially devastating impacts on a group's cultural history, traditional practices, and sense of place. These cascading impacts are more likely to occur if there are cumulative effects from other human activities contributing to pressures on indigenous cultural resources. Thus, destruction of the land can damage and even destroy cultures that remain closely tied to their environments, which in turn can lead to mental, psychological, and social pathologies (Pilgrim et al. 2009).

Impacts on culture, like all impacts, occur through these direct, indirect, and cumulative processes. Direct impacts are ones specifically caused by the development in question, such as the aforementioned mine construction requiring destruction or removal of physical heritage resources. Indirect impacts are changes caused or contributed to by a development through a chain of events. For example, increased employment of indigenous people at a large mine may reduce their reliance on, and interest in, spending time on the land conducting traditional

Table 17.3-3 Common cultural impacts related to mining projects

Valued Cultural Component(s)	Impact Concern
Cultural heritage resources	• Physical loss of heritage resources and associated spiritual loss
Cultural landscapes and other special spiritual spaces/places	• Visual impacts redefining the way a place/landscape is "seen" in culture (inappropriate use or alteration of landscape can create loss of value and meaning regarding what places on the land can teach us about life)
Culture and land tenure	• Social organization may be tied closely to the system of land tenure, which may also reflect transmission of rights, property, and gender relationships. Changes to land tenure through mining or changes in property ownership may interfere with these relationships.
Overall relationship to land and traditional activities on the land (including practice of traditional economy)	• Sense of disconnection from traditional lands and wellspring of cultural well-being • Less time on the land conducting traditional practices can cause cascading impacts on well-being across a variety of categories (everything from poor diet to erosion of language and ways of learning) • Often linked to perceived and real physical degradation of habitat and lower harvesting success for key wildlife species • Less time in kinship relationships that are critical to transmission of knowledge
Values and belief systems	• "Culture free" working conditions can erode cultural values or lead to alienation • Large-scale in-migration can alter cultural norms and values • Spiritual/religious change, loss of faith in traditions
Methods of cultural transmission (including language, oral history, intergeneration relations, social networks)	• Contributions to changes in social structures leading to cultural loss (e.g., decline of intergenerational culture transmission) • Contribution to cumulative loss of indigenous language, even to extinction • Linked to changing political structures and power relations (e.g., elders vs. politicians or business-people as leaders)
Sense of self; sense of place; overall well-being	• Loss of sense of control over one's own fate • Health impact outcomes caused by cultural impacts (e.g., increased suicide, alcohol and drug addiction, poor diet)

pursuits. This can in some cases contribute to loss of connection to the land, reduction in intergenerational knowledge transfer, and other damage to maintenance of traditional culture. Alternatively, increased employment at a fly-in/fly-out mine may provide the added income and free time required to spend time on the land, creating an increase in practice of the traditional economy and beneficial cultural outcomes (O'Faircheallaigh 1995). The lesson here is twofold:

1. The term *impact* simply means "change." Impact outcomes are not inherently unidirectional. They can be beneficial or adverse, or either for different people, depending on the cultural context and the way people react to change.
2. Pathways by which impacts occur may be simple and involve single-chain cause–effect relationships (e.g., a bulldozer physically destroying an artifact), or they may involve complex causal chains with a variety of contributing factors along multiple pathways (e.g., less practice of traditional economic activity caused by a combination of increasing involvement in wage economy, fewer harvestable animals, changing demand for country food, and decreasing time on the land). The incorporation of culture holders in predicting both initial changes and their eventual outcomes may be required to create a realistic picture of the most likely impact outcomes in a specific cultural context.

Mine developments do not occur in cultural, economic, or social vacuums where they are the only cause of change. Cultural change is always occurring, and for multiple reasons. A common argument is that the most dominant forces of change in modern industrial society are beyond the capacity of individual developers to manage. This is true. Television, for example, brings into the homes of indigenous people the values and world views of dominant societies. Developers are not being asked to take full responsibility for the cumulative effects of these multiple contributors to cultural change. They are expected instead to recognize the pressures these cumulative effects may be placing on valued cultural components, as part of understanding the local operating environment to which the developer needs to adapt. The "weight of recent history" (Gibson et al. 2008) may need to be considered, as many indigenous communities have been bombarded with the negative effects associated with intrusions by settler cultures. Cumulative effects are often measured through trends in language retention rates, time spent on the land, and traditional harvesting levels. Equally important may be the more-qualitative measures of change that can only be captured by talking to the culture holders about their sense of place, their attachment to special sites and cultural landscapes, and their hopes and fears for the future.

It can be argued that impacts must be interpreted through the filter of the culture holders themselves. Impact significance is socially defined, and these social perceptions are linked to the values of the culture holder. Consider a situation where a defined cultural landscape of importance to an indigenous group is to be used for an open-pit mine. In absolute terms, the physical footprint of the mine and associated infrastructure may account for 1% of the cultural landscape. But that 1% may represent some of the most important hunting grounds and encompass an important spiritual site, and its importance is therefore much greater than its area would suggest. In addition, impacts on that 1% may affect the perceived value of a much wider area. Indeed, in some cases the destruction of one site can impact the values associated with the entire cultural landscape and have even further ramifications—the inability of traditional owners to protect a site may change relationships within the community. In addition, if certain sacred sites are destroyed, rituals and ceremonies may no longer be practiced, which will have far-reaching ramifications for cultural transmission.

The nature of cultural impacts may change over the mine life cycle. During initial exploration, concerns may focus on the cultural appropriateness of using a certain area for any sort of industrial development (MVEIRB 2007b). During mine development, increased activity on the ground may cause increased concern about immediate destruction or damage to specific physical heritage resources, spiritually important landscape features, and key wildlife species. There may also be increased concern about the effects on culture of increased in-migration, of increased indigenous involvement in the wage economy, and of associated changes to the traditional economy. During mining operations, concerns may focus on changing social network structures and familial tensions associated with long-distance commuting (Barker and Brereton 2005) or on workplace cross-cultural conflict (G. Gibson 2008).

Determining a Project's Potential for Cultural Impacts

Determining the potential cultural impacts of a particular mining development requires project- and culture group-specific analysis. Only by examining both the "site" (the project) and the "situation" (the cultural context within which it is embedded) can the developer answer these important questions:

- How much further work is required to fully understand potential cultural impacts?
- What should that work focus on?

The proper identification of impact issues allows a refining of the level of detail required and of the focus of cultural impact assessment work. For example, it makes no sense to conduct additional heritage resource assessments in areas where they have already been conducted or where all parties agree they are unlikely to be found. The exercise becomes especially fruitless if the real issue for the community is the potential for high levels of in-migration to exacerbate pressures on traditional culture. Therefore, although scoping for cultural impact assessment can be done in-house by the mining company, it is preferable to involve indigenous culture holders (e.g., through key contact interviews, focus groups, scoping sessions, or meeting with leaders or other groups) in the initial identification of potential impacts.

A range of warning signs is related both to the nature and scale of the proposed development and the cultural context. The following lists identify some attributes to consider when determining the likelihood that a project may cause or contribute to adverse cultural impacts. The first list identifies characteristics of a proposed project that are indicative of a higher impact potential on culture. The second focuses on aspects of the physical and human environmental context surrounding the proposed development. Again, these lists are far from exhaustive, and context-specific variables will emerge during dialogue between parties.

Site, or project, attributes associated with higher cultural impact potential may include situations such as the following:

- The physical and impact footprint of the project is large (e.g., in terms of the extent to which land, air, water, and wildlife may be adversely impacted).
- New or improved transportation access routes are involved.
- The land management status of an area would need to be changed.
- New, complex, or controversial technology is necessary.
- A high level of in-migration to a region is required.
- Long-distance commuting for all workers is required.
- The land use will lead to irrevocable visible changes (e.g., large-scale open-pit mining).
- It is a brand new development (known as a greenfield).

Situational, or cultural, contexts include the following:

- Indigenous communities/traditional lands/trails are nearby.
- Proposed or existing protected areas are in the vicinity.
- There is known or high potential for physical heritage resources or spiritual sites.
- More than one indigenous group is using the area.
- Wildlife intensity is high in the area, traditional pursuits are widely undertaken, and traditional harvesting is an important part of the economy.
- The area contains unique/rare landscape forms.
- The location is important to indigenous oral histories; that is, there is a high density of indigenous place names in the area.
- People practice agriculture in the region and have local forms of land tenure that is not recognized by the government.
- Other past, present, or reasonably foreseeable future developments contribute to cumulative effects in the area.
- Potentially affected communities are already dealing with the intensity of preexisting social, economic, or cultural problems.

The likelihood of major impacts and the level of effort required to manage them will grow as the number of attributes increases.

DIGGING DEEPER: DATA AND METHODS FOR DETERMINING CULTURAL IMPACTS

This section deals with data collection, impact identification, and predicting the significance of the determined impact. While each cultural impact assessment needs to be developed around the specific cultural context and expectations of the case at hand, there may be guidelines for social and cultural impact assessment from the jurisdiction where the work is being done or from prospective lending agencies (e.g., MVEIRB 2007b; MVEIRB forthcoming; IFC 2007).

In considering levels of effort for cultural impact assessment, a distinction is made between basic, moderate, and comprehensive. Consider, for example, a situation where the only potential cultural impacts identified during scoping are the physical disruption of heritage resources.

- Where there is a low risk of adverse impacts, a *basic* cultural impact assessment may require a summary of existing data on heritage resources and heritage resources potential, and inclusion of respected local knowledge holders to understand the impacts.
- Where there are no known heritage resources but medium-to-high potential for unfound heritage resources, a *moderate* impact assessment may require archaeological studies on the project site and development of a site-specific heritage resource protection plan(s).
- Where there are known heritage resources and high potential for unfound resources, a *comprehensive* impact assessment may require incorporation of traditional knowledge studies alongside local archaeological studies, a regional heritage resources cumulative effects assessment, and identification of a regional heritage resource protection plan, in concert with the responsible government authorities and indigenous groups.

For projects with potential impacts on less-tangible elements of culture, for example, whether they be effects on broader cultural landscapes, key wildlife species and traditional harvesting, or contributions to effects on language retention, a similar distinction from basic to comprehensive can be determined based on the results of scoping. In essence, a basic assessment looks to confirm that significant adverse impacts are unlikely, through secondary data collation and dialogue with potentially affected parties. A moderate assessment is used where there might be significant adverse impacts that require, at a minimum, adaptive management plans to be in place in case they do occur. A comprehensive assessment is based on the premise that adverse cultural impacts are likely, that there may be cumulative effects beyond the scope of the project that need to be factored in, and that aggressive mitigation and a dedicated monitoring system for project effects must be set up by the parties involved.

The discussion in this section assumes that the project in question has at least a moderate additional cultural impact assessment requirement.

Involving Cultural Resource Experts and the Culture Holders

Many approaches can be used to help understand impacts on indigenous cultures, including

- Traditional land-use studies (e.g., Tobias 2000);
- Traditional ecological knowledge studies;
- Physical anthropology or archaeology studies;
- Collection of oral histories from elders and other traditional knowledge holders;
- Linguistic and kinship studies;
- Place names research and other ethnogeographic studies;
- Cultural landscape studies;
- Land-use planning, including proposals for protected areas;
- Focus groups, interviews with key contacts, and town hall sessions geared toward identifying valued cultural components and concerns for them;
- Analysis of statistical trends in appropriate cultural indicators, usually collected by a bureau of statistics or similar body (e.g., land usage, language proficiency); and
- Community wellness surveys including indicators of cultural resilience and vulnerability.

Effective cross-cultural engagement and cultural impact assessments require input from trained practitioners, just as mining requires application of engineering and geological skills. The concept of a cultural impact assessment itself is often perceived as being "soft" or "fuzzy," a mistake compounded by the fact that this emboldens nonexperts to try to become practitioners without any training. As a result, the

right professionals are often not brought on board to assist with community engagement and cultural impact assessment. Often, the engineer or geologist who seems best suited to consulting with communities—in other words, those with some empathy and communication skills—are given the added task of understanding culture, writing about it in an impact assessment, and finding mitigation strategies to avoid damaging it. The lack of engagement of anthropologists, linguists, or traditional knowledge specialists can lead to real problems down the road through misunderstandings and lack of rigor.

Technical experts alone are not enough. When it comes to documenting archaeological and other tangible impacts, there is no replacement for the expertise of archaeologists and cultural anthropologists, but their work should incorporate (or itself be used to augment) the local expertise of elders and traditional knowledge holders. Although archaeologists can determine the presence of ancestors via marks of their physical passing, they may not be aware of mythological events that have been worked into indigenous imaginations and cultural landscapes. When defining the cultural importance of a place or space, there is no substitute for the words of the culture holders themselves.

Despite this, companies often commission consultants to do desktop studies that include little or no engagement of culture holders themselves and rely entirely on archaeological evidence. This focus on desktop studies has led to an emphasis on summarizing available statistical and physical data to the neglect of the rich oral and indigenous narratives that can emerge locally. These narratives are laden with context and predictive power. Elders, leaders, spiritual healers, and cultural resource experts within a region can help to identify sacred areas, or values that are at stake in a region, and only local people can verify the meaning and extent of impacts on the cultural environment.

The developer should keep in mind that most indigenous communities want to maximize the control they have over cultural impact studies, especially ones that deal with sensitive traditional knowledge about land use, spiritual places and spaces, and other aspects of their culture. Research shows that control over cultural maintenance is linked to population health among indigenous groups. Chandler and Lalonde (2007) show how Aboriginal communities in British Columbia that have strong showings in eight indicators of control over "cultural continuity" (e.g., control over education, cultural facilities) have lower suicide rates than communities that lack control over cultural continuity. Lack of control over aspects of culture can be an impact in itself, one the developer should avoid exacerbating. However, there are often few financial or technical resources available for potentially affected communities to properly engage in studies of cultural impacts during EIA processes. The developer's role may be to assist in funding such studies rather than imposing them on the group from outside.

The parties should, early in their relationship, establish mechanisms for ensuring substantial indigenous control over research processes and cultural information. A basic start might be to give communities the authority to select archaeologists and other nonindigenous technical staff. Further, communities should be in charge of the information flow surrounding cultural heritage identification and protection. In Australia, for example, it is standard practice for indigenous organizations to enter into "cultural studies agreements" or "cultural heritage protocols" that

- Establish agreed processes for cultural heritage work;
- Allow for recognition of indigenous intellectual property in cultural heritage and indigenous control of sensitive information; and
- Provide for provision of reports that meet the developer's need for project permitting, yet avoid inappropriate disclosure or use of information.

Public consultation that occurs during a formal EIA requires specific engagement with expert knowledge holders, but these people may not be able to share knowledge with all indigenous community members, let alone the nonindigenous public. For example, women may only be able to pass knowledge on to young women during their puberty rites, and to pass knowledge to any men would be to violate norms and rules (O'Faircheallaigh 2008). Given the potential for information to be sacred or secret, early consultation should be undertaken with communities before public consultation, so that protocols can be arranged for control, content, and flow of sensitive information (King 2004).

Counting What Counts: Collecting and Analyzing the Right Data

The developer should have identified valued cultural components, the cultural assets people most want to protect and enhance. Appropriate ways to study changes to these cultural assets now need to be determined. Table 17.3-4 identifies some potential indicators for consideration when conducting a cultural impact assessment on mining projects.

Quantitative tools should be used where appropriate to augment qualitative approaches and vice versa. While quantitative indicators (e.g., percentage of language speakers) can surface some trends, the meaning and culturally relevant issues will only emerge from locally situated narratives. Community experts and outside experts can tend to weigh significance differently, prioritize certain issues, and account for change in culturally specific ways. As a result, it is important to include local experts in the definition of the appropriate issues, indicators, and narratives of impact and change. In some cases, quantitative indicators will be developed with and vetted by an appropriate cross section of the residents of potentially affected communities prior to the developer adopting them during any formal impact assessment process. Thresholds of acceptable change for all valued cultural components must also be established, and they cannot be effectively developed without community input. The same applies to qualitative narratives of the impacts and significance of these impacts.

As mentioned earlier, impacts on tangible culture, especially physical heritage resources, are the ones most often studied during a formal environmental impact assessment, and impact assessments for mining projects have largely tackled the concept of culture through a technical, "scientific" lens. As a result, only a limited range of indicators (almost entirely quantitative) have emerged to "capture" the concept of what constitutes culture and to assess potential cultural impacts (e.g., number of historic sites, numbers of traplines, language retention rates). Qualitative means of measuring cultural change have largely been neglected, often because they are difficult and time-consuming to measure, in part because relevant data are not available from published sources. Another reason for the limited use of qualitative data is a bias in the EIA business (of which cultural impact assessment is but a part) against what some professionals and EIA professionals

Table 17.3-4 Example valued cultural components, criteria, and indicators

Valued Component(s)	Indigenous Group Goal/Priority	Criteria or Indicator
Heritage resources	Protecting heritage and cultural resources	• Historic properties, including buildings, sites, structures, and signs • Cultural items, such as sacred objects and funerary items • Archaeological resources • Sites, features, or artifacts that are old (the specific required age is often determined by archaeological resource protection acts the developer needs to be familiar with) • Archaeological, historic, and scientific data that is usually found in association with archaeological resources and historic properties but may exist independently • Sacred sites
Cultural landscapes and other special spiritual spaces/places	The aesthetic, cultural, archaeological, and/or spiritual value attributed to places	• Reported time spent on the land for transmission of practices, language, oral histories • Types of cultural uses of and associations with the potentially affected area • Place names • Continuity of traditions • Relationship of site to other cultural sites in the landscape
Land tenure	Maintenance of traditional economic structures and forms of land tenure	• Landownership or traditional patterns of agriculture
Overall relationship to the land and traditional activities on the land (including practice of traditional economy)	Continuity of knowledge and practice	• Recreational and traditional economy that involves access to the land • Valuation of alternative land uses (i.e., tourism vs. hunting vs. industry) • Percentage of country food in diet • Number of traplines, or traditional use registered or known in area
Values and beliefs	Protection of indigenous culture in occupational sites and at home site	• Cultural norms, values, and beliefs that might be impacted at work sites
Methods of cultural transmission (including language, oral history, inter-generation relations, social networks)	Maintenance of traditional language, education, laws, and traditions; maintenance of cultural ties	• Practices: rituals, religious rites, and ceremonial activities • Percentage speaking indigenous language at home • Percentage retention of education, laws, and traditions • Density of relationships and networks in the culture
Sense of self, sense of place, overall well-being	Ability to influence decisions on critical cultural issues	• Percentage of gambling, reported crimes, substance abuse • Reports of sense of control over decision making

regard as "anecdotal" evidence. The stories, oral histories, and words of the culture holders are often discounted in favor of evidence that is easier to quantify.

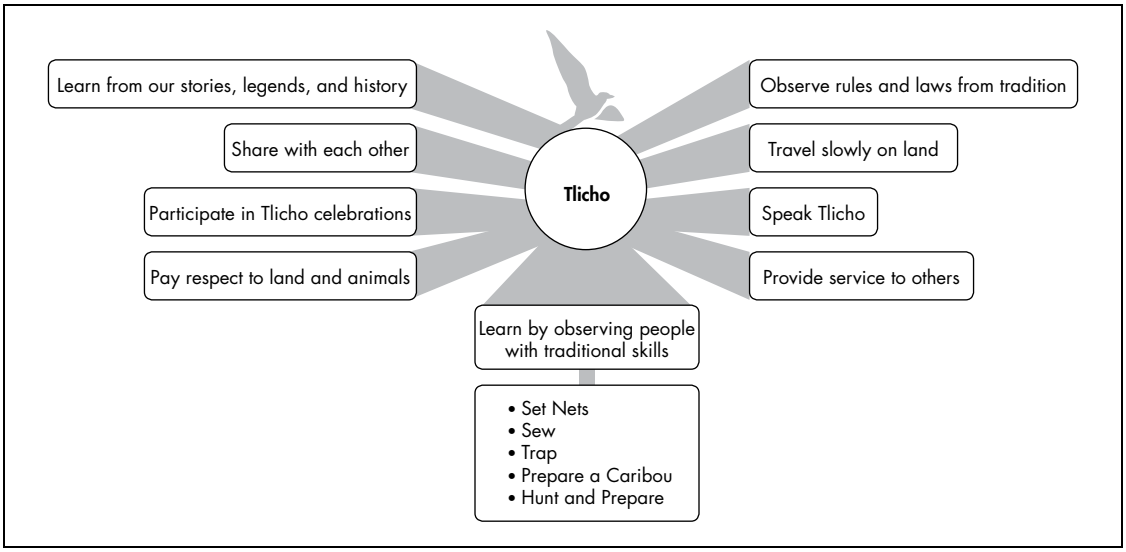
When cultural practices are quantified (e.g., language, hunting, trapping, and food consumption), the indicators may be used as inappropriate proxies for what culture *is*, so that when one is not practicing this activity, one is not a "cultural" being. Local accounts of culture are usually about much more than participation in "traditional" practices. Elders will often tell complex stories that relate to values, roles, and relationships that are reinforced in meetings, at church, and in daily activities. These things cannot be reduced to numbers. They are complex concepts that are only reflected and reinforced through talk and practice. What holds the group together is more than participating in activities and speaking the language; it is also engaging and reengaging relationships to other people and animals, both past and present. Forces that interfere with unity, relationships, and the continuity of roles are what elders often define as threats or impacts (G. Gibson 2008). This reveals the importance of surfacing both qualitative and quantitative measures and narratives of impact.

A downside of focusing exclusively on quantitative data is that it can tell you *what* is happening, but rarely if ever can it tell you *why* change is occurring, especially on intangible assets of culture. Researchers tend to use available quantitative data on valued cultural components to get a sense of status and trends, looking for cultural "hot spots"—those valued cultural components that are in decline. Researchers then dig deeper using qualitative assessment tools to get at the reasons behind the phenomena.

A community's issues and concerns are often best surfaced through in-depth interviews, focus groups, or community meetings. Some training and knowledge of local context and politics is required to be effective. Talking to the wrong person about sensitive issues and places (e.g., a member of a family who does not use a certain area) can result in inaccurate information and inadvertently even raise tensions within the community.

In identifying appropriate criteria and indicators of cultural impacts and cultural resilience, it is important to consider not only readily available statistical data on culture collected by settler government agencies, but also community-based data and analysis. Figure 17.3-4 reveals the kinds of criteria for cultural transmission that an elder (in this case, a group of Tlicho elders from Canada's Northwest Territories) might raise with respect to the possible impacts from a proposed development. These are the criteria of cultural preservation that are often raised in a collective setting—the next step is connecting these criteria to the key issues and concerns that emerge from the proposed project. For example, if these are the areas the community most values and wants to enhance, the question of how mining can embrace these aspects might be addressed. Thus, Canadian mines operating in indigenous territory can bring in elders to tell stories, encouraging spiritual and cultural celebrations on site.

While these criteria do not lend themselves to quantification (e.g., it makes little sense to count the number of stories told over a year), the collective reflection in community settings on the issues and practices helps to imagine mitigation measures for a proposed development. Such community-based initiatives are more likely to be focused on criteria, stories, and indicators deemed most relevant to cultural health and maintenance by the culture holders themselves (G. Gibson 2008). Community-based initiatives may also have the added



Source: G. Gibson 2008.

Figure 17.3-4 Community-defined criteria of cultural preservation

benefit of linking data collected to goal statements or thresholds of acceptable change that can assist in the determination of whether additional impacts will be significant. Where thresholds are discussed, there is the possibility of then defining when and how action will be taken, if significant changes come to pass.

Impact Prediction and Significance Considerations

A robust and defensible picture of potential impacts must be developed before effective mitigation strategies can be identified. Consider a situation where the question at hand is, What is the potential for impacts on heritage resources from Development X?, yet no on-the-ground archaeological studies have yet been conducted. How can a credible answer be derived?

The absence of physical evidence cannot be assumed in this case to be "evidence of absence" in terms of a location's cultural importance. In this case, different sources on the area's cultural resource potential can be consulted. Government officials responsible for assessing heritage resource potential across the jurisdiction, traditional land users, elders, and other traditional knowledge holders can be consulted to establish warning signs of the potential presence of unfound heritage resources or other cultural assets, which may include identification of

- The prevalence of trails, important hunting locations, meeting places, or historic habitations in the vicinity of the proposed development;
- Historic and current use of the area;
- Identification of heritage resources found around, rather than within, the project area;
- The role of the place and paths to it in the oral history of the culture, including density of place names;
- The ranges of important wildlife;
- Types of landforms, water bodies, and other physical characteristics that make it more likely to be used for cultural purposes;
- Whether the location fits into a larger cultural landscape, and how it relates to the whole; and
- The expressed desire by the culture holders to protect the location as it is, as seen, for instance, in public statements of concern, or attempts to withdraw the area from development or find other legal protections.

Using multiple indicators of cultural impact potential triangulated by several sources allows for an estimation of the importance of the area to a culture including, but not limited to, the presence of physical heritage resources. And this initial estimation can be accomplished in some cases without requiring a costly heritage resource impact assessment.

After cultural impact potential is identified, determination of cultural significance must be embraced as a social rather than a technical process. Because culture is a complex and always evolving human phenomenon, it is essential to focus not only on what impacts culture can absorb (a technically estimated threshold of *manageable* change), but also on what the culture holders are willing to take (a socially derived threshold of *acceptable* change). Therefore, when the time comes to assign significance to an impact on culture, a central voice should be that of the culture holders (Winds and Voices Environmental Services Inc. 2000). They are the primary experts on what will change or potentially be affected, because the significance of a site or space may not be apparent to the outsider, to a developer, assessment body, or regulatory agency. As the MVEIRB put it when deciding that a de facto burial ground under Great Slave Lake was not an appropriate location for exploration drilling:

> The Review Board relied on the most reliable and accurate experts it could when making its determination about cultural impacts—the Aboriginal cultural groups themselves. Cultural impacts are best identified and addressed when relayed by the holders of the cultural knowledge, the community members themselves (MVEIRB 2005b).

In indigenous cultures where oral history is the canon of proof (Andrews and Buggey 2008), the evidence of traditional knowledge holders such as elders and other land users,

Table 17.3-5 Factors to consider when determining cultural impact significance

Significance Factor	Questions the Assessors May Want to Consider
Nature of impact	• Which valued cultural components does the impact threaten? How does it threaten them? What pathway would it occur by? Knowledge of pathways of impact that can be blocked or managed by mitigation is essential to minimizing or eliminating impacts.
Situational context	• What is the evidence for use, bequest, and existence values associated with the potentially affected area? • How sensitive are the valued cultural components to change—will a little bit of change fundamentally alter the values associated with the place? • Is the place or landscape one of many similar important places or is it the "one and only"? • Is there the potential for cascading impacts from this cultural change? • Are there cumulative effects that need to be considered from other developments? • Does the area that will be impacted have multiple uses by many different groups? • How vulnerable or resilient are the valued cultural components that may be affected? What is the current status and trends for indicators related to them? Less-resilient elements of culture, if they are deemed important by culture holders, should be prioritized for mitigation.
Trade-offs	• Will there be beneficial effects to offset adverse effects? Will the beneficial effects be on the same valued components or different? For example, economic growth at the expense of cultural decline may require mitigation that takes financial capital and converts it to cultural assets. • Are there any activities or values associated with the location to be impacted that will be inevitably constrained, altered, or eliminated if the development proceeds?
Capacity to manage and social acceptability	• Will any identified thresholds of acceptable change be approached or breached if the development proceeds? Are there thresholds already breached that increase the significance of any additional changes? Does the predicted change exceed the existing capacity of the community and government services to absorb the change? • Are there management plans in place and can they handle the likely increased change?
Geographic area and distribution	• How many communities, culture groups, areas of cultural importance, and valued cultural components will be impacted by this development?
Likelihood	• Is there a greater than 50% chance the impact will occur? Does the estimated magnitude of the change make a 50% likelihood too high to consider? What is the estimate based on and what assumptions were used? Who made the estimation? Are there people who disagree with it?
Impact equity	• Are certain cultural groups or subgroups within a culture more likely to be impacted? For example, elders, people who rely on the traditional economy and live on the land, women, and youth, in that order, are often more likely to be impacted by adverse cultural change.
Public concern	• Is there a high level and wide cross section of the culture group (and others) expressing concern about the proposed activity?

affirmed by other culture holders, is not anecdotal evidence but rather the lived reality.

In assessing the authenticity of a claim about cultural impact, it is telling to look at the range of culture holders who will attest to the significance of the affected cultural site, space, or activity. Thus, significance is clearly established if an impact surfaces again and again in interviews, community meetings, focus groups, or public hearings.

A range of factors may be considered in weighting cultural impact significance, as identified in Table 17.3-5.

The situational context is often more important than the project context for determining the potential for significant adverse cultural impacts. The scale and meaning of cultural impacts cannot be foretold solely on the basis of project size. Just as severe water quality issues can emerge from the smallest of mines if acid mine drainage occurs, so too can a cultural impact from a small drilling program be significant if it is in a location of high sensitivity. Further, impacts on one part of a cultural landscape may have wider and more profound impacts on the whole, regardless of the scope or scale of the activity that causes the impact. Both of these issues were addressed in an MVEIRB decision to recommend rejection of an exploration drilling program in the Upper Thelon River basin of Canada's Northwest Territories:

> The Review Board is of the view that the degree of biophysical impact on the area is not always commensurate with the magnitude of the cultural impact experienced by the people who value it. The scale of the project is not the main consideration in this case. The project is small, but the issues are much bigger because the proposed development is located in a landscape of such vital cultural importance. In this case, the Review Board heard that this development, regardless of its size, cannot be reconciled with the values placed on the area where it is proposed (MVEIRB 2007a).

Because of rare instances like those noted here where developments are not allowed to proceed, the term *significance* can create fear in the minds of developers because there is an impression that significance is all about *whether* a project should proceed ("go" versus "no-go" zones). It is not always possible to mitigate, and as a result, indigenous groups will in some cases accept nothing but complete rejection of a project. Nonetheless, such outright rejection is the extremely rare exception rather than the rule. Significance determination is almost always more about helping to determine *how* a project should proceed, under what conditions, and with what level of cultural protection. Far from being "project breakers," most adverse impacts on culture initially deemed significant can be dealt with through proactive, cost-effective mitigation. The next section examines how appropriate mitigation measures can both minimize adverse impacts and maximize benefits to all parties.

CULTURAL IMPACT MITIGATION AND CULTURAL ENHANCEMENT STRATEGIES

General Considerations

Mitigation measures to avoid, reduce, or compensate for adverse cultural impacts and to enhance cultural resilience run

the gamut from complex re-routing of the proposed activity to small, day-to-day accommodations at the work site. For example, the type of food, the norms for recreation, the support for workers unfamiliar with the wage economy, the involvement of community-based elders and spiritual leaders and healers, and the naming of features within the site can all become a focus of cultural enhancement measures (Barker and Brereton 2005; G. Gibson 2008).

In many jurisdictions legislation nominally designed to protect indigenous heritage affords few if any enforceable tools to protect the complex and varied constituents of culture. This is particularly true for impacts on less-tangible elements of culture, such as cultural landscapes, which are often only given commemorative designations without attendant substantive land-use restrictions (NWT Cultural Places Program 2007), and impacts on cultural maintenance factors like language preservation. In the absence of formal statutory legal requirements, developers face two choices: They can negotiate effective mitigation measures with indigenous groups or they can face the risks of project delay and ultimately abandonment created by these groups as they use direct action and alternative legal or regulatory mechanisms, such as EIAs, to help make up for the absence of legislative protection.

It is in the interest of developers to prioritize cultural impact mitigation, not only because it helps companies gain the required social license to operate from the cultural group, but because successful mitigation, and cross-cultural engagement more generally, offer other substantial benefits. For instance, it can help make the work environment a welcoming one for indigenous people alongside the settler culture work force, which minimizes costly turnover and time spent on workplace conflict resolution. It can also help to demonstrate the corporate social responsibility expected by shareholders, funders, and the general public alike.

Finding appropriate means to minimize impacts on culture from a mining development requires asking

- What is the likely impact outcome?
- What are the pathways/triggers by which this impact might occur?
- How can this outcome be avoided, minimized, or compensated for (in that order of prioritization)?
- Does the proposed mitigation deal with the root of the problem (pathway/trigger)?
- What alternative mitigations are available and what are the advantages and disadvantages of each? What is the best mitigation option?
- Is the proposed mitigation acceptable to the impacted group?
- What have the cultural holders themselves proposed for avoidance, minimization, or compensation?

Mitigation should focus on altering the triggers of adverse change so that valued cultural resources are, at a minimum, not eroded and, at best, enhanced. Effective mitigation prioritizes elimination of the impact, but if this cannot be done, then impacts are controlled or minimized. Compensation should always be the last resort. One of the main reasons why impact avoidance or minimization is much preferred to compensation is that it is extremely hard and usually inappropriate to place a monetary value on cultural assets. Another is that monetary compensation can in some cases actually exacerbate other social, economic, or cultural impacts on indigenous communities rather than minimize them (Gibson and Klinck 2005). If the pathways by which culture is eroding are sought out, however, targeted investments of compensatory funds can go directly into avoiding root causes or bolstering cultural resilience strategies, such as bush camps, funding for traditional harvesters to get out on the land, documentation and dissemination of elders' knowledge and stories, and grade school cultural curriculum development.

Looking beyond adverse impact mitigation, developers can obtain substantial benefits by seeking to incorporate indigenous culture into the everyday life of mining projects. In particular, this can help create a workplace conducive to indigenous recruitment and retention, with major economic benefits for the mining company (O'Faircheallaigh 2010).

Specific Examples

Different types of mitigation measures should be established and managed at different stages of project development. For example, early consultation may serve to identify areas that are sacred, and the developer may remove these areas from its development plans. Cultural protection provisions may also be recommended in EIA reports issued by regulators and attached to licenses and permits. Mitigation commitments should be sought early and redefined as necessary as the development and relations between parties mature, and when potential impacts emerge via the monitoring systems established.

Table 17.3-6 comprises a range of mitigations that have been used in projects. However, mitigation will always have a site-specific element, and so new measures will emerge tailored to the local context.

In evaluating the strength of cultural heritage protections of a large number of agreements, O'Faircheallaigh (2008) identified two key criteria: the level of protection sought; and the activities, processes, and resources applied to ensure that this level of protection is actually realized on the ground. In relation to the first, he identified five distinct and alternative levels of protection. At one extreme, sites or areas of significance may be damaged or destroyed by project development without any reference to indigenous people. At the other, there is an unqualified requirement to avoid damage to cultural heritage, even if this involves major changes to project design and/or a decision not to develop specific ore bodies. In relation to activities, processes, and resources, he identified six key areas. In this case the criteria do not represent alternatives but are cumulative; that is, the more of them that are present, the more likely it is that the level of protection sought will actually be realized. The six key areas are

1. Provisions that maximize indigenous control of site clearance and heritage management processes;
2. Provision of financial and other resources to support cultural heritage identification and management;
3. Measures to enhance an indigenous community's general capacity for cultural heritage protection;
4. Explicit protection of any cultural knowledge provided by indigenous people as part of the cultural heritage protection regime;
5. Provisions that allow traditional owners to temporarily stop project activities to protect previously unknown sites; and
6. More general measures designed to support a *system* of cultural heritage protection.

O'Faircheallaigh (2008) found that the highest levels of cultural heritage protection were required in only one-fifth of

Table 17.3-6 Cultural and social measures from environmental assessments and contractual agreements

Area	Principles and Options
Cultural heritage protection	• Principle of avoiding damage, then minimize and compensate as last resort • Measures to avoid damage to cultural sites • Description of meaning of damage to, or interference with, site/object • Site clearance guidelines including time frame for site clearance reports • Prohibitions on interference with cultural spaces and archaeological sites • Indigenous access to areas that contain sacred sites (e.g., traditional owners at the Rio Tinto Argyle diamond mine in Western Australia have access to areas of the mining lease that contain such sites) • Employment of local cultural heritage protection monitors • Monitoring guidelines • Protocols for site or object management • Confidentiality of culturally sensitive information • Consequences if mining party defaults • Process or compensation for site/object damage
Cultural landscapes and other special spiritual spaces/places	• Protection of sacred sites • Continued practice of rituals to maintain site or protect visitors from harm
Culture and land tenure	• Protection or creation of alternative sites or livelihoods for agriculturalists
Overall relationship to the land and traditional activities on the land (including practice of traditional economy)	• Compensation for lost revenues from trapping and fishing caused by damage to equipment, loss of animals, including direct loss if animals are not in area or if there is increased cost associated with additional travel • Assistance in location of new areas • Rotation change such as moving from a 3-day rotation schedule to a 2-week rotation schedule to allow for time out harvesting on the land
Values	• Measures to control interactions of "outside" workers from large accommodation camps with small, primarily indigenous communities • Family and community support (e.g., money management, training) • Counseling in locally appropriate way and in own language for worker and for entire family • Flexible human resource policies that acknowledge different models of kinship • "Country" or "bush" food kitchens, and the ability to prepare food in the culturally appropriate style • Provisions for country food on the main dining menu • Possibilities or spaces for the practices, rituals, and spirituality of the indigenous peoples
Methods of cultural transmission (including language, oral history, intergeneration relations)	• Site visits by elders and conduct of ceremonial activities on site (e.g., the cultural practice of mantha, a ritual done to protect workers at the Argyle diamond mine or naming of landscape features using Aboriginal place names) • Special leave for culture and for family or community crisis • Creation of community–corporation relationship mechanisms such as family visits and out-on-the-land trips • Language-promotion activities • Community involvement in defining, monitoring, and analyzing cultural impact
Social networks	• Family accommodations and spousal visits • Develop family separation, life skills, drug and alcohol or gambling addictions programs; prepare people for lifestyle associated with development or rotation schedule management • Policies on consumption and use of alcohol and drugs • Training for and access to computers and availability of phone and computer access to family
Sense of self, sense of place, overall well-being	• Cultural exchanges of senior managers and senior indigenous leaders • Cultural sensitivity training for all incoming senior managers to ensure awareness and sensitivity to culture • Funding traditional knowledge studies—information collection and dissemination in an appropriate manner and led by the cultural group • Broader support for cultural activities (e.g., support for cultural activities) • Antiracism training and gender sensitivity training • Flexible work schedules and equal time on and off of rotations • Cross-cultural training workshops

the agreements he evaluated. This suggests that negotiated agreements are only able to achieve protection in some cases, given that the bargaining power of some indigenous communities is limited. Where the community is well-resourced and politically powerful, there is often a strong ability to use agreements to achieve cultural heritage protection. In other cases, indigenous peoples will need to use all available avenues for protection.

O'Faircheallaigh's research also highlighted the fact that even where a system is created for cultural heritage protection, there is often a systematic failure to implement it effectively. There are many reasons for this failure to implement agreements, such as changing circumstances, lack of resources, failure to review and renegotiate mechanisms over time, failure to anticipate all the issues that emerge, and inappropriate allocation of responsibility or authority for cultural heritage protection (O'Faircheallaigh 2003). This highlights the need to focus carefully on the mechanisms and resources needed to ensure that cultural heritage protection provisions continue to operate and work as previously envisioned.

Mining companies must develop, in collaboration with affected indigenous groups, specific measures to mitigate cultural impacts and support cross-cultural engagement, and ensure that these are not put in place solely to secure project

approval but are maintained over the longer term. This requires substantial and ongoing commitment by senior management of the time and financial resources required to support indigenous participation and implementation of protective measures.

Workplace-specific mitigation is an essential element of any cultural impact management plan. Some of the measures that increase workplace satisfaction include antiracism training, gender sensitivity training, and policies on harassment, racism, and cultural leave for bereavement (G. Gibson 2008). In addition, the nature and length of a rotation will be of critical importance (Shrimpton and Storey 2001, 1989); an equal rotation (e.g., 2 weeks on and 2 weeks off) is often the most sustainable for families and workers. Exit interviews with Aboriginal miners at the Century mine in north Queensland found that appropriate support and mentoring for workers was essential (Barker and Brereton 2005). The commitment involved in providing this support is well justified in terms of the role it plays in securing an ongoing license to operate, in avoiding the serious investment risks that can occur if indigenous heritage is damaged, and in terms of the value that can accrue to the enterprise from the positive effects of cross-cultural engagement.

CONCLUSION

Using the tools outlined in this chapter will help developers get past the fear of the unknown and embrace the opportunities presented by cultural impact assessment and cross-cultural engagement. A mindset change is at least as important as a new toolkit. It is critical to remember that finding an adverse cultural impact does not in most cases mean a project will not proceed, and that avoidance of thorough and early cultural impact assessment is much more likely to create a high-risk situation than avoid one.

A good cultural impact assessment can do more than merely provide evidence for the EIA processes. Corporations can gain public prestige both by effectively engaging with indigenous communities and by supporting strong community-based identification, management, and protection of cultural assets. Cross-cultural engagement can also generate substantial value for enterprises by, for instance, allowing them to gain access to stable, skilled work forces that are committed to the regions in which minerals are developed.

Cooperative indigenous–company action to enhance culture can also generate benefits for indigenous culture holders. It can provide opportunities for indigenous people to document and appreciate their own history and culture. Where potential impacts on sacred land are being studied, the stories, songs, and practices that are central to this place may be archived and rejuvenated for younger generations of indigenous people.

The following general rules can make cross-cultural engagement and cultural impact assessments effective, risk-minimizing, and benefit-maximizing processes for all parties.

Key considerations for effective cross-cultural engagement:

- Proceed with caution and prudence. Don't rush in.
- Follow community engagement protocols.
- Be ready to put on the shoes of the "other"; identify and respect differences between your culture and that of the group you are interacting with. Suspend your disbelief early—just because a place has no cultural value or meaning to you, it does not follow that it has no meaning to others.
- Talk to indigenous people and their organizations well in advance of actions that may have impacts; provide information early, fully, and in a form that makes sense to people. Recognize that withholding information is itself a major cause of negative sociocultural impacts.
- Don't assume your own interpretation of what will or will not generate impacts is soundly based in the indigenous context.
- Share control and work cooperatively. For most indigenous people, a process someone else controls is a process they don't trust.
- Provide resources for communities to engage; communities are not in a position to do so, and it is an investment that will yield dividends.
- Use all available avenues, not just the formal EIA process, to engage with communities on cultural issues.
- Do not view cross-cultural engagement as a discrete stage in project life, or simply as one of a series of regulatory hurdles. See it as an integral part of all activities, like health and safety.

Key considerations for effective cultural impact assessment:

- Do early work—acclimatize to the indigenous culture, and understand what mining can do to culture.
- Include traditional knowledge as much as possible, but respect culture-group-defined protocols around gathering, control, and distribution of culturally sensitive information.
- Avoid the "fear factor"; realize that project-threatening impacts are relatively rare. Treat cultural impact assessment instead as an opportunity for proactive relationship building, not as a regulatory hurdle.
- Get beyond "bones and stones"; identify both tangible and intangible valued cultural components.
- See beyond the site to the situation; learn about cultural context.
- Recognize that wisdom sits in places, spaces, and people—all merit attention.
- Adopt the proper scope and scale for your project's cultural impact assessment based on robust scoping.
- Use experts where necessary and appropriate degrees of rigor.
- Don't study for study's sake; exhaust your analysis of existing data sources before starting additional fieldwork.
- Gradually ramp up effort, investment, and specialist advice as the project develops and potential impacts expand.
- Project size is only one factor in the equation—do not assume that the biophysical impact footprint is the only driver of cultural impact potential.
- Support and augment qualitative data with quantitative data, and vice versa.
- The culture holders themselves must play the key role in defining cultural impact significance.
- Consider whether community-defined thresholds of acceptable change are likely to be exceeded for valued cultural components, and work with other responsible parties to minimize additional impact loads.
- Recognize that the work site itself is not culture-free, and develop mitigation to avoid cross-cultural conflict.

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CHAPTER 17.4

Management of the Social Impacts of Mining

Daniel M. Franks

INTRODUCTION

The value of a "social license to operate" is increasingly recognized within the mining industry. Unmitigated negative social impacts have the potential to result in negative publicity, increased litigation, and reputational damage, or to delay, prevent, or close down mining in existing and prospective areas as a result of community concerns. On the other hand, the positive impacts associated with mining projects can be welcomed by communities and governments. Assessment and ongoing management of environmental impacts are relatively common fixtures in the mining industry, as evidenced through formal environmental impact assessment processes and environmental management plans. Only more recently have approaches emerged to link social impact assessment with ongoing management and to proactively respond to social and community issues.

This chapter outlines techniques and processes that assist first in identifying and responding to social issues during planning and then in guiding and monitoring projects during operation through to postclosure. The term *management* is used in this chapter to refer to the coordination of activities in responding to social impacts and social risks. Effective management requires an understanding of social issues, which can be gained through ongoing assessment. Through both assessment and management, the design and implementation of mining activities can be shaped to enhance environmental and community outcomes.

Social impact assessment and management are the responsibility of dedicated community relations practitioners at most mining operations. In addition, however, there is a need for mining engineering professionals to be familiar with such approaches, since effective management requires integration across all aspects of the operation. In recent years, mining companies have increasingly come to recognize the value of using management systems to manage all aspects of their operations. Commencing with the introduction of quality systems and environmental management systems based around key international standards (ISO 9000 and ISO 14000, respectively), the trend now is to bring these specialized areas together into integrated management systems. It is essential that the management of social issues be part of this trend and not considered in isolation from the other parts of the business.

This chapter begins with a discussion of the social impacts of mining and then looks at each of the phases of social impact assessment and management that make up current practice in the field (Figure 17.4-1). These phases are consistent with an adaptive management framework or the plan-do-check-act cycle. Adaptive management emphasizes continuous improvement through an iterative process of planning, implementation, monitoring, and adjustment. The phases of social impact assessment and management include (1) the scoping and formulation of alternatives, (2) profiling and baseline studies, (3) predictive assessment and revision of alternatives, (4) management strategies to avoid and mitigate negative social impacts and enhance positive impacts, (5) monitoring and reporting, and (6) evaluation and review. The chapter concludes with lessons that can inform technical and management roles. Through greater awareness and consideration of social impacts and risks, mining engineering professionals will find their projects are more acceptable to communities and have a greater prospect of success.

SOCIAL IMPACTS AND RISKS

A social impact is something that is experienced or felt (real or perceived) by an individual, social group, or economic unit. Social impacts are the effect of an action (or lack of action) and can be both positive and negative. Social impacts can vary in type and intensity, and over space and time. Examples of social impacts associated with mining operations include employment effects; changes to social services, such as health and childcare or the availability and cost of housing; and cultural change, such as changes in traditional family roles as a result of the demands of mining employment, or even the breakdown of traditional economies due to the introduction of a cash economy. Environmental impacts also have social implications. Mining activities can result in changes to community amenities, health, or the availability and quality of water and land (Table 17.4-1).

Impacts can be direct, such as the impact of noise and dust, or result from indirect pathways, such as road fatalities

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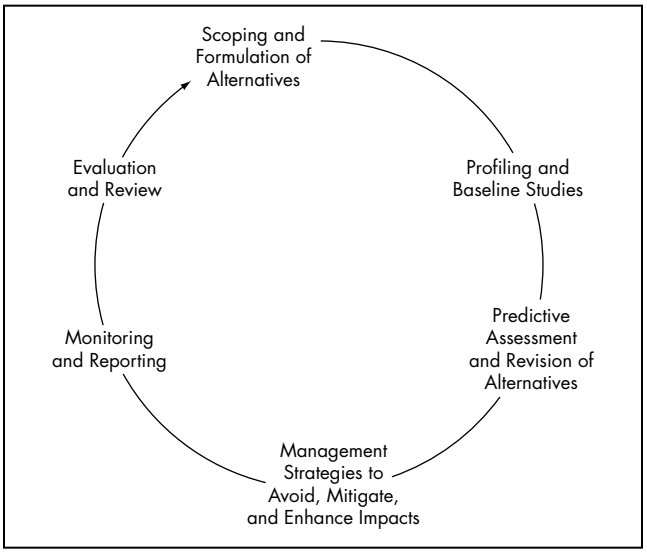


Figure 17.4-1 Phases of social impact assessment and management. The phases should be applied to each stage of the mining life cycle as part of an adaptive management cycle.

resulting from increased traffic in a nearby town servicing a mine. Impacts often accumulate and interact such that they trigger or become associated with other impacts. Cumulative impacts arise from the compounding activities of a single operation or multiple mining and processing operations, as well as from the interaction of mining impacts with other past, current, and future activities that may not be related to mining (Franks et al. 2009a).

A social risk is the potential for an existing or planned project to have an impact on individuals or groups or, conversely, to be impacted by them. Like impacts, social risks are both positive and negative because of the potential for mining to generate social and economic opportunities, such as economic and community development and employment.

The fear of an action can often be as important a generator of impact as the action itself. Perceptions of impact, opportunity, or risk are subjective, and public understandings do not necessarily correspond with scientific perspectives. There are a number of reasons for this. Nontechnical factors, such as personal value systems, previous experience, levels of trust in information sources and methods, and openness to change, all influence the way individuals and communities perceive and respond to change.

PHASES OF SOCIAL IMPACT ASSESSMENT AND MANAGEMENT

Social impact assessment (SIA) is a process for understanding the social issues associated with development. SIA is focused on how to identify, avoid, mitigate, and enhance outcomes for communities and is most effective as an iterative process, rather than a one-off activity at the outset of mining (Vanclay 2003; Becker and Vanclay 2006). Social impact management refers to systems and strategies undertaken during the implementation phases of a development (including exploration) to monitor, report, evaluate, review, and proactively respond to change. Together, SIA and impact management provide a very effective tool to address impacts, if the team conducting the work are well integrated with the overall management of the entire operation.

Not all social impacts are predictable. Because communities and the external environment are dynamic, an element of uncertainty will always be present. Similarly, not all impacts can be avoided or mitigated. However, SIA can provide insights, focus attention, and identify key issues as perceived by stakeholders to predict and anticipate change, and social impact management can assist in proactively responding to the intended and unintended consequences of mining developments.

SIA and impact management are most effective when undertaken across the life cycle of mining and resource processing, encompassing all of the activities from exploration, construction, extraction, and processing, through to postclosure, as well as including recycling and waste management. The varied social impacts across the mine life-cycle phases and the extraction and resource processing stages demand a varied range of approaches to assessment and management. For example, the exploration phase often entails a need to carefully manage community expectations, a challenge in maintaining continuity, and a risk that the outcomes at this early stage can greatly influence the future of relations. At the construction phase, there is invariably a large influx of temporary contract workers into a region that may pose challenges for communities; at closure, impacts are generated by the withdrawal of economic activity and employment, and management requires long-term planning to support alternate futures. Similarly the nature and scale of impacts, and thus the scale and scope of assessment and management, will vary with the type of extraction. Opencut pits involve the removal, processing, storage, and rehabilitation of large amounts of material that can pose risks to waterways or affect the visual amenity of a region; underground mines can create subsidence at the surface, which can affect other land users; heap leach and in-situ leach operations can pose risks to groundwater that may be used by other people and industries; and the various stages of processing will demand varied inputs and create by-products and hazardous materials that can pose risks to communities. The form and level of assessment and management is determined by the significance and scale of the action, as well as by the sensitivity of the community, the location, and its environment.

SIA is a common requirement of regulatory approval processes at the project approvals phase for mining and processing stages in many jurisdictions. These SIAs may be independent or integrated within environmental impact assessments. Beyond this regulatory requirement, SIAs are undertaken by mining operations in response to an actual (or potential) major change in the project or impact on a community—for example, in preparation for closure or for a major expansion. Social impact assessment can also address any particular issues faced during exploration, resettlement, temporary scale-down of operations, or a community development program. Assessments are most useful when maintained and periodically updated as "living" documents to inform decision making. Social impact management approaches, such as community reference panels, social management plans, grievance mechanisms, and community development initiatives, are increasingly required by government legislation, as well as by industry and corporate policies/standards.

The following sections introduce and describe each of the phases of SIA and impact management. Consistent with an adaptive management approach, many of the techniques mentioned may be useful in several of the phases. For example,

Table 17.4-1 Common changes induced by mining activities that can lead to social impacts and risks

Social and Cultural Change	
Population and demographics	In-migration, out-migration, workers' camps, social inclusion, growth or decline of towns, conflict and tensions between social groups
Social infrastructure and services	Demands on and investment in housing, skills (shortages and staff retention), childcare, health, education, and training
Crime and social order	Corruption, domestic violence, sexual violence, substance abuse and trafficking, prostitution, change in social norms, pace of change for vulnerable communities
Culture and customs	Change in traditional family roles, changing production and employment base, effect of cash economy, reduced participation in civil society, community cohesion, sense of place, community leadership, cultural heritage
Community health and safety	Disease, vehicle accidents, spills, alcohol and substance abuse, pollution, interruption to traditional food supply, awareness and treatment programs
Labor	Health and safety, working conditions, remuneration, right to assemble, representation in unions, labor force participation for women
Gender and vulnerable groups	Disproportionate experience of impact and marginalization of vulnerable groups (e.g., women, disabled, aged, ethnic minorities, indigenous, and young), equity in participation and employment
Human rights and security	Abuses by security personnel (government, contractor, company), social disorder in camps, suppression of demonstrations, targeting of activists, rights awareness programs
Economic Change	
Distribution of benefits	Employment, flow of profits, royalties and taxes, training, local business spending, community development and social programs, compensation, managing expectations, equitable distribution across state/regional/local/ethnic/family groups, cash economy
Inflation/deflation	Housing (ownership and rents), food, access to social services
Infrastructure	Demands on, and investment in, roads, rail, ports, sewerage, telecommunications, power and water supplies
Socio-Environmental Change	
Pollution and amenity	Air (e.g., dust), water (e.g., acid and metalliferous drainage, cyanide, riverine and submarine waste disposal), noise, scenic amenity, vibration, radiation, traffic, government capacity to monitor and regulate
Resources (access/competition)	Land, mobility, water (groundwater, river, ocean), mineral resources (artisanal and small-scale mining), cultural heritage, forest resources, human, postmining land use
Resettlement	Consent and consultation for resettlement, compensation, ties to land, adequacy of resettlement housing and facilities, equity, postsettlement conditions, livelihoods
Disturbance	Disruption to economic and social activities (including by exploration), consultation for land access, frequency and timing, compensation
The Process of Change	
Community engagement	Consultation, communication, participation, empowerment, access to decision makers, transparency, timing, inclusiveness—particularly for vulnerable and marginalized groups—respect of customs and authority structures, reporting
Consent	Indigenous sovereignty/title (free, prior, and informed consent), community consent
Participation	Planning, development of programs, monitoring, selection of alternatives and technologies, operational aspects
Remedy	Grievance and dispute resolution, acknowledgment of issues, compensation, mitigation
Agreements	Equity, timely honoring of commitments, issues with delivery, duress, clarity of obligations, capacity and governance (including government capacity to respond to and manage change)
Community development	Participation, adequacy, appropriateness, capacity to facilitate, consistency, prioritization

management approaches such as community reference groups can be effectively used to identify issues during scoping. To avoid duplication, the techniques will be discussed in the phase in which they are most commonly used. Table 17.4-2 presents a variety of social research methods and impact assessment techniques that are often, but not exclusively, applied at the scoping and predictive assessment phases.

Scoping and Formulation of Alternatives

When developing a new project, planners are usually faced with a number of alternative options. Impact assessments provide an opportunity to investigate these alternatives in greater detail to assist decision makers in choosing the most appropriate option.

The scoping phase sets the parameters for the later phases of assessment and management by determining the scale, timing, and focus of the assessment, ascertaining who is likely to be impacted and identifying the actions that are likely to result in impacts.

Scoping will begin by defining the purpose of the assessment and identifying background material that may influence the assessment. This includes regulations and legislation, government and corporate policy and programs, standards, and operating procedures. After these have been taken into account, the next step is to identify project stakeholders.

Early public involvement is crucial to identifying attitudes and perspectives, harnessing local knowledge, and defining processes for further public involvement (more information on the identification of stakeholders appears in "Profiling and Baseline Studies"). With the assistance of stakeholders, the actions that are likely to result in impacts and the extent to which these impacts may be felt can be initially identified. Case studies of related actions may assist in this regard, and any gaps in knowledge should be recorded.

Alternative options should be formulated for later analysis and an initial appraisal of the impacts of these alternatives undertaken. Impacts can be prioritized, in consultation with stakeholders, to narrow down the analysis

Table 17.4-2 Common social and economic research methods and assessment techniques

Method	Description
Literature review	A critical summary of current knowledge on a topic drawn from an existing body of literature.
Interview	A method of primary data collection that consists of in-depth questioning. Interviews may vary according to the type of informant, the type of medium (telephone or face-to-face, individual or group), the setting and recording, and the type of questioning (structured, semistructured, and unstructured).
Focus group	A group interview method where a facilitator poses questions to generate discussion among participants.
Survey/questionnaire	A form with questions used to solicit information from a statistically significant group of respondents. Surveys may be used in SIA to provide data on the characteristics and opinions of a population and may vary according to the choice and wording of questions, the type of instrument (e.g., mail-out, telephone, or face-to-face), the sample size, and sample frame.
Stakeholder analysis	A stakeholder is anyone who can affect, or is affected by, an action. Stakeholder analysis consists of the identification of stakeholders, analysis of their underlying attitudes and motivations, a determination of which stakeholders are most significant, an understanding of their networks and relationships, and the development and implementation of an engagement plan.
Social and economic profile	A process to collect relevant primary and secondary data about a community. The profile is a detailed description of the community, environment, and economy of a region and provides insight into values, priorities, and trends.
Social baseline	An appraisal of the current state of a community or social group including a consideration of trends.
Case study/comparative analysis	Detailed analysis of an example, often used to identify patterns and causal relationships. Case studies can be developed from multiple sources of information and can be supplemented by data gathering.
Cultural heritage mapping	A process for identifying and recording the meaning ascribed to landscapes by cultural groups.
Ethnography	The description of human societies, usually informed by field methods that include interviews, participant observation, and surveys.
Impact pathway analysis	A process-mapping exercise used to predict the pathway of impacts resulting from an action. The method prompts insights into the direct and indirect impacts of actions and their interaction. Also known as change mapping.
Social risk assessment	A participatory technique to identify, prioritize, and respond to the social risks and opportunities faced by an organization or communities. Through a facilitated workshop, key stakeholders determine the consequence and likelihood of each identified risk and develop controls to avoid, mitigate, or enhance priority risks. Also known as social risk and opportunity analysis.
Scenario analysis	A tool used to anticipate change under different plausible future situations. Scenario analysis assists the development of a proactive policy response through the testing of assumptions.
Trend analysis	The collection and analysis of historical and contemporary data to inform the prediction of the future.
Cost–benefit analysis	An economic technique that compares the costs and benefits, usually quantified in monetary terms, for scenarios with and without an action.
Cost-effectiveness analysis	An economic technique that compares the cost-effectiveness of alternative options for achieving an outcome. It is used to identify the alternative with the lowest direct financial cost.
Input-output analysis	An investigation of the relationships and interdependences of an economy through an analysis of the flow of resources. It considers the inputs to industry, transfers between sectors, household consumption, and the outputs of goods produced.
Choice modeling	An experimental technique used in economics to frame trade-offs between different options. The technique estimates the value of options by revealing how respondents are willing to trade them. Information is usually gathered through surveys.

and investigation of the later phases to address key issues of relevance. Cumulative impacts should be considered during this prioritization stage. Although an issue may appear acceptable when considered in isolation, the impact may have greater significance when combined with the impacts of other overlapping activities.

Future and simultaneous proposals should be considered during the scoping phase. Methods range from collation of known projects (announced, approved, or under construction) and government forecasting to industry surveys of potential future activities. Efforts to forecast potential future developments can demonstrate that the proposal is serious about (and investing in) regional futures. Issues of commercial confidentiality have been overcome in other resource provinces through the use of anonymous industry forecasting surveys, usually undertaken by a peak industry body. Such information can also help the industry negotiate with government on future infrastructure needs and priorities.

The output of the scoping phase may be the definition of the objective, scope, scale, priority issues, and terms of reference for the phases of assessment and management to follow.

Profiling and Baseline Studies

Social profiling consists of understanding the communities and stakeholders potentially impacted by the activity through social and economic research. Profiling involves analysis of the social and economic characteristics of a region at a given point of time. Baselines are an appraisal of the state of a community or social group before an activity takes place. Baseline studies provide a benchmark against which potential impacts can be anticipated and change measured. They are also valuable for building mutual understanding. The foundational

information and understandings provided by profiling are useful across all of the phases of extractive projects. Leading companies now routinely require their operations to undertake such studies and update them at regular intervals, particularly when there is any significant change to the scale or shape of a project or community. For example, Anglo American PLC has developed its own Socio-Economic Assessment Toolbox process to assist and encourage their operations to regularly review social impacts at different stages through the life of the project. Through regular review of profiles and baseline studies, or by approaching profiling as a "living" output, longitudinal trends may be observed over time and a more accurate picture of the change processes will be developed.

Knowing the community assists in anticipating how people might respond to change. Actions, even well-meaning ones, that are not socially, culturally, or environmentally grounded in the people and places (potentially) affected by a development, invariably result in poor outcomes. Understanding communities involves an analysis of their relationships and networks and the values that may shape attitudes and behaviors.

Stakeholder analysis is a common profiling technique that may be useful across a number of phases. A stakeholder is someone that has, or potentially has, an interest in an issue—any entity that is affected by, or can affect, a project. Stakeholders can be individuals, groups, neighborhoods, or organizations. Stakeholder analysis consists of the identification of stakeholders, analysis of their underlying attitudes and motivations, a determination of which stakeholders are most significant, an understanding of their networks and relationships, and the development and implementation of an engagement plan. Targeted consultations (or focused plans) may need to be developed for each stakeholder, especially vulnerable groups. Stakeholders have varying degrees of power, legitimacy, and interest in an issue or a project. They may include communities located in the vicinity of mining operations, employees, shareholders, financial institutions, indigenous peoples, nongovernmental organizations, trade unions, and governments and their departments. Stakeholders also include people within resource companies that may be important to the planning, development, and implementation of the action.

Profiling can be assisted by establishing meaningful avenues of two-way dialogue with stakeholders. Dialogue can reveal stakeholder histories and decode cultural meanings and symbols. Knowing the operations or proposed activity is a crucial, but often missing, element of profiling. This step requires an understanding of how the various elements of the proposal fit together and how the work groups involved in planning and implementation make decisions.

Profiling includes analysis of demographic patterns and trends; population characteristics; ethnicity and culture; the local economy; the labor market; land-use and ownership patterns; social and political organization; family and community organization; matters involving health, nutrition, and disease; community infrastructure and services (housing, health, childcare, etc.); expectations and concerns community members have about the project; community needs and desired futures and the capacity to meet these needs; and the vulnerability of social groups.

After a review of secondary information, and the identification of knowledge gaps, a program for the collection of primary data is developed. It is important to know the purpose for the collection of each piece of data. The accumulation of vast amounts of information of marginal use can be overwhelming and leave little time to do analytical work. Methods of primary data collection include quantitative, qualitative, participatory, and technical methods, such as interviews, focus groups, literature reviews, and surveys and case studies, among others.

Predictive Assessment and Revision of Alternatives

In this phase, likely impacts are identified and predicted, and their scale and significance evaluated, using technical and participatory methods. Impact prediction is an opportunity to analyze issues in more detail and to undertake broader consultation and engagement. A number of techniques can be used in social impact assessment to inform this phase (Table 17.4-2). The choice of methods will depend on the nature of the activity and the phase within the mining life cycle. The focus of the analysis will be informed by the prior scoping and profiling phases, and the baseline data will form a foundation for anticipating and predicting risks and impacts. Predictive assessment usually requires the collection of additional data.

Where possible, analysis should consider associated facilities, policies, or programs, such as roads, power transmission lines, and community development initiatives. Predictions should both describe how the proposed activity will contribute to the existing situation and assess the capacity of environmental and social systems to absorb impacts. The methodology and level of confidence should also be clearly explained, as well as any gaps in knowledge.

The outcomes of predictive assessment and analysis are usually prioritized by their scale and level of significance (sometimes as a matrix). They are used to provide feedback to engineers and project developers in order to modify and revise the project, as well as to enable them to decide which proposed project alternative best achieves the objectives of the project while still enhancing social outcomes and avoiding negative impacts.

As part of the trend toward integrated management systems, most mining companies have now adopted extensive risk-management processes that encompass many different elements. It is common to find consequence tables in risk-assessment procedures that include both environmental and community dimensions (Barclay et al. 2009). Social risk assessment, also known as social risk and opportunity analysis, is a technique that can be undertaken during the predictive assessment phase, but it may also be periodically employed to identify and update key social risks and opportunities, and to respond with appropriate strategies, during the impact management phases. Social risk assessments usually involve a workshop of key stakeholders (internal and/or external to the organization) and may be focused on the social risks and opportunities faced by the company or communities. The key tasks are to

1. List the potential social risks and opportunities that may affect (a) the company, and (b) the community;
2. Think about the causal factors for each of the risks/opportunities identified;
3. Make an assessment of the likelihood of the risk/opportunity occurring;
4. Make an assessment of the consequence if the risk/opportunity were to materialize;
5. Prioritize the risks and opportunities based on the assessment; and

6. Develop controls (ways to mitigate or enhance) for the most significant risks and opportunities (Evans et al. 2007).

The outcomes of such a social risk-assessment process should be captured as part of the overall "risk register" for the project. Social risks can be among the highest priority category for mining projects.

Management Strategies to Avoid, Mitigate, and Enhance Impacts

In many circumstances, there is a lack of integration between SIA and the ongoing management of social and economic issues after a project commences or after an operation closes. This can happen for many reasons. SIAs are often conducted by external consultants to serve the regulatory need for independent analysis. In such cases, there can be a lack of continuity between impact assessment and relationship development with community and stakeholders. Another possible reason for a lack of integration is that impact assessment might be the responsibility of a small team, most likely in the community relations section, whereas management requires coordination across all aspects of the operation.

It is important that the outcomes of SIA be embedded across all aspects of the business, similar to the way that health and safety have been embraced at a corporate level in recent years. Social impact management can be formalized into management systems, typified by various series of the International Organization for Standardization, site plans, agreements, development of standard operating procedures for high-risk issues, and systems to handle complaints and grievances. Examples of management procedures to address social issues include formal complaint handling systems, cultural heritage management plans, human rights and cultural awareness training (linked to human resources systems), and local sourcing and purchasing policies.

For some impacts, particularly cumulative impacts, the most effective approach may not be to target a particular impact generated from mining but to invest "off-site" to ameliorate or enhance impacts generated by other activities. At the broader level, there are four approaches to the management of impacts. These are to (1) mitigate or enhance the impacts of past and existing activities; (2) mitigate or enhance the impacts of the project or activity under consideration; (3) mitigate or enhance the impacts of potential future projects; or (4) consider whether and how these projects or activities should proceed (Duinker and Greig 2006).

This section has introduced a number of strategies for ongoing management. There is a great deal of overlap between the strategies outlined here and the practice of community relations, community development, and community engagement. The following discussion details a sample of approaches including social management plans, scenario planning, community reference groups, community development and social investments, networking and working groups, and complaints and grievance handling mechanisms.

Social Management Plans

Social management plans (also known as "environmental and social management plans," "social and labor plans" or "environmental and social action plans") summarize the findings of the impact assessment; they outline the measures adopted to enhance positive impacts and to avoid, mitigate, and (as a last resort) offset and compensate negative impacts (Franks et al. 2009b). In addition, the plans provide estimates of the timing, frequency, duration, and cost of management measures. They also establish monitoring and reporting procedures (discussed later in greater depth).

Social management plans are usually developed in partnership with regulatory agencies, investors, and the community; they identify the responsibilities of each party in the management of impacts, opportunities, and risks. Management plans also provide an opportunity to link activities with local and regional planning processes and, if developed with reference to the management plans of other operations, can assist in addressing cumulative impacts. They also provide the facility to coordinate project activities with service and infrastructure planning by government.

The plans may explicitly refer to capacity-building activities, where the institutional or community capacity to undertake such activities is lacking, and may include details of community development and social investments. Finally, social management plans outline the procedures for how social issues will be addressed in site management systems and plans, the processes for ongoing public participation and information disclosure, and the mechanisms for handling community grievances and feedback. Measures that are the responsibility of other parties are recorded and form a basis for ongoing partnerships (Franks et al. 2009b).

Scenario Planning

Scenario planning can assist organizations in preparing for unplanned activities. Scenario analysis is a tool to anticipate change under different plausible future situations. It assists the development of a proactive policy response through the testing of assumptions. If conducted with communities, scenario planning can help to inform the public of risks and manage expectations. For example, the boom-and-bust nature of the industry can increase the risk of premature or temporary closure and downsizing of operations. The effects of mine closures (where impacts are generated by the absence of activities) can be a significant challenge for regional communities and economies. Planning for closure should proceed well before an operation starts, and measures should be put in place to prepare communities and companies for such an eventuality, which will most definitely occur (however large the deposit or valuable its product).

Community Reference Groups

Community engagement is an important component of social impact management. It involves activities such as the communication of the project proposal to stakeholders; the incorporation of stakeholder views in order to modify project details; the ongoing involvement of stakeholders in community boards and reference groups; and stakeholder participation in the submission of ideas for, and implementation of, community projects.

Regardless of the form of engagement, it is important to be upfront and straightforward with stakeholders when communicating the potential impacts of a project. False impressions can distort expectations and become an ongoing point of contention and a breach of trust when the true nature of impacts becomes clear (Franks 2009).

Community reference groups provide a forum for ongoing consultation and engagement. Representation can include groups such as youth and aged organizations, local business,

tourism, health, welfare, policing, and education, in addition to environment, government, and community groups. Broad representation helps to ensure that a range of issues are covered, (although reference groups should be kept to a manageable size). For community reference groups to be at their most effective, there needs to be strong mechanisms for feedback to the broader community to provide an opportunity for input and to report on outcomes.

Community Development and Social Investments

A key strategy adopted by mining companies to manage social impacts involves programs to support community development. These may include health and education programs or support of organizations such as schools, clubs, and societies. It is important to focus and coordinate investments to target community priorities and identified needs. Community development may be prioritized by the outcomes of the scoping, profiling, and predictive assessment phases and, more importantly, through community participation. Partnerships are often the best way to facilitate local capacity-building and development programs, social services, and infrastructure (Kemp 2003, 2010). Partnerships with organizations, service providers, governments, other mining companies, and peak industry bodies can be effective in mobilizing greater resources, leveraging investments, and coordinating activities. Many mining organizations prefer an approach where community-led initiatives are supported by industry. This approach seeks to build the capacity of communities and their organizations to undertake activities and avoid overdependence on mining companies. Examples of community trusts include BHP Billiton's Minera Escondida Foundation in Chile and Rio Tinto's Coal and Allied Community Trust in Australia.

Networking and Multistakeholder Working Groups

Informal and formal networks can provide important opportunities to exchange experiences at the operational and strategic level to better manage the impacts of activities. Professional networking is an opportunity to exchange ideas and advice, as well as to communicate approaches (both successes and failures). Formal networking arrangements, such as forums of mine managers and professional staff, provide an ongoing opportunity to discuss common issues and coordinate activities. Examples include the Central Queensland Mining Rehabilitation Group, which is a forum for sharing experiences about environmental management on mine sites in central Queensland, Australia, and the Muswellbrook Mine Managers forum, which is a regular meeting between the general managers of multiple coal mines and local government in the Hunter Valley, Australia.

Multistakeholder working groups are an opportunity to facilitate partnerships around a particular goal. Working groups can share strategic information, develop and coordinate solutions, undertake research into best practice, and facilitate cross-sector communication. Multistakeholder working groups are well placed to focus on the management of social issues at a regional scale.

Complaints and Grievance Handling Mechanisms

Complaints and grievance handling mechanisms are becoming more common at the operational level within the mining industry as a means to actively respond to community concerns. A grievance is a concern, issue, or problem that is usually expressed through a complaint or protest by individuals or groups. According to Kemp and Gotzman (2009) grievance mechanisms include

- A dedicated pathway (or pathways) and processes of engagement for handling grievances;
- Procedural elements, such as a documented procedure outlining steps to be taken to handle community grievances;
- Records, such as complaints/grievance logs and data, evidence of communication about the process, and documentation of outcomes;
- Dedicated resources, such as human and financial resources, as well as formally defined responsibilities for grievance handling;
- Evidence of dialogue with aggrieved parties and/or use of alternative dispute-resolution techniques (negotiation, mediation, arbitration, etc.) where direct dialogue is not possible or has little potential (through normal channels) of leading to a resolution of issues; and
- Substantive outcomes, such as improved organizational practice and relationships, and conflict resolution (validated by aggrieved parties).

Monitoring and Reporting

The monitoring and reporting phase involves collection, analysis, and dissemination of information on social impacts, opportunities, and risks over time. This phase can (1) assist in refining assessments, (2) track the progress of social impact management approaches and identify changes needed (adaptive management), (3) assess the return that a company is getting on its community investments, (4) report to communities on how they are being impacted, and (5) facilitate an informed dialogue around these issues.

Monitoring

Social impact monitoring can be challenging for many reasons: time lags between actions and outcomes, difficulty in isolating impacts, difficulties in obtaining and comparing data, differences in the way people experience impacts, the risk of information overload for managers, and the complexity of the way impacts interact. Not all impacts can be separated and analyzed independently because they do not exist in isolation of the social and ecological context. It is important, therefore, to have ongoing points of intervention and monitoring.

Monitoring should be undertaken over meaningful time scales and spatial extent. Regional monitoring can help to address the cumulative impacts of multiple actions (Franks et al. 2009a). Monitoring should also be designed to facilitate community participation. The participation of community members can assist many aspects of monitoring, including the collection of data, participation in the development of indicator frameworks, or multistakeholder monitoring organizations. Meaningful participation can assist in building public confidence and trust in the monitoring and resolution process.

Monitoring consists of a number of discrete activities. These include the following:

1. Decide what to monitor. Prioritization can be based on community engagement and the profiling, assessment, and impact management phases. It is important to monitor outcomes, not just effort and activity.
2. Define targets, limits, and thresholds. What are the desired and undesired outcomes? Thresholds refer to

scientifically defined points where undesirable changes result if exceeded. Limits consider what may be acceptable to the community as determined through consultation and participation, with targets being designed as the desired future outcomes.
3. Select indicators. What would indicate that change is occurring or has occurred?
4. Establish measures. What data are preferred, what data are available, how will data be collected, and how often will it be collected?
5. Report and interpret data. Identify and verify trends (or nontrends), interpret trends (attribute change), and communicate results.

The collection of irrelevant information can be avoided through planning and prioritization, and the measurement load can be minimized through the appropriate design of administrative systems and the use of existing data whenever possible. Qualitative data can provide exceptionally useful information if the methodology employed is consistent and robust.

Reporting

Sustainability reporting has become a significant activity for the mining industry to communicate performance at the operational and corporate levels. Reporting consists of the documentation and communication of information on numerous activities and outcomes. In recent years there has been a trend toward standardized reporting requirements, such as the Global Reporting Initiative (GRI). The GRI is a sustainability reporting framework that requires reporting on the economic impacts on project stakeholders and systems; environmental inputs, outputs, and expenditure; labor practices; human rights; and social risks to communities. The GRI has developed, in collaboration with the International Council on Mining and Metals, a mining and metals sector supplement that details specific disclosures and indicators for the industry (GRI 2010). The Extractives Industry Transparency Initiative (EITI) is another such approach. The EITI is a global standard for transparency in oil, gas, and mining that is implemented by both businesses and governments. EITI requires industry to publish all of the payments made to governments, such as through royalties and taxes, and for governments to disclose the revenues they receive from resource developments (EITI 2009).

The documentation of social decisions and agreements (internal and external to the operations) is a key aspect of reporting that is often overlooked. The nondelivery of agreed outcomes is a salient issue at many operations. Nondelivery can create unaccounted-for future liabilities, breach trust with communities, and damage the reputation of the operation.

For some operations and issues, reporting is best addressed at a regional scale. In circumstances where multiple mining operations are located in close proximity to a single town or community, there is often an absence of information that provides a comprehensive overview of industry investments, activities, aggregate impacts, and the state of the environment. Collective reporting to the community on the economic, social, and environmental performance of the industry may be more effective at communicating the overall contribution of the industry and the totality of activities and impacts (Brereton et al. 2008; Franks et al. 2009a). Regional organizations and industry bodies are best placed to coordinate such efforts; however, the absence of a representative organization is not necessarily prohibitive.

Evaluation and Review

The final phase of social impact assessment and management is to evaluate and review the assessment and management processes. An active process of evaluation and review—and importantly, the adjustment of actions—are fundamental features of adaptive management and should be integrated into each of the previous phases. The reconciliation of impacts predicted during the assessment phase with the actual impacts experienced during implementation will assist in refining and improving future approaches. Structured review should be built into ongoing programs, policies, projects, and agreements in order to reflect contemporary conditions.

LESSONS FOR THE EFFECTIVE MANAGEMENT OF SOCIAL IMPACTS

This section summarizes key lessons that can inform mine managers, engineers, and technical professionals to effectively manage the social impacts of mining. It is important to note that communities are complex and, at times, difficult to predict, that local context is paramount, and that the transferability of approaches is often not straightforward. At the same time, consideration of these issues will assist mining professionals to design and implement projects that are more acceptable to communities and therefore have a greater prospect of success.

1. Engage with community stakeholders and build relationships through understanding and goodwill. Be upfront and straightforward about potential risks. Establish meaningful and timely avenues for two-way dialogue, and use this to understand stakeholder histories, relationships, and networks, as well as the values that shape attitudes and behaviors. Assist stakeholders to articulate community concerns and visions. Listen to and design the project or activity within the parameters of such perspectives. Be responsive and adaptive; respect customs and political and authority structures; and, where appropriate, gain informed consent.
2. Align and integrate the outcomes of assessment and monitoring into site management systems and plans. Ensure company-wide understanding and respect of community expectations, concerns, and future visions. SIA and impact management are only effective to the extent that the responsible staff and the outcomes are well integrated into the overall management of the entire operation.
3. Draw on specialist skills and social and economic research to build knowledge and understanding. The use of specialists, however, should not come at the expense of integrating understandings into the organization, nor should it confuse accountability or responsibilities. Relationships with community stakeholders are most effective when stakeholders feel the people they are dealing with have the authority to make decisions and changes.
4. Build the capacity of stakeholders. Tailor community development activities to enhance independence and postmining legacies.
5. Monitor what matters. Design monitoring in such a way that the community's concerns are identified and acted on. Acknowledge and remedy (past) issues and grievances,

and fulfill outstanding commitments. Use appropriate systems to record issues and to coordinate and deliver outcomes.
6. Continuously review activities and programs, and periodically update assessments as "living" documents.
7. Assess and manage social impacts and risks across the life cycle of mining and resource processing activities, including all of the activities from exploration through to postclosure, as well as recycling and waste management. The scale and focus of activities will vary depending on the stage, and local and operational context.

The processes of social impact assessment and impact management detailed in this chapter will assist in identifying key stakeholder issues, predicting and anticipating change, and embedding these understandings into ongoing systems and strategies to proactively respond to the consequences of mineral exploitation. Trust is an important feature of managing social impacts. Mining companies may not originate from the region, or be familiar with the local culture, customs, and lifestyles, but nevertheless they have the power to transform the environment and society. As outsiders, mining companies may be viewed with suspicion by communities and must earn community trust. Even well-meaning actions, when not socially, culturally, or environmentally relevant, may result in poor outcomes. However, by proactively responding to community issues, facilitating meaningful participation, and shaping mutually beneficial futures, mining companies can avoid conflict with communities and the associated costs.

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APPENDIX A

Web Sites Related to Mining

UNITED STATES PROFESSIONAL ORGANIZATIONS

American Coal Ash Association (ACAA): www.acaa-usa.org
American Coal Foundation: www.teachcoal.org
American Geological Institute: www.agiweb.org
American Geophysical Union: www.agu.org
American Institute of Mining, Metallurgical, and Petroleum Engineers (AIME): www.aimehq.org
American Institute of Professional Geologists (AIPG): www.aipg.org
American Rock Mechanics Association (ARMA): www.armarocks.org
American Society for Testing and Materials (ASTM): www.astm.org
Association of American State Geologists (AASG): www.stategeologists.org
Association of Environmental and Engineering Geologists (AEG): www.aegweb.org
China Clay Producers Association (CCPA): www.kaolin.com
Coal Operators and Associates, Inc.: www.miningusa.com/coa
Electric Power Research Institute (EPRI): www.epri.com
Energy and Mineral Law Foundation (EMLF): www.emlf.org
(The) Environmental and Engineering Geophysical Society (EEGS): www.eegs.org
Gems, Rocks and Minerals: www.gemsrocks.com
GEOindex: www.geoindex.com/geoindex
Geo-Institute: www.geoinstitute.org
(The) Geological Society of America (GSA): www.geosociety.org
Illinois Clean Coal Institute (ICCI): www.icci.org
International Center for Aggregates Research (ICAR): www.icar.utexas.edu
International Society for Soil Mechanics and Geotechnical Engineering (ISSMGE): www.issmge.org
International Society of Explosives Engineers (ISEE): www.isee.org
Kentucky Coal Education: www.coaleducation.org
Kentucky Mining Institute (KMI): www.miningusa.com/kmi
Lignite Energy Council: www.lignite.com
Mineral Economics and Management Society: www.minecon.com
Minerals and Geotechnical Logging Society (MGLS): http://mgls.spwla.org
(The) Minerals, Metals and Materials Society (TMS): www.tms.org
Mining Industry Council of Missouri: www.momic.com
Mississippi Valley Trade and Transport Council (MVTTC): www.mvttc.com
National Association of Geoscience Teachers (NAGT): www.nagt.org
National Council of Examiners for Engineering and Surveying (NCEES): www.ncees.org
National Mine Land Reclamation Center: http://wvwri.nrcce.wvu.edu/programs/nmlrc/index.cfm
(The) National Mining Association (NMA): www.nma.org

National Research Center for Coal and Energy (NRCCE): www.nrcce.wvu.edu
National Stone, Sand, and Gravel Association (NSSGA): www.nssga.org
North Carolina Coal Institute (NCCI): www.nccoal.org
Nuclear Energy Institute (NEI): www.nei.org
Rocky Mountain Association of Geologists (RMAG): www.rmag.org
Rocky Mountain Mineral Law Foundation (RMMLF): www.rmmlf.org
Seismological Society of America (SSA): www.seismosoc.org
Society for Mining, Metallurgy, and Exploration, Inc. (SME): www.smenet.org
Society for Sedimentary Geology (SEPM): www.sepm.org
Society of Economic Geologists (SEG): www.segweb.org
Society of Exploration Geophysicists (SEG): www.seg.org
Society of Independent Professional Earth Scientists (SIPES): www.sipes.org
Society of Petroleum Engineers (SPE): www.spe.org
Soil Science Society of America (SSSA): www.soils.org
Solution Mining Research Institute (SMRI): www.solutionmining.org
United States Universities Council on Geotechnical Education and Research (USUCGER): www.usucger.org

UNITED STATES REFERENCE, INFORMATION, AND PUBLICATIONS

Aggregates and Roadbuilding: www.rocktoroad.com
Argus: www.energyargus.com
Chemical Engineering: www.che.com
Coal Age: http://www.coalage.com
Coal Information Network: www.coalinfo.org
Dredging News Online: www.sandandgravel.com
Earth: www.earthmagazine.org
Electronic Journal of Geotechnical Engineering: www.ejge.com/Index_ejge.htm
Energy Market Report: www.econ.com/emrindex.html
Engineering and Mining Journal (E&MJ): www.e-mj.com
Engineering News-Record (ENR): www.enr.com
GoldAndSilverMines.com: www.goldandsilvermines.com
Goldsheet Mining Directory: www.goldsheetlinks.com
ICMJ's Prospecting and Mining Journal: www.icmj.com
InfoMine: www.infomine.com
Journal of Metals (JOM): www.tms.org/jom.html
Metal World: www.metalworld.com
Mine-On-Line: www.mine-on-line.com
Mineral Information Institute: www.mii.org
Miners News: www.minersnews.com
Minesite.*com:* www.minesite.com
Mining Engineering: http://me.smenet.org
Mining.com: www.mining.com
(The) Mining Record: www.miningrecord.com
(The) Northern Miner: www.northernminer.com
Pit and Quarry: www.pitandquarry.com
Platts: www.platts.com
Rock and Dirt: www.rockanddirt.com
Skillings Mining Review: www.skillings.net
World-Wide Web Virtual Library of Geotechnical Engineering: www.ejge.com/GVL

INTERNATIONAL MINING ECONOMICS

AME Mineral Economics (Australia): www.ame.com.au
American Metal Market: www.amm.com
Bloomberg: www.bloomberg.com
Bloomsbury Minerals Economics (United Kingdom): www.bloomsburyminerals.com
Chicago Stock Exchange: www.chx.com

CME Group: www.cmegroup.com
Coal Trading Association: www.coaltrade.org
Commodity Research Bureau: www.crbtrader.com
Denver Gold Group: www.denvergold.org
Dow Jones: www.dowjones.com
Financial Times Energy (United Kingdom): www.ftenergy.com.companies/energy
London Metal Exchange (United Kingdom): www.lme.co.uk
Metals Economics Group (United Kingdom): www.metalseconomics.com
MineMarket.com: www.minemarket.com
NASDAQ: www.nasdaq.com
New York Mercantile Exchange (NYMEX/COMEX): www.nymex.com/index.aspx
New York Stock Exchange (NYSE Euronext): www.nyse.com
OTC Bulletin Board: www.otcbb.com
Reuters: www.reuters.com
Standard and Poor's (S&P): www.standardandpoors.com/home/en/us
Wall Street Journal: www.wsj.com
World Mine Cost Data Exchange, Inc.: www.minecost.com

MINING COMPUTING ORGANIZATIONS

Engineering Software Center: www.engsoftwarecenter.com
Geotechnical and Geoenvironmental Software Directory: www.ggsd.com
Macintosh Geological Software: http://geowww.geo.tcu.edu/faculty/geosoftware.html
Mining Internet Services, Inc.: www.miningusa.com

UNITED STATES GOVERNMENT

Federal Agencies

Army Corps of Engineers (ACE): www.usace.army.mil/pages/default.aspx
Bureau of Land Management (BLM): www.blm.gov/wo/st/en.html
Bureau of Reclamation: www.usbr.gov
Commodity Futures Trading Commission: www.cftc.gov
Council on Environmental Quality (CEQ): www.whitehouse.gov/administration/eop/ceq
Department of Commerce: www.commerce.gov
Department of Energy (DOE): www.energy.gov
Department of the Interior (DOI): www.doi.gov
Energy Information Administration (EIA): www.eia.doe.gov
Environmental Protection Agency (EPA): www.epa.gov
Fossil Energy: www.fe.doe.gov
Geological Survey (USGS): www.usgs.gov
Mine Safety and Health Administration (MSHA): www.msha.gov
Minerals Information: http://minerals.usgs.gov/minerals
Minerals Management Service (MMS): www.mms.gov
National Geophysical Data Center: www.ngdc.noaa.gov
National Institute for Occupational Safety and Health (NIOSH): www.cdc.gov/niosh/mining
National Science Foundation (NSF): www.nsf.gov
National Technical Information Service (NTIS): www.ntis.gov
Nuclear Regulatory Commission (NRC): www.nrc.gov
Office of Surface Mining (OSM): www.osmre.gov
State Minerals Statistics and Information: http://minerals.usgs.gov/minerals/pubs/state

Mining Agencies and Associations (by State)

Alabama Surface Mining Commission: www.surface-mining.state.al.us
Alaska Department of Natural Resources, Division of Mining, Land and Water: http://dnr.alaska.gov/mlw
Alaska Miners Association: www.alaskaminers.org
Arizona Department of Mines and Mineral Resources: www.admmr.state.az.us

Arizona State Mine Inspector: www.asmi.state.az.us
Arkansas Department of Environmental Quality: www.adeq.state.ar.us/mining
California Mining Association: www.calmining.org
California Office of Mine Reclamation: www.conservation.ca.gov/omr
California State Mining and Geology Board: www.conservation.ca.gov/smgb
Colorado Division of Reclamation Mining and Safety: http://mining.state.co.us
Colorado Mining Association: www.coloradomining.org
Connecticut Department of Environmental Protection: www.ct.gov/dep/sit/default.asp
Delaware Geologic Survey: www.dgs.udel.edu
Florida Mining and Minerals Regulation: www.dep.state.fl.us/water/mines/index.htm
Georgia Environmental Protection Division: www.gaepd.org
Hawaii Department of Lands and Natural Resources: http://hawaii.gov/dlnr/land
Idaho Department of Lands: www.idl.idaho.gov
Idaho Mining Association: www.idahomining.org/ima
Illinois Office of Mines and Minerals: http://dnr.state.il.us/mines
Indiana Department of Natural Resources, Division of Reclamation: www.in.gov/dnr/reclamation
Iowa Soil Conservation Division: www.agriculture.state.ia.us/soilConservation.asp
Kansas Department of Health and Environment: www.kdheks.gov/mining
Kentucky Coal Association: www.kentuckycoal.org
Kentucky Division of Mine Reclamation and Enforcement: www.dmre.ky.gov
Kentucky Office of Mine Safety and Licensing: www.omsl.ky.gov
Louisiana Office of Conservation, Surface Mining Section: http://dnr.louisiana.gov/CONS/CONSERIN/Surfmine.ssi
Maine Bureau of Land and Water Quality: www.maine.gov/dep/blwq
Maryland Bureau of Mines: www.mde.state.md.us/Programs/WaterPrograms/MiningInMaryland/BOM
Massachusetts Department of Conservation and Recreation: www.mass.gov/dcr
Michigan Department of Natural Resources: www.michigan.gov/dnr
Minnesota Division of Lands and Minerals: www.dnr.state.mn.us/lands-minerals/index.html
(Minnesota) Iron Mining Association of Minnesota: www.taconite.org
Mississippi Mining and Reclamation Division: www.deq.state.ms.us/mdeq.nsf/page/geology-mining-and-reclamation
Missouri Department of Natural Resources: www.dnr.mo.gov
Montana Department of Environmental Quality: www.deq.state.mt.us
Nebraska School of Natural Resources, Conservation and Survey Division: http://snr.unl.edu/csd/
Nevada Bureau of Mines and Geology: www.nbmg.unr.edu
Nevada Division of Minerals: http://minerals.state.nv.us
Nevada Mining Association: www.nevadamining.org
New Hampshire Department of Environmental Services: http://des.nh.gv
New Jersey Geological Survey: www.state.nj.us/dep/njgs
New Mexico Bureau of Mine Safety: www.bmi.state.nm.us
New Mexico Mining and Minerals Division: www.emnrd.state.nm.us/mmd
New York Division of Mineral Resources: www.dec.ny.gov/about/636.html
North Carolina Division of Land Resources: www.dlr.enr.state.nc.us
North Dakota Geological Survey: www.dmr.nd.gov/ndgs
North Dakota Public Service Commission, Coal Mine Reclamation: www.psc.state.nd.us/jurisdiction/reclamation.html
North Dakota State Land Department: www.land.nd.gov
Northwest Mining Association: www.nwma.org
Ohio Aggregates and Industrial Minerals Association: www.oaima.org
Ohio Coal Association: www.ohiocoal.com
Ohio Division of Mineral Resources Management: www.dnr.state.oh.us/mineral/default/tabid/10352/Default.aspx
Oklahoma Department of Mines: www.ok.gov/mines
Oregon Department of Geology and Mineral Industries: www.oregongeology.org/sub/default.htm

Pennsylvania Office of Mineral Resources Management: www.dep.state.pa.us/dep/deputate/minres/minres.htm
Rhode Island Bureau of Natural Resources: www.dem.ri.gov/programs/bnatres
South Carolina Geological Survey: www.dnr.sc.gov/geology
South Dakota Minerals and Mining Program: http://denr.sd.gov/des/mm/mmprogram.aspx
Tennessee Department of Environment and Conservation: www.state.tn.us/environment
Texas Bureau of Economic Geology: www.beg.utexas.edu
(Texas) Railroad Commission of Texas: www.rrc.state.tx.us
Utah Department of Natural Resources: http://naturalresources.utah.gov
Utah Division of Oil, Gas, and Mining: http://ogm.utah.gov
Utah Mining Association: www.utahmining.org
Vermont Agency of Natural Resources: www.anr.state.vt.us
Virginia Department of Mines, Minerals and Energy: www.mme.state.va.us
Washington Geology and Earth Resources Division: www.dnr.wa.gov/ResearchScience/GeologyEarthSciences
West Virginia Coal Association: www.wvcoal.com
West Virginia Department of Environmental Protection: www.wvdep.org
West Virginia Office of Miners' Health Safety and Training: www.state.wv.us/mhst
Wisconsin Waste and Materials Management: www.dnr.state.wi.us.org/aw/wm
Wyoming Department of Environmental Quality: http://deq.state.wy.us
Wyoming Mining Association: www.wma-minelife.com

INTERNATIONAL PUBLICATIONS AND ASSOCIATIONS

Africa

Chamber of Mines of South Africa: www.bullion.org.za
MBendi (South Africa): www.mbendi.com

Asia

Asian Journal of Mining: www.asianmining.com
Chamber of Mining Engineers of Turkey: www.maden.org.tr/index_en.php
China Coal Information Network: www.coalinfo.net.cnChina Coal Research Institute, Xi'an Branch: http://xianccri.en.alibaba.com/
Japan Coal Energy Center: www.jcoal.or.jp/index-en.html
Mineral Resources Authority of Mongolia: www.mram.gov.mn/en/index.php
Mining India: www.miningindia.com

Australia and New Zealand

Association of Mining and Exploration Companies: www.amec.asn.au
Australasian Institute of Mining and Metallurgy (AusIMM): www.ausimm.com.au
Australian Centre for Mining Environmental Research (ACMER): www.acmer.uq.edu.au
Australian Coal Association Research Program: www.acarp.com.au
Australian Institute of Geoscientists: http://aig.org.au
Australian Journal of Mining: www.theajmonline.com.au
Australian Mines and Metals Association.: www.amma.org.au
Chamber of Mines and Energy of Western Australia, Inc.: www.cmewa.com
New South Wales Minerals Council: www.nswmin.com.au
New Zealand Minerals Industry Association: www.minerals.co.nz/html/index.html
OzGold Database Technology: www.comcen.com.au/~ozgold/main.html
Queensland Resources Council: www.qrc.org.au/default.asp

Canada

Association for Mineral Exploration British Columbia: www.amebc.ca/Home.aspx
CAMESE (Canadian Association of Mining Equipment and Services for Export): www.camese.org
Canadian Diamond Drilling Association: www.canadiandrilling.com
Canadian Institute of Mining, Metallurgy and Petroleum (CIM): www.cim.org/splash

Canadian Mining Industry Research Organization (CAMIRO): www.camiro.org
Canadian Mining Journal: www.canadianminingjournal.com
Coal Association of Canada: www.coal.ca/content
International Council on Mining and Metals: www.icmm.com
Mining Association of Canada: www.mining.ca
Natural Resources Canada: www.nrcan-rncan.gc.ca.com
NWT and Nunavut Chamber of Mines: www.miningnorth.com
Ontario Mining Association: www.oma.on.ca/en
Ontario Ministry of Northern Development, Mines and Forestry: www.mndm.gov.on.ca
Prospectors and Developers Association of Canada: www.pdac.ca
Yukon Chamber of Mines: www.ycmines.ca

Europe

Coal Services International (Ireland): www.coalservices.com
EuSalt (European Salt Producers' Association) (France): www.eusalt.com
German Brown Coal Association: www.braunkohle.de
German Coal Importers Association: www.verein-kohlenimporteure.de
GInfoServer: www.geo.uni-bonn.de/members/haack/gisinfo.html
IEA Coal Industry Advisory Board (France): www.iea.org/ciab
International Energy Agency (IEA): www.iea.org
International Organization for Standardization (ISO)(Switzerland): www.iso.org
International Society for Rock Mechanics (Portugal): www.isrm.net

Latin America

Editec (Chile): www.editec.cl
La Camara de Mineria del Ecuador: www.cme.org.ec/portal
National Society of Mining Petroleum and Energy (Peru): www.snmpe.org.pe
Panorama Minero (Argentina): www.panoramaminero.com.ar

United Kingdom

Association of Geotechnical and Geoenvironmental Specialists: www.ags.org.uk/site/home/index.cfm
British Geological Survey: www.bgs.ac.uk
(The) Coal Authority: www.coal.gov.uk
Coal International: www.quarryworld.co.uk/coalinternational.php
(The) Geological Society: www.geolsoc.org.uk/index.html
Geopages: www.geopages.co.uk/about/aboutus.html
globalCOAL: www.globalcoal.com
IEA Clean Coal Centre: www.iea-coal.org.uk
Industrial Minerals: www.mineralnet.co.uk
Institute of Materials, Minerals and Mining: www.iom3.org
Institute of Quarrying: www.inst-of-quarrying.org/iq
International Association of Hydrogeologists: www.iah.org
International Mining: www.im-mining.com
(The) McCloskey Group: www.mccloskeycoal.com
Minerals and Energy—Raw Materials Report (Sweden): www.tandf.co.uk/journals/SMIN
Mining and Quarry World: www.quarryworld.co.uk/miningquarryworld.php
Mining Journal Online: www.mining-journal.com
Mining Magazine: www.miningmagazine.com
World Coal: www.energyglobal.com/magazine/latestissue/world-coal.aspx
World Coal Institute: www.wci-coal.com
World Mining Equipment: www.wme.com
World Nuclear Association: www.world-nuclear.org

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